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Technical Report

Prairie Creek Property Feasibility Study NI 43-101 Technical Report
Canadian Zinc Corporation

NORTHWEST TERRITORIES, CANADA

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

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AMC Project 716057

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1 Summary

Figure 1.1 The Prairie Creek Mine Site

Executive summary

Introduction

This Technical Report on the Prairie Creek Property, NWT, Canada (the Property), has been prepared by AMC Mining Consultants (Canada), Ltd. (AMC) of Vancouver, Canada, in conjunction with Ausenco Engineering Inc. (Ausenco), Vancouver, with contributions by Global Mineral Resource Services Ltd., Allnorth Consultants Limited and F. Wright Consulting Inc., on behalf of Canadian Zinc Corporation (CZN) of Vancouver, Canada in accordance with the requirements of National Instrument 43-101 (NI 43-101) *"Standards of Disclosure for Mineral Projects",* of the Canadian Securities Administrators (CSA) for Filing on CSA's "System for Electronic Document Analysis and Retrieval" (SEDAR).

This report discloses the results of a Feasibility Study ("2017 FS") based on the 2015 Mineral Resources, updated Mineral Reserves, updated metallurgy test work, ongoing optimization and other engineering studies completed since the Prefeasibility Study 2016 ("2016 PFS") AMC Report dated 30 September 2016.

Economic summary

This Feasibility Study indicates a base case Pre-Tax Net Present Value ("NPV") of $344M using an 8% discount rate, with an Internal Rate of Return ("IRR") of 23.8%, and a post-tax NPV of $188M with an IRR of 18.4%. Corresponding pre-tax and post-tax payback periods from mill start-up are 4.4 and 4.6 years respectively. The Base Case metal price assumptions used in the model are: Zn US$1.10/lb., Pb US$1.00/lb., Ag US$19.00/oz., with a foreign exchange rate of C$1.25=US$1.00.

The pre-tax and post-tax net present values and internal rates of return, at 5% and 8% discount rates, are illustrated in the table below at a Canadian / US dollar exchange rate of C$1.25=US$1.00, except where noted. The table also illustrates the sensitivities of the Prairie Creek Project to zinc, lead and silver prices and to the Canadian / US dollar exchange rate.

Table ES.1.1 Economic sensitivities of the Prairie Creek Project

Metal prices		Pre-tax				Post-tax [1]			
Zinc / lead US$/lb	**Silver US$/oz**	**Undiscounted $M**	**NPV (5%) $M**	**NPV (8%) $M**	**IRR %**	**Undiscounted $M**	**NPV (5%) $M**	**NPV (8%) $M**	**IRR %**
0.80	17.00	139	10	(39)	5.5	75	(29)	(68)	3.3
0.90	18.00	452	211	120	14.4	282	109	43	10.6
1.10/1.00	**19.00**	**899**	**497**	**344**	**23.8**	**562**	**291**	**188**	**18.4**
1.20/1.00	19.00	1,033	582	410	26.2	644	344	230	20.4
1.10	20.00	1,077	614	437	27.3	671	364	247	21.3
1.20	21.00	1,390	815	596	32.7	863	489	346	25.7
1.30	22.00	1,703	1,017	755	37.7	1,053	612	444	29.8
1.10/1.00 [2]	**19.00 [2]**	**1,208**	**696**	**501**	**29.5**	**752**	**416**	**287**	**23.1**
1.20/1.00 [2]	**19.00 [2]**	**1,355**	**789**	**574**	**31.9**	**842**	**473**	**332**	**25.0**
1.10/1.00 [3]	**19.00 [3]**	**589**	**298**	**188**	**17.4**	**371**	**166**	**88**	**13.2**

1. Post-tax results include all taxes, royalties, aboriginal participation costs and the Sandstorm 1.2% NSR.
2. Foreign exchange assumed to be C$1.375:US$1.00 on these rows.
3. Foreign exchange assumed to be C$1.125:US$1.00 on this row.

Using the base case metal prices and exchange rate of C$1.375 = US$1.00 would increase the pre-tax $NPV_{8\%}$ to $500M and the IRR to 29.5% relative to the base case. Using a zinc price of US$1.20 per lb., with all other base case inputs and a foreign exchange rate of C$1.25 = US$1.00, the pre-tax $NPV_{8\%}$ would be $410M with an IRR of 26.2%. Using a zinc price of US$1.20 per lb., with all other base case inputs and a foreign exchange rate of C$1.375 = US$1.00 would increase the pre-tax $NPV_{8\%}$ to $574M and the IRR to 31.9% relative to the base case. Using the base case metal prices and exchange rate of C$1.125 = US$1.00, the pre-tax $NPV_{8\%}$ would be $188M with an IRR of 17.4%, and the post-tax $NPV_{8\%}$ would be $88M with an IRR of 13.2%

During the first 10 years of concentrate production the 2017 FS indicates average annual production of approximately 65,000 tonnes of zinc concentrate and 72,000 tonnes of lead concentrate, containing an average of approximately 95 million pounds of zinc, 105 million pounds of lead and 2.1 million ounces of silver.

The 2017 FS indicates average annual earnings before interest, taxes, depreciation and amortization ("EBITDA") during the first 10 full years of production as $111M per year, and cumulative EBITDA of $1,294M over the projected LOM of 15 years, using base case metal prices.

Location, ownership, and history

The Property consists of two surface leases and 12 mining leases totalling 7,487 hectares in area. The Property is situated in the Northwest Territories approximately 500 km west of Yellowknife in the Mackenzie Mountains at an elevation of 850 m above mean sea level. The Property is surrounded by, but is not included in, the Nahanni National Park Reserve (NNPR).

Year-round access to the Property, at this time, is provided by aircraft utilizing a 1,000 m gravel airstrip immediately adjacent to the camp. The Property has also, in the past, been accessible by a winter road that extended 180 km from the Property to the Liard Highway 7; most of this access road route is now planned to be all season and will be constructed to support full-time operation of the mine.

The Prairie Creek Property contains a high-grade, silver-lead-zinc-copper vein, and other lead-zinc deposit types that have been explored since the early 1900s and were developed by Cadillac Explorations Limited (Cadillac) from 1966 to 1983. The Cadillac Mine was targeting silver production and was developed and fully permitted. A processing plant, along with other surface infrastructure, was built in the early 1980s. A sudden decline in metal prices resulted in the closure of the Mine in 1983 prior to commencement of production. San Andreas Resources Corporation exercised its option on the Property in the 1990s and, through a series of agreements, together with a name change to Canadian Zinc Corporation in 1999, established an increasing interest in the Property, culminating with the acquisition of a 100% interest in the Property and mine site in 2004, now referred to as the Prairie Creek Mine.

Geology and mineralization

The Property 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 Property, north-south trending faulting and folding is apparent. The most significant fold structure is the fault-bounded, north-south doubly-plunging Prairie Creek anticlinal structure, which is the host to the Prairie Creek mineralization.

Four styles of base metal mineralization have been identified on the Property: 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 underground workings have been developed, containing the bulk of the currently defined Mineral Resource. The Main Quartz Vein (MQV) has been exposed in detail by underground development and diamond drilling over a strike length of 2.1 km (Main Zone). The MQV trends at an azimuth of approximately 20º and dips between vertical and 40º east, with an average dip of 65º. 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 located, through surface trenching, throughout the entire 16 km length of the mining leases.

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

Stratabound Massive Sulphide (SMS) mineralization occurs intermittently at the base of the trend of the Prairie Creek vein system over a strike length of more than 3 km. SMS mineralization occurs as semi-massive sphalerite-galena-pyrite replacement located close to both the vein system and the axis of the Prairie Creek 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.

Mississippi Valley-type (MVT) lead-zinc mineralization is exposed on the Property within surface showings of rock formations 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. This type of mineralization does not form part of the current resource.

Exploration and data management

CZN, including its former entity as San Andreas Resources Corporation, has been involved with mineral exploration activity across the Prairie Creek Property since 1992. Somewhat limited exploration drilling had occurred and most of the existing underground development had been undertaken prior to CZN's initial involvement. From 1992 to the end of 2015, CZN completed 296 surface and underground exploration diamond drillholes with an aggregate length of 78,587 m. In addition, 1,032 underground channel samples forming 365 composites from the three existing underground levels have been collected and analysed.

The main exploration and underground development work has been focused on the Main Zone mineralization, where approximately 80% of the total drilling has been carried out.

Mineral Resource estimate

The most recent Mineral Resource estimation was undertaken by AMC and announced in a press release dated 17 September 2015. It followed completion of the successful 2015 underground exploration program at Prairie Creek and resulted in an increase in Measured and Indicated Mineral Resource tonnages of 32%. Upon further review of the Mineral Resource, and since there were no changes to the database within the resource area, the same Mineral Resource Estimate was used as the basis for the 2017 Feasibility Study.

A single block model was created to encompass the three mineral domains: MQV, STK, and SMS. The summary results of the Mineral Resource estimate for the three zones combined, at a cut-off of 8% Zn Equivalent (ZnEq), are shown below.

Table ES.1.2 September 2015 Mineral Resources Prairie Creek Mine

Mineral zone	Classification	Tonnes (t)	Silver (g/t)	Lead (%)	Zinc (%)
Main quartz vein (MQV)	Measured	1,313,000	211	11.5	13.2
	Indicated	4,227,000	168	11.6	9.2
	Measured & Indicated	5,540,000	178	11.6	10.2
	Inferred	5,269,000	199	8.7	12.9
Stockwork (STK)	Measured	169,000	116	5.3	12.6
	Indicated	1,953,000	61	3.5	6.6
	Measured & Indicated	2,122,000	66	3.6	7.1
	Inferred	1,610,000	70	4.6	6.2
Stratabound (SMS)	Indicated	1,042,000	54	5.2	10.8
	Measured & Indicated	1,042,000	54	5.2	10.8
	Inferred	170,000	60	6.3	11.2
Total	Measured	1,482,000	200	10.8	13.2
	Indicated	7,222,000	123	8.5	8.7
	Measured & Indicated	8,704,000	136	8.9	9.5
	Inferred	7,049,000	166	7.7	11.3

Mineral Resources are stated as of 10 September 2015.
Mineral Resources include those Resources converted to Mineral Reserves.
Stated at a cut-off grade of 8% ZnEq based on prices of US$1.00/lb for both zinc and lead and US$20/oz for silver.
Average processing recovery factors of 78% for zinc, 89% for lead, and 93% for silver.
Average payables of 85% for zinc, 95% for lead, and 81% for silver.
ZnEq = (grade of Zn in %) + [(grade of lead in % * price of lead in US$/lb * 22.046 * recovery of lead in % * payable lead in %) + (grade of silver in g/t * (price of silver in US$/Troy oz/ 31.10348) * recovery of silver in % * payable silver in %)] / (price of zinc in US$/lb*22.046 * recovery of zinc in % * payable zinc in %).
$ Exchange rate = 1 C/US.
Numbers may not compute exactly due to rounding.

The September 2015 Prairie Creek Mine Mineral Resource estimate was completed by Gregory Z. Mosher, P.Geo, Qualified Person (QP), as defined by National Instrument 43-101 (NI 43-101), of Global Mineral Resource Services Ltd.

Mineral Reserve estimate

The 2017 FS has a new Mineral Reserve estimate of 8.1 million tonnes of Proven and Probable Reserves at a combined grade of 16.75% Pb and Zn plus 124 g/t Ag, which represents a 6% increase in Mineral Reserve tonnage compared to the 2016 PFS.

The increase is due to marginally lower Zinc Equivalent cut-off grades, reflecting the final 2016 PFS operating cost estimate, a small increase in projected Zn prices and further optimization of the stoping design. The 2017 Mineral Reserves have slightly lower average metal grades than those estimated in the 2016 PFS, but increased overall metal content. The estimation of Mineral Reserves by AMC is shown in Table ES.1.3.

Table ES.1.3 August 2017 Mineral Reserves Prairie Creek Mine

Mineral zone	Classification	Tonnes (t)	Silver (g/t)	Lead (%)	Zinc (%)	ZnEq (%)
Main quartz vein (MQV)	Proven	1,524,171	161.43	8.90	10.22	26.84
	Probable	4,190,187	144.76	9.96	8.20	25.70
	Total	**5,714,358**	**149.21**	**9.67**	**8.74**	**26.00**
Stockwork (STK)	Proven	188,173	108.19	4.84	11.56	21.22
	Probable	1,188,366	63.81	3.54	6.86	13.46
	Total	**1,376,539**	**69.88**	**3.72**	**7.50**	**14.52**
Stratabound (SMS)	Proven	-	-	-	-	-
	Probable	980,566	54.90	5.06	9.64	17.97
	Total	**980,566**	**54.90**	**5.06**	**9.64**	**17.97**
Total	Proven	1,712,344	155.58	8.45	10.36	26.22
	Probable	6,359,119	115.78	8.00	8.17	22.22
	Total	**8,071,463**	**124.22**	**8.10**	**8.64**	**23.07**

The Mineral Reserves are as of 2 August 2017, and based on a design cut-off grade of 11% ZnEq for longhole open stoping ("LHOS"), 11% ZnEq for mechanized drift-and-fill ("DAF"), an incremental stoping cut-off grade of 10% ZnEq, and 6% ZnEq cut-off grade for development ore. Cut-off grades are based on a zinc metal price of US$1.00/lb, recovery of 75% and payable of 85%; a lead metal price of US$1.00/lb, recovery of 88% and payable of 95%; and a silver metal price of US$18/oz, recovery of 92% and payable of 81%. Exchange rate used is C$1.25= US$1.00. Average planned dilution, unplanned dilution and mining recovery factors of 13%, 11% and 95%, respectively, for LHOS; and 18%, 6% and 98%, respectively, for DAF are assumed.

The August 2017 Prairie Creek Mineral Reserve estimate was prepared by H. A. Smith, P.Eng., Qualified Person ("QP"), as defined by National Instrument 43-101 ("NI 43-101") of AMC Mining Consultants (Canada) Ltd.

These Mineral Reserves are based upon a Measured and Indicated Resource of 8.7 million tonnes grading 9.5% Zn; 8.9% Pb and 136 g/t Ag, and represent an initial mine life of 15 years.

Prairie Creek also hosts an additional Inferred Mineral Resource of 7.0 million tonnes grading 11.3% Zn, 7.7% Pb, and 166 g/t Ag, which has the potential, through further exploration and development, to be upgraded to Measured or Indicated Mineral Resources and increase the initial 15 year mine life.

Mining

The mine will be an underground operation, based primarily on the MQV and mining an average of 1,600 tonnes per day at steady state, over a 15-year mine life (16 years including development prior to mill start-up). During full production, approximately 584,000 tonnes of ore per year will be mined.

Adits were previously driven on three levels: the 970 mL, the 930 mL, and the 883 mL, totalling approximately 5 km of underground workings. Access for mining will be through an enlarged 883 mL portal and adit, with secondary access through the 930 mL. The 970 mL penetrates the topmost limits of the MQV only and is not part of the current mine plan. As mining on the MQV progresses to depth, ore mined will be supplemented by ore from the STK and SMS deposit zones.

Mining in the MQV and STK zones will be by longhole open stoping (LHOS) with paste backfill. Mechanized drift-and-fill (DAF) will be used for the SMS ore, also with paste fill. The plan and objective is to use 100% of flotation tailings as backfill.

Ground conditions in existing development underground are generally good and the existing workings have stood unsupported for over thirty years with minimal bolting. CZN commissioned a geotechnical program at the end of 2013, including mapping and examination of drill core. This program and subsequent assessment in both PFS and FS studies indicated that the ground is amenable to longhole open stoping, with the results of the assessment being used for rock support design.

The 2017 FS mine plan envisages slashing out of some of the existing development and establishing two spiral ramps to access deeper levels. Major goals in development and production sequencing are to access higher grade sulphide ore as early as practicable, while minimizing development costs as much as possible.

The 2016 PFS targeted both the higher grade oxide above the 883 mL and the higher grade sulphide below this horizon, which in turn had lower recoveries. The new mine plan increases the mining rate to 1,600 tonnes per day, versus 1,350 tonnes per day in the 2016 PFS. The existing 883 mL adit will be enlarged to 5 x 5 m and access to the ore below the 883L will be via the twin ramps. A single ramp will provide access to the ore above 883 mL.

Priority will be given to ramp development to establish dewatering sumps in advance of mining. Ore drifts will be driven on the MQV north and south from the ramp access points to the strike limits of the ore body. Stoping will begin at the ore limits and retreat to the ramp access points. Pre-production development is anticipated for approximately 15 months prior to mill start-up. This work will be performed by a contractor. On completion of the contracted scope of work, CZN will have the option of taking over the work itself or continuing with contract mining.

Managing groundwater will be a key aspect of the operation. At peak levels, it is estimated that the mine will produce up to 200 L/s of water, but with the majority of this water to be collected through advanced dewatering boreholes and pumped to surface, and avoiding any contamination from mine workings. All water discharged from the mine will either be sent to the mill as process water, pumped into the existing impoundment pond that was originally planned for tailings storage and which will now be modified into a two-cell water storage pond, or directly treated in a new water treatment plant.

CZN anticipates that, because of the high concentrate mass pull, even with 100% disposal of tailings underground as paste fill, some shortfall in backfill volume will occur. Any shortfall will be made up with Dense Media Separation (DMS) float material or waste rock. When no stopes are available for backfill, filtered tailings will be stored in an active tailings building or in an adjacent passive stockpile. Development waste and DMS float material will be stored in a newly created waste rock pile north of the plant site away from the Prairie Creek floodplain.

The mine will be ventilated by an exhausting type ventilation system. The primary exhaust fans will be located on 930 mL, one adjacent to the 930 portal and the other at the base of the existing raise to surface. Fresh air will be drawn entirely through the 883 portal where a duplex propane and liquified natural gas (LNG) fired mine air heating system will be installed to heat the air during the winter months. The planned airflow through the mine is 142 m^3/s, with fresh air being distributed through the ramps and exhausting to internal return air raises feeding up through to the 930 mL exhaust fans. For level ventilation, fresh air will be delivered along each ore drive by auxiliary fan and duct installations. The exhaust air raises will be fitted with ladderways to serve as the second means of egress through to the 930 mL, where egress will be through the 930 mL portal.

Metallurgy and processing

Metallurgical tests conducted to date on MQV and SMS material have proved positive, as have initial metallurgical tests on STK material. Reasonably good metal recoveries have been achieved with both sulphide and oxide material with a cyanide-free reagent suite. A new metallurgical testing program was completed in 2017, focusing on MQV material, which had a lower oxide component than historical samples, and demonstrated improved recoveries and metallurgical performance based on a simplified process flow sheet, part of which is incorporated into this study.

According to the test results on MQV composited material only, the overall average grade of the blended lead sulphide / oxide concentrate is anticipated to be 65% lead, with an approximate 90% average recovery of lead in the plant feed. The zinc sulphide concentrate is estimated to be 59% zinc, with an approximate 90% recovery of zinc in the plant feed. An average of 86% of the total silver values in the plant feed is estimated to be recovered within the lead and zinc concentrates. Impurities of antimony, arsenic and mercury are expected to report to both concentrates.

A processing plant / concentrator was substantially constructed prior to project shutdown in 1982, together with a 1.5 million tonne capacity tailings impoundment, power plant, and water treatment plant. CZN plans to rehabilitate and upgrade the processing plant and site infrastructure.

The current crushing facilities have a 1,750 tpd capacity, with an installed jaw crusher, short-head cone crusher, double-decked screen, and conveyor systems feeding an 1,800 t fine ore bin.

A new dense media separation (DMS) plant, with a nominal feed rate of 1,600 tpd, will be installed downstream of the crushing circuit and is expected to reject an average of approximately 25% of the feed as waste with minimal metal losses. The milling circuit is designed for 1,200 tpd at nominally 80% passing 156 µm. The ROM ore is expected to be softer initially and then harder as the mine develops deeper and an increase in the Bond Work Index is anticipated. As such, a new secondary ball mill (200 kW tyre mill) is planned for installation, costed as sustaining capital after 5 years of operation. Precise timing of this additional milling power will need to be optimized based on the work index progression over time and other economic factors.

The ground material will be subjected to three stages of sequential flotation: lead sulphide flotation followed by zinc sulphide flotation and lead oxide flotation. Each flotation circuit will consist of rougher flotation and multiple-stage cleaner flotation to upgrade the rougher concentrates to marketable grades. The existing regrind ball mill will be refurbished and utilized to further grind the lead sulphide rougher flotation concentrate in order to maximize grade of the final lead sulphide concentrate. The rougher tailings from the lead oxide flotation plant will be discharged as final tailings to the tailings thickener (new equipment) before being pumped to the paste plant. The lead sulphide and lead oxide flotation concentrates will be pumped to the lead dewatering system, while the zinc sulphide flotation concentrate will be sent to the zinc dewatering system. Both dewatering systems will consist of conventional thickening and pressure filtration circuits. The dewatered concentrates will be temporarily stored in the on-site storage facility prior to being loaded for transport to off-site smelters.

The process plant will require new equipment including modernization of the electrical system, addition of a thickener, new flotation cells to complement the existing cells, a concentrate storage and loadout facility and an on-stream analyzer and control system. The new DMS circuit will be added to the north side of the mill and a reagent mixing area and concentrate storage and loadout facility will be added to the south side of the mill building. A new lead oxide circuit will be added to the eastern side of the mill building and will also include space to store reagents. A new paste backfill plant is proposed to be built to the south of the mill building along with an active tailings storage facility.

Site infrastructure

In 1982, the mine was fully permitted and construction almost complete, but never achieved production. The existing site infrastructure is substantial and these facilities will be utilized and upgraded as necessary. This includes upgrading the mill building, administration building, workshops, sewage treatment plant, diesel storage tank farm, warehouses and part of the accommodation facilities. New facilities needed for operations will include the DMS plant, a paste backfill plant, tailings stockpile shed, LNG facility, water treatment plant, lead-oxide building, heated warehouse and concentrate load-out facility.

Four new 2.77 MW dual-fuel powered low-speed power generator units will provide power and heat for the site. These power generator units will be located within the existing mill powerhouse after removal of the obsolete units currently in place, to provide the installed power of 11.1 MW with an expected running load of 6.7 MW. The energy source for the power generation will be provided by a combination of Liquified Natural Gas (LNG) from the newly installed site LNG storage / vaporization facility and diesel fuel from the existing diesel storage tank farm adjacent to the mill. The FS incorporates a turn-key type power-by-the-hour operation proposal received from the Northwest Territories Power Corporation. The new generators will be outfitted with glycol heat recovery systems in order to maximize energy efficiency. The waste heat from the generators will be used to heat the surface facilities.

Tailings from the mill will be placed permanently underground as paste backfill, produced in a new 55 m^3/hr paste backfill plant, and augmented by DMS reject material in the event of any volume shortfall. An active stockpile of tailings will be stored in a building with heating capability next to the paste backfill plant to provide feed to the plant. An adjacent outdoor area will accommodate any additional tailings on a temporary basis. The majority of DMS reject and mine development material will be placed in a newly created waste rock pile facility located north of the mill off the Prairie Creek floodplain and accessed by trucks on a reconstructed internal site road. Although the waste rock is classified as non-acid generating due to its high content of carbonate material, appropriate precautions will be taken to prevent and mitigate any leaching that may occur from surface run-off through the waste rock pile.

A 150-person camp and cookhouse exists on the site, but most of the buildings have deteriorated beyond economical repair. They will be demolished and will be replaced by a modular camp adjacent to the upgraded administration building complex to be used during construction and operations. A portion of the existing accommodation camp will be salvaged and upgraded for re-use during construction.

The site water management plan for the Prairie Creek Mine proposes the reconfiguration of the present tailings impoundment pond into a two-celled water storage pond connected to the mine and mill via piping and to a new water treatment plant. An exfiltration pipe below the bed of Prairie Creek will discharge treated waters and site run-off that collects in the final site pond. Water treatment rates will be based on a load-based water management system incorporating real-time flows measured in Prairie Creek upstream of the site regulated by the existing Water Licence.

Access road and transportation plan

The construction of the process plant and site infrastructure will be initially serviced via a winter road. Site production operations will be supported via an all season road, which has the following benefits but an increased capital cost:

- Decreased working inventory.
- More timely delivery of product and consistent supply of materials.
- Lower logistical risk of transporting concentrate and supplies.
- A smaller trucking fleet throughout the year.
- Facilitates the use of alternative energy sources such as LNG.

The all season road will reduce energy costs and also enable the consideration of more environmentally friendly alternative energy sources. Local gas fields in the area are producing LNG at this time, which provides an opportunity to reduce reliance on diesel fuel for power generation. An all season road would also have environmental and safety benefits, in that, spreading out the trucking schedule over the full year would avoid high or congested traffic in winter months, therefore lowering the risk of accidents or spills. On 12 September 2017, the Mackenzie Valley Review Board concluded the Environmental Assessment (EA) of the proposed All Season Road to the Prairie Creek Mine by recommending approval to the Federal Minister of Crown-Indigenous Relations and Northern Affairs subject to the implementation of measures described in the EA report.

The current transport logistics system envisages shipping mineral concentrate over an all season road from the mine site in 20 tonne bulk containers. This would involve the creation of a bulk handling load-out facility at the mine and transport by B-line trucks, each carrying two containers along the all season road and highway to Fort Nelson. Containers would be offloaded from the trucks at Fort Nelson and loaded onto four-container-capacity rail flat-cars for transport by CN Rail to the port of Vancouver for shipment to smelters overseas. Inbound freight will be trucked as backhaul over the same route. A marshalling area will be developed in Fort Nelson near the rail siding.

Concentrate marketing

The Prairie Creek Project will produce three types of concentrate: zinc sulphide, lead sulphide and lead oxide. CZN plans to combine the two lead concentrates into one concentrate at the mill site.

Canadian Zinc has signed a Memorandum of Understanding (MOU) with each of Korea Zinc and Boliden for the sale of zinc and lead concentrates. The MOUs set out the intentions of CZN and each of Korea Zinc and Boliden to enter into concentrate sales agreements for the concentrates to be produced from the Prairie Creek Mine on the general terms set out in the MOUs, including commercial terms which are confidential.

The sales agreements will account for all of the planned production of zinc concentrate and about half of the planned production of lead concentrate for the first five years of operation at the Prairie Creek Mine. The sales agreements will provide that treatment charges will be set annually at the annual benchmark treatment charges and scales, as agreed between major smelters and major miners.

Payables, penalties, and quotational periods will be negotiated in good faith annually during the fourth quarter of the preceding year, including industry standard penalties based on indicative terms and agreed limits specified in each MOU.

Treatment and refining charges, including deductibles, payable and penalties, vary with smelter location and individual smelter terms and conditions. The Economic Model used in the 2017 FS has been prepared assuming average blended indicative treatment charges of US$172 per tonne for zinc sulphide concentrates and US$130 per tonne for lead concentrates, with industry standard penalties, including mercury penalties of US$1.75 for each 100 ppm above 100 ppm Hg per tonne of concentrate.

Project execution

The mine start-up schedule is significantly influenced by the seasonal weather conditions in the Northwest Territories. Target start-up for commencement of production / milling operations at Prairie Creek Mine is 1 August 2020, with commissioning of the mill for three months prior to this date. The first year of the path to production project schedule mostly comprises detailed on-site and off-site engineering design, initial site / portal preparation, and the completion of permitting and design of the all season road. Later during the first year, procurement of long-lead items would be completed in order to have the required equipment and supplies available to be brought in on the winter road of the second year; thereby to commence main construction, begin mine development, further prepare the site and advance the all season road. The third year would involve continuing mine development, completing site construction, continuing construction of the all season road and commissioning the mill to production. It is projected that a pre-production on-site workforce will peak at approximately 211 people in August of 2019 (Project Year -01).

Mobilization to site will initially be by winter road and air, concurrent with construction of the all season road. The subsequent shipment of concentrates and production supplies will be on the all season road.

Permitting, environmental, and community

The Prairie Creek Mine is located in an environmentally sensitive watershed of the South Nahanni River and proximal to the Nahanni National Park Reserve (NNPR). As a result, particular attention has been paid by the Company and by regulators to potential impacts on water quality that may be caused by Project construction and operations.

CZN currently has a number of permits and licences for both exploration and mine operations issued by the Mackenzie Valley Land and Water Board (MVLWB) under the *Mackenzie Valley Resource Management Act*. In addition, CZN has a Land Use Permit (LUP) and Water Licence from Parks Canada for the portion of an operations winter road that crosses the NNPR.

The main Licence is the Type “A” Water Licence (MV2008L2-002), which was issued by the MVLWB on 8 July 2013 and permits CZN to conduct mining, milling and processing activities at the Prairie Creek Mine site, use local water, dewater the underground mine and dispose of waste from mining and milling. Other Land Use Permits and Water Licences provide for winter road, mine site and transfer related facilities.

Water Licence MV2001L2-0003 and LUP MV2012C0008 allow CZN to continue with underground exploration prior to operations. LUP MV2012C0002 provides for surface exploration and diamond drilling at sites throughout the Prairie Creek property.

A Land Use Permit and Water Licence for an all season road were applied for in April 2014 to the MVLWB and Parks Canada and were referred to EA with the Mackenzie Valley Review Board. On 12 September 2017 the Mackenzie Valley Review Board recommended to the Federal Minister of Crown-Indigenous Relations and Northern Affairs that the Project proceed to permitting subject to implementation of measures described in the Report of Environmental Assessment.

Prior to the main operating licences being issued in 2013, CZN had been involved in numerous regulatory processes to obtain various Land Use Permits and Water Licences for normal-course exploration and development at the Prairie Creek Mine site.

Innovative water management practices are necessary at the Prairie Creek Mine during operations due to the seasonal nature of the discharge and the receiving environment upstream of a national park. The volume of water for discharge will vary seasonally, being greatest in summer. Flows in Prairie Creek are also variable, being very low in winter and fluctuating in summer. Therefore, storage of water in a large pond on site will be maximized in winter, and treated water discharge will be proportionately tied to creek flows to minimize receiving water concentrations, meet Water Licence limits and protect the ecosystem downstream. A variable load discharge (VLD) approach to water management was developed and accepted during the regulatory process. A Water Licence to operate the mine was issued in 2013 by the MVLWB. The Water Licence will regulate discharge by 'end-of-pipe' effluent quality criteria as well as by VLD to meet receiving water objectives during operations. Real-time flow measurements upstream in Prairie Creek are planned in order to track the allowable load for discharge. A seasonal schedule for treated mine and mill water discharge will apply based on the site water balance; although the actual discharge rates will be based on the daily on-site analysis of treated water sentinel parameters, and on flows in Prairie Creek, which may vary on an hourly basis. Discharge via exfiltration trench below the bed of Prairie Creek will promote mixing and attenuation of parameter concentrations to meet site specific water quality objectives.

In June 2009, the NNPR was expanded to include the entire watershed of the South Nahanni River. However, the Prairie Creek site and a 300-km^2 surrounding area were excluded from the Park. An amendment to the Canada National Parks Act provided for a right of access through the expanded Park into the Prairie Creek area. Recognizing the need to work closely together, in 2008 CZN and Parks Canada entered into a MOU that formalized the intent of both parties to work collaboratively, within their respective areas of responsibility, authority and jurisdiction, to achieve their respective goals of an expanded NNPR and an operating Prairie Creek Mine. The MOU was renewed in 2015 for another five years.

CZN has completed a detailed socio-economic assessment in support of the Project. The study concluded that the Prairie Creek Mine will be a relatively modest project in a region of the NWT that has limited economic prospects. The majority of the economic and social benefits will be generated through the participation of local labour and businesses in the area, including the communities of Nahanni Butte, Fort Simpson, and Fort Liard.

In 2011, Canadian Zinc signed Impact and Benefits Agreements with each of the Nahanni Butte Dene Band and Liidlii Kue First Nation (Fort Simpson), both in the Dehcho Region. Later that year, CZN negotiated a Socio-Economic Agreement with the Government of the Northwest Territories (GNWT), covering social programs and support, commitments regarding hiring and travel, and participation on an advisory committee to ensure commitments are effective and are carried out.

Employment

Over the course of the construction path to mill production, projected to be initiated in 2018 (Y-02) and to extend into 2020 (Y01), a maximum of approximately 211 people is expected to be employed on site during 2019 (Y-01).

During steady state operations the mine will employ a total of approximately 330 people on-site working in two alternating shifts on a two-week-in / two-week-out basis, including truck operators, with half of the employees being on-site at any one time. An additional 33 personnel, mostly related to trucking, will be employed off-site in the Fort Liard and Fort Nelson areas. Personnel will work a regular rotation on site, with rest periods off site, with transport by charter flights to the existing on-site 1,000 m gravel airstrip. CZN's hiring policy and commitments under its signed Impact and Benefits Agreements are to give preference to qualified local community residents, followed by northern residents. Training programs will be organized to further promote and maximize local aboriginal employment.

Project metrics

Table ES.1.4 Project metrics – Prairie Creek Mine

Mine and mill parameters			**Concentrates**			
			Type	10 yr W. Avg. Tonnes	Average grade	Payability
Total ore mined (million tonnes)		8.07	Zinc concentrate	64,800	Zinc: 59%	Zinc: 85%
Mining rate (tonnes / day)		1,600			Silver: 136 g/t[3]	Silver: 70%
Milling rate (tonnes / day) post-DMS		1,200	Lead concentrate	71,600	Lead: 62%	Lead: 95%
LOM (years)		15			Silver: 800 g/t	Silver: 95%
Mine and mill statistics						
Metal	10 yr ore grade (weighted average)		Ore grade LOM (weighted average)		Mill recoveries LOM (weighted average)	10 yr average annual contained metal
Zinc	8.50%		8.70%		83%	95M lbs[4]
Lead	9.30%		8.10%		88%	105M lbs[4]
Silver	139 g/t		124 g/t		87%	2.1M oz[4]
Project assumptions base case						
Zinc price	**US$1.10/lb**		**Treatment charges**		Exchange rate	C$1.25:US$1.00
Lead price	**US$1.00/lb**		US$172/tonne Zn Con		Discount rate	8%
Silver price	**US$19.00/oz**		US$130/tonne Pb Con			
Operating and capital costs						
Operating costs[2]	LOM $/t ore mined		Capital costs			$M
Mining	58		Pre-production capital			253
Processing	47		Contingency			26
Site services	19		Total pre-production capital			279
G&A	30		Sustaining capital			117
Total on-site costs	154		Working capital			36
Transportation[1]	69					
Total operating costs[2]	223					
[1] Includes truck, rail, handling and ocean shipping			[3] Subject to a deduction of 3 oz. per tonne of concentrate			
[2] Does not include treatment, refining charges, royalty			[4] Total metal contained in both lead and zinc concentrates			
Economic results (LOM)					Pre-tax	Post-tax
Cash flow undiscounted ($M)					899	562
NPV @ 8% ($M)					344	188
NPV @ 5% ($M)					497	291
IRR (%)					23.8	18.4
Payback period (years from first revenue)					4.4	4.6
Average annual EBITDA ($M)					81	

Capital cost estimates

The general breakdown of the Pre-Production Capital cost estimate for the Prairie Creek Project is indicated in the following table:

Table ES.1.5 Capital cost estimate – Prairie Creek Mine

	Project year			
Description (costs in $M)	**-02**	**-01**	**01**	**Total cost**
Mine development	2.6	13.6	21.5	**37.7**
Site preparation	4.3	12.5	2.6	**19.4**
Mill process plant	9.0	18.9	3.2	**31.1**
Paste tailings plant and process	2.9	16.6	3.4	**22.9**
Indirects including EPCM	10.9	7.8	5.1	**23.8**
Other site infrastructure	6.7	7.7	1.5	**15.9**
All season road	13.0	41.6	13.9	**68.5**
Owner's costs	6.8	15.3	11.5	**33.6**
Total (excluding contingency)	**56.2**	**134.0**	**62.7**	**252.9**
Contingency	5.5	12.3	8.2	**26.0**
Total pre-production capital	**61.7**	**146.3**	**70.9**	**278.9**

Pre-Production Capital cost refers to capital costs incurred until the first processing of mined ore, and has been estimated at a total of $252.9M, excluding contingency, and $278.9M including a contingency of $26.0M.

Based on proposals received, several capital items will be supplied on a lease-to-purchase basis, including the accommodation camp, paste plant, flotation cells and thickeners. The lease costs of such items incurred during the pre-production period are included in Pre-Production Capital costs, and lease costs incurred after production start-up are included in Sustaining Capital costs.

Contingency for the process plant and site infrastructure portion was estimated using a Monte Carlo simulation model with an overall contingency of 13.2% based on 80% confidence level. Mine development costs are largely based on contractor quotes for the detailed scope of work, but with an overall 13.0% contingency allowance. The all season road estimation used an overall contingency of 8.0% and owner's costs were assigned a contingency factor of 10.0%. The overall Project contingency is 10.3%.

Sustaining capital over the life of the mine has been estimated at $117M and relates largely to ongoing mine development as the mine is expanded to deeper levels, ongoing maintenance of the all season road, and includes leasing costs of capital items in the amount of $11M.

Working capital of $36 million is estimated to be required over the first six months subsequent to the start of commercial production.

Operating cost estimates

The breakdown of the Operating Cost Estimate for the Prairie Creek Mine, on a Canadian dollar per tonne mined basis, is shown in the following table.

Table ES.1.6 Operating cost estimate – Prairie Creek Mine

Total operating cost	**($/t mined)**
Mining	58.23
Milling / processing	46.76
General and administrative	30.32
Site services	18.55
Sub-total	**153.86**
Transportation[1]	68.73
Total	**222.59**

1. Includes truck / rail / handling / shipping.

Mining operating costs for the first two years of operation are largely based on contractor quotes. Operating cost estimates for mining beyond the contractor period have been developed from first principles and using direct supplier quotes.

The mining contractor quotes for the first two years of operation, based on a detailed scope of work and schedule, provide a high level of confidence in the estimated mining costs. The indicative proposal from the Northwest Territories Power Corporation to supply turnkey type power generation provides further support in the key area of power costs.

The following list summarizes key project assumptions used to develop the operating costs, which are in 2017 constant dollars:

- All electrical power will be produced by generators operating on LNG and provided by Northwest Territory Power Corporation on a flat rate for the life of mine and using an estimated LOM power cost of $0.25/kWhr for the main power generation.
- A delivered price of diesel of $0.82/L and LNG of $15.50/GJ was used to estimate power costs other than for the main generator supply.
- Mill, surface and G&A operating costs are generally deemed to be steady-state per tonne milled LOM, based on recent labour and materials costs.
- Manpower costs for road maintenance and concentrate haul are included in total transport costs.

Economic analysis

The Base Case economic model has been developed using long-term metal price assumptions of US$1.10/lb zinc, US$1.00/lb lead, US$19.00/oz silver and an exchange rate of C$1.25:US$1.00. Determination of metal prices for use in the 2017 FS has included consideration of consensus price forecasts published by Consensus Economics Inc. as at September 2017, and a review of market commentary published by various services, including the International Lead and Zinc Study Group, CRU, Metals Bulletin Research, Wood Mackenzie, and other industry sources as discussed in Section 19. Current metal prices, rolling three-year averages, and prices used in recent similar mining project studies were also considered for the Prairie Creek economic evaluation.

A sensitivity analysis was conducted on the Project model to evaluate its robustness against variations in financial parameters, specifically Base Case metal prices +/- 10% and the Base Case foreign exchange rate +/- 10% and +4%. The financial analysis centering on the Base Case, showing average annual EBITDA, NPV (at 8% and 5% discount rates), IRR and payback periods, on a pre-tax and post-tax basis is presented in the following table.

Table ES.1.7 Financial analysis – Prairie Creek Mine

Metal price scenario[1]	90%	100%	110%
Average Annual EBITDA ($M)	59	81	103
Pre-Tax Cash Flow Undiscounted ($M)	546	899	1,251
Pre-Tax NPV @ 8% discount ($M)	166	344	523
Pre-Tax NPV @ 5% discount ($M)	270	497	724
Pre-Tax IRR	16.5%	23.8%	30.2%
Post-Tax Cash Flow Undiscounted ($M)	345	562	779
Post-Tax NPV @ 8% discount ($M)	74	188	301
Post-Tax NPV @ 5% discount ($M)	148	291	433
Post-Tax IRR	12.4%	18.4	23.7%
Post-Tax Payback Period (years from first revenue)	5.7	4.6	4.0
Exchange rate scenario[2]	**C$1.125:US$1.00**	**C$1.30:US$1.00**	**C$1.375:US$1.00**
Average annual EBITDA ($M)	62	89	100
Pre-Tax Cash Flow Undiscounted ($M)	589	1,022	1,208
Pre-Tax NPV @ 8% discount ($M)	188	407	501
Pre-Tax NPV @ 5% discount ($M)	298	577	696
Pre-Tax IRR	17.4%	26.2%	29.5%
Post-Tax Cash Flow Undiscounted ($M)	372	638	752
Post-Tax NPV @ 8% discount ($M)	88	228	287
Post-Tax NPV @ 5% discount ($M)	166	341	416
Post-Tax IRR	13.2%	20.3%	23.1%
Post-Tax Payback Period (years from first revenue)	5.5	4.4	4.1

1. Metal prices varied plus / minus 10% and exchange rate unchanged.
2. Exchange rate varied plus / minus 10% and plus 4%, and metal prices unchanged.

A 'stressed case' sensitivity analysis using assumed metal prices of US$0.80/lb for zinc and lead and US$17/oz for silver, and an exchange rate of C$1.40:US$1.00 indicates a pre-tax $NPV_{8\%}$ of $104M and IRR 14% (post-tax $NPV_{8\%}$ of $32M and IRR 10%). Using the average metal prices for the three years ended 30 June 2017 of US$0.98/lb for zinc, US$0.88/lb for lead and US$16.82 for silver, and an exchange rate of C$1.27:US$1.00 indicates a pre-tax $NPV_{8\%}$ of $161M and IRR 16% (post-tax $NPV_{8\%}$ $71M and IRR 12%).

Recommendations

As a result of the feasibility study assessment, AMC recommends the following:

- Early completion of engineering and mine development programs to facilitate achievement of scheduled access development, initial dewatering and first ore.
- Completion of permitting of the all season access road.
- Study of opportunities for enhanced mine operation through use of automation and advanced technology.
- Further underground paste backfill strength and flow property studies.
- Additional study of paste binder requirements and backfill methodology.
- Further hydrology study to enhance understanding of water ingress to the mine and optimize dewatering strategy and water treatment.
- Selection of appropriately qualified and experienced mining contractor for the pre-production phase and the first phase of production operations.
- Completion of a detailed trade-off study further examining the economic merits of recovering lead oxide and oxide recovery at low grades.
- Further detailed mine design and schedule to optimize grades, and balance ore and tailings stockpile requirements.

- Detailed design of key underground infrastructure including magazines, dewatering and sump set-ups, service bays, main fan and heating set-ups, electrical infrastructure.
- Finalization of arrangements for active and passive tailings stockpiles.
- Ordering of key long-lead items for underground operation including main fans and heaters.

Ausenco recommends the following in the execution phase of the project:

- Further investigation of the use of LNG (liquefied natural gas) for heating of buildings and underground mine.
- Further investigation of the use of excess heat from generators to supplement underground heating.
- Investigation of the use of used construction equipment and mobile equipment for operations.
- Optimization of the paste plant design, to include best use of recent test results.
- Complete detailed engineering and IFC drawings to support the procurement and construction of the process plant and site infrastructure.
- Completion of early works site activities including removal of existing generators from the power house, repair of the mill roof, initial work on the water storage pond and waste rock pile, site clearance of derelict buildings, equipment and scrap material.
- Selection of appropriately qualified and experienced contractor(s) to construct the surface works.
- Selection and engagement of appropriately qualified and experienced contractor(s) to carry out the construction of the process plant and site infrastructure.

Total associated preliminary cost estimate for the above recommendations from AMC and Ausenco is $18.2M, of which the majority is included in the FS Capital Cost estimate.

Conclusions

The Prairie Creek Property contains a high-grade, silver-lead-zinc-copper vein along with other lead-zinc deposits and deposit types.

The 2017 FS indicates a Mineral Reserve of 8.1 Mt and an LOM from mill start-up of 15 years at a steady-state production rate of 584,000 tpa.

Mill start-up is projected for August 2020, with a pre-production period during which detailed engineering, mill and camp refurbishment, underground development from existing workings, and construction of key surface infrastructure items, including a paste plant and all season road, will take place.

The 2017 FS indicates a base case Pre-Tax Net Present Value (“NPV”) of $344M using an 8% discount rate, with an Internal Rate of Return (“IRR”) of 23.8% and a post-tax NPV of $188M with a post-tax IRR of 18.4%. The Base Case metal price assumptions used in the model are: Zn US$1.10/lb., Pb US$1.00/lb., Ag US$19.00/oz., with a foreign exchange rate of C$1.25=US$1.00.

The development of the Prairie Creek Mine is projected to offer significant economic advantages on a wider scale. Canadian Zinc has indicated that there is broad support among aboriginal organizations and communities in the Dehcho region for the direct benefit and economic stimulus that the mine would bring to this region of the Northwest Territories. Its envisaged operation presents a significant opportunity for potential enhancement of the social and economic well-being of the surrounding communities. During construction there will be approximately 211 jobs, and during steady state operations over the life of the mine there will be approximately 330 direct full-time jobs. In addition, the Project offers other potential indirect business and employment opportunities, related to transport, supply of the mine site and environmental monitoring and management.

The Prairie Creek Mine is shown to be a viable project, based on the Mineral Reserves, mine plan, and production and economic parameters determined within the 2017 FS. AMC recommends that Canadian Zinc advance the Project to the next stage, which will include: detailed design and planning of the required services, construction of the all season road, refurbishment of the mill, ordering the long-lead equipment for power generation, portal refurbishment, access widening, and development of ramp declines and underground infrastructure in preparation for ore production and processing.

Contents

Tables

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LIST OF ABBREVIATIONS

AANDC	Aboriginal Affairs and Northern Development Canada (recently changed to Indigenous and Northern Affairs Canada)
Ag	Silver
AHJ	Authority Having Jurisdiction
AMC	AMC Mining Consultants (Canada) Ltd.
AMSL	Above Mean Sea Level
ANFO	Ammonium Nitrate-Fuel Oil blasting agent.
ARD / ML	Acid Rock Drainage / Metal Leaching
ASR	All Season Road
BC	British Columbia
BOD	Biological Oxygen Demand
BWI, BMWi	Ballmill Work Index
Cadillac	Cadillac Explorations Ltd.
CBH4	Navigation Canada Designation for the Prairie Creek Airstrip
CN	Canadian National railway
CRP	Closure and Reclamation Plan
CZN	Canadian Zinc Corporation or Canadian Zinc or the Company
CFM	Cubic feet per minute
Cd	Cadmium
Cu	Copper
d	Day
DAF	Drift and Fill mining method
DAR	Developer's Assessment Report
DCFN	Dehcho First Nations
DHC	de Havilland Canada
DMS	Dense Media Separation
dmt	Dry metric tonne
EA	Environmental Assessment
EBITDA	Earnings before interest, taxes, depreciation, and amortization
EPCM	Engineering, Procurement and Construction Management
ER	Emergency Response
Fe	Iron
FOB	Free On Board
FS	Feasibility Study
g	Gramme
g/t	Grammes per tonne
G&A	General and Administration
GNWT	Government of the Northwest Territories
GVW	Gross Vehicle Weight
h, hr	Hour
Ha	Hectare (area 100 m by 100 m)
Hg	Mercury
HLS	Heavy Liquid Separation
HW	Hanging Wall
INAC	Indigenous and Northern Affairs Canada
IBA	Impact and Benefits Agreement
ID2	Inverse Distance Squared
IFC	Issued For Construction drawing
ILZSG	International Lead and Zinc Study Group
IND	Indicated Mineral Resource
INF	Inferred Mineral Resource
IRR	Internal Rate of Return
IT	Information Technology

kg	Kilogramme
km	Kilometre
kt	Kilotonne
kVA	Kilovolt Amp
kW	Kilowatt
kWh	Kilowatt-hour
L	Litre
LCT	Locked Cycle Test
LHD	Load Haul Dump machine
LHOS	Long Hole Open Stoping mining method
LKFN	Liidlii Kue First Nation
LNG	Liquified Natural Gas
LOM	Life of Mine
LTF	Liard Transfer Facility
LUP	Land Use Permit
L/sec, L/s	Litres per second (flow volume)
m	Metre
m^3	Cubic metre
mm	Millimetre
M	Million
MBP	Mid-size Backfill Plant
MEA	Measured Mineral Resource
MMER	Metal Mining Effluent Regulations
MW	Megawatt
MWh	Megawatt-hour
mg/L	Milligrammes per litre
µg/L	Microgrammes per litre
µm	Micron or micrometre
MOU	Memorandum of Understanding
MQV	Main Quartz Vein mineralization
MTS	Mine Training Society
Mt	Million tonnes
MVLWB	Mackenzie Valley Land and Water Board
MVRMA	Mackenzie Valley Resource Management Act
MVRB	Mackenzie Valley Review Board
MVT	Mississippi Valley Type mineralization
NAG	Non Acid Generating materials
NBDB	Nahanni Butte Dene Band
NI 43-101	National Instrument 43-101
NNPR	Nahanni National Park Reserve
NPV	Net Present Value
NSR	Net Smelter Return
NT or NWT	Northwest Territories
NTPC	Northwest Territories Power Corporation
NU	Nunavut
PAG	Potentially Acid Generating materials
Pb	Lead
PbOx	Lead Oxide
P. Eng.	Professional Engineer
PFS	Pre-Feasibility Study
P. Geo.	Professional Geoscientist
PC	Parks Canada
PCA	Prairie Creek Anticline
PRB	Probable Mineral Reserve
PRV	Proven Mineral Reserve
QA/QC	Quality Assurance/Quality Control

QP	Qualified Person
ROM	Run of Mine
RQD	Rock Quality Designation
SARC	San Andreas Resources Corporation
Sb	Antimony
SEIA	Socio-Economic Impact Assessment
SG	Specific Gravity
SMS	Stratabound Massive Sulphide mineralization
STP	Sewage Treatment Plant
STK	Stockwork mineralization
t	Tonne (metric)
st	Short ton
t/d, tpd	Tonnes per day
tph, t/h	Tonnes per hour
tpy, t/a	Tonnes per year
TDS	Total Dissolved Solids
ToR	Terms of Reference
TOC	Total Organic Content
TSX	Toronto Stock Exchange
TSS	Total Suspended Solids
TT	Tetra Tech Inc.
V	Volt
VFR	Visual Flight Rules
VLD	Variable Load Discharge
VP	Vice President
WBS	Work Breakdown Structure
WMP	Wildlife Management Plan
wmt	Wet metric tonne
WSC	Water Survey of Canada
WSP	Water Storage Pond
WTP	Water Treatment Plant
WRF	Waste Rock Facility
Zn	Zinc
ZnEq	Zinc equivalent

2 Introduction

2.1 General and terms of reference

This Technical Report on the Prairie Creek Property (the Property), located approximately 500 km west of Yellowknife in the Northwest Territories, Canada, has been prepared by AMC Mining Consultants (Canada) Ltd. (AMC) of Vancouver, Canada, in conjunction with Ausenco Engineering Canada Inc. (Ausenco), Vancouver, and with input from other experts as disclosed in Table 2.1 and Section 3 on behalf of Canadian Zinc Corporation (CZN or the Company or Canadian Zinc) of Vancouver, Canada in accordance with the requirements of National Instrument 43-101 (NI 43-101), “Standards of Disclosure for Mineral Projects”, of the Canadian Securities Administrators (CSA) for lodgment on CSA’s “System for Electronic Document Analysis and Retrieval” (SEDAR).

This report discloses the results of a Feasibility Study (FS) based on the previous Mineral Resources, and ongoing optimization projects and other engineering studies completed since the date of the previous report in 2016. The previous report is titled “Prairie Creek Property Prefeasibility Update NI 43-101 Technical Report (Amended and Restated)” for Canadian Zinc Corporation, prepared by AMC and with an effective date of 31 March 2016 (2016 AMC Report) and a revised date of 30 September 2016.

CZN is the 100% owner of the Property, which consists of two surface leases and twelve mining leases. The Property assets include the Prairie Creek Mine, a processing plant, various mine and plant-related surface infrastructures, various earth moving and mining equipment, and numerous mineralized occurrences that are at various stages of exploration and development.

2.2 The issuer

CZN is a publicly traded mining exploration company that is based in Vancouver, Canada and with offices in Toronto and Fort Simpson (NWT). CZN is listed on the Toronto Stock Exchange under the trading symbol “CZN”, on the OTCQB Venture Marketplace in the United States under trading symbol “CZICF”, and under the symbol "SAS" on the Frankfurt Exchange. The prime asset controlled by CZN is the Prairie Creek Property.

Prior to 25 May 1999, CZN was named San Andreas Resources Corporation.

2.3 Report authors

The names and details of persons who prepared, or who assisted the Qualified Persons (QPs) in the preparation of this Technical Report are listed in Table 2.1.

Table 2.1 Qualified Persons responsible for the preparation of this Technical Report

Qualified Person	Position	Employer	Independent of CZN	Date of Last site visit	Professional designation	Sections of report **
Mr G.Z. Mosher*	Principal Geologist	Global Mineral Resource Services Ltd.	Yes	No visit	P.Geo.	Sections 4-12, 14, 23; part of 1, 3, 25, 26, and 27.
Mr H.A. Smith	Senior Principal Mining Engineer	AMC Mining Consultants (Canada) Ltd.	Yes	July 2017	P.Eng.	Sections 2, 15, 16, 19, 20, 22, 24; part of 1, 3, 18, 21, 25, 26, 27.
Mr F. Wright	Metallurgical Engineer	F. Wright Consulting Inc.	Yes	May 2017	P.Eng.	Section 13; part of 1.
Mr D. Williams	Division Manager, Structural Transportation	Allnorth Consultants Limited	Yes	No visit	P.Eng.	Part of 1 and 18.
Mr L. P. Staples	Global Practice Lead – Minerals and Metals	Ausenco Engineering Inc.	Yes	July 2017	P.Eng.	Section 17; and part of 1, 18, 21, 25, 26, and 27.
Mr S. Elfen	Global Lead – Geotechnical Services	Ausenco Engineering Inc.	Yes	No visit	P.E.	Part of 1 and 18.

*Formerly employed by AMC Mining Consultants Ltd.
**Note: Where QPs accept responsibility for parts of sections, that responsibility is limited to their areas of expertise: Mr G.Z. Mosher – Geology and Mineral Resource aspects; Mr H.A. Smith – Mining, Mineral Reserves, and associated costs and commentary; Mr F. Wright – Metallurgical aspects; Mr D. Williams – All Season Road; Mr L.P. Staples – Processing, Surface Infrastructure, and associated costs and commentary; Mr S. Elfen – Surface Geotechnical aspects.

Table 2.2 Persons who assisted the Qualified Persons in preparation of this Technical Report

Expert	Position	Employer	Independent of CZN	Visited site	Sections of report
Mr A.B. Taylor, P.Geo.	COO & VP Exploration	Canadian Zinc Corporation	No	October 2017	Part of 4-12
Mr T.L. Cunningham, CPA, CMA	Chief Financial Officer & Vice President, Finance	Canadian Zinc Corporation	No	August 2012	19, 21 and 22
Mr D. Harpley, P.Geo.	VP Environment & Permitting Affairs	Canadian Zinc Corporation	No	August 2017	Part of 4; 20; part of 24
Mr K. Cupit, P.Geo.	Project Geologist	Canadian Zinc Corporation	No	October 2016	Part of 4-12
Mr A. Grice P.Eng.	Principal Backfill Consultant	AMC Mining Consultants (Canada) Ltd.	Yes	No	Part of 16
Mr J. Huang, P.Eng.	Senior Metallurgical Consultant	Tetra Tech	Yes	January 2014	Part of 1-2; 13; 17; part of 21, 25, and 26.
Mr. C. Wels, P.Eng.	Principal and Senior Hydrogeologist	Robertson Geoconsultants Inc.	Yes	2014	Part of 16
Mr T. Morrison**, P.Eng.	Consultant	Self-employed	Yes	June 2017	Part of 16, 18, 21, 22; 24; part of 25, and 26.
Mr W. Pitman, P.Eng.	Principal Geotechnical Engineer	Adiuvare Geology & Engineering Ltd.	Yes	No	Part of 16
Mr A. Smith	Principal Mining Consultant	AMC Mining Consultants (Canada) Ltd.	Yes	July 2017	Part of 15-16
Mr Paul Salmenmaki, P.Eng.	Senior Mining Engineer	AMC Mining Consultants (Canada) Ltd.	Yes	No	Part of 15-16
Mr Mo Molavi, P.Eng.	Principal Mining Engineer / Mining Services Manager	AMC Mining Consultants (Canada) Ltd.	Yes	No	Part of 1, 2, 16, 18, 24, 25, 26, 27, and 28.
Mr Ben Patterson	Project Manager	Ausenco Engineering Inc.	Yes	May 2017	Part of 18
Mr William Hughes*, P.Eng.	Principal Mechanical / Infrastructure Engineer	Prairie Machine & Parts	Yes	July 2017	Part of 16 and 18

*Formerly employed by AMC Mining Consultants Ltd.
**Independent consultant who has been engaged by CZN to assist with this FS and preparation of this report.

Frequent visits are made to the Property by CZN personnel. Recent visits are noted in Table 2.1 and Table 2.2 for those persons who are taking responsibility for Sections of this report or who have assisted in its preparation.

2.4 Source of information

This report relies in part on previous reports on the Property, previous Technical Reports prepared by AMC and others disclosing Mineral Resource estimates, and the results of a PFS in 2012 and an update in 2016. These Technical Reports are titled *Prairie Creek Property, Northwest Territories, Canada, Technical Report for Canadian Zinc Corporation* and with an effective date of 15 June 2012, (2012 AMC Report), and *Prairie Creek Property Prefeasibility Update NI 43-101 Technical Report (Amended and Restated) For Canadian Zinc Corporation* and with an effective date of 31 March 2016, revised date 30 September 2016 (2016 AMC Report).

In addition to numerous previous studies and reports, additional programs and studies were carried out during 2016-2017. These additional assessments include:

- Detailed underground hydrology modelling.

- Mining / milling maximization capacity analysis.
- Metallurgical programs including DMS, grinding and flotation studies based on a new composite MQV bulk sample, initial flotation study of STK material, variable flotation studies, abrasion testing, and grinding index testing.
- Material handling and flow property tests of ore
- Underground paste backfill binder testing and analysis.
- Further trade-off and optimization studies for the mill.
- Underground mine trade-off studies including ventilation, heating, haulage and development.
- Analysis of energy alternatives for the operation.
- Further assessment of site facilities to maximize utilization, minimize risk and analyze reliability.
- Advancement of transport routing and logistics.
- All season road studies and permitting.

A number of these reports are referenced and highlighted in Section 27.

2.5 Other

The Company reports its financial information in Canadian dollars and all monetary amounts set forth herein are expressed in Canadian dollars unless specifically stated otherwise.

This report has an effective date of 28 September 2017.

3 Reliance on other experts

The Qualified Persons have relied, in respect of legal aspects, upon the work of the issuer's expert listed below. To the extent permitted under NI 43-101, the Qualified Persons disclaim responsibility for the relevant section of the Report.

- Expert: Mr A. Taylor, Vice President Exploration and Chief Operating Officer of CZN.
- Report, opinion or statement relied upon: information on mineral tenure and status, title issues, royalty obligations, etc.
- Extent of reliance: full reliance following a review by the Qualified Persons.
- Portion of Technical Report to which disclaimer applies: Section 4.

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

- Expert: Mr A. Taylor, Vice President Exploration and Chief Operating Officer of CZN.
- Report, opinion or statement relied upon: information on environmental studies, permitting, social and community impact, site monitoring remediation and reclamation, and closure plan.
- Extent of reliance: full reliance following a review by the Qualified Persons.
- Portion of Technical Report to which disclaimer applies: Section 20.

The Qualified Persons have relied, in respect of taxation aspects, upon the work of the issuer's expert listed below. To the extent permitted under NI 43-101, the Qualified Persons disclaim responsibility for the relevant section of the Report.

- Expert: Mr Trevor L. Cunningham, CPA, CMA, Chief Financial Officer and Vice President, Finance of CZN.
- Report, opinion or statement relied upon: information on tax issues.
- Extent of reliance: full reliance following a review by the Qualified Persons.
- Portion of Technical Report to which disclaimer applies: Section 21.

4 Property description and location

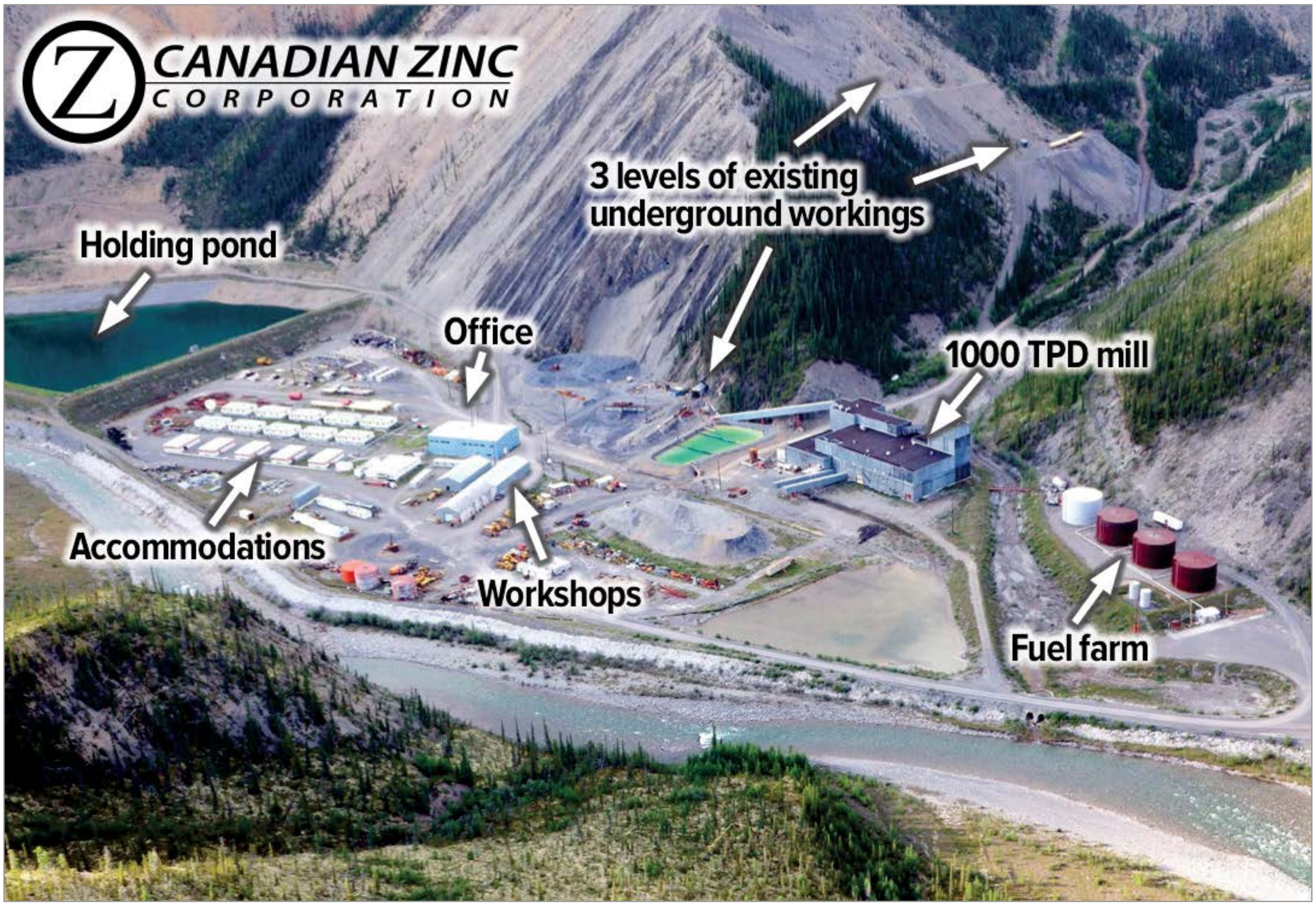
4.1 Property location

The Property is located in the Northwest Territories (NWT), Canada, near the Yukon border, at latitude 61° 33' North and longitude 124° 48' West. The nearest communities include Nahanni Butte, approximately 90 km to the southeast, Fort Liard approximately 170 km to the south, and Fort Simpson, approximately 185 km to the east. Yellowknife, the capital and administrative centre of the NWT, is approximately 500 km to the east. The town of Fort Nelson, British Columbia, which is located approximately 340 km to the south of the Mine, is the nearest point Figure 4.1 shows the location of the Property within the Northwest Territories and relative to various population centres and mining operations.

Figure 4.1 Location of the Prairie Creek property

The Mine site, which is highlighted in Figure 4.1, is located within the watershed of the South Nahanni River, approximately 48 km upstream of the point where Prairie Creek joins the South Nahanni River. The current boundary of the expanded Nahanni National Park Reserve (NNPR) is approximately 7 km downstream and 18 km upstream of the mine site. Since the expansion of the NNPR, the Property is located within an approximate 300 km² area of crown-owned land that is now surrounded by, but not included in, the expanded NNPR. Figure 4.2 shows an overview of the mine site as of March 2017.

Figure 4.2 The Prairie Creek Mine site

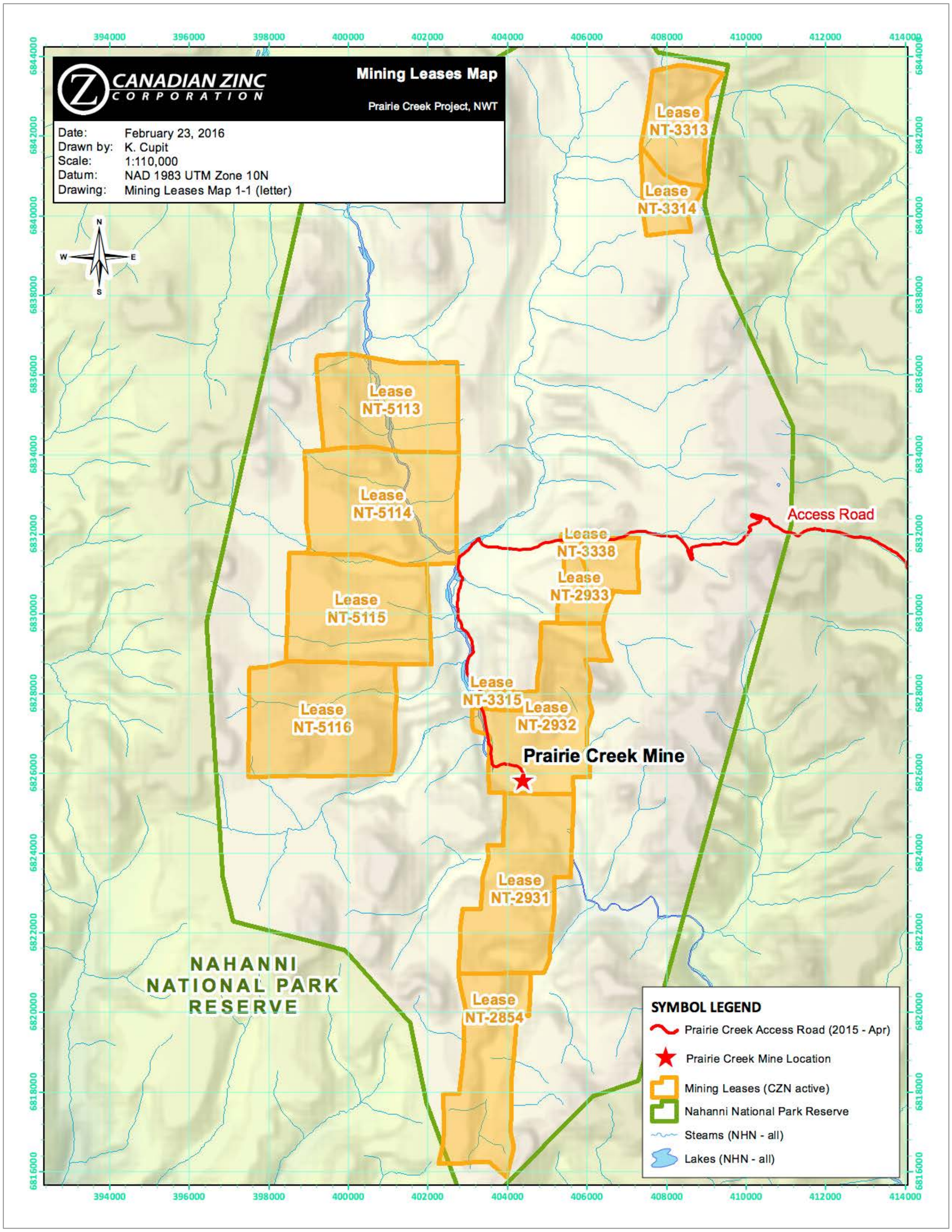


4.2 Property description and ownership

The Property consists of mining leases and surface leases which are leased by CZN from the Government of the Northwest Territories and further described in Section 4.3.

The total area of all land holdings, including mining leases and surface leases at Prairie Creek, is 7,487 hectares. All of the leases, as listed in Table 4.1, are currently in good standing and are located on Crown Land, which is in turn surrounded by NNPR boundary, as shown in Figure 4.3.

Figure 4.3 Plan of leases and claims relative to Nahanni National Park Reserve boundary



4.3 Land tenure

The Mining Leases are renewable on a 21-year basis and currently have expiry dates ranging from August 2019 to July 2032.

The Surface Leases, containing the mine infrastructure, were originally granted by Aboriginal Affairs and Northern Development Canada (AANDC) on a renewable, ten-year basis and, since devolution of some Federal powers to the Northwest Territories on 1 April 2014, are now administered by the Government of the Northwest Territories (GNWT). Presently the surface leases are held in a recurring annual overholding tenancy triggered 31 March of each year until they are converted into operating leases. A minimum six months' notice as to initiation of construction activities related to future mine operations has to be given to GNWT to allow time to prepare and negotiate new leases for mining.

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 hectares. New mining leases for the Gate Claims were received on 16 February 2011, are dated 9 September 2009, and have a term of 21 years, until 9 September 2030.

There is a 1.2% Net Smelter Royalty payable to Sandstorm Gold on the Property.

The Prairie Creek Mine is located on land claimed by the Nahanni Butte Dene Band of the Dehcho First Nations (DCFN) as its traditional territory. The DCFN is engaged in ongoing land settlement negotiations with the Government of Canada and the Government of the Northwest Territories in what is referred to as the Dehcho Process.

Table 4.1 Summary of Canadian Zinc land holdings

Property type	File number	Name	Expiry date	Area (ha)
Surface leases	95F/10-5-5	Mine site	31 March 2019	113.6
	95F/10-7-4	Airstrip	31 March 2019	18.2
Total surface lease area	-	-		131.8
Mining leases	ML 2854	Zone 8-12	22 August 2019	743
	ML 2931	Zone 4-7	5 August 2020	909
	ML 2932	Zone 3 / Main Zone	5 August 2020	871
	ML 2933	Rico West	5 August 2020	172
	ML 3313	Samantha	13 July 2031	420.05
	ML 3314	West Joe	13 July 2031	195.86
	ML 3315	Miterk	13 July 2031	43.7
	ML 3338	Rico	17 July 2032	186.16
	ML 5113	Gate 1	9 September 2030	794.4
	ML 5114	Gate 2	9 September 2030	1,039.64
	ML 5115	Gate 3	9 September 2030	944.13
	ML 5116	Gate 4	9 September 2030	1,036.00
Total mining lease area	-	-		7,354.94
Grand total	**-**	**-**		**7,486.74**

4.4 Existing environmental liabilities

Existing environmental liabilities at the Property are covered through various security deposits posted to the relevant government agencies. The two surface leases, which contain all existing and proposed infrastructure, are covered by a reclamation and closure plan together with a security deposit.

On 22 May 2015 the Mackenzie Valley Land and Water Board (MVLWB) approved amendments to security payments, relating to the recently issued Land Use Permit and Water Licence for operations at Prairie Creek, as proposed by CZN. The amended payments reflected CZN's existing liability at the site based on the Closure Plan associated with the Surface Leases and staged future payments based on specific developments. CZN made an immediate additional payment to cover the existing liability. On 19 August 2015 the Government of the Northwest Territories confirmed that CZN had posted an additional security of $1,550,000 (additional to the previously posted $250,000 security) consistent with the MVLWB ruling.

The existing surface leases will be converted from a care and maintenance status to a production status prior to operations. A new reclamation and closure plan associated with the operations Water Licence forms the basis for future security payments tied to construction and operations (refer to Table 21.5).

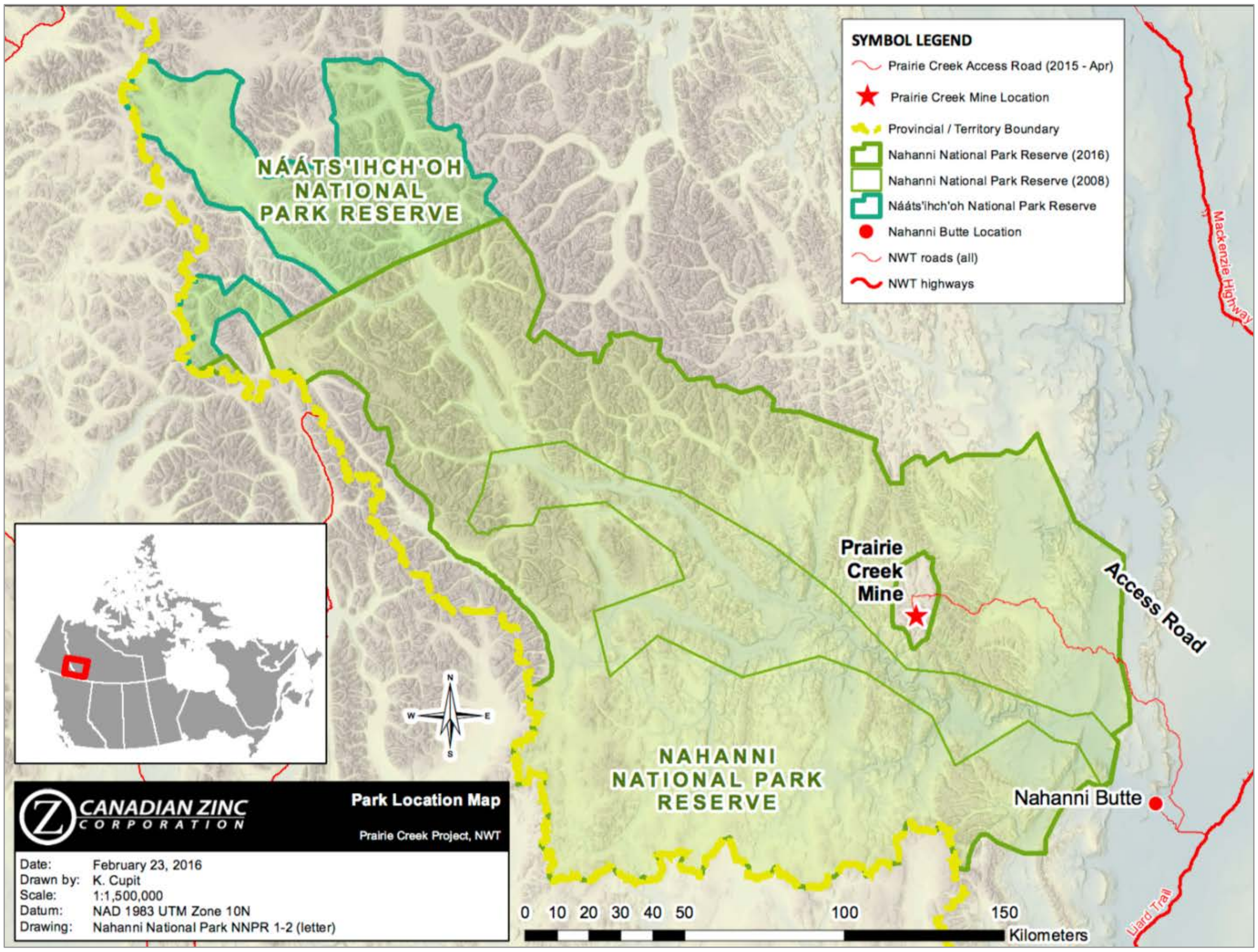
Existing exploration land use permits and water licences issued by the Mackenzie Valley Land and Water Board also have separate security deposits associated with them to ensure reclamation is carried out.

4.5 Nahanni National Park Reserve

The NNPR was created in 1972 specifically for the purpose of setting aside the South Nahanni River for wilderness recreational purposes. Exploration activity at Prairie Creek Mine 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 NNPR. The NNPR was expanded by 30,000 km^2, making it the third largest National Park in Canada. The enlarged park covers most of the South Nahanni River watershed and completely encircles the Prairie Creek Mine. However, the Mine itself and a large surrounding area of approximately 300 km^2 are specifically excluded from the expanded NNPR, as depicted in Figure 4.4, and the Parks Act was amended in parliament to allow the rights of access into the Prairie Creek Mine. The Nááts'ihch'oh National Park Reserve was proclaimed in 2014 and adjoins NNPR to the northwest to further protect the South Nahanni watershed.

Figure 4.4 Property in relation to the expanded Nahanni National Park Reserve



The exclusion of the Prairie Creek Mine from the NNPR expansion area has brought clarity to the land use policy objectives for the region and facilitated various aspects of the environmental assessment process. In July 2008, Parks Canada Agency (Parks Canada) and CZN entered into a Memorandum of Understanding (MOU), valid for three years, with regard to the expansion of the NNPR and the development of the Prairie Creek Mine. In March 2012, the MOU was renewed for a further period of three years wherein Parks Canada and CZN agreed to work collaboratively to achieve their respective goals of managing the NNPR and an operating Prairie Creek Mine. Subsequently the MOU was renewed in November 2015 for a period of five years.

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

5.1 Physiography and vegetation

The Property is located 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 Mine 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, 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.

5.2 Accessibility

Year-round access to the Mine site is provided by charter aircraft, generally from Fort Nelson, BC. or Fort Simpson, NWT, 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 Mine site, and is shown in Figure 5.1.

Figure 5.1 Prairie Creek Mine 1,000 m airstrip (Registered as CBH4 Navigation-Canada)

The Liard highway, which connects Fort Nelson, BC to Fort Simpson, NWT, 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.

5.3 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 is established immediately to the south of the mine site, which measures precipitation, temperature, wind speed and wind direction. A mean annual temperature of minus 2.8º Celsius was recorded during 2005-2006 (maximum 13º Celsius, minimum minus 25º Celsius), with annual rainfall of 350 mm.

5.4 Local resources and infrastructure

5.4.1 Local resources

The hamlet of Nahanni Butte is the closest settlement to the Property (90 km by air). It has an airstrip but it is remote and can offer only a limited labour force. Fort Liard and Fort Simpson are the next closest NWT communities and can provide moderate support services such as labour, catering services, some heavy equipment and supplies. Fort Nelson, BC (340 km south of Mine site) is located adjacent to both a railhead and the Alaska Highway and it is able to provide additional support.

5.4.2 Utilities

Electrical power on-site is provided by diesel-powered generators. There are a number of generators available to bring on-line, depending on demand, and include a CAT 3412 750 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. In the past, potable water was extracted from fresh-water wells. A sewage treatment plant exists on-site but is not commissioned at this time.

Previously, extraction up to 1,159 m^3 per day of water was permitted from the Prairie Creek Valley aquifer for process and potable purposes. The original water licence has expired and future water extraction from the Prairie Creek Valley aquifer is now covered by a new Class A Water Licence MV2008L2-0002, which was issued by the Mackenzie Valley Land and Water Board in September 2013. This allows for the extraction of 14,300 m^3 per year of potable water. All process water will be recycled through the water management scheme.

5.4.3 Tailings impoundment area

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

The tailings impoundment design originally formed an integral part of Water Licence N3L3-0932 issued by the NWT Water Board in 1982, which authorized the use of water and the disposal of waste associated with mining and milling operations at the Mine.

Current plans are that the existing large pond, originally intended for tailings disposal, will be reconfigured, relined and recertified to form a two-celled water storage pond. Mine drainage, treated sewage water and waste rock-pile runoff will report to the first cell. Water for the mill process will be taken from this first cell. Excess waters from the first cell will overflow into the second cell. Used water from the Mill will also report to the second cell. The second cell will feed a water treatment plant for eventual discharge to the bed of Prairie Creek.

5.4.4 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.

5.4.5 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. Trailer accommodations and kitchen facilities along with full shop facilities were built to support a 200-man construction crew. A tank farm to store diesel was engineered and constructed onsite. The various buildings and tank farm remain in good condition and only regular maintenance is required to keep them in good order.

5.4.6 Processing plant

A processing plant, also constructed in 1982, consists of a crusher at that time rated to handle 1,500 short tons per day of material, and a grinding and flotation circuit to produce separate lead and zinc concentrates that is rated at 1,000 short tons per day. 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 tails in preparation for a tailings backfill circuit that was never completed. Upon mine closure in 1982, the processing plant was incomplete.

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

The mill building and covered primary crusher feed conveyor (to the left of the plant complex) that extends from the 883 mL portal is shown in Figure 5.2.

Figure 5.2 Mill Processing plant complex, 870 mL Portal in upper left, temporary polishing pond on left

5.5 Underground

5.5.1 Development

Underground development was carried out on the Main Zone between the 1970s and the early 1980s, initially for purposes of exploration and later in preparation for production at a planned rate of 1,000 stpd. In 2006 / 2007, CZN completed a new decline parallel to the Main Zone to facilitate underground Mineral Resource definition drilling. This work included the installation of a new ventilation system and electrical sub-stations, a track upgrade and general rehabilitation. Note that where historical extraction was reported in ‘tons’, they are reported as such.

Main Zone mineralization is currently the primary target for underground mining as it is adjacent to the processing plant and contains the most extensive underground workings. The Main Zone is presently accessed by three adit levels that are referenced to metres above mean sea level and are historically known as the 970 mL, 930 mL, and 870 mL, (or 880 mL). The lowest adit level will be referred to as the 883 mL for mine planning purposes, and throughout this report, as that is the elevation of the first mining block.

These levels contain the following development:

- 970 mL: 220 m of footwall drift with six crosscuts at 30 m intervals. This level is not connected to either the 930 mL or the 870 mL.
- 930 mL: 940 m of trackless footwall haulage drift with 32 crosscuts at 10 m centres consisting of 630 m of vein drifting and 480 m of other development. A number of shrinkage stopes with active drawpoints were developed in the early 1980s that allowed production at a rate of about 500 tpd using trackless methods. Some vein material was mined from 930 mL in 1981 / 1982. This material is currently stockpiled next to the processing plant.
- 883 mL: 610 m of tracked footwall haulage drift, 380 m of vein drifting and approximately 150 m of other development. The portal for the 883 mL is adjacent to the mill feed conveyor.

Limited workshop storage facilities were completed and a mine air heater was installed. Concrete pads for substations were also installed.

In preparation for the 2006 / 2007 underground development and drilling activities, new support was installed at the portal entrance, some timbers were stripped-out, rock bolts were installed where required, and a 75 horsepower ventilation fan was installed at the portal. The fan forces air down a manway to 883 mL where the new decline was excavated.

5.5.2 Production equipment

A significant amount of heavy mobile equipment, suitable for surface earthworks, remains on site from original construction in 1980-1982. This includes D-6 and D-8 dozers, an excavator, an air-track quarry drill, rock trucks, cement mixer trucks, front-end loaders and similar machinery. Some of this equipment is in need of repair. Two 2-yd scoop trams are on site of which one is known to be usable. Some track-bound underground mining machinery is on site, which CZN does not intend to use. One diesel locomotive is in working order and is available for temporary use with 5 tonne Granby cars. CZN plans to contract mine start-up and anticipates that the selected contractor will bring a complete fleet of equipment to the site.

6 History

6.1 Activities and ownership – 1928 to 1970

The original discovery of mineralization on the Property was made by a local trapper in 1928, at what is now known as the Zone 5 showing, a mineralized vein exposed in the 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 Limited in 1966. Cadillac also acquired an 182,590 acre, regional Prospecting Permit.

Between 1966 and 1969, trenching was carried out on a number of 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 collared approximately 325 vertical metres below the surface trenches. A small amount of drifting and crosscutting from the main drive was completed, however only low metal values were encountered. The portal has been blocked by sloughed debris for many years and the drive is inaccessible. Similarly, in Zone 8 a 240 m long underground drive was collared in 1969 and driven north, opposite the Zone 7 portal, to attempt to undercut the surface vein showings exposed in the trench 300 m vertically above the tunnels. A vein was reportedly intersected in the drive but carried only low metal values. The portal in Zone 8 is presently blocked by debris and is inaccessible. Zones 7 and 8 have not been explored in sufficient detail to support estimates of Mineral Resources.

Cadillac's Prospecting Permit expired in 1969 and 6,659 acres (210 claims) were selected by Cadillac and brought to lease. The Property was optioned to Penarroya Canada Limited (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 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 Property, drilled eight holes and subsequently dropped its option in the same year. Cadillac, 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 by Kilborn Engineering Limited (Kilborn), the results of which prompted the decision to put the Mine (then called Cadillac Mine) into production. In December 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 1,000 stpd.

Between 1980 and 1982, extensive mine development took place. Cadillac acquired a 1,000 stpd 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 Mine 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 mine received a Class A Water Licence and Land Use Permit and 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 into bankruptcy in May 1983, after a total of approximately C$64M (1982 value) had been expended on the Property. Thereafter, site maintenance and operations were taken over by Procan, which acquired Cadillac's interest in the Property through bankruptcy proceedings in 1984.

6.3 Ownership post – 1991

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

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

6.4 Historical Mineral Resource estimates

A number of historical estimates have been reported for the Main Zone deposits. The Main Zone in this report refers to Zones 1, 2, and 3, the locations of which are shown in Figure 7.2. Initially these estimates were for the Main Quartz Vein only, but later incorporated the stratabound and stockwork mineralization, as they were discovered. The chronology of historical resource estimates is shown in Table 6.1.

Table 6.1 History of Mineral Resources / Reserve estimates

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

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

The most recent Mineral Resource estimate, completed by AMC in September 2015, is discussed in Section 14.

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 an ore 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 weathered for over 30 years, has been given no value as a Mineral Resource at this time.

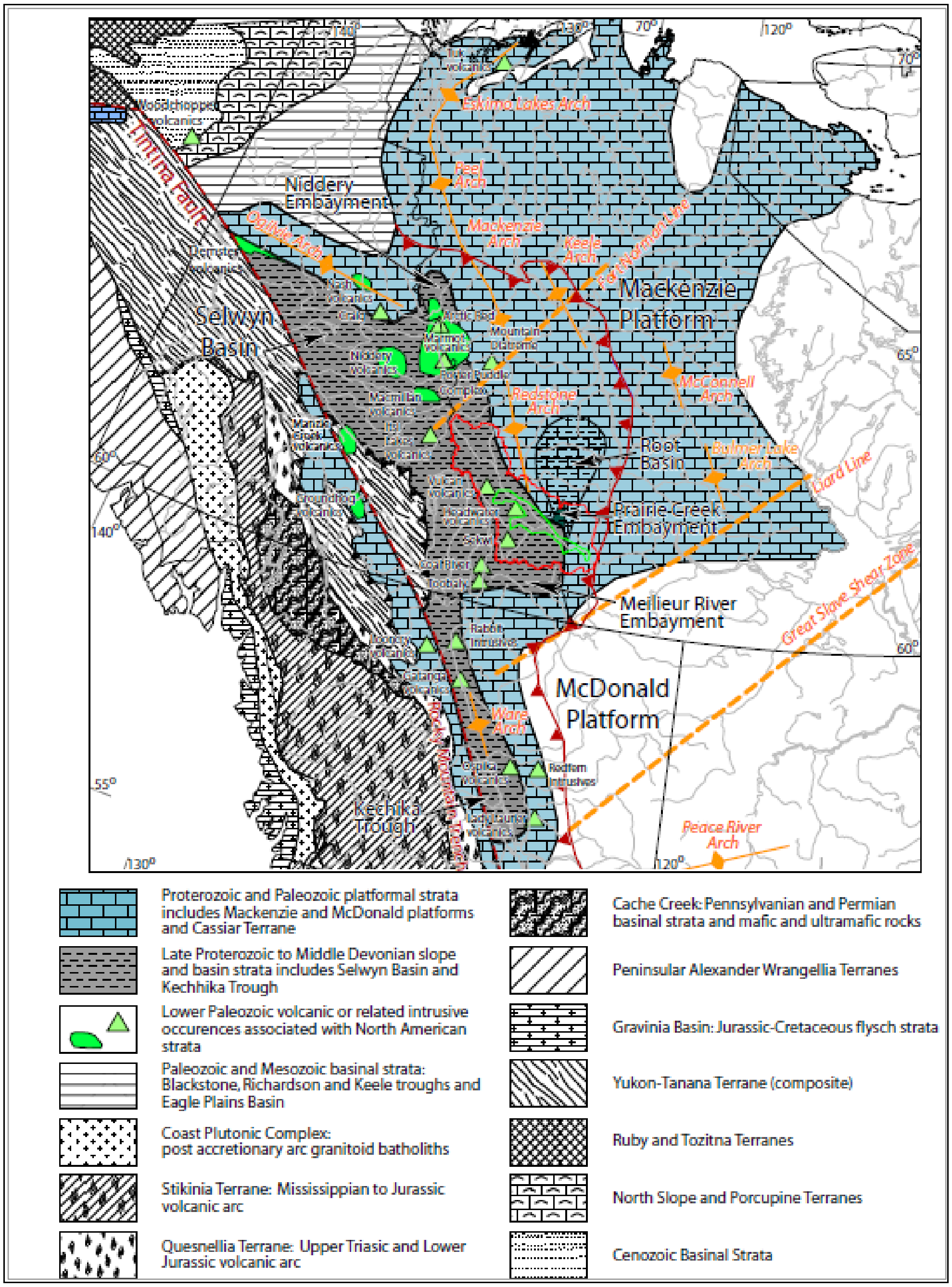
7 Geological setting and mineralization

7.1 Regional geology

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

The Property 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 Prairie Creek regional geology



Source: Paradis, 2007.

During the period from Ordovician to Devonian time, 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 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 thinly 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 steeply-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 basinal sedimentary rocks of the Embayment.

7.2 Property geology

7.2.1 Stratigraphy

The lower Paleozoic strata exposed in the area of the Property 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 Prairie Creek area the Sunblood Formation is unconformably overlain by dolostones of the Whittaker Formation and, west of Prairie Creek, the Sunblood Formation is conformably overlain by basinal 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 Prairie Creek area. The Whittaker Formation is divided into three members: 1) lower dark-grey silty to sandy limestone of middle to upper Ordovician age (muOw1), 2) fine-grained quartzite of middle to upper Ordovician age (muOw2), and 3) laminated, dark-grey fine-crystalline dolostone of upper Ordovician to Silurian age (OSW3) that is the host rock of the stratabound mineralization at Prairie Creek. The Silurian-Devonian age Road River Formation conformably overlies the Whittaker Formation and is comprised of graptolite-bearing shale and argillaceous dolostone. 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 Formation.

The Upper Whittaker has been divided into seven lithological sub-units on the basis of detailed information obtained from diamond drilling (Table 7.1). From stratigraphic top to bottom these sub-units are the Inter-bedded Chert-Dolomite (OSW3-7), Upper Spar (OSW3-6), Upper Chert Nodular Dolomite (OSW3-5), Lower Spar (OSW3-4), Lower Chert Nodular Dolomite (OSW3-3), Mottled Dolomite (OSW3-2), and Massive Dolomite (OSW3-1). 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 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 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 Formation.

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

Table 7.1 summarizes the Prairie Creek stratigraphy. Figure 7.2 is a simplified geological map of the Property area and Figure 7.3 is a representative cross-section through the main mine area showing both stratigraphy and mineralization.

Table 7.1 Summary of the Prairie Creek stratigraphy

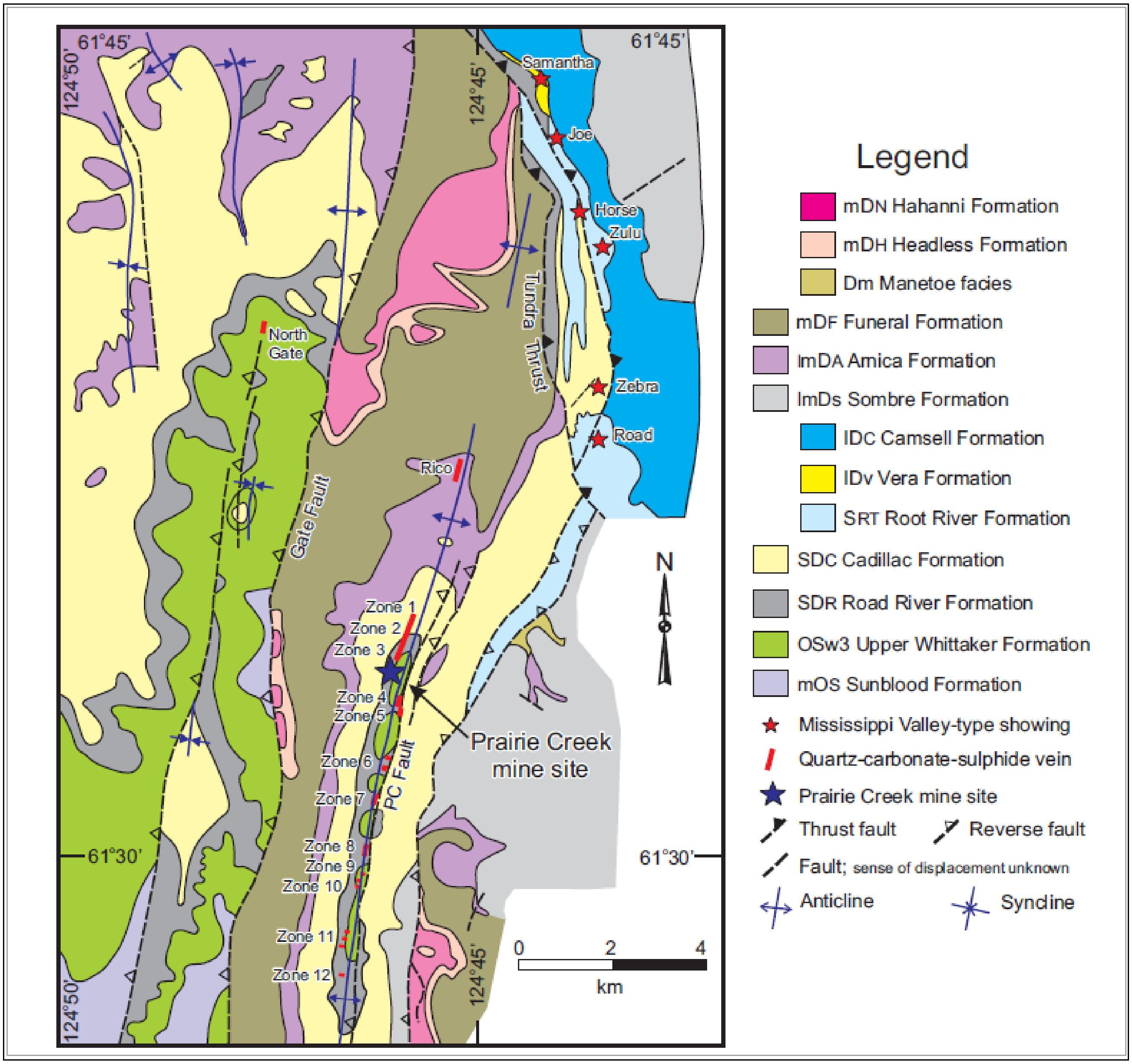
Formation	Code	Thickness (m)	Description
Arnica	ImDAb	200 to 250	Finely crystalline black nodular and banded cherty dolostone and limestone with white quartz-carbonate crackle veining.
Cadillac	SDC	300 to 350	Grey, thinly banded siltstone / shale with minor debris flow.
Road River	SDR	230 to 280	Mid-dark 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 (similar to the Upper Spar unit).
	OSW3-3	40 to 60	Lower Chert Nodule-dolostone unit (similar to 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 SD-1, SD-2 stratabound sulphide 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.

7.2.2 Structure

In the immediate area of the Property, faulting 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 Prairie Creek mineralization. Windows of Upper Whittaker and Road River Formation strata are exposed through the overlying Cadillac Formation along the axis of the PCA.

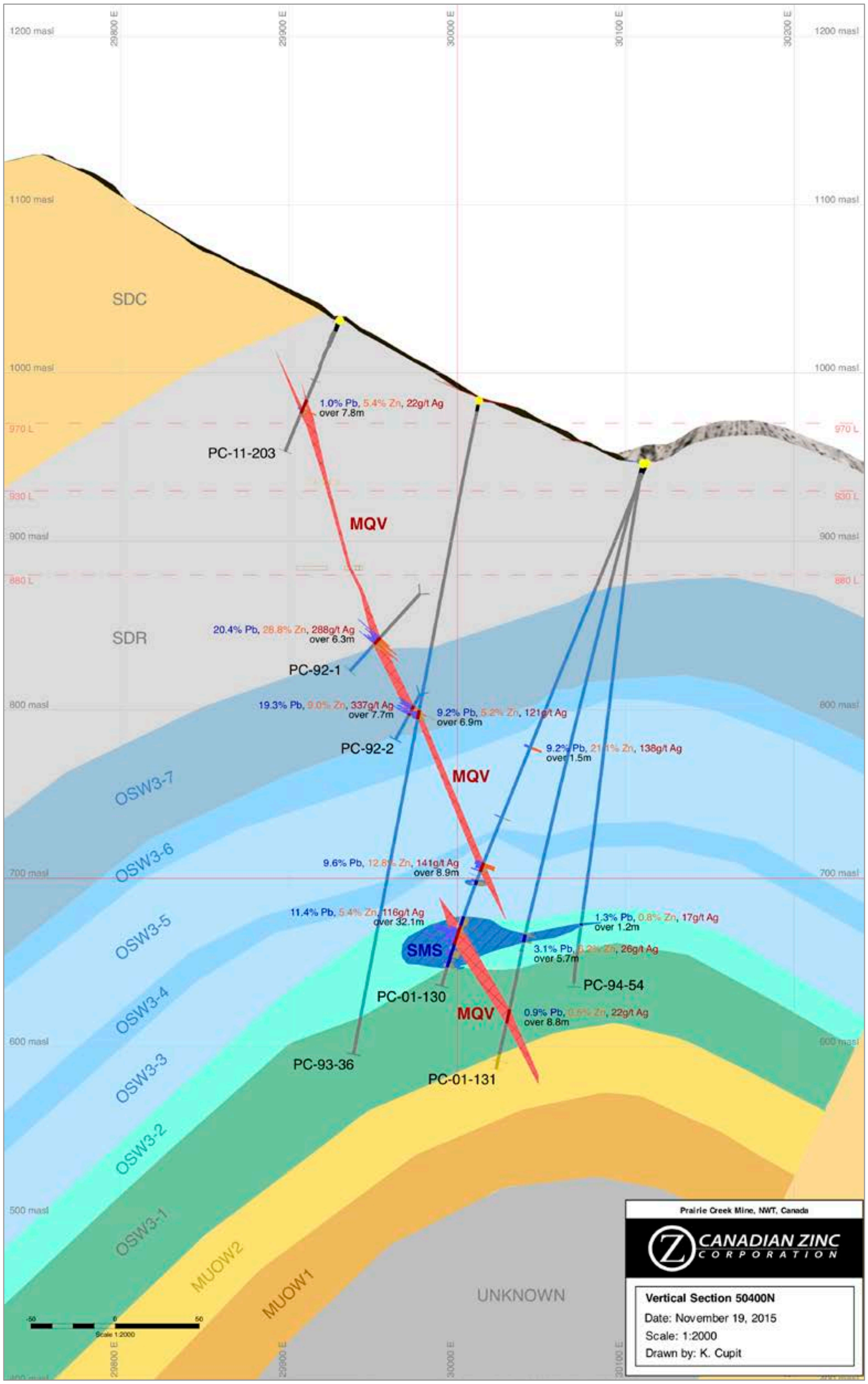
The PCA is bounded to the east by the Prairie Creek Fault and to the west by the Gate Fault. Both are west-dipping thrusts. The Prairie Creek Fault is up to 40 m thick and has a dextral displacement of approximately 1,500 m.

Figure 7.2 Prairie Creek property geology



Source: Paradis, 2007.

Figure 7.3 Simplified vertical cross section of Prairie Creek Main Zone



Source: Canadian Zinc.

Leases in the northern part of the Property straddle the Tundra Thrust, which separates strata of the Prairie Creek Embayment 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 by the Root River, Camsell and Sombre Formations (listed from oldest to youngest). MVT mineralization is hosted in biohermal reefs of the Root River Formation, or facies equivalent.

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

To the west of the Mine area, four contiguous Gate mining leases overlie rock assemblages similar to those found in the PCA (Figure 7.2). Grassroots exploration was completed on this ground to test for mineralization similar to that found in the 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 many base metal occurrences on the Property that can be grouped into four styles of mineralization.

Quartz veins containing base metal mineralization occur in a north-trending, 16 km long corridor in the southern portion of the Property, where the occurrences are exposed on surface. Vein showings were referred to historically as Zones 1 through 12, as shown on Figure 7.2. The Main Zone, which includes the Main Quartz Vein (MQV) and other styles of mineralization, 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.

Stockwork (STK) type mineralization is associated with the MQV and does not appear to represent a true stockwork but rather a series of tensional splays from the MQV. STK mineralization is exposed underground in the 880 mL and has been intersected in drillholes.

Stratabound (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.

Mississippi Valley type (MVT) showings in the northern section of the Property 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 Main Quartz Vein (MQV) mineralization

Vein mineralization developed within the 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 comprises massive to disseminated galena and sphalerite with lesser pyrite and tetrahedrite-tennantite in a quartz-carbonate-dolomite matrix. Secondary 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 between less than 0.1 m and 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 strike length of 2.1 km. The MQV trends approximately north-south and dips between vertical and 40º east (average dip is 65º 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 m above mean sea level.

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. As well, the vein does not appear to be well developed in the shales of the Cadillac Formation lying stratigraphically above the Road River Formation.

Figure 7.4 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. The presence of an en échelon vein structure could offer a simple explanation for the apparent off-sets between the various vein showings.

7.3.2 Stockwork Mineralization (STK)

Towards the end of 930 mL at Crosscut 30, a series of narrow (average 0.5 m wide), massive sphalerite-galena-tennantite veins are developed at about 40º to the average trend of the MQV. These sub-vertical veins range in thickness from 0.1 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 stockwork, although it does not appear to represent a true stockwork but rather a series of splays off the MQV. To date, STK-style mineralization has only been located in the immediate area surrounding the exposure in the 930 mL workings and through local diamond drilling.

Figure 7.5 Stockwork Mineralization showing separate distinct high-grade sub-vertical veins in 880 mL

7.3.3 Stratabound Mineralization (SMS)

Stratabound mineralization was discovered by CZN 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 3 km 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, 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 and half as much galena as, but substantially more iron sulphide / pyrite than, typical vein material. Fragments of SMS material occur in vein material indicating that the SMS predates the vein material.

The majority of SMS mineralization occurs within the Mottled Dolomite unit of the Whittaker Formation (OSW3- 2, see Table 7.1), which the mineralization totally replaces without any significant alteration. SMS sulphides are developed close to both the vein system and the axis of the PCA and are probably older than the vein mineralization (Figure 7.3). An apparent thickness of 28 m of SMS mineralization has been intersected in MQV drillholes, approximately 200 m below 883 mL.

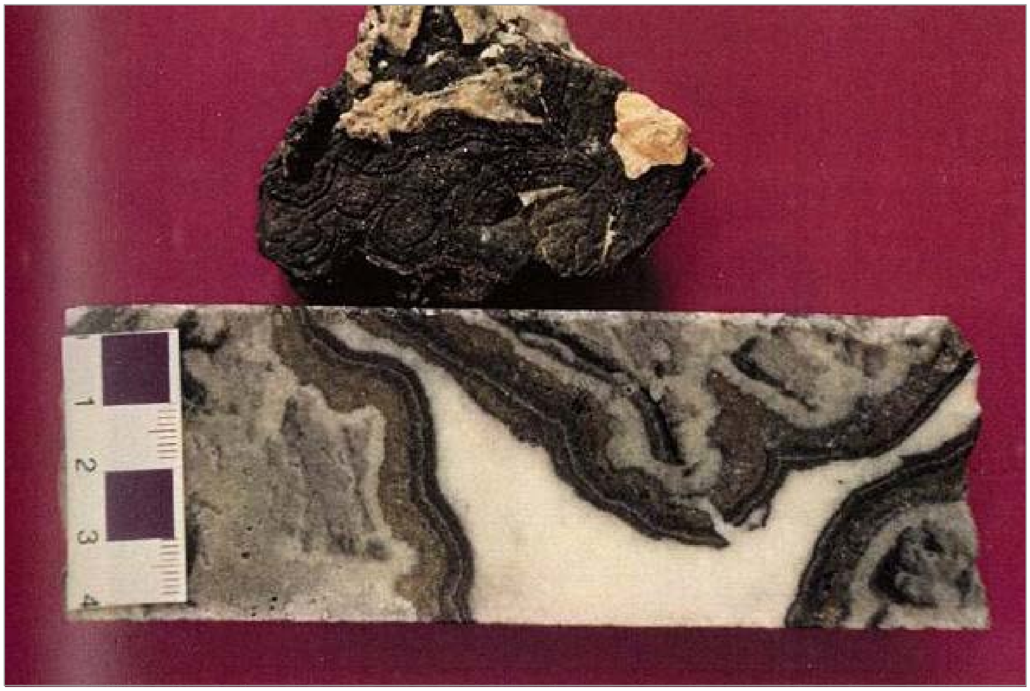
Figure 7.6 Stratabound Mineralization showing massive sphalerite and pyrite in drill core

7.3.4 Mississippi Valley Type Mineralization (MVT)

MVT mineralization found on the Property 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.

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



8 Deposit types

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

- Hydrothermal Quartz Veins (MQV)
- Stockwork (STK)
- Stratabound (SMS)
- Mississippi Valley type (MVT)

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

8.1 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 occur as sets of parallel and offset veins. Individual veins vary from centimetres 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. Veins may widen to tens of metres in stockwork zones.

Texture / structure: Compound veins with a complex paragenetic sequence are common. A wide variety of textures exist, including cockade texture, colloform banding and crustifications, and locally druzy. 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.

Ore Mineralogy: (Principal and subordinate): Galena, sphalerite, tetrahedrite-tennantite, other sulphosalts including pyrargyrite, stephanite, bournonite and acanthite, native silver, chalcopyrite, 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 chalcopyrite 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.

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.

8.2 Stratabound Mineralization (SMS)

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 1-2 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 (Navan), thin laminated, graded and cross-bedded sulphides, with framboidal pyrite, occur above more massive sulphide lenses.

Ore Mineralogy: (Principal and subordinate): Sphalerite, galena; barite, chalcopyrite, pyrrhotite, tennantite, sulfosalts, tetrahedrite, and chalcopyrite.

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. Mn halos occur around some deposits; silicification is local and uncommon. Fe in iron formations is distal.

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.
- 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.

8.3 Mississippi Valley Type (MVT)

Salient characteristics of MVT mineral deposits are presented below (modified after Alldrick 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 peneconcordant as planar, braided or linear replacement bodies. May be discordant in roughly cylindrical collapse breccias. Individual ore 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.

Ore Mineralogy: (Principal and subordinate): Galena, sphalerite, barite, and fluorite. Some mineralization contain 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 finely 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 ore 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.

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°- 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:

- Compaction-driven fluid flow is generated by over-pressuring of subsurface aquifers by rapid sedimentation, followed by rapid release of basinal fluids.
- 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 with 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.
- Reduced sulphur: Base metals are transported together with reduced sulphur. Precipitation is brought about by change in pH, dilution, and / or cooling.

9 Exploration

Table 9.1 summarizes all the work carried out by CZN since 1991. A full discussion with tables of results is contained in earlier reports that are referenced in Section 27. Drilling is further discussed in Section 10.

Table 9.1 Summary of exploration work, 1992 to 2015

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. 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	3,393	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, 696	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	5,926	Deep drilling in Casket Creek and proximal to resource drilling, Geophysical Gravity & EM surveys, LIDAR survey of property.
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.
Well	1	183	Hydrology well.

9.1 Channel sampling

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, and 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 between 1980 and 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 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; 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 hangingwall 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.

The weighted average grades (by estimated tonnes) of the rounds excavated in MQV compare reasonably well with weighted average for the channel samples: Rounds: 19.0% Zn, 16.4% Pb, 250 g/t Ag, 0.5% Cu; Channels: 21.3% Zn, 17.0% Pb, 413 g/t Ag, 1.2% Cu, (all samples); Channels: 20.6% Zn, 15.4% Pb, 302 g/t Ag, 0.7% Cu (excluding one outlier). No documentation was seen by AMC describing the sampling, which in some reports is referred to as ‘chip sampling’.

In 2015, CZN 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 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.

9.2 Gate mining leases

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 favourable geology on the Gate 3 Mining Lease.

During 2007, CZN carried out a helicopter-supported diamond drill exploration program to further test the previously defined soil anomalies within the Gate group and Zones 8, 9, and 11. The results from this program returned very few significant mineral intersections.

10 Drilling

10.1 General

The metres drilled during the drill programs completed since 1992 are summarized in Table 10.1.

Table 10.1 Summary of diamond drilling carried out by CZN / SARC

Year	DDHs	Length
1992	22	6,322
1993	31	8,432
1994	31	11,113
1995	36	10,082
2001	5	1,711
2004	27	5,944
2006	19	2,393
2007	53	11,141
2010	4	2,694
2011	30	5,926
2012	11	5,628
2013	5	1,472
2015	21	5,548
Well	1	183
Total	**296**	**78,587**

Approximately 19,244 m of drilling was carried out on the Property prior to 1992. None of those drill results has been used in the current Mineral Resource estimate.

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 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.

10.2 2010 drill program

During 2010 three holes with an aggregate length of 2,703 m were drilled in the Casket Creek area approximately 1.7 km north of the 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 into the wedged hole, 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 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 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 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.

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

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 similar type mineralized structure to the MQV in the main zone does exist 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.

10.4 2012 drill program

Nine holes with an aggregate length of 3,446 m were drilled in 2012. Eight of these tested the Main Zone and one (PC12-213) 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.

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
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	2059.00	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

10.5 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. At the same time, hole PC-13-220 was also designed to intercept projections of previously-defined vein and STK mineralization within the upper parts of the hole.

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 likely 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

10.6 2015 drill program

During 2015 CZN drilled 21 holes from the 883 mL decline as a series of vertical fans, all with an approximate azimuth of 290°. These holes were intended primarily to test the MQV Zone but also tested the STK and provided 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 indicates that the STK 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 prior to the emplacement of the MQV. Representative assay results from the 2015 drill program are shown in Table 10.5.

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
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

10.7 Drilling procedure

10.7.1 Drills

Since 1992, surface diamond drilling has been carried out using skid-mounted Longyear Super 38 drills owned by CZN to recover NQ diameter (47.6 mm) core. Core size was reduced to BQ size (36.5 mm) where difficult downhole conditions are encountered. In 2010 a new, higher-capacity HTM-2500 diamond drill rig was airlifted to the property 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 CZN drills. In 2007 Titan Drilling Limited of Yellowknife, NWT was contracted to carry out a surface drilling program using a Boyles helicopter-portable drill to recover NQ diameter (47.6 mm) 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, CZN 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 CZN drills. More recently in 2014-2015, Procon was contracted to manage the underground program and subcontracted DMAC Drilling Ltd of Aldergrove, BC to carry out the diamond drilling.

Figure 10.1 HTM-2500 skid-mounted diamond drill rig at Casket Creek

10.7.2 Field procedures

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

Drilled core is placed in wooden boxes with depth markers placed in the boxes at the beginning and end of each drilling run. The markers are labelled by the drillers in feet or metres, to correspond with units used for the drill rods. Full drill core boxes are individually sealed with wooden lids that are securely nailed in place to prevent any spilling or shuffling during transit of the boxed core.

10.7.3 Surveying

The collars of completed surface drillholes are surveyed by qualified surveyors using a transit. Both UTM coordinates and local mine grid co-ordinates are calculated. The collars of underground holes are surveyed using mine grid coordinates that are 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 to the present, downhole survey measurements have been completed using a

Reflex EZ-Shot and are taken every 15 m instead of every 60 m as was previously the case. The completion of individual surveys is dependent on downhole conditions.

Raw survey data is processed by the software that accompanies the survey tools. Output such as Depth in Feet, Depth in M, Azimuth, Dip, Magnetic Field Strength and Magnetic Dip are captured from the processed data and copied to a master spreadsheet of all drillhole surveys. The spreadsheet is then used to prepare traces of the drillholes in three-dimensions, using Geovia GEMS software. Paper and electronic data files are stored at CZN's head offices in Vancouver, BC.

10.7.4 Core logging

All drill core logging is carried out at the Mine site in a secure facility. Received core is laid out and a quick assessment is done to verify that all the boxes are intact, confirm the drillhole identification data and that the drillers' depth markers are in good order (i.e. drill core mixing or displacement has not occurred during transport). If disruption is identified (which rarely occurs), the core is "fitted" together and the depth markers are placed at the appropriate points by means of direct measurement and identification of the start / end points of successive drilling runs. The depth markers are then converted, if necessary, to metre measurements and aluminium tags are stapled to each box-end noting drillhole number and the box-start and end depths. Drill core recovery is calculated by comparing the drilled length with the actual core length between depth markers. Rock Quality Description (RQD) is calculated from the sum of the length of full-diameter drill core pieces over 10 cm, divided by the total length of the run. Rock mass ratings are then calculated for 10 m envelopes around individual mineralized intersections, using industry standard methods.

All drill core is 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 lithology, alteration, oxide / sulphide mineralization and structure, are compiled and recorded. Prior to core photography, which is done for two or three boxes at a time, sample intervals are marked on the core by the geologist responsible for that hole. Core photographs are archived in CZN's electronic files.

Prior to 2011, core logs were transposed into Excel spreadsheet format for copying into a central database. Starting in 2011, CZN switched to inputting data directly to a computer, 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 (overlapping intervals, missing intervals and duplicate samples) are performed via automated software checks nearer to the end of the season, and problems are resolved as they are identified, referring to the core as needed.

10.7.5 Core recovery

Core recoveries have been consistently recorded since 2006. Average recoveries are approximately 80% for the MQV and 97% for the SMS mineralization. No recovery information was provided for the STK mineralization. Intervals of poor recovery in the MQV are associated with shearing and faulting. Rates of recovery were based on drill run lengths; where poor recoveries were encountered during sampling, sample lengths were increased to ensure sufficient material for proper analyses was obtained.

10.7.6 Bulk density

No bulk density measurements have been collected from the drill core recently. 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. This is discussed further in Section 14.2.4.

10.7.7 Drilling results

In addition to the results tabulated in Table 10.4 and Table 10.5, a cross section shown as Figure 14.7 demonstrates the general intersection angle of the surface drillholes with the MQV. Underground drillholes have an intersection angle which is generally near normal to the planar vein. As the SMS is sub-horizontal, the surface holes have an intersection angle which is near true width.

11 Sample preparation, analyses, and security

11.1 Chain of custody

11.1.1 Underground channel samples

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

11.1.2 Drill core samples

Drill core is boxed at the drill rig by the drillers' helpers who securely nail a wooden lid onto each filled drill core box. Underground core is transported by the drillers to the portal. Both underground and surface drill core boxes are picked up by a Company geologist and then transported in pick-up trucks, driven by a CZN geologist, to the secure, on-site drill core logging and sampling facility.

The sealed drill core boxes are laid out 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 will mark appropriate sample intervals on the drill core of approximately 1 m in length. After logging, core boxes are photographed three at a time and then cross-piled outside or set aside for sample processing within the core shack. A geotechnician will then cut the marked sample intervals of drill core in half with a diamond saw blade, placing half of the material into a sample bag and the other half back into the core box. A tag is placed into the sample bag listing only an ID number, and another tag with the same ID number is stapled into the core box at the start of the sampled interval for later reference if needed.

11.1.3 Sample sacks

All drill core logging and sampling is supervised by a senior geologist. Only authorized personnel or those accompanied by an authorized person are allowed into the core shack. The shed is locked at all times when geologists or their assistants are not present.

Individual drill core sample bags are sealed with plastic ties and placed in rice sacks (50 pounds per bag). Individual rice sacks, containing either channel samples or drill core sample bags, are labelled with the shipping address and requisition sheets are inserted. The sacks are securely fastened and then stored in the secure, on-site drill core and sampling facility, prior to their transport off-site.

11.1.4 Transport

Samples are air-freighted in charter aircraft from the Mine site to Fort Nelson, BC. Prior to 2011, samples were transported by Greyhound bus to the Acme Labs assay laboratory in Vancouver, BC. From 2011 onwards, samples were delivered to the AGAT Laboratories sample drop-off facility in Fort Nelson, and then entered their sample preparation and assay chain.

11.1.5 Drill core storage

Boxes containing the main mineralized drill core intersections are stored in trailers adjacent to the core shack facility (Figure 11.1) to ensure their security, to facilitate their ready access, and to protect the core from weathering. Boxes containing unmineralized drill core are square-piled in stacks in the core storage area next to the boneyard near Harrison Creek.

Figure 11.1 Stored unmineralized drill core at Harrison Creek site

Figure 11.2 Stored mineralized drill core intersections at main site

11.2 Assay method

Acme Labs (ISO 9001-2000 accredited) has carried out the majority of the sample assaying since CZN's first involvement with the Property in 1992, and was used up until 2011. From 2011 onwards, sample assaying has been conducted by AGAT Laboratories (ISO/IEC 17025:2005 accredited).

11.2.1 Sample preparation

Samples are sorted and inspected for quality of use (quantity and condition); wet or damp samples are dried at 60º Celsius. Samples are then crushed to 70% passing ten mesh (2 mm), homogenized, riffle split (250 g sub-sample) and pulverized to 95% passing 150 mesh (100 microns). The crusher and pulverizer are cleaned by brush and compressed air between routine samples. A granite wash is used to scour equipment after high-grade samples, between changes in rock colour and / or at end of each file. Granite is crushed and pulverized as the first sample in each sequence and each granite sample is carried through to analysis to monitor background assay grades.

11.2.2 Assay procedure

The grades of silver, copper, lead, and zinc, as well as 30 additional elements, are determined for all samples by aqua regia digestion followed by an ICP-ES finish. Lead and zinc oxides are assayed by ammonium acetate leach and AAS finish.

11.3 QA/QC procedures

CZN submits Quality Assurance / Quality Control (QA/QC) blanks, duplicates and standards for analysis with the regular samples to ensure accuracy of the analysis. Blanks, duplicate samples, or standards are inserted on average after approximately 20 drill core samples, and are randomly pre-designated to be inserted up to five samples ahead of or behind this mean value in order to reduce predictability of QA/QC sample occurrences in the sample stream.

11.3.1 Blanks

The blank material used is common landscaping gravel.

11.3.2 Duplicate samples

Duplicate samples comprise half of the core halves remaining after normal splitting and sampling: the half core is split longitudinally using a diamond saw; the remaining quarter core is returned to its core box for storage and reference and the quarter core sample is placed in a sample bag for transport and assaying. The same procedures as those outlined for half drill core samples are followed as regards labelling, storage and transport of duplicate samples.

11.3.3 Standard samples

CZN has 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 CZN 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 grams 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), Actilabs 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).

The remainder of the packaged standards was returned to CZN for insertion into the sample stream, as earlier outlined. Certificates for each of CZN's three standards (as compiled by Smee) are available.

11.3.4 Check samples

As an additional quality control measure, a number of check samples were selected from the 2015 drill program and forwarded to Met-Solve Laboratories Inc. 27 sample pulps were chosen using a random number generator on a list of samples that excluded duplicates, standards and blanks. Using the same analytical techniques as AGAT Laboratories, Met-Solve Laboratories returned values that were within acceptable ranges for Pb, Zn, Ag, and Cu from those obtained by AGAT Laboratories.

11.4 Conclusion

AMC reviewed QA/QC data for four sampling campaigns: 2006 – 2007 underground drilling, 2007 surface drilling, 2011 – 2013 surface drilling and 2015 underground drilling. Collectively these programs included 21 duplicate pairs, 82 blank samples, and 120 standards comprising 36 CZN Standard-1, 43 CZN Standard-2 and 41 CZN Standard-3 for a total of 223 control samples equal to approximately 6% of the samples collected for analysis. The temporal distribution of these control samples is set out in Table 11.1.

Table 11.1 Prairie Creek QA/QC control samples by type and year

Year	**Blank**	**Duplicate**	**STD CZN-1**	**STD CZN-2**	**STD CZN-3**
2006-2007	47		10	19	15
2007	6	6	4	1	
2011-2013	18	6	7	8	4
2015	11	9	15	15	22
Total	**82**	**21**	**36**	**43**	**41**

The following observations were made of the various control samples.

Four blanks exceeded the background values of lead and zinc. All were from the 2006-2007 underground program and all were immediately preceded by samples containing high values of lead or zinc or both. Analytical data for the samples following the contaminated blanks is available for only two of the four; one is of sufficiently high-grade that the contribution from the level of contamination in the blank would have been trivial and the other sample is of very low grade and not obviously contaminated.

The duplicate samples are in general, although not always close, agreement. However, given the coarse nature of much of the mineralization, close agreement between split samples should not necessarily be expected.

Most lead, zinc and silver assays of standard samples fell within two standard deviations of the expected mean; four lead assays (5%) exceeded three standard deviations and all of the zinc and silver assays were within three standard deviations.

The QP believes that the data collection and handling followed normal industry practice and the data is fit for purpose of Mineral Resource estimation. However, although QA/QC samples were inserted during pre-2010, drill programs and the results have been observed in the assay certificates, it is not clear if any analysis of the data was carried out or whether any remedial action was taken for out-of-bounds results, if any. This deficiency has been remedied in the programs that have taken place since 2010.

12 Data verification

Raw and final assay data undergo final verification by a British Columbia Certified Assayer, who signs all Analytical Reports before they are released to CZN.

12.1 Historical drill core data

None of the assay data from surface holes completed prior to the inception of CZN's involvement with the Property in 1992 was included in any recent Mineral Resource estimates because of uncertainties relating to their collar positions, a lack of downhole surveys, poor recovery factors and / or a lack of laboratory certificates.

12.2 Pre-AMC work on post-1991 data

MRDI verified the 1992 to 1998 assay database as part of its January 1998 resource estimate program. The integrity of assay data transfer and organization into Excel spreadsheets by CZN for the 2001 to July 2007 assay data (i.e. all data post-MRDI's Mineral Resource estimates, up to and including the 2006 / 2007 Phase I underground drilling program), was verified by MineFill by means of manual and digital comparison of original laboratory datasets and CZN's Excel spreadsheet database.

The results of MineFill's data verification program are summarized in Table 12.1. Only verified assay data was used for subsequent Mineral Resource estimates.

Table 12.1 Summary of results, MineFill July 2007 data verification program

Year	Number of assays	Verified	Corrected
2001	91	85%	15%
2004	143	97%	3%
2006	201	76%	24%
2007	778	95%	5%

12.3 AMC verification

AMC reviewed the data that was verified by MineFill and is satisfied with the results. AMC performed a random check of approximately 5% of the drillhole assays that have been generated since the 2012 verification program by comparing assay values in the database against the laboratory certificates. No discrepancies were found.

Data was also verified in GEMS software. The verification procedure included checks for duplicates, overlaps, sample intervals beyond the end of the hole, and collar coordinates. Collars, down-hole surveys, assays, composite, and lithology tables were verified. No errors were found.

The QP considers that the data is fit for the purpose of estimating a Mineral Resource.

13 Mineral processing and metallurgical testing

13.1 Introduction

There are three principal types of mineralization that represent the resource from the Prairie Creek Project. These consist of the main quartz vein (MQV), stockwork (STK), and stratabound massive sulphide (SMS). The MQV is described as the principal resource zone for the project.

There has been a considerable amount of historical metallurgical test work performed beginning in the 1960s and continuing sporadically up to the 1980s and beyond, when more focused test programs were initiated, including those undertaken as recently as 2016. This historic test work was performed on MQV and SMS mineralization. For MQV the historic testing had been limited to samples readily available from the bulk sampling areas in mine adits closer to surface. These contained more highly oxidized portions of this zone. 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.

Among the recommendations of the related pre-feasibility engineering study based on the historical metallurgical test work was to evaluate processing for other areas of the resource. This included obtaining samples from deeper, less oxidized, portions of MQV, as well as from the STK mineralization. Consequently, laboratory test work was performed over the first half of 2017, and focused on confirming the mineral processing response of these previously untested areas. These samples were obtained from drill core, originating from the 2015 exploration program. The test work also investigated process methods that could further optimize the flotation procedures and flowsheet.

A summary of the historical and 2017 metallurgical test programs is separately discussed below, respectively as Sections 13.2 and 13.3.

13.2 Historical test work

There has been a significant amount of test work done by a variety of laboratories, as well as related metallurgical reviews done historically on the Prairie Creek project. This historical testing was conducted on the SMS material, and on areas of the MQV zone where samples were obtained closer to surface, in the pre-existing underground mine workings. Historic test work was summarized during pre-feasibility 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	✓	-	-	✓

Kamloops = Metallurgical Services Ltd / Kamloops Research & Assay Laboratory Ltd. / G&T Metallurgical Services Ltd.
De Randt Corp = De Randt Corp Mineral Technologies Group, Division of De Randt Corp Enterprises.
Terra = Terra Mineralogical Services.
Harris = Harris Exploration Services.
Hazen = Hazen Research Inc.
Cominco = Cominco Exploration Research Laboratory.
Lakefield = Lakefield Research of Canada Limited.
SGS Lakefield = Lakefield Research Limited.
CSMRI = The Colorado School of Mines Research Institute.
UBC = University of British Columbia.
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 1980s. 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. 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 (DMS), followed by flotation. Other laboratories performed additional programs as described in the following sub-sections.

13.2.1 Mineralogy

13.2.1.1 Main Quartz Vein (MQV) Mineralization

A number of different laboratories included mineralogical studies of the head samples, as well as some of the concentrates from the MQV. An examination by Terra in 2006 was done on material collected from the 930 m and 883 m level adits, showing the principal lead mineral was galena accounting for about 60% of the lead, followed by cerussite (lead carbonate) at 30%, and remainder being anglesite (lead sulphate), along with trace lead sulfosalts. Most of the mineral textures were described as coarse-grained and simple, implying that lead-bearing minerals would liberate well at a coarse primary grind. A minor amount of galena-sphalerite, galena-quartz, and cerussite-dolomite was present locally at finer more complex textures, which could require a finer grind size to liberate the target minerals.

For the MQV, sphalerite was found to be the main zinc carrier, accounting for 77% of the zinc mineralization, and occurring mostly as liberated grains, or forming simple coarse-grained intergrowths, predominantly with galena and quartz. The balance is principally accounted for by smithsonite, is which is commonly intergrown with sphalerite, but is also noted with finer mineral intergrowths with the dolomite.

The main copper carrier is a combination of tetrahedrite and azurite / malachite; minor to trace amounts of covellite and enargite were also identified. Non-opaque gangue is mainly comprised of quartz and dolomite. Dolomite can be intergrown with smithsonite and / or cerussite, as well as quartz.

Follow-up mineralogy by G&T laboratories undertook mineral composition for the MQV by weight, identifying ~25% sphalerite, ~18% galena, 1.6% each of tetrahedrite and pyrite, with 54% gangue minerals. The corresponding liberation is provided in the following tables.

Table 13.2 G&T: MQV sulphide mineral liberation

Mineral class	Mineral distribution (%)				
	Tetrahedrite	Galena	Sphalerite	Pyrite	Gangue
Liberated	77	83	82	65	95
Binary with Tetrahedrite	-	1	1	<1	<1
Binary with Galena	4	-	3	<1	1
Binary with Sphalerite	3	7	-	7	2
Binary with Pyrite	1	<1	1	-	1
Binary with Gangue	5	3	10	22	-
Multiphase	10	6	3	6	1

An examination at de Randt Laboratories consisted of mineralogical analysis of fourteen MQV rock samples. The study showed that most of the metals occur as sulphide minerals, although some carbonates of zinc, lead, and copper were also found. The metals in the carbonates only accounted for a small portion of the total metals. Most of the value minerals occurred in either coarse liberated sulphide grains or sulphide middling particles, it was suggested that a fine grind size may be required to separate effectively the metal bearing minerals from each other. No native silver was observed nor was it detected by scanning electron microscopy.

Hazen's 1997 study was on a MQV bulk head sample, as well as corresponding flotation concentrates. The assay of the head samples is provided in Table 13.3 indicating a high portion of oxidized lead and zinc.

Table 13.3 Hazen sample – MQV head assay

Element	Total	Oxide
Pb	9.70%	6.70%
Zn	6.10%	3.90%
Cu	0.30%	-
Hg	0.04%	-
Ag	1.76 g/t	-
Au	<0.07 g/t	-

The Hazen results indicated that main value minerals are galena, sphalerite, cerussite, and smithsonite, with lesser amounts of tetrahedrite-tennantite, minor amounts of pyrite, and traces of chalcopyrite and covellite. Gangue minerals were quartz and dolomite. Oxide minerals showed great diversity in occurrence and texture, varying from independent, liberated single crystals and crystal aggregates, through various stages of sulphide replacement, and also complex intergrowths with quartz and dolomite.

Examination of gravity separation products, at a grind size < 10 mesh, showed the target minerals to be generally coarse and mostly liberated from the quartz, although finer sub-hedral to euhedral quartz inclusions (typical size

20 to 80 µm) are fairly common, particularly in galena. Most of the sulphides were liberated from each other. However, mutual intergrowths typically ranging from about 50 to 150 µm were relatively frequent.

Occurrence of mercury in flotation concentrates was found to be as follows:

Zinc flotation concentrate: mercury occurs sub-microscopically in sphalerite. Mercury concentrations vary from 1,500 to 3,300 ppm in individual particles.

Lead flotation concentrate: mercury occurs in tetrahedrite-tennantite, ranging from 800 to 4,200 ppm in individual particles.

Copper flotation concentrate (low copper grade): mercury occurs in sphalerite and tetrahedrite-tennantite (typically 1,600 to 2,400 ppm Hg, and 1.0 to 1.3% Ag).

A study by Cominco on MQV sulphide flotation tailings concluded that high losses of lead and zinc to tailings were primarily as carbonates, such as cerussite and smithsonite. Further study by Cominco on lead and zinc concentrates showed that about 75% of the galena in the lead concentrate occurred as liberated grains, while 50% of the pyrite and 25% of the sphalerite were liberated. The zinc concentrate contained 95% sphalerite, 4% pyrite, and 1% galena. Approximately 80 to 85% of the sphalerite occurred as liberated grains, while 50% of the pyrite was in a liberated form; most galena was associated with other minerals. It was indicated that mercury and cadmium may occur in the sphalerite lattice, and arsenic and antimony are associated with tennantite.

CSMRI mineralogical work found the liberation at approximately 80% for cerussite and smithsonite in the 100 to 150 mesh particle size fraction. As a part of 1982 pilot plant test work, CSMRI conducted mineralogical examinations on various products. The results indicated that all samples contained both galena and sphalerite, with most having detectable cerussite and smithsonite. Tetrahedrite was in all samples and with silver concentrations greater than 850 g/tonne.

13.2.1.2 Stratabound (SMS) Mineralization

Work on SMS was limited in scope, although some samples were obtained from drill core. The SMS sample studied by G&T outlined pyrite as the dominant sulphide, and accounted for most of the iron sulphides present. Some marcasite was also observed, but no appreciable pyrrhotite was detected. Galena was present as well-formed crystals and accounted for virtually all of the lead. Sphalerite was dominant for zinc. Trace quantities of chalcopyrite and tetrahedrite were seen in intimate association with the galena, and to a lesser extent with sphalerite. The interstitial iron content of the sphalerite was estimated at approximately 4%, indicating that the maximum zinc concentrate grade that could be produced from the sphalerite would be about 63%. No lead or zinc oxides were observed, although very small amounts of oxides were detected by chemical assay techniques.

The non-sulphides in the sample consisted of quartz and apparently colloform silica, together with some calcite and dolomites. The mineral composition of the sample generated by two sets of modal data is shown in the following table.

Table 13.4 G&T: mineral composition of SMS

Sample	Mineral composition				
	Galena	Sphalerite	Pyrite*	Tetrahedrite	Non-sulphide Gangue
Flotation Feed – 80% 79 µm	7.4	17.8	40.2	<0.1	34.6
Flotation Feed – 80% 44 µm	7.5	17.0	41.7	<0.1	33.8

Mineral liberation data indicated that a fine association between galena and sphalerite, with binary assemblages between galena and sphalerite, pyrite and gangue observed even in the sub-sieve fractions. Approximately 20% of sphalerite was as binary composites with pyrite, displaying complex structures with multiple, small pyrite inclusions and adhesions on larger sphalerite particles. Typically, these composites had equal pyrite and sphalerite weights, assaying about 30% Zn and 25% Fe. Multiphase particles containing near equal amounts of sphalerite and pyrite, with smaller and highly variable galena and gangue contents, accounted for approximately 5% of the lead and 5% of the zinc in the flotation feed stream.

G&T's testwork showed the stratiform sample as a mixture of sulphides in a dolomite host rock. In relative abundance order, dominant sulphides were pyrite, sphalerite, and galena. Trace tetrahedrite group minerals and minor amounts of arsenopyrite were also detected. At a grind level of 80% passing 50 µm, more than 80% of the galena, pyrite, and non-sulphide gangue, and about 65% of sphalerite were liberated. At least one third of the sphalerite in the feed stream was locked, mostly in binary and multiphase composites rich in non-sulphides. Only 3% of the sphalerite was locked with galena in structurally simple binary assemblages, containing about 50% galena by weight.

13.2.2 Comminution

Six Bond ball mill work index (BMWi) tests were performed on MQV material in 2007 by SGS Lakefield. The results showed a moderately soft work index of 8.5 to 11.1 kWh/tonne, averaging 9.7 kWh/t. The hardest material was for a composite with higher oxidation than the others. The results are shown in the following table with asterisks indicating if the test was performed on sink product obtained from heavy media separation.

Table 13.5 SGS Lakefield – MQV bond ball mill work index

Sample	Screen aperture (mesh)	Bond work index (kWh/t)
Master Composite – w/HLS*	150	8.5
Master Composite + Dilution	100	10.2
Low Oxidation Composite – w/HLS*	150	8.8
Low Oxidation Composite + Dilution	150	8.7
High Oxidation Comp – w/HLS*	150	11.1
High Oxidation Composite + Dilution	150	10.0

* 2.8 SG Sink + Fines; HLS: heavy liquid separation.

The findings were supported by G&T testing an MQV sample indicating 9.7 kWh/tonne at a closing sieve size of 105 µm (150 mesh). An SMS sample that was tested gave a measurement of 9.2 kWh/tonne.

13.2.3 Dense media separation

Initial dense media separation (DMS) testing was performed by Hazen in 1997 for MQV material that contained a relatively high proportion of oxide lead and zinc minerals. The material was crushed to different particle sizes, resulting in 55% - 60% of the material being rejected. The resulting metal losses with corresponding weight rejected are provided in the following table.

Table 13.6 Losses of metals in DMS tailings, Hazen (1997)

Particle size (finer than)	Weight (%)	Distribution (%)			
		Pb	Zn	Cu	Ag
$^{1}/_{2}$ Inch	59.5	6.8	11.8	16	15
$^{1}/_{4}$ Inch	57.7	5.3	7.2	7.9	11
6 Mesh	56.9	4.8	6.7	9.5	11.3
10 Mesh	55.6	3.7	6	7.2	8.5

Analyses of the minus 10 mesh DMS rejects showed that the majority of the lead and zinc losses occurred as oxide mineralization and amounted to 86.5% of the total lead and 70% of the total zinc present in the reject. Microscopic examination showed that the oxides occurred primarily as intergrowths with dolomite and quartz. The lead, zinc, silver and copper reporting to the < 200 mesh fraction recovered screened fines that ranged between 21.2 and 35.5% of these metals.

In 2005, SGS Lakefield did DMS tests simulated by using heavy liquid separation (HLS). A composite of 50% MQV and 50% SMS was crushed to two product sizes of nominal 12.7 mm (½") and 6.4 mm (¼") to be used for the testing. Both of the sized materials were pre-screened to remove minus 3.36 mm (6 mesh) particles, which were analyzed separately. The results are presented for both fractions in the following table at three varying heavy liquid densities of 2.6, 2.8, and 3.0 g/cm^3.

Table 13.7 SGS – HLS test data for a 50 wt.% MQV & 50wt.% SMS feed

Product	Weight (%)	Assays						Distribution (%)					
		Pb (%)	Zn (%)	Cu (%)	Ag (g/t)	Pb Oxide (%)	Zn Oxide (%)	Pb	Zn	Cu	Ag	Pb Oxide	Zn Oxide
Heavy liquid separation ¼"													
Minus 6 mesh	57.3	10.2	9.5	0.24	137	3.51	2.05	65.4	59.8	65.2	67.9	71.5	67.9
¼" Sample 3.0 SG Sink	15.3	19.4	22.6	0.48	226	4.78	3.38	33.2	38	34.8	29.9	26	29.9
¼" Sample 2.8 SG Sink	15.6	0.52	0.92		9.3	0.27	0.48	0.9	1.6		1.3	1.5	1.3
¼" Sample 2.6 SG Sink	11.4	0.38	0.47		9.3	0.24	0.26	0.5	0.6		0.9	1	0.9
¼" Sample 2.6 SG Float	0.4	0.71	0.54		22.5	0.41		0.03	0.03		0.08	0.1	0.1
Head (calculated)	100	8.93	9.09	0.21	115.6	2.81	1.8	100	100	100	100	100	100
Heavy liquid separation ½"													
Minus 6 mesh	30.8	12.1	19	0.41	201	6.29	3.46	39.5	52.8	47.2	45.6	49.5	52.1
½" Sample 3.0 SG Sink	26.6	19.8	18.2	0.53	253	6.26	2.85	55.8	43.7	52.8	49.6	42.6	37.1
½" Sample 2.8 SG Sink	22.7	1.41	1.33		20.4	1.03	0.78	3.4	2.7		3.4	6	8.7
½" Sample 2.6 SG Sink	19.7	0.63	0.39		8.7	0.38	0.22	1.3	0.7		1.3	1.9	2.1
½" Sample 2.6 SG Float	0.2	0.52	0.62		59.8	0.32					0.1		
Head (calculated)	100	9.44	11.1	0.27	135.7	3.91	2.04	100	100	100	100	100	100

Follow-up work in 2005 by Confidential Metallurgical Services (CMS) and by SGS Lakefield undertook HLS testing on 13 MQV samples from the 883 m and 930 m level adits. The samples were crushed to minus 12.7 mm (½") particle size and screened to remove the minus 1.4 mm (12 Tyler mesh) fraction. The −12.7 +1.4 mm fraction was tested at heavy liquid specific densities of 2.8 and 3.0 g/cm^3. For the 883 m level samples, weight percentages of the HLS rejects (floats) ranged from 14% to 53% at a specific density of 2.8 g/cm^3, averaging 34.3%. Average metal losses were 2.6% Pb and 4.6% Zn. HLS rejects for the 930 m level samples accounted for 8% to 34% of the feed weight, averaging 21.6%. Average metal losses were 1.5% Pb and 2.1% Zn. The higher specific density media (3.0 g/cm^3) produced 17% more rejects for the 930 m level samples, and 23% more rejects for the 883 m level samples, than the 2.8 g/cm^3 density media. However, more metal losses were seen at the higher specific density.

In 2007, SGS Lakefield did four sets of HLS tests on four different composite samples at media specific densities of 2.6 and 2.8. At specific density 2.8, between 19.0 % and 26.5% of the total feed weight was rejected. The loss of lead, zinc and silver to the HLS rejects was similar among these samples, ranging from 0.9% to 2.1% Pb, 1.4% to 2.6% Zn, and 1.3% to 3.2% for Ag. Samples with higher sulphide oxidation averaged slightly higher metal losses to the rejects.

In 2009, SGS did large-scale DMS testing on a composite generated from the 883 m and 930 m adits. A 530 kg sample was processed through media with a specific density of 2.8. Results are provided in Table 13.8.

Table 13.8 SGS (2009) – DMS testing @ SG2.8 on MQV (883 m + 930 m level adits)

Product	Weight (%)	Assays						Distribution (%)					
		Pb (%)	Pb Oxide (%)	Zn (%)	Zn Oxide (%)	Cu (%)	Ag (g/t)	Pb	Pb Oxide	Zn	Zn Oxide	Cu	Ag
Sink	31.9	28.3	6.44	26.9	4.48	1.1	453	73.3	50	57.1	42	66.4	59.5
-14 Mesh	27.1	11	6.81	21.3	5.64	0.6	339	24.2	45	38.5	45	30.8	37.9
Sink plus -14 Mesh	59	20.4	6.61	24.3	5.01	0.9	401	97.5	94.9	95.6	94.9	97.2	97.4
Float	41	0.75	0.51	1.6	1.07	0.04	15.2	2.5	5.1	4.4	5.1	2.8	2.6
Head (Calc.)	100	12.3	4.11	15	3.4	0.5	242	100	100	100	100	100	100

The test work showed 41% of the feed weight was rejected as waste with a corresponding loss of economic metals at 2.5% Pb, 4.4% Zn, and 2.6% Ag.

Later work performed in 2013 by SGS Lakefield did DMS on a composite sample also obtained from the 883 m level adit. The sample was stage-crushed to < 6.35 mm and screened to remove the minus 20 mesh fraction. The coarse fraction underwent DMS upgrading at a media specific density of 2.8, with results as shown in Table 13.9.

Table 13.9 SGS (2013) – DMS testing @ SG2.8 on MQV (883m level adit)

Product	Weight (%)	Assays				Distribution (%)			
		Pb (%)	Zn (%)	Cu (%)	Ag (g/t)	Pb	Zn	Cu	Ag
Sink	29.2	23.1	29.6	0.87	341	40	38	35.3	35
Sink plus -14 mesh	79.3	21.0	28.2	0.88	350	98.8	98.1	97.3	97.6
Intermediate	2.2	0.96	2.01	0.07	24.6	0.1	0.2	0.2	0.2
Float	18.5	1.00	2.05	0.10	33.8	1.1	1.7	2.5	2.2
Head (Calc.)	100	16.9	22.8	0.72	284	100	100	100	100

The results showed a recovery of 99% of the lead, 98% of the zinc and 98% of the silver, which includes the sink recombined with the screened fine fractions. Approximately 21% of the feed weight was rejected as waste.

Following up in 2014, GMR laboratory conducted HLS tests at a media specific density of 2.8 on various samples, including a master composite sample and 17 variability test samples. Results gave ~20.2% of the feed weight from the master composite as rejected into the float fraction with losses of the lead, zinc, and silver respectively at 1.7%, 1.7%, and 1.6%. The floats rejected from the 17 variability samples ranged from 9.0% to 44.6%, averaging 22.7% of the feed weight.

13.2.4 Preliminary flotation studies (1980 to 2000)

13.2.4.1 Main Quartz Vein (MQV) Mineralization

Lead and zinc response

Among the earliest tests with good documentation available were those from Lakefield Research in 1980. Preliminary flotation studies were performed on a sulphide composite and an oxide composite, both obtained from the MQV zone with a near surface bulk sampling program. The test results were described by Lakefield as providing a poor separation between lead and zinc. Lakefield indicated that the slime gangue minerals (dolomite and graphitic materials) complicated the flotation, and used a combination of gangue depressants and zinc mineral depressants in an effort to suppress zinc minerals and slime at the lead flotation stage. The test work also investigated the effect of primary grind size on lead and zinc flotation performance, indicating that lead recovery at both the rougher and cleaner flotation stages reduced slightly at a fine primary grind size. However, the selectivity between lead and zinc improved at a fine primary grind size. The test conducted on the oxide composite showed that sufficiently high-grade lead and zinc concentrates could be produced. However, metal recoveries decreased significantly in the sulphide flotation stages, although it appeared that lead oxide minerals were able to be recovered after the zinc sulphide flotation tailings were conditioned by sodium sulphide.

In 1980, Kamloops Research & Assay Laboratory Ltd (Kamloops, KM 019) did further tests on the samples tested by Lakefield. Soda ash was used to adjust slurry pH and sodium cyanide and sodium sulphite were used for suppressing zinc minerals (some tests used zinc sulphate to replace sodium cyanide). The work also evaluated effects of primary grind sizes on lead and zinc differential flotation. The better results were attained at a primary grind size between 70% and 80% passing 74 µm (200 mesh). The tests also indicated that the degree of regrinding of lead rougher concentrate would be a key factor to achieve satisfactory metallurgical performance, and that substantial addition of sodium cyanide to the primary grinding circuit would permit acceptable zinc suppression at the lead flotation stages. Zinc flotation responded well to the conventional reagent scheme. Lead and zinc concentrates produced good grades, but contained significant deleterious elements. Projected metallurgical performance is shown in Table 13.10.

Table 13.10 Kamloops Lab 1980 MQV flotation data

Product	Grades (%)		Distribution (%)			
	Pb	Zn	Mass	Pb	Zn	Ag
Feed	12.5	15.5	100.0	100.0	100.0	100.0
Lead concentrate	55.0	10.0	15.5	68.0	10.0	59.0
Zinc concentrate	5.0	55.0	21.1	8.0	75.0	16.0

In 1980, Kamloops carried out separate test work using a sample identified from the lower adit to compare metallurgical performance with that achieved with the upper adit sample in the previous studies. A composite sample was generated from three cross-cut samples. In general, the metallurgical responses of both samples tested were similar, with high-grade lead and zinc concentrates being produced. Both samples showed zinc minerals active in the lead flotation circuits. It was concluded that use of strong zinc depressants may be necessary.

Two years later potential alternatives to the cyanide-based reagent scheme were examined. Kamloops indicated that complete exclusion of cyanide from the reagent scheme for the mineralization would produce unsatisfactory results. Optimum primary grind size continued to appear to be about 75% passing 74 µm (200 mesh). Regrinding benefits on lead rougher concentrate were described as being of marginal value to lead metallurgical performance. However, zinc rougher concentrate regrinding was seen to be potentially beneficial. Subsequent testing by Kamloops laboratory showed regrinding zinc rougher concentrates would not significantly impact the zinc response.

In 1982, Kamloops (KM081) conducted two tests on a sample that was being used for a pilot test program at CSMRI. Reagents used for zinc suppression were 1,000 g/t soda ash and 200 g/t sodium cyanide in the primary grinding. Collector dosage and flotation retention time varied. To better reject zinc minerals from lead concentrates, a lower collector dosage (30 g/t vs. 60 g/t Z-11) and shortest possible flotation retention time (4 minutes vs. 7 minutes) was deemed to be required. The resulting lead concentrate assayed 63.1% lead and 11.9% zinc. The zinc concentrate assayed 55.8% zinc and 5.0% lead. Again, results confirmed cyanide is warranted for rejecting zinc from lead concentrate for the MQV mineral samples.

In 1982, CSMRI did pre-pilot plant flotation testing to determine effects of various conditions on metallurgical performances. Results were achieved at a primary grind that varied from 50% to 94% passing 74 µm (200 mesh). Metal recoveries to lead and zinc rougher concentrates were not significantly affected, although lead rougher recovery was reduced slightly at the coarsest grind investigated, regrinding was not incorporated. A primary grind size of ~ 75% passing 74µm was used for the remainder of the laboratory flotation tests.

Testing of cyanide dosage on lead rougher flotation showed that the addition of 1000 g/t (2.0 lb/ton) of soda ash in the primary grind, with 500 g/t (1.0 lb/ton) sodium cyanide, produced the highest-grade lead rougher concentrate (49.0% lead) with the lowest zinc content (16.1% zinc). Using a lower dosage of cyanide, or partially replacing with sodium sulphite, resulted in decreased response. Addition of sodium cyanide in lead cleaner flotation stages did not improve final lead concentrate grade. The effect of regrinding rougher lead concentrate on lead cleaner flotation was not conclusive, with two sets of tests generating different results. Also, the test program showed that an extended conditioning with sodium cyanide for 60 minutes did not improve zinc depression in lead cleaner flotation. When sodium sulphide dosages were increased above 1.5 kg/t (3.0 lb/ton), lead grade of the lead rougher concentrate improved with more efficient rejection of zinc. The lead recovery to the concentrate also improved by approximately 2%. CSMRI also tested ammoniacal zinc cyanide as a replacement for sodium cyanide without effective rejection of the zinc minerals to improve lead recovery. Lime was used in place of soda ash to modify slurry pH. At a similar pH level to soda ash, the lime produced inferior results; lead metallurgical performance also deteriorated at pH 12.0 as compared to pH 9.5.

Earlier investigation into zinc flotation looked at altering retention time, pH and related modifiers, along with various depressants, with minor apparent effect as zinc grade and recovery appeared challenging, with significant quantities of zinc reporting to the lead circuit. In 1994, G&T did further testing to evaluate a cyanide-free processing scheme to minimize potential environmental impact and avoid silver dissolution. Flotation response of the sample remained relatively consistent, despite relatively large changes in reagents and treatment conditions. In addition to sodium cyanide as zinc mineral suppressant, sodium metabisulphite (SMBS) additions of up to 5,000 g/t resulted in very similar concentrate grades and recoveries during differential flotation. Regrinding of cleaner feed streams did not appear to enhance lead circuit performance, but regrinding of zinc cleaner feed stream was considered potentially beneficial.

Figure 13.1 and Figure 13.2 show the G&T test results in terms of recovery and grade relationships for lead and zinc cleaner open circuits.

Figure 13.1 G&T lead grade recovery curves for MQV open cycle cleaning

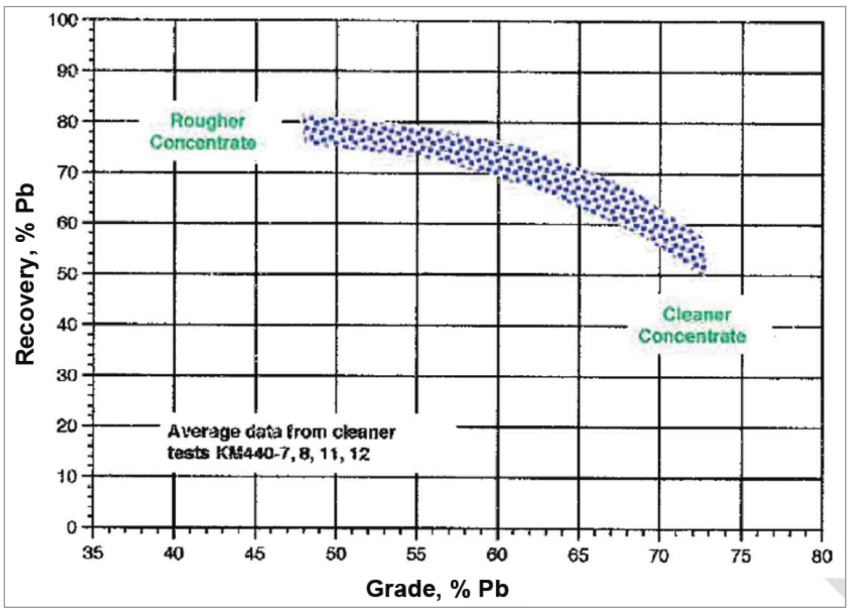
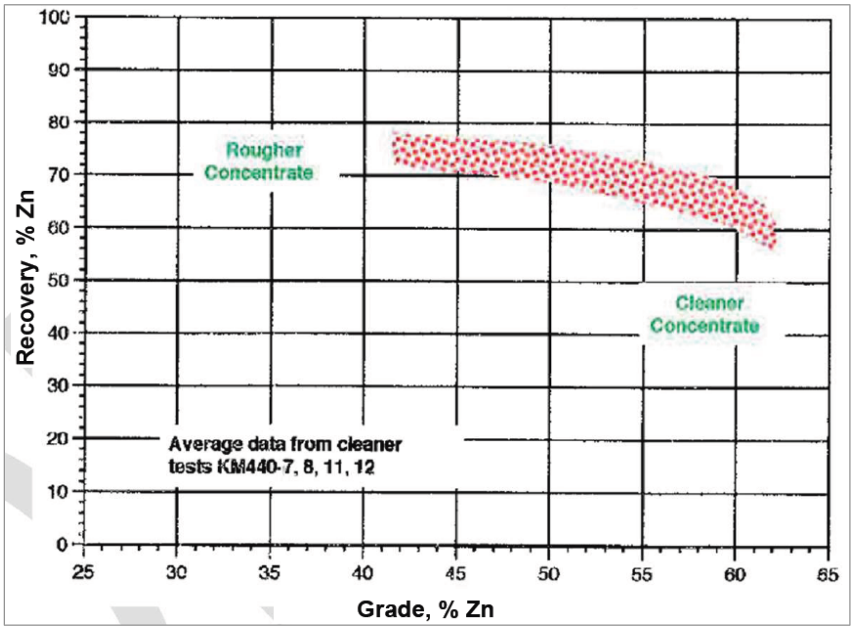


Figure 13.2 G&T zinc grade recovery curves for MQV open cycle cleaning



The results indicated that acceptable lead concentrate grades resulted in approximately 70% Pb recovery, while for zinc a recovery of 65% might be expected.

Copper separation

G&T tested a batch differential flotation procedure to produce separate copper, lead and zinc concentrates for smelting studies. SMBS was used as lead and zinc depressant in copper and lead flotation circuits, with M2030 as copper collector and ethyl xanthate as lead collector. Lime, copper sulphate, and potassium amyl xanthate (PAX) were used for zinc flotation. Although reasonably good-grade concentrates could be achieved, lead and zinc recoveries were apparently low; lead flotation circuits used complex reverse flotation, which is likely not practicable in industrial operations.

A high-grade head sample, containing approximately 0.77% copper, 20.8% lead, and 26.6% zinc was used for further evaluation. The open circuit tests produced 28.2% copper concentrate at 47% copper recovery, 75.5% lead concentrate at 71.7% lead recovery, and 56.6% zinc concentrate at 61.6% zinc recovery. The response is considered modest and the high feed grades did not well represent the MQV mineralization.

Further studies investigated copper metallurgical response with sequential copper and lead flotation, as well as testing copper separation from copper-lead bulk concentrates. Lakefield did three tests on a sulphide composite in 1980 in an effort to produce a silver-rich copper concentrate. The tests included one sequential flotation using sulphur dioxide to suppress lead minerals. Best results were obtained using soda ash and a mixture of oxide zinc and cyanide in the copper-lead bulk flotation circuit and dichromate for lead depression in the copper separation circuit. The test produced a copper concentrate grading 24% copper, 24% lead, 8.4% zinc, 5,964 g/t silver, and 4.5% arsenic.

Due to high lead losses to copper-lead bulk flotation tailings and high lead content (24% lead) in copper concentrate, concentrates and tailings from the test were mineralogically examined. The majority of galena (~66%) in copper concentrate and about 50% of sphalerite particles in copper and lead concentrates were seen in liberated forms. Separation conditions were seen as not optimum. Further work was recommended to reduce lead content of the copper concentrate and to improve cleaning efficiency for copper-lead bulk concentrate. A copper separation test was also done using sequential flotation with sulphur dioxide to suppress lead minerals. The resulting copper concentrate assayed 8.5% copper and 57.3% lead.

In 1980 Kamloops Laboratories investigated potential methods to produce a silver-rich copper concentrate from the composite sample tested by Lakefield. Copper and lead separation was performed on bulk copper-lead concentrates that were produced by a soda ash / cyanide procedure. The rougher copper-lead concentrate was reground, cleaned, and subjected to two copper and lead separation techniques. The first procedure used potassium dichromate with sodium silicate and sulphur dioxide to depress galena, and the other procedure conditioned with sulphur dioxide, then pH was increased with lime and using a selective collector. Both procedures apparently gave similar results with a moderate response. The procedures were deemed complex and seen as unlikely to be able to be controlled in a commercial environment and applicable for the typical feed grades expected.

Follow-up testing performed on copper and lead separation was conducted on samples from the lower and upper adits. Both sequential flotation and copper-lead bulk flotation followed by copper-lead separation were tested. A maximum of 30% copper recovery at a grade of 10.6% copper was obtained using a combination of sulphur dioxide / lime / dichromate / silicate to suppress lead and gangue minerals. A concentrate with 19% copper was obtained at a copper recovery of 18% using sulphur dioxide / lime as lead depressants. These results were not consistent as compared to those obtained by Lakefield. Kamloops concluded that producing a silver-rich copper concentrate from either adit samples was not technically feasible. This may be due to high lead to copper ratios, which are 64:1 for the lower adit sample and 22:1 for the upper adit sample.

In 1981 Kamloops conducted further testing to evaluate copper and lead separation by using SMBS to suppress the galena in the copper-lead bulk concentrate. The work appeared to show that a copper-rich concentrate could be produced from the sample tested, although conditioning time and reagent dosages were not optimized. The copper concentrates produced contained 20 to 30% copper. Silver reporting to copper concentrate ranged from 20% to 30% for the upper adit sample and 13% to 28% for the lower adit sample. However, the copper concentrates produced contained about 10 to 40% lead. This program also investigated effect of recycled water on the selectivity of copper-lead separation. It appeared that separation selectivity decreased with using recycled water.

In a later program a locked cycle test was done using the SMBS method to separate copper and lead. Copper and lead separation efficiency was relatively good until the final cycle of the test. This instability was attributed to an increased load into the copper and lead separation stage. Lead metallurgy was quite stable after the initial cycle, but zinc contents of the final lead concentrates were high. A 25.3% copper concentrate with a copper recovery of 43.7% was produced from the locked cycle test. A further batch copper and lead separation test was conducted using starch and sulphur dioxide as lead mineral depressants at an enhanced pulp temperature (60 to 65°C). Little difference from the SMBS method resulted. A subsequent study investigated the effect of adding various dosages of SMBS in primary grinding on copper metallurgical performance. It was found that copper grade and recovery did not suffer using 2,000 g/t SMBS compared to 5,000 g/t SMBS. The results from the test work using 3,500 g/t SMBS were anomalous in that copper grade and recovery were significantly lower than when adding 2,000 or 5,000 g/t SMBS. In 2000, MQV samples tested at the University of British Columbia (UBC) showed metallurgical performance sensitive to grind size, with a fine primary grind possibly required. This was contrary to some previous results. Further evaluation was performed in 1981 by Kamloops Laboratories with mine water obtained from the upper adit that indicated that a negative impact, particularly for the copper and lead, was experienced unless some dilution with fresh water was incorporated.

In 1994, G&T did further sequential flotation studies on a MQV sample in an effort to produce a separate copper concentrate. Three collectors were tested, with Minerec 2030 giving the better metallurgical performance. The open circuit test results showed that 47% to 61% of the copper was recovered to the copper concentrates, grading about 20 to 28% copper. A locked cycle test was conducted on the sample using the SMBS and Minerec 2030 reagent regime in the copper flotation. The test results showed that 58.7% of the copper was recovered to a 20.6% copper concentrate. Using the differential flotation method tested by the earlier program, a large-scale, open-circuit flotation was carried out to produce copper, lead, and zinc concentrates for smelting testing from a MQV bulk sample. The copper concentrate produced assayed 28.2% copper, 9.8% lead, and 13.1% zinc. The lead concentrate assayed 75.5% lead, 5.3% zinc. The zinc rougher concentrate assayed 56.8% zinc.

Cerussite and Smithsonite Recovery

In 1980, Lakefield Research did preliminary oxide mineral flotation tests on a sulphide composite and an oxide composite. About 20% of the lead and zinc minerals in the sulphide composite and about 50% of the lead and zinc minerals in the oxide composite were in oxide forms. The oxide mineral flotation was conducted separately on the two samples.

The sulphide composite in reality was still highly oxidized. The sulphide flotation tailings produced after flotation of galena and sphalerite were filtered, re-pulped, and then conditioned with sodium sulphide, and cerussite was floated with potassium amyl xanthate collector (Z6). The results gave 9.4% of the lead reporting to an oxide concentrate with 18.5% lead and 4.2% zinc. The oxide lead flotation tailings were then conditioned with 500 g/t copper sulphate and then floated for smithsonite. The flotation failed to recover oxide zinc minerals and insignificant zinc was recovered. For the more highly oxidized composite the treatment was similar except that no oxide zinc flotation was conducted. The results gave an oxide lead concentrate averaging 11.5% lead and 7.8% zinc. The previous additional lead flotation recovered about 22% of the lead.

In 1982, CSMRI did flotation tests on a gravity concentrate sample containing cerussite and smithsonite. Sodium sulphide was added to sulphidize zinc tailings. Smithsonite was depressed with copper sulphite and sodium cyanide. Selective flotation was not achieved, possibly due to excess sodium cyanide depressing both lead and zinc minerals. Similar tests were done on zinc tailings with the tailings sulphidized by sodium sulphide. Essentially all of the lead carbonate was lost to the zinc concentrate and final tailings.

In 1983, CSMRI did a series of open-circuit tests to determine if cerussite and smithsonite could be recovered by a variety of procedures including flotation, leaching, or gravity concentration. Soda ash was used for conditioning and galena was concentrated by flotation; the tailings were subjected to non-sulphide flotation. The feed sample did not contain significant sphalerite. The oxide lead and zinc minerals flotation response to various flotation reagent regimes was investigated. For cerussite, sodium sulphide, sodium hexametaphosphate, and copper sulphate were examined. For smithsonite, a tallow amine collector and potassium dichromate were tested. Sodium hexametaphosphate had a depressing effect on both cerussite and smithsonite flotation; copper sulphate reduced the effectiveness of sodium sulphide, and dichromate depressed smithsonite flotation. Potassium dichromate depressed smithsonite flotation to a lesser extent with the use of a stronger primary amine. For the cerussite flotation collector suites that incorporated xanthate, fatty acid, or petroleum sulphonate, only xanthate produced a selective float.

Among the best results from the 1983 CSMRI test work was Test #15 that gave a cerussite third cleaner concentrate of 57.7% lead and 8.1% zinc using sodium silicate / sodium sulphide for flotation conditioning and Aero 350 as the cerussite collector. From the head sample, 36.2% of lead and 22.8% of silver was recovered. The effect of sphalerite flotation reagents on subsequent cerussite flotation should be studied. Using primary coco amine as a zinc collector, (Test #12) produced an oxide zinc concentrate that assayed 19.1% zinc and 0.96% lead and 1.52 oz/ton silver. The zinc and silver recoveries reporting to the concentrate were 62.8% and 10.4%, respectively. Further testing showed that napthenic acid as a collector for smithsonite was not effective, and silica flotation prior to smithsonite flotation was not successful.

A process patented by the New Jersey Zinc Company to concentrate smithsonite was also tried. This involved dispersion of silica, selective flocculation of carbonate materials, and smithsonite flotation with an organic ester of carboxylic acid. A smithsonite concentrate of 39.6% zinc was produced, but only 1.1% of the zinc in the sample was recovered. Using a similar procedure, (Test #18) produced a concentrate of 32.9% zinc with a zinc recovery of 13%.

In 1994, Kamloops Laboratory initiated testing for oxide lead and zinc recovery. However, the sample had a non-sulphide content that was lower than expected and few conclusions resulted from the test program.

13.2.4.2 Stratabound Mineralization

Stratabound mineralization (SMS) has less copper, and silver with a lower extent of sulphide oxidation than the MQV. A principal mineralogical difference is that about half the sulphides occur as 50% pyrite (including marcasite). Lead and zinc minerals mainly occur as galena and sphalerite.

Kamloops undertook a test program in 1992 to establish a preliminary flowsheet and to select an appropriate reagent regime. Two primary grind sizes of 80% passing 80 µm and 50 µm were tested. Lead circuit performance was seen to be insensitive to change in grind size. However, the zinc circuit was influenced by the grind size variation. Tests of two sphalerite / pyrite / marcasite suppression reagent schemes, lime-cyanide, and lime alone, showed similar response of the lead and zinc. No test work to optimize reagent addition levels was conducted. Locked cycle tests, at a primary grind size of 80% passing 80 µm, produced a lead concentrate with 57.8% Pb grade and 80.6% recovery, and a zinc concentrate with 52.0% Zn grade at 87.4% recovery. With the same reagent conditions at 80% passing 50 µm, lower metal recoveries resulted. Mineralogical analysis of concentrates produced from the finer grind showed that 70% to 75% of the lead concentrate was galena, with pyrite comprising 15% to 20%, of which 50% was liberated. The zinc concentrate consisted of 95% sphalerite, with the remainder pyrite, galena, and traces of tennantite. In follow-up testing, three samples were tested, using 250 g/t calcium oxide and 500 g/t sodium cyanide as sphalerite / pyrite / marcasite suppression reagents. The three samples produced similar metallurgical performance as previous results.

Further testing at Kamloops used a simple two-product process at a primary grind of 80% passing 50 µm. However, it was noted that a coarser primary grind of 80% passing 100 µm may suffice for mineral separation in lead and zinc rougher flotation stages. Sulphoxy (SMBS or sulphur dioxide), lime-cyanide, and lime alone were tested for sphalerite / pyrite / marcasite suppression. The sulphoxy scheme developed for MQV mineralization did not produce an acceptable metallurgical response for the SMS mineralization. Process selectivity was poor due to uncontrolled pyrite flotation. Metallurgical response improved slightly, using lime-cyanide. For lead rougher concentrate, a combination of 500 g/t lime and 250 g/t sodium cyanide gave the highest lead grade (41.5% lead) and lowest zinc grade (7.7% zinc). Lime-cyanide and lime-alone reagent regimes were tested in locked cycle tests. Similar results for sphalerite / pyrite / marcasite suppression were achieved.

In 2000, a University of British Columbia study showed an SMS sample to be sensitive to overgrinding. This resulted in elevated losses for both lead and zinc, especially in the less than 44 µm (325 mesh) fraction of the flotation tailings. It was recommended to evaluate more grind sizes, including staged grinding with flash flotation.

13.2.5 Flotation testing (2001-2016)

13.2.5.1 Sample origin and characterization

A more comprehensive testing program was performed by SGS Lakefield beginning in 2004 and was performed in several phases. The work was performed on composites generated from samples collected from several underground adit crosscuts. In Phase 1 and 2 of the program, three samples were generated from material from the MQV upper (930 m level) and lower (883 m level) adits, as well as from the SMS zone. Composites were labeled as Lower Zone and Upper Zone composites and Stratabound composite. From the zone composite samples, two master composite samples were generated:

- Master Composite 1 - 50% Upper Zone Composite and 50% Lower Zone Composite.
- Master Composite 2 - 50% Master Composite 1 and 50% Stratabound Composite.

The head analyses for major elements of interest are provided for the composites in Table 13.11.

Table 13.11 SGS composite head analyses (Phase 1 & 2 - 2005 test program)

Sample	Assays					
	Pb (%)	Pb oxide (%)	Zn (%)	Zn oxide (%)	Ag (g/t)	Cu (%)
Upper Zone Composite	21.3	7.63	20.3	5.57	242	0.76
Upper Zone Composite w Dilutions	11	-	11.4	-	193	-
Lower Zone Composite	16	6.48	15.9	4.01	350	0.46
Lower Zone Composite w Dilutions	11.5	-	11.5	-	137	-
Master Composite 1	18.2	6.9	17.9	5.5	320	0.58
Master Composite 1 w Dilutions*	10.8	-	11.7	-	180	-
Master Composite 2	15.5	5.63	16.5	3.34	255	0.44
Master Composite 2 w Dilutions*	11.2	-	12.2	-	175	-
Stratabound Composite	5.16	0.33	10.5	0.11	52.4	0.025

* Back-calculated head assay from locked cycle tests.

For Phase 3, flotation test samples were diluted Lower Zone (870 m) and Upper Zone (930 m) composites from Phases 1 and 2. The samples were upgraded by heavy liquid separation (HLS). Flotation head, which consists of HLS sink and prescreened fines, had the assay results shown in Table 13.12.

Table 13.12 SGS composite head analyses (Phase 3 - 2005 test program)

Sample	Assays					
	Pb (%)	Pb oxide (%)	Zn (%)	Zn oxide (%)	Ag (g/t)	Cu (%)
Upper Zone Composite HLS*	19.6	6.92	21.5	6.47	263	13.4
Lower Zone Composite HLS*	15	5.55	12.5	4.3	174	11.8

* 2.8 SG Sink + Fines.

In Phase 4, three additional composite samples were tested, including a master composite made from 11 individual samples containing dilution material, and two sub-composites identified as low-oxide composite and high-oxidation composite. The main test work was carried out on the Master Composite sample after HLS upgrading. Additional tests were done on the non-pre-concentrated Master Composite and the two sub-composites treated by the HLS pre-concentration. Head assay data are summarized in Table 13.13.

Table 13.13 SGS composite head analyses (Phase 4 - 2007 test program)

Sample	Assays					
	Pb (%)	Pb oxide (%)	Zn (%)	Zn oxide (%)	Ag (g/t)	Cu (%)
Master Composite	18	8.59	16.4	4.12	258	8.01
Master Composite HLS*	21.9	7.58	21.9	5.5	304	-
Low Oxidation Composite**	21.5	4.84	19.1	2.86	350	-
Low Oxidation Composite HLS*	24.6	5.37	23	3.37	407	12.9
High Oxidation Composite**	12.4	4.48	12.8	4.15	194	-
High Oxidation Composite HLS*	15.5	5.82	15.9	4.76	237	6.38

* 2.8 SG Sink + Fines; **: back-calculated head grades.

Phase 5 testing was done on a 503 kg composite sample made from nine individual samples. The key objective was to produce a quantity of process water for environmental testing purposes. Head assay data for the composite sample are shown in Table 13.14.

Table 13.14 SGS composite head analyses (Phase 5 - 2009 test program)

Sample	Assays						
	Pb (%)	Pb oxide (%)	Zn (%)	Zn oxide (%)	Ag (g/t)	Cu (%)	S (%)
ROM Composite	12	6.81	15.3	5.64	219	0.51	8.33

In 2013, SGS did tests for environmental and other aspects using water collected from the mine adits. The water was used along with composite material collected from the 883 m adit to generate flotation concentrates, flotation tailings, and supernatants for evaluation. Head analyses of the composite are provided in the following table.

Table 13.15 SGS composite head analyses (2013 test program)

Sample	Assays			
	Pb (%)	Zn (%)	Ag (g/t)	Cu (%)
Composite	16.9	22.8	284	0.72

Between 2014 and 2016, GMR laboratories in Burnaby conducted a test program to generate concentrate samples for a bio-leaching program (due to Hg smelter penalty concerns) and an attempt to simplify the flotation reagent regime. One master composite sample and 17 variability test samples from the mine adits were collected for the study. The head grades for the samples ranged from 8.0% to 43.5% for total lead, and 4.3% to 28.9% for total zinc. However, the average oxidation rate was high at approximately 47% for lead, and 24% for zinc, indicating the samples are not representative of the more global MQV mineralization.

13.2.5.2 DMS and open cycle flotation testing

Between 2004 and 2013 SGS Lakefield did flotation test work that used DMS to pre-concentrate flotation feed. The test work objectives included optimization studies for flowsheet development and developing a cyanide free flotation reagent scheme. This would provide a mass balance for the metals of interest, including oxide portions of the feed material, and in determining the process response if MQV and SMS materials are blended when mining.

Phase 1 & 2 – SGS Lakefield (2004 - 2005)

The results of the first two phases of test work showed that the sulphide and oxide minerals can be recovered using a sequential flotation process, consisting of sulphide lead flotation, sulphide zinc flotation, oxide lead flotation and oxide zinc flotation. The process flowsheet development tests investigated various reagent schemes, especially types and dosages of depressants and dispersants. Reagent scheme findings for the Lower Zone composite are as follows:

Cyanide-free reagent schemes were tested, including several depressant combinations: sodium sulphide / zinc sulphate, sodium sulphide / zinc sulphate / ferric sulphate, sodium sulphide / MQ1 (sodium metabisulphite (Na2S2O5) / zinc sulphate) and sodium sulphide / MQ2 (sodium metabisulphite / sodium thiosulfate (Na2S2O3) / zinc sulphate). Results showed that zinc minerals and pyrite can be successfully suppressed by cyanide-free reagent schemes, such as sodium sulphide / MQ1 or sodium sulphide / MQ2. The programs also tested the effect of slime dispersants on lead flotation. MKF (60% sodium silicate, 20% Acumer 9000, 20% thiourea) gave best results among sodium silicate and polyacrylamide dispersants, and was used for remaining tests. Collector testing for sulphide and oxide lead flotation included using sodium isobutyl xanthate (SIBX) as primary collector and several other collectors as secondary collectors. A combination of SIBX and DF067 collectors was selected. For oxide lead flotation, selected reagents were sodium sulphide for sulphurization, SIBX and DF067 as collectors, and MKF as slime dispersant.

The pH levels used ranged from 7.5 to 8.5 for sulphide zinc rougher flotation and 10 to 10.5 for sulphide zinc cleaner flotation. PZ1 (40% Dextrin W9524, 40% disodium hydrogen phosphate (Na_2HPO_4) and 10% Tamol 819), were used as gangue minerals and pyrite suppressors in sulphide zinc flotation. SIBX and Cytec 3894 were used as collectors for sulphide zinc flotation. The oxide zinc flotation used an emulsified mixture of PAX and Armeen C as a collector. The typical reagent scheme developed is summarized in Table 13.16.

Table 13.16 SGS initial reagent scheme for Prairie Creek Mineralization

Reagent	Reagent dosage (g/t)			
	Sulphide lead flotation	Oxide lead flotation	Sulphide zinc flotation	Oxide zinc flotation
Modifier and depressant				
Na2S 9H20	500	800	-	600
MQ2	475	-	-	-
MKF	175	175	-	-
PZ I	-	-	400	-
CaO	-	-	~1300	-
CuSO4 5H20	-	-	900	-
Na2SiO3	-	-	-	400
Collector and frother				
Dinafloat DF067	22	12	-	-
SIBX	30	65	50	-
Cytec 3894	-	-	6	-
PAX / Armeen C (50:50)	-	-	-	50
MIBC	4	-	-	24

MQ2: 60% $ZnSO_4$, 30% $Na_2S_2O_5$ and 10% $Na_2S_2O_3$.
MKF: 60% Na_2SiO_3, 20% Acumer 9000 and 20% Thiourea.
PZ1: 40% Dextrin W9524, 40% Na_2HPO_4 and 10% Tamol 819.
PAX / Armeen C: 44% PAX/44% Armeen C/12% Ethofat 242/12 (emulsifying agent).

Grind size used for the Lower Zone composite was 80% passing about 80 µm, although there is limited information to show how this was selected. Overall the samples selected responded reasonably well to the process flowsheet developed, and it was reported that the different types could be co-mingled in processing.

Phase 3 – 2006 SGS Lakefield

The test objective was to optimize flotation on HLS pre-concentrated samples using the general flowsheet and reagent scheme developed in the Phase I and II test program. Effects of process variables tested on the pre-concentrated samples indicated a number of findings. Among these was that use of soda ash as a pH modifier for sulphide lead flotation circuit gave improved lead recovery, as well as better selectivity between lead and zinc. Sodium hydroxide to adjust pulp pH for sulphide zinc flotation gave much poorer results.

Sulphide lead collector testing showed that alternative collectors, such as modified dithiophosphates, did not give improved sulphide lead metallurgical performance compared to the collectors selected in the previous test programs. A combination of N-type sodium silicate and DV177 (short chain polyacrylamide) gave better results in suppressing gangues in the oxide lead and zinc circuits.

Primary grind size testing showed the Lower Zone composite to be harder than the Upper Zone composite. Change in primary grind size did not significantly affect the overall metallurgical responses.

For Upper Zone testing, results were better with pre-concentration than without. The principal open cycle metal flotation recoveries were given as 91% lead, 87% zinc, and 98% silver into the lead and zinc concentrates. For the Lower Zone, lead metallurgical performance was also improved with HLS pre-concentration than without. Metal recoveries achieved were 94% lead, 82% zinc, and 98% silver in lead and zinc concentrates. Zinc grade for the combined sulphide and oxide zinc concentrate was 46.4% Zn, lower than that obtained from the as-received sample.

Phase 4 – 2007, SGS Lakefield

Principal objectives were to improve selectivity between lead and zinc mineral flotation in the lead flotation circuit, and improve oxide zinc concentrate grade.

Sulphide lead / zinc and oxide lead flotation

SGS Lakefield further examined the effect of primary grind size on the metallurgical performance of target minerals, using the conditions developed in the Phase 3 test program. Results indicated better lead sulphide metallurgical performance through increasing primary grinding fineness from 80% passing 60 µm to 80% passing 117 µm. However, this did not result in a significant change in the oxide lead and zinc sulphide metallurgical results.

The effect of lime on sulphide lead flotation rather than soda ash as a pH modifier was studied. Lime gave a significant loss in selectivity between lead and zinc differential flotation. The soda ash dosage did not significantly affect sulphide lead metallurgical performance. Several zinc depressants were tested on the Master Composite sample for sulphide lead flotation. The previously developed MQ3 was not as effective as a modified version, P82 (50% zinc sulphate, 25% sodium thiosulfate, and 25% sodium metabisulphite).

The Master Composite sample was seen to contain high levels of clay-type slimes, with this having a negative effect on lead and zinc flotation selectivity. To reduce this effect, a new slime dispersant / depressant, AQ4 (33% Accumer 9000, 34% sodium silicate and 33% trisodium phosphate (Na_3PO_4), was developed and tested; it reportedly showed better metallurgical performance than MKF. Consequently, reagents P82 and AQ4 were selected for suppressing zinc minerals and dispersing slimes for the rest of the test program.

For sulphide lead flotation, instead of recycling the first cleaner scavenger concentrate to primary grinding, a modified flowsheet eliminated this stage and sent the first cleaner flotation tailings to the lead rougher scavenger flotation. The rougher scavenger flotation concentrate was cleaned and the tailings were floated again, the concentrate produced being sent to primary grinding and the tailings to the zinc flotation circuit. It was stated that the modified flowsheet appeared to give improved lead selectivity in the locked cycle tests.

Oxide zinc sulphide and oxide lead flotation

Various collector and gangue dispersant / depressant combinations were tested. A SIPX and Normac S (amine acetate) combination gave better metallurgical performance in oxide zinc flotation and was retained for the rest of the test program. Secondary gangue depressant testing in the oxide zinc circuit included starch, a Calgon / Dispersogen mixture, and polyacrylamide. Highest zinc concentrate grade obtained was 34.7% Zn.

Both starch and the Calgon / Dispersogen mixture produced good concentrate grade in batch tests. The Calgon / Dispersogen mixture performed better in locked cycle tests. Regrinding oxide zinc rougher concentrate prior to cleaning flotation had a negative effect on zinc concentrate grade, which was reduced from 32% to 20% zinc. Also indicated from the batch testing when using process recycle water was a negative effect on flotation showing a deteriorated selectivity between lead and zinc minerals when LCT test flotation water was used.

Phase 5 – 2009, SGS Lakefield

SGS tested a bulk composite sample to generate flotation products for environmental and concentrate marketing review. It was noted that secondary copper minerals in the ROM sample resulted in zinc minerals activation, which caused a sulphide lead flotation selectivity problem. In using the Phase 4 reagent regime, some further selectivity problems were seen because of a larger proportion of fine slimes. For satisfactory slime depression and flotation selectivity, the reagent scheme was modified, including increasing sodium sulphide dosages and changing slime depressant from AQ4 to SQ4 (40% Aqumer 9400, 45% sodium silicate), with 15% EDTA (ethylene diamine tetra acetic acid).

Mine water was not shown to affect sulphide lead and zinc flotation, but oxide lead flotation deteriorated. By adjusting reagent dosages and modifying depressant AQ4, oxide lead floatability was restored. The final reagent scheme that was developed, including those eventually used for the locked cycle testing, is shown in the following table.

Table 13.17 SGS optimized 2009 reagent scheme

Reagent	Reagent dosage (g/t)		
	Lead sulphide flotation	Oxide lead flotation	Zinc sulphide flotation
Modifier and depressant			
Na_2CO_3	4,800	-	1,900
$Na_2S\ 9H_2O$	500	1,000	-
SQ4	550	-	400
P82	1,200	-	-
Sodium Silicate ‘N’	-	900	-
$CuSO_4\ 5H_2O$	-	-	1,800
Collector / frother			
DF067	20	12	-
SIBX	36	65	75
3894	-	-	18
MIBC	4	-	-

SQ4: 40% Aqumer 9400, 45% Na_2SiO_3, 15% EDTA (Ethylene Diamine Tetra Acetic Acid).

2013 test work, SGS Lakefield

SGS did flotation test work on a pre-concentrated DMS sample, including two batch flotation tests and two locked cycle tests. The first batch flotation test used potable tap water; the remaining tests used water from the 883 m level adit decline.

The test work objective was to generate supernatants for environmental tests and concentrates for marketing assessments. The reagent regime was similar to that of the Phase 5 testing, done in 2009. Concentrate grades and metal recoveries of the sulphide lead and zinc concentrates produced were inferior to these generated in previous testing. Locked cycle testing results are described below.

2014-2016 test work, GMR

The GMR laboratory located in Burnaby, BC conducted a series of flotation test programs to generate concentrate samples for a bio-leaching program and to simplify the flotation reagent regime. The open bench test results appear to show similar metallurgical performance results using a modified reagent regime, compared to the previous test results.

13.2.6 Flotation locked cycle test work

13.2.6.1 MQV oxidized material

The historical locked cycle work was performed on samples collected from bulk sample areas of the resource located closer to surface. Locked cycle tests (LCT) done prior to 2001 did not use DMS pre-concentration and focused on flotation of galena and sphalerite with little attention paid to the oxide minerals. Some tests included copper and lead separation. During 2004 and 2013 SGS did extensive LCT testing, including sulphide, as well as investigating recovery of oxide lead and zinc flotation

Specific locked cycle testing done on MQV material prior to 2001, with corresponding tabulated results, is shown below. This includes work by Kamloops Laboratories (KM) along with the related project number, which is provided in Table 13.18. Those LCTs performed later by SGS are summarized in Table 13.19.

Table 13.18 MQV locked cycle test data (Kamloops Labs, prior to 2001)

Product	Weight (%)	Grade				Distribution			
		Pb (%)	Zn (%)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)	Ag (%)	Cu (%)
KM019 – 1980 Lakefield sample									
Feed	100	12.5	15.5	226	-	100	100	100	-
Lead concentrate	15.5	55	10	857	-	68	10	59	-
Zinc concentrate	20.1	5	55	171	-	8	75	16	-
Tailings	64.4	4.7	3.7	89	-	24	15	25	-
KM048 - upper adit sample									
Feed	100	11.6	15.4	203	0.58	100	100	100	100
Copper concentrate	1	22.8	6.6	5,281	25.3	2	0.4	26.7	43.7
Lead concentrate	15	49.9	21	538	1.2	64.7	20.5	42.4	31.1
Copper + lead concentrate	16	48.2	20.1	834	2.7	66.7	20.9	69.1	74.8
Zinc concentrate	19.8	5.7	51.3	178	0.4	9.8	66.1	17.9	13
Tailings	64.2	4.3	3.1	40	0.11	23.5	13	13	12.2
KM440 - composite									
Feed	100	15.4	15.8	-	0.56	100	100	-	100
Lead concentrate	15.8	68.3	7.03	-	2.77	70.1	7.1	-	78.9
Zinc concentrate	19.9	4.17	59.1	-	0.17	5.4	74.5	-	6.1
Zinc retreat tail	13	13.1	11	-	0.39	11	9	-	9
Tailings	51.3	4.07	2.89	-	0.07	13.5	9.4	-	6
KM454 – vein ore sample									
Feed	100	15.8	16.1	-	0.6	100	100	-	100
Copper concentrate	1.8	17.9	16	-	20.6	2	1.8	-	58.9
Reverse tailings	13.4	69.4	12.4	-	0.3	58.7	10.3	-	6.4
Reverse concentrate	7	19.4	35	-	0.9	8.6	15.3	-	10
Copper + lead concentrate	22.2	49.5	19.8	-	2.1	69.3	27.4	-	75.3
Zinc concentrate	18.5	6.5	52.7	-	0.3	7.6	60.7	-	8.8
Zinc retreat tailings	13.5	10.9	5.4	-	0.4	9.3	4.5	-	8.6
Tailings	45.7	4.8	2.6	-	0.1	13.8	7.4	-	7.3

The pre-2001 processing flowsheet did not include recovery of oxide lead and zinc minerals, which lowered recovery. Some testing investigated separate copper recovery. Tests for copper were performed on samples with above average Cu grades and not considered representative of the resource as a whole. There was also significant loss of other payable metal distribution into the copper concentrate, as well as a high level of detrimental elements such as arsenic and antimony. While most of the silver reported with the copper concentrate, the low copper grades achieved may inhibit being able to achieve an easily saleable product.

The recovery and grade ranges varied depending on composite and the procedure. Many of the target metals were distributed into other concentrates or lost to tailings. The optimized lead concentrate appeared to be test KM440, which recovered approximately 70% of the lead, with a grade of ~68% Pb, with the zinc concentrate recovering about 75% with a concentrate grade of ~ 60% Zn.

Subsequent locked cycle tests performed by SGS used procedures to recovery oxide minerals, with most tests using HLS for pre-concentration. The flotation flowsheet used a similar procedure to the earlier work, followed by oxide flotation. However, as outlined previously in the open cycle procedures, the reagent scheme was much more complex. The SGS locked cycle results are highlighted in the following table.

Table 13.19 MQV locked cycle test data (SGS)

Product	Weight (%)	Grade				Distribution			
		Pb (%)	Zn (%)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)	Ag (%)	Cu (%)
LR10916-001 / Test 56 / Master Composite 2*									
Feed	100	9.6	11.5	152	0.4	100	100	100	100
PbS concentrate	7.6	63.5	7.6	1,145	2.64	50.2	5	56.9	56.5
PbO concentrate	4.5	55.1	4.8	385	0.6	25.6	1.9	11.3	7.7
PbS + PbO concentrate	12	60.4	6.6	864	1.9	75.8	6.9	68.2	64.2
ZnS concentrate	15	2.1	58.3	89.7	0.2	3.3	75.5	8.8	7.1
ZnO concentrate	2.5	2.4	33.7	155	0.5	0.6	7.4	2.6	3.9
ZnS+ZnO concentrate	17.5	2.1	54.7	99.1	0.2	3.9	82.8	11.4	11
Total tailings	70.5	2.8	1.7	44.2	0.1	20.3	10.3	20.4	24.8
LR10916-001 / Test 57 / Upper Zone Composite									
Feed	100	9	10.8	162	0.4	100	100	100	100
PbS concentrate	7.5	60.9	4.5	1,258	3.5	50.7	3.1	58.6	64.8
PbO concentrate	6.3	49	5.9	394	0.6	34.1	3.4	15.3	8.6
PbS+PbO concentrate	13.8	55.5	5.1	864	2.1	84.8	6.5	73.9	73.4
ZnS concentrate	13.4	4	59.5	132	0.2	6	73.8	10.9	7.8
ZnO concentrate	2	4.8	37.6	307	1.2	1.1	6.9	3.8	5.7
ZnS + ZnO concentrate	15.4	4.1	56.6	155	0.4	7	80.7	14.7	13.6
Total tailings	70.8	1	2	26	0.1	8.2	12.8	11.4	13
LR10916-001 / Test 58 / Lower Zone Composite									
Feed	100	14	10.7	152	0.3	100	100	100	100
PbS concentrate	11.5	84.7	8.6	847	1.5	69.1	9.2	63.9	54.9
PbO concentrate	5.8	46.2	5.9	383	0.6	19	3.2	14.6	10.4
PbS + PbO concentrate	17.3	71.7	7.7	691	1.2	88.2	12.4	78.5	65.3
ZnS concentrate	11.9	4.2	60.2	109	0.3	3.6	66.7	8.6	9.6
ZnO concentrate	4.6	10	27	139	0.5	3.3	11.6	4.2	7
ZnS + ZnO concentrate	16.5	5.9	50.9	118	0.3	6.9	78.3	12.8	16.5
Total tailings	66.2	1	1.5	20	0.1	4.9	9.3	8.7	18.2
LR10916-001 / Test 59 / Master Composite 1									
Feed	100	9.5	10.9	160	0.4	100	100	100	100
PbS concentrate	7.6	64.4	8.8	1,261	3.3	51.1	6.1	59.8	60.3
PbO concentrate	5.8	44.8	6	377	0.7	27.2	3.2	13.6	9.7
PbS + PbO concentrate	13.4	55.9	7.6	879	2.1	78.3	9.3	73.5	70
ZnS concentrate	13.2	3.9	60.9	112	0.2	5.5	73.8	9.3	7.3
ZnO concentrate	2.4	26.5	20.6	297	0.7	6.7	4.5	4.5	4.3
ZnS + ZnO concentrate	15.6	7.4	54.7	141	0.3	12.1	78.3	13.7	11.5
Total tailings	71	1.3	1.9	28.8	0.1	9.6	12.4	12.8	18.5
LR11098-001 / Test 25 / HLS Upper Zone Composite									
Feed	100	18.2	21.2	259	-	100	100	100	-
PbS concentrate	16.4	71.2	5.3	1,010	-	64	4.1	63.9	-
PbO concentrate	10.3	46.4	8.7	311	-	26.2	4.2	12.4	-
PbS + PbO concentrate	26.7	61.6	6.6	740	-	90.2	8.3	76.3	-
ZnS concentrate	23.5	3.8	60.3	148	-	4.9	67.1	13.4	-
ZnO concentrate	12.8	3.6	31.5	144	-	2.6	19	7.1	-
ZnS + ZnO concentrate	36.3	3.7	50.2	146	-	7.5	86.1	20.6	-
Total tailings	37	1.1	3.2	21.9	-	2.3	5.6	3.1	-
LR11098-001 / Test 26 / HLS Lower Zone Composite									
Feed	100	15.4	11.7	175	-	100	100	100	-
PbS concentrate	16	66.9	8.5	784	-	69.7	11.6	71.8	-
PbO concentrate	7.9	47.8	8.8	298	-	24.7	6	13.5	-

Product	Weight (%)	Grade				Distribution			
		Pb (%)	Zn (%)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)	Ag (%)	Cu (%)
PbS + PbO concentrate	23.9	60.6	8.6	623	-	94.3	17.6	85.3	-
ZnS concentrate	12.4	5.4	55.5	149	-	4.3	59.1	10.6	-
ZnO concentrate	8.1	1.3	29.8	43.8	-	0.7	20.7	2	-
ZnS + ZnO concentrate	20.6	3.8	45.3	108	-	5	79.8	12.6	-
Total tailings	55.5	0.2	0.6	7.9	-	0.7	2.6	2.5	-
LR11098-002 / Test 24 / HLS Master Composite									
Feed	100	20.9	23.5	332	0.7	100	100	100	100
PbS concentrate	20.2	67.7	7.2	1,078	2.4	65.4	6.2	65.5	70.5
PbO concentrate	8.2	38.5	10.3	413	0.4	15.2	3.6	10.2	5.4
PbS + PbO concentrate	28.4	59.2	8.1	885	1.8	80.6	9.8	75.7	75.9
ZnS concentrate	28.3	5.5	60.4	182	0.3	7.5	72.7	15.5	10.9
ZnO concentrate	10.1	12.2	23	133	0.4	5.9	9.9	4	6.2
ZnS + ZnO concentrate	38.4	7.3	50.6	169	0.3	13.3	82.6	19.5	17.1
Total tailings	33.2	3.83	5.4	47.5	0.14	6.1	7.6	4.8	7
LR11098-002 / Test 27 / HLS Master Composite									
Feed	100	17.8	22.7	277	-	100	100	100	-
PbS concentrate	20.4	56.2	16	871	-	64.4	14.4	64.1	-
PbO concentrate	8.3	47.9	8.1	413	-	22.3	3	12.4	-
PbS + PbO concentrate	28.7	53.8	13.7	739	-	86.7	17.4	76.5	-
ZnS concentrate	23.9	4.4	60.2	161	-	6	63.5	13.9	-
ZnO concentrate	7.7	4.8	31.1	137	-	2	10.5	3.8	-
ZnS + ZnO concentrate	31.6	4.5	53.1	155	-	8	74	17.6	-
Total tailings	39.7	2.39	4.95	41.2	-	5.3	8.6	5.9	-
LR11098-002 / Test 30 / HLS Master Composite									
PbS concentrate	17.2	75	4	1,032	-	62.8	3.1	58.7	-
PbO concentrate	9.3	57	6.1	440	-	25.7	2.6	13.5	-
PbS + PbO concentrate	26.5	68.7	4.7	825	-	88.5	5.7	72.2	-
ZnS concentrate	27.3	4.4	59.8	202	-	5.8	73.8	18.2	-
ZnO concentrate	8.1	1.9	33.3	116	-	0.7	12.2	3.1	-
ZnS + ZnO concentrate	35.4	3.8	53.8	182	-	6.5	86	21.3	-
Total tailings	38.1	2.69	4.85	51.5	-	5	8.3	6.5	-
LR11098-002 / Test 31 / Master Composite w/o HLS									
Feed	100	16	16.8	221	-	100	100	100	-
PbS concentrate	14.4	69.1	6.2	982	-	62.1	5.3	63.7	-
PbO concentrate	7.6	44.6	7	388	-	21.2	3.2	13.3	-
PbS + PbO concentrate	22	60.6	6.5	777	-	83.3	8.5	77	-
ZnS concentrate	20.5	5.9	59	130	-	7.6	72	12.1	-
ZnO concentrate	7.1	8.7	30	137	-	3.9	12.7	4.4	-
ZnS + ZnO concentrate	27.6	6.65	51.5	132	-	11.5	84.7	16.5	-
Total tailings	50.4	1.63	2.27	28.7	-	5.2	6.8	6.5	-
LR11098-002 / Test 37 / HLS High Oxide Composite									
Feed	100	13.6	16	223	-	100	100	100	-
PbS concentrate	11.3	59	12.7	1,069	-	49	9	54.2	-
ZnS concentrate	15.8	5.2	57.9	208	-	6	57.1	14.6	-
Total tailings	72.9	8.4	7.44	95.4	-	45	33.9	31.2	-
LR11098-002 / Test 38 / HLS Low Oxide Composite									
Feed	100	20.7	23.7	397	-	100	100	100	-
PbS concentrate	20.3	64.9	7.8	1,423	-	64	6.7	72.9	-
ZnS concentrate	28.5	3.4	62	114	-	4.6	74.4	8.2	-
Total tailings	51.2	12.7	8.7	147	-	31.4	18.9	18.9	-

Product	Weight (%)	Grade				Distribution			
		Pb (%)	Zn (%)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)	Ag (%)	Cu (%)
LR11098-002 / Test 40 / HLS Master Composite									
Feed	100	20	22	306	-	100	100	100	-
PbS concentrate	15.5	73.6	4.8	1,193	-	57.1	3.4	60.5	-
ZnS concentrate	25.1	4	62.2	186	-	5	71.1	15.3	-
Total tailings	59.4	12.8	9.5	125	-	37.9	25.5	24.2	-
LR12018-001 / Test 9 / HLS ROM Composite									
Feed	100	19.7	24.7	331	-	100	100	100	-
PbS concentrate	17.4	71.8	4.9	1,158	-	63.3	3.5	60.7	-
PbO concentrate	6.8	53.9	8.3	321	-	18.7	2.3	6.6	-
PbS + PbO concentrate	24.1	66.8	5.9	922	-	82	5.8	67.3	-
ZnS concentrate	32.2	5.1	58	261	-	8.3	75.6	25.3	-
Total tailings	43.7	4.4	10.5	56.3	-	9.7	18.6	7.4	-
LR12018-001 / Test 10 / HLS ROM Composite									
Feed	100	18.8	24.8	318	-	100	100	100	-
PbS concentrate	16.3	75.1	4.5	1,172	-	65	3	60	-
PbO concentrate	5.9	52.5	8.9	382	-	16.6	2.1	7.1	-
PbS + PbO concentrate	22.2	69.1	5.6	961	-	81.6	5.1	67.1	-
ZnS concentrate	32.4	4.6	57.6	251	-	7.9	75.4	25.6	-
Total tailings	45.4	4.3	10.6	51.5	-	10.5	19.5	7.3	-
LR12018-001 / Test 11 / HLS ROM Composite									
Feed	100	19.8	24.4	337	-	100	100	100	-
PbS concentrate	18.8	71.4	6.5	1,091	-	67.5	5	60.8	-
PbO concentrate	6.7	49	9.1	343	-	16.6	2.5	6.8	-
PbS + PbO concentrate	25.5	65.5	7.1	895	-	84.1	7.5	67.6	-
ZnS concentrate	31.7	5.3	57.4	267	-	8.4	74.7	25.2	-
Total tailings	42.8	3.5	10.1	56.8	-	7.5	17.8	7.2	-
LR50242-001 / Test 2 / DMS Composite									
Feed	100	19.5	31.6	320	0.84	100	100	100	100
PbS concentrate	19.8	50.7	24	999	2.7	51.3	15	61.7	63.2
PbO concentrate	1.78	44.6	11.5	662	1.95	4.1	0.6	3.7	4.1
PbS + PbO concentrate	21.58	50.2	23	971	2.6	55.4	15.6	65.4	67.3
ZnS concentrate	34.7	8.1	62.3	159	0.24	14.5	68.5	17.3	9.8
Total tailings	43.7	13.4	11.5	127	0.44	30.1	15.9	17.3	22.8
LR50242-001 / Test 3 / DMS Composite									
Feed	100	20.4	27.7	351	0.85	100	100	100	100
PbS concentrate	19.8	53.6	19.4	1,036	2.47	52	13.9	58.6	57.8
PbO concentrate	4.1	52.2	7.02	598	1.45	10.4	1	6.9	6.9
PbS + PbO concentrate	23.9	53.4	17.3	961	2.3	62.4	14.9	65.5	64.7
ZnS concentrate	33.3	7.62	57.3	191	0.36	12.4	68.8	18.1	14.2
Total tailings	42.8	12	10.5	134	0.42	25.2	16.2	16.4	21

Generally, the results for recovery of sulphide lead and zinc minerals were similar to the pre-2001 testing. Flotation of oxide lead and zinc minerals improved overall lead and zinc recoveries, but with lower combined resulting concentrate grades. The tests showed better slime suppression control was required for oxide flotation circuits, especially for oxide zinc.

The sulphide lead concentrate recovered about 50% to 70% of the lead and 57 to 72% of the silver. Concentrate lead grades (mainly between 60% and 75%), were higher than pre-2001. Oxide lead flotation further recovered about 15% to 34% of the lead to a concentrate containing about 48% lead (range from 38 to 57%). Silver recovery to oxide lead concentrate ranged from 7% to 15% and averaged about 11%, if removing anomalous test results.

Concentrate produced from sulphide zinc flotation recovered between 57% and 76% of the zinc and between 8% and 25% of the silver. Zinc concentrate grades ranged from 55% to 62% zinc. Oxide zinc flotation was performed after oxide lead flotation, recovering an additional 5% to 20% of the zinc (averaging 12%). Average oxide zinc concentrate grade was 30% zinc, with a range 23% to 38%. The oxide concentrate further recovered about 4% of the silver.

In 2015 GMR conducted a multi-cycle LCT test. It appeared that the lead and zinc performances were stable in the initial cycles and then more zinc reported to the lead concentrate. This implies that suppression of zinc minerals in the lead flotation circuit should be further optimized.

13.2.6.2 Stratabound (SMS) Mineralization

The 2005 testing by SGS had included a locked cycle test using a sample of 50% MQV and 50% SMS. The work appeared to confirm additional open cycle testing that co-processing of the two types of mineralization would not cause a significant impact on metallurgical performance.

The locked cycle procedures for SMS did not include an attempt to recovery an oxide mineral portion due to the low content present. Otherwise the testing looked at recycling the streams, following earlier developed procedures, although grind was reported to be coarser. The results are presented in the Table 13.20.

Table 13.20 SMS locked cycle test data

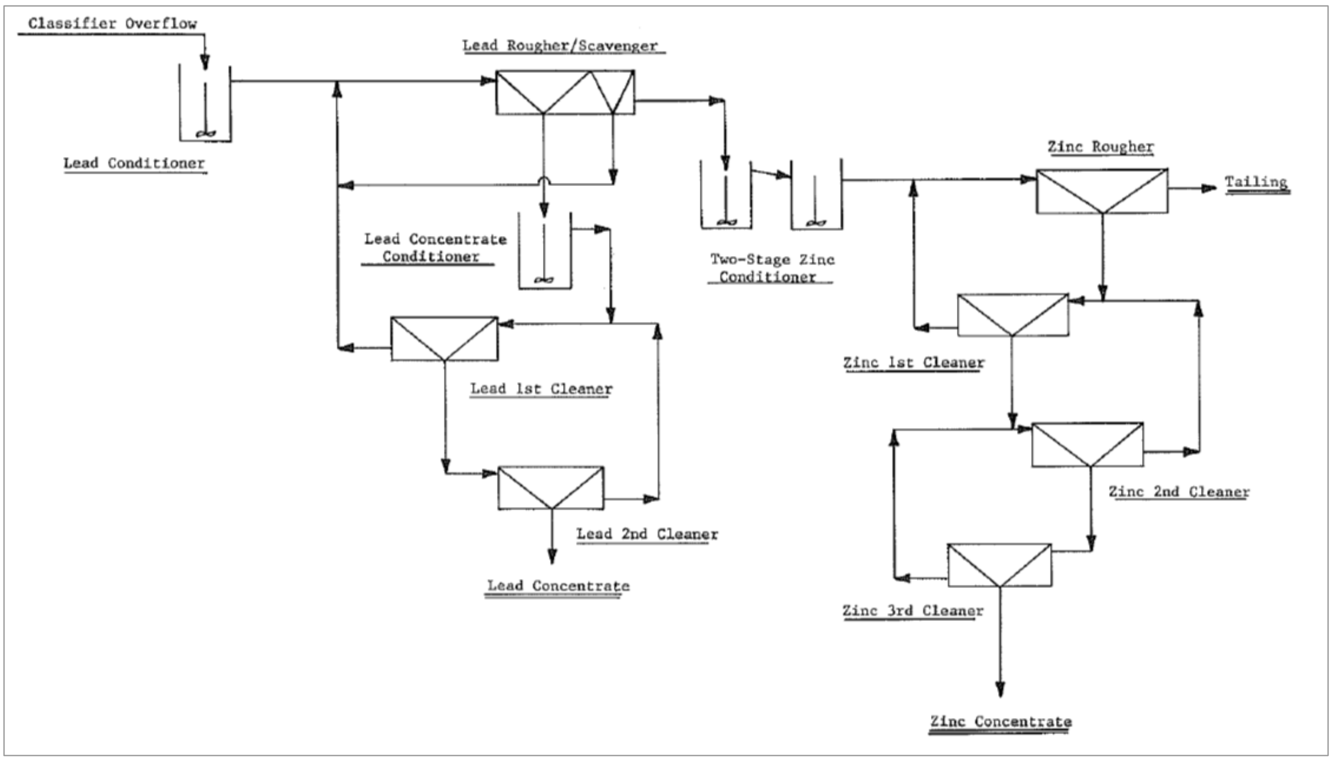
Product	Weight (%)	Grade			Distribution		
		Pb (%)	Zn (%)	Ag (g/t)	Pb (%)	Zn (%)	Ag (%)
KM370 / Test 8							
Feed	100	6.3	11.4	53	100	100	100
Lead concentrate	8.3	57.2	8.9	361	75.6	6.5	57.2
Zinc concentrate	15.3	1.6	57.7	60	4	77.3	17.4
Tailings	76.4	1.7	2.4	17	20.4	16.2	25.4
KM370 / Test 9							
Feed	100	6.3	11.5	55	100	100	100
Lead concentrate	8.8	57.8	7.5	372	80.6	5.8	59.7
Zinc concentrate	19.2	1.9	52	62	5.7	87.4	21.7
Tailings	72	1.2	1.1	14.1	13.7	6.8	18.6
KM462 / Test 13							
Feed	100	9.2	15.4	-	100	100	-
Lead concentrate	13.6	53.1	6.4	-	79	5.6	-
Zinc concentrate	22.5	3.3	59	-	8	85.9	-
Tailings	64	1.9	2.1	-	13	8.5	-
KM462 / Test 14							
Feed	100	9.6	16	130	100	100	100
Lead concentrate	13.8	53.3	4.3	349	76.7	3.7	37.1
Zinc concentrate	24.8	4.3	59.2	265	11.2	91.7	50.7
Tailings	61.4	1.9	1.2	25.4	12.1	4.6	12.2
KM462 / Test 15							
Feed	100	9.1	16.2	115	100	100	100
Lead concentrate	11	60.4	3.7	422	73.3	2.5	40.6
Zinc concentrate	23.8	2.9	62	199	7.6	91	41.4
Tailings	65.2	2.7	1.6	32.1	19.2	6.5	18
LR10916-001 / Test 60							
Feed	100	4.9	9	46.8	100	100	100
Lead concentrate	7.3	59.8	5.4	404	89.5	4.4	63.1
Zinc Concentrate	14.9	0.8	53.9	82.8	2.5	89.3	26.4
Tailings	77.7	0.5	0.7	6.3	8	6.3	10.5

While the data is difficult to evaluate due to the feed containing 50 wt.% oxidized MQV, the SMS appeared to provide for a modestly improved performance as compared to MQV alone. This is likely due to the lower extent of sulphide oxidation for the SMS deposit. The lead concentrate recovered about 73% to 89% of the lead at grades ranging from ~53% to 60% Pb. Corresponding silver recovery ranged from 17% to 50%, with grades of 350 g/t to 422 g/t Ag. The silver recovery to the lead concentrate is noted to be low with significant losses reporting with the zinc or lost to final tailing, probably because a lead oxide concentrate was not included. Approximately 77% to 92% of the zinc reported to the concentrate, which graded from 53% to 61% Zn.

13.2.7 Pilot plant testing

In 1982, CSMRI undertook flotation pilot plant testing to simulate the then proposed mill process. Test objectives were to provide operating and metallurgical data from the continuous operation of three flow schemes (Runs # 101, 102, 103) and produce lead and zinc concentrates representative of the full-scale mill operation. The flowsheets tested were similar, with the final two runs including a regrind prior to cleaning the lead. A flowsheet, not showing the regrind step, is provided in Figure 13.3.

Figure 13.3 CSMRI Pilot Plant Flowsheet - base case run #101



Run 101 is shown as in the preceding figure, while Runs 102 and 103 had minor modifications to the flowsheet. Modifications for Run 102 were that it undertook regrinding of the first lead cleaner tailing and lead rougher scavenger concentrate. Run 103 reground all rougher and scavenger concentrates and lead cleaner flotation with three stages of flotation instead of the two used in the previous two runs. The pilot plant feed rate was approximately 227 kg/h (500 lb/h), with sampling every 45 minutes for a minimum of three hours to produce composite samples for chemical and particle size analyses. Primary grind size was approximately 70% passing 74 µ (200 mesh). Soda ash / sodium cyanide was used in the lead circuit and lime / copper sulphate in the zinc circuit. A summary of the results is provided in Table 13.21 below.

Table 13.21 CSMRI pilot plant test results

Run	Process stream	Feed rate (lb/h)	Weight (%)	Grade				Distribution			
				Cu (%)	Pb (%)	Zn (%)	Ag (oz/t)	Cu (%)	Pb (%)	Zn (%)	Ag (%)
101	Flotation feed	492	100	0.37	10.1	11.9	5.61	100	100	100	100
	Lead 2nd cleaner concentrate	52.4	10.6	2.8	63.6	10.7	36.9	79.9	67.1	9.6	70
	Zinc 3rd cleaner concentrate	73.1	14.9	0.23	4.23	59.2	5.87	9.1	6.2	73.8	15.6
	Zinc rougher flotation tailings	366.5	74.5	0.06	3.62	2.67	1.09	11	26.7	16.7	14.5
102	Flotation feed	463	100	0.37	9.58	11.6	5.49	100	100	100	100
	Lead 2nd cleaner concentrate	43.3	9.4	2.97	64.5	9.44	40.4	74.8	62.9	7.6	68.7
	Zinc 3rd cleaner concentrate	65	14	0.26	2.96	61	5.76	9.7	4.3	74.2	14.7
	Zinc rougher flotation tailings	354.7	76.6	0.08	4.09	2.75	1.19	15.5	32.7	18.2	16.6
103	Flotation feed	442	100	0.35	9.5	12.8	5.27	100	100	100	100
	Lead 2nd cleaner concentrate	46.6	10.5	2.79	66.3	9.03	37.7	85	73.5	7.4	75.4
	Lead 3rd cleaner concentrate	34.9	7.9	2.57	69.1	7.74	35.5	58.8	57.4	4.8	53.2
	Zinc 3rd cleaner concentrate	79.8	18.1	0.42	5.4	56.8	7.9	22.2	10.3	79.9	27.1
	Zinc rougher flotation tailings	327.3	74	0.09	4.15	2.66	1.4	19.1	32.3	15.3	19.7

Pilot plant runs produced lead concentrates containing 64% to 69% lead, with 8% to 11% zinc, with incorporation of regrind showing a slight improvement to grade. Lead recovery based on the second cleaner concentrate was 63% to 73% into about 10% of the feed weight.

The zinc concentrates graded from 57% to 61% zinc, with 3% to 6% lead. Less lead was present if regrinding had been incorporated during prior lead cleaning. Zinc recovery was 74% up to 79% in Run 103.

13.2.8 Other test work

13.2.8.1 Settling tests

Flotation tailing

In 2011 Outotec performed a thickening test on two flotation tailings samples. No significant difference was observed. Particle size was 80% passing 91 µm. MF10 as flocculant at 30 g/t dosage gave a thickener underflow solid density of 69 to 71% at loading rates of 0.54 and 0.84 t/m²/h respectively. Results are summarized in Table 13.22, with no significant differences noted.

Table 13.22 Outotec settling test data – flotation tailing

Flocculant		Solid loading rate (t/m²/h)	Rise rate (m/h)	Underflow (w/w%)	Overflow (ppm)	Vane yield stress (Pa)
Type	Dosage (g/t)					
MF10	30	0.54	2.4	71	36	67
		0.84	3.8	69	56	47

The Phase 3 work performed in 2006 by SGS included dewatering tests on tailings and subsequently the concentrates. Tailings from both Upper Zone and Lower Zone MQV composites contained appreciable clay-like slimes, thus settling rates were relatively low. The thickener overflow also contained appreciable suspended solids. To achieve reasonably good settling rates, tailings pH was reduced to about 7.0 and with a significant amount of the flocculent, Magnafloc (MF) 351 added, as outlined in Table 13.23.

Table 13.23 SGS settling test data – flotation tailing

Test No.	pH	Flocculant		Initial settling rate $m^3/m^2/d$	Thickener U/F unit area $m^2/mt/d$	Thickener hydraulic unit area $m^2/mt/d$
		Type	g/t			
Upper zone composite						
S-3	7	Mag 351	34.4	27.59	0.154	0.089
S-4	6.8	Mag 351	50	412.5	0.057	0.006
Lower zone composite						
S-1	7	Mag 351	8.5	49.44	0.085	0.048
S-2	7	Mag 351	17.3	63.52	0.073	0.039

In 1982 CSMRI piloting settling tests were conducted on Run 103 flotation tailings. The results showed a clear supernatant was produced. The tailings settled rapidly and rate improved with the addition of lime or flocculant (Superfloc 1202, non-ionic), as shown in the Table 13.24.

Table 13.24 CSMRI settling test data - flotation tailing

Flocculant		Critical time* (min)	Unit rate ($ft^2/st/d$)
Type	Dosage (lb/ton)		
-	-	32	3.03
$Ca(OH)_2$	2.02	27	3.05
$Ca(OH)_2$	5.08	25	2.36
Superfloc 1202	0.035	17	1.76
Superfloc 1202	0.1	13	1.23

Flotation concentrates

SGS undertook settling tests on oxide and sulphide concentrates from the first two phases of their work performed in 2006. The concentrates had been frozen and were thawed and re-cleaned prior to the settling tests, which were performed at natural pH, with and without flocculant added. Table 13.25 shows the results. Good settling rates were achieved for all concentrates. Addition of Magnafloc 351, improved settling rate and supernatant clarity.

Table 13.25 SGS settling test data – flotation concentrates

Test No.	pH	Flocculant		Initial settling rate $m^3/m^2/d$	Thickener U/F unit area $m^2/mt/d$	Thickener hydraulic unit area $m^2/mt/d$
		Type	g/t			
Lead sulphide concentrate, particle size: 80% 46 µm						
S-5	8.3	-	-	47.64	0.022	0.035
S-6	8.3	Mag 351	6.4	494.4	0.008	0.004
Oxide lead concentrate, particle size: 80% 84 µm						
S-9	10.2	None	0	20.46	0.112	0.092
S-10	10.2	Mag 351	2.7	49.44	0.06	0.038
S-11	10.2	Mag 351	6.8	79.4	0.035	0.024
Zinc sulphide concentrate, particle size: 80% 90 µm						
S-7	9.6	-	-	28.94	0.034	0.059
S-8	10	Mag 351	6.6	300.9	0.009	0.006
Oxide zinc concentrate, particle size: 80% 74 µm						
S-12	10	-	-	57.17	0.029	0.037
S-13	10	Mag 351	7.8	349	0.007	0.006

The CSMRI 1982 pilot plant study included settlings tests by the modified Kynch method on flotation concentrates produced from Run 103. The addition of Superfloc 1202 was seen to improve concentrate settling rates. The addition of lime did not improve settling rates. Without flocculant, the conventional unit settling rate requirements were estimated to be 0.086 $m^2/mt/d$ (0.84 $ft^2/st/d$) for the lead concentrate and 0.128 $m^2/mt/d$ (1.25 $ft^2/st/d$) for the zinc concentrate, indicating good settling of both the concentrates – see Table 13.26.

Table 13.26 CSMRI settling test data – flotation concentrates

Test No.	Flocculant		Critical time* (min)	Unit rate ($ft^2/st/d$)
	Type	Dosage (lb/st)		
Zinc concentrate				
1	-	-	32	1.25
2	$Ca(OH)_2$	0.61	35	1.37
4	$Ca(OH)_2$	1.25	38	1.49
5	Superfloc 1202	0.034	22	0.85
Lead concentrate				
8	-	-	20	0.84
10	$Ca(OH)_2$	0.11	21	0.88
11	$Ca(OH)_2$	1.09	20	0.87
12	Superfloc 1202	0.01	15	0.55
14	Superfloc 1202	0.04	11	0.48

*Time to reach a solid density of 65% w/w.

13.2.8.2 Filtration testing on flotation concentrates

SGS performed filtration tests on the oxide and sulphide concentrates generated from the first two phases of study in 2006. The procedure used a vacuum pour-on method. Good cake production rates were achieved on each concentrate as shown in Table 13.27. Filter cake moisture ranged from 9.5% to 10.9%.

Table 13.27 SGS settling test data – flotation concentrates

Sample	Slurry			Total filtration cycle time (min)	Filter cake		
	Percentage solids (%)	pH	Mag 351 (g/t)		Thickness (mm)	Moisture (%)	Filtration rate $kg/m^2/h$
Oxide lead concentrate	59.6	10.2	10	4.5	10	9.5	365.9
Sulphide zinc concentrate	61.0	10.0	10	1.76	12	10.3	666.7
Sulphide lead concentrate	61.4	8.3	10	1.4	10	9.8	909.1
Oxide zinc concentrate	56.4	10.0	13	1.58	13	10.9	5882

Dry kg/ m^2/h; filter cloth: Neatex 3670/13 Total Filtration Cycle Time.

13.2.9 Miscellaneous testing procedures

Historically a number of miscellaneous testing procedures were attempted on a scoping level basis to address potential problems with treating the various products that were generated. However, none of these processes were developed to advanced consideration for the flowsheet. The work included:

CSMRI in 1983 undertook leaching of sulphide flotation tailings using ammonium hydroxide, sodium hydroxide, and sulphuric acid as lixiviates for lead and zinc extraction. Both ammonia and sulphuric acid extracted over 97% of zinc, but less than 1% of lead. Caustic leaching of cerussite flotation tailings extracted over 90% of lead and over 95% of zinc, but no silver. Caustic leaching of galena flotation tailings extracted about 95% of lead and zinc.

In a related CSMRI study whole ore caustic leaching extracted about 50% of lead and 80% of zinc. A precipitate with 52% lead, 14% zinc, and 25 oz/ton silver resulted from the caustic leach liquor. Subsequent precipitation of zinc hydroxide with carbon dioxide (CO_2) gave a concentrate of 64% zinc, 0.03% lead. Caustic soda regeneration was only partially successful, reportedly due to the method complexity.

Gravity concentration studies were performed by CSMRI in 1983 on both head sample and sulphide flotation tailings. The head sample procedures used shaking tables that recovered 75.8% of lead and 63.8% of zinc into a concentrate grading 24.4% lead and 20.1% zinc originating from feed with 11.8% lead and 10.6% zinc. A cerussite flotation tailings table test produced three products: concentrate, middling, and tailings. 56.9% of zinc was recovered into a table concentrate with 34.6% zinc. A related earlier test undertaken by O'Kane was also performed using hydrocyclone principles to recover non-sulphide values from sulphide zinc flotation tailings, but was shown to be unsuccessful.

Pyrite flotation following zinc flotation was performed on several SMS samples, due to its higher iron content. The product was obtained from zinc flotation tailings which followed upstream treatment of lime and cyanide addition in the lead flotation circuit, as well as copper sulphate in the zinc flotation circuit. The sulphide zinc flotation tailings were then conditioned with sulphur dioxide and floated using butyl xanthate collector. Pyrite rougher concentrates from three different SMS samples graded 37.5 to 40.4 % Fe. A similar procedure used on an MQV sample produced a pyrite concentrate of 9.3% Fe and 26.1% Zn.

A few procedures to reduce mercury content in the flotation concentrates were evaluated, although limited documentation remains. One procedure evaluated a polysulphide leach procedure on both the lead and zinc concentrates, using a 4% sodium sulphide / 2% sodium hydroxide (NaOH) solution. The results reduced mercury levels in the concentrates by ~16%. A second procedure used sodium cyanide (NaCN) resulting in reducing mercury content by 23%, to 5 g/l, in the lead concentrate. Residue from the zinc concentrate showed no reduction in mercury levels using NaCN. A later scoping study heated the zinc concentrate to 750°C to volatilize mercury. Results showed the mercury content was significantly reduced from 1,988 to 155 ppm on MQV zinc concentrate, and from 477 to 157 ppm Hg for SMS concentrate. A more thorough review of the options was reported by Thibault and Associates Inc. in a report issued to Canadian Zinc on 5 February 2015.

Hydrometallurgical procedures have been performed on the zinc concentrate, in large part due to the elevated mercury content that results in potentially significant smelter penalty charges. This has included pressure oxidation procedures which are used commercially in other operations worldwide. A report by Dynatec Corp. dated October 2002 indicated that ~98% zinc dissolution could be expected with the mercury remaining in the residue. The pregnant zinc solution would then be recovered by solvent extraction – electrowinning (SX-EW). However, due to significant capital costs, as well as some processing complications including loss of silver credit the program has not been followed up. Preliminary bioleaching was performed on the zinc concentrate by GMR in 2015 with some success. However there are no comparable commercial plants and operating costs are deemed to be excessive due to the high power requirements for extended periods to achieve dissolution of sphalerite.

13.2.10 Flotation concentrate characteristics

13.2.10.1 Lead concentrates

The grade achieved into most of the MQV sulphide lead concentrate produced is considered good, generally ranging from 55% to 70% Pb. The concentrates also contained from 800 to 1,200 g/t silver. Oxide lead concentrates from MQV gave significantly lower grade ranging from 38% to 57% Pb, with 300 to 430 g/t Ag. Impurity levels are significant, and of special concern are arsenic and antimony, which are potential penalty elements from the smelters. Mineralogical studies showed that arsenic and antimony are intimately associated with the copper minerals, which typically report into the lead concentrate. The analyses of the MQV lead concentrates generated from the bulk sampling zone are shown in Table 13.28 and Table 13.29.

Table 13.28 MQV analyses of sulphide and oxide lead flotation concentrates

Concentrate		**Sulphide lead concentrate**								**Oxide lead concentrate**		
Test program	**LR10916 -001**	**LR11098-001**		**LR11098 -002**	**KM 440**	**CSMR**	**LR2252**		**LR1091 6-001**	**LR11098-001**		**LR11098 -002**
Head sample	**Comp 1**	**HLS 930 Comp**	**HLS 883 Comp**	**HLS M Comp**	**Comp ***	**-**	**Comp**	**Comp**	**Comp 1**	**HLS 930 Comp**	**HLS 883 Comp**	**HLS M Comp**
Lead (Pb) %	63.3	70.9	69	71.5	70	67.5	54.3	54.3	44.2	50.5	50.8	56.5
Zinc (Zn) %	8.55	6	8.22	3.93	7.5	8.56	19.2	15.4	5.8	8.58	7.97	6.09
Copper (Cu) %	3.11	1.75	1.41	1.96	2.8	2.97	1.8	1.9	0.17	0.31	0.69	0.42
Iron (Fe) %	0.48	0.71	0.76	0.14	1	0.33	0.56	0.87	2.03	0.91	2.87	1.06
Cobalt (Co) %	<0.02	<0.02	<0.02	<0.02	-	<0.001	-	-	<0.02	<0.02	<0.02	<0.02
Arsenic (As) %	0.53	0.29	0.24	0.35	0.5	0.54	0.29	0.34	0.096	0.028	0.054	0.27
Antimony (Sb) %	1.43	0.94	0.67	1.09	1.2	1.4	-	-	0.16	0.075	0.14	0.67
Tin (Sn) %	<0.002	<0.002	<0.002	<0.002	-	<0.01	-	-	<0.002	<0.002	<0.002	<0.002
Sulphur (S) %	14.3	13.1	14.1	12.9	-	16.6	-	-	2.19	1.61	3.22	1.27
Carbon (total) %	0.43	0.34	0.53	0.17	-	0.39	-	-	4.3	4.81	4.37	4.09
Germanium (Ge) g/t	<4	<4.0	<4.0	<4.0	-		-	-	<4	<4.0	<4.0	<4.0
Selenium (Se) g/t	<10	<20	<20	<10	-	<30	-	-	<15	<20	<20	23
Fluorine (F) %	<0.005	<0.01	0.01	<0.01	-	0.024	-	-	0.014	0.02	0.01	<0.01
Chlorine (Cl) g/t	303	210	105	90	-	1100	-	-	47	450	30.3	52
Titanium (Ti) g/t	69	58	90	<40	-		-	-	220	308	150	90
Calcium (Ca) %	0.25	0.28	0.07	<0.04	-	0.082	-	-	2.4	1.69	1.19	0.43
Magnesium (Mg) %	0.1	0.11	0.1	0.024	-		-	-	1.2	0.929	0.64	0.21
Manganese (Mn) g/t	31	30	60	<20	-	20	-	-	130	170	160	70
Aluminum ($A_{12}0_3$) %	0.1	0.18	0.4	<0.08	-	0.051	-	-	0.42	0.73	<0.4	0.24
Silica ($Si0_2$) %	1.2	1.07	0.92	0.48	-	0.52	-	-	12	8.41	9.49	6.07
Bismuth (Bi) g/t	<400	20	<20	<20	-	100	-	-	<400	30	<20	<20
Cadmium (Cd) %	0.08	0.044	0.06	0.034	0.045	0.069	0.17	0.15	0.036	0.073	<0.09	0.047
Mercury (Hg) g/t	1120	550	810	562	550	830	360	360	660	40	310	936
Gold (Au) g/t	0.11	0.04	0.12	0.03	-	0.062	-	-	0.06	0.13	0.06	0.05
Silver (Ag) g/t	1,246	791	815	1,034	1,100	1,126	737	813	374	309	297	438

* Estimated concentrations by laboratory.

The grades of the lead concentrate from the SMS were lower than those from the MQV at about 55% Pb, with the levels of impurities also generally reduced. The grade range for metal payables and the principal impurity elements occurring in an SMS lead concentrate blend are shown in the following table, with Comp. 2 actually being a 50:50 blend of SMS and MQV material.

Table 13.29 SMS analyses of sulphide and oxide lead flotation concentrates

Concentrate	Sulphide lead concentrate		Oxide lead concentrate
Test program	**KM462**	**LR10916-001**	**LR10916-001**
Head sample	**Comp***	**Comp 2****	**Comp 2 ****
Lead (Pb) %	55	67	49.2
Zinc (Zn) %	5	7.32	4.82
Copper (Cu) %	<0.1	2.64	0.6
Iron (Fe) %	15	1.89	1.5
Cobalt (Co) %		<0.02	<0.02
Arsenic (As) %	<0.01	0.42	0.091
Antimony (Sb) %	0.045	1.28	0.17
Tin (Sn) %		<0.002	<0.002
Sulphur (S) %		15.9	0.79
Carbon (total) %		0.42	4.92
Germanium (Ge) g/t		<7	<4
Selenium (Se) g/t		<15	<15
Fluorine (F) %		0.005	0.009
Chlorine (Cl) g/t		449	592
Titanium (Ti) g/t		53	140
Calcium (Ca) %		0.42	3.2
Magnesium (Mg) %		0.18	1.6
Manganese (Mn) g/t		43	220
Aluminum ($A_{12}0_3$) %		0.11	<0.075
Silica ($Si0_2$) %		1.1	9.2
Bismuth (Bi) g/t		<400	<400
Cadmium (Cd) %		0.07	0.047
Mercury (Hg) g/t	40	676	56
Gold (Au) g/t		0.04	0.12
Silver (Ag) g/t	450	1,102	341

* Estimated concentrations by laboratory.
** Blended sample (50%S MS and 50% Stratabound).

13.2.10.2 Zinc concentrates

For both the MQV and SMS material the zinc sulphide concentrates ranged from 55% to 62% Zn, although much lower grades were evident for subsequent zinc oxide concentrates (when produced). There is also less iron and copper, while some impurities, such as arsenic and antimony, could be a concern. However, mercury content is elevated, averaging over 2,000 ppm for MQV sulphide zinc concentrate. These levels are likely to be penalized by most smelters and may limit marketability. Mineralogical investigation showed the mercury to be intermittently associated with zinc minerals. Concentrate from the SMS showed lower mercury content, as did oxide zinc concentrates. Cadmium present in the MQV mineralization was also concentrated into the zinc sulphide concentrates. Average cadmium concentration would be expected to be in the range of 0.2% to 0.5% Cd. The analyses of the zinc concentrates are outlined in Table 13.30 and Table 13.31.

Table 13.30 MQV analyses of sulphide and oxide zinc flotation concentrates

Concentrate		Sulphide zinc concentrate							Oxide zinc concentrate			
Test program	LR109 16-001	LR11098-001		LR11098 - 002	KM 440	CSMR	LR2252		LR109 16 - 001	LR11098-001		LR11098 -002
Head sample	Comp 1	HLS 930 Comp	HLS 883 Comp	HLS M Comp	Comp *	-	Comp	Comp	Comp 1	HLS 930 Comp	HLS 883 Comp	HLS M Comp
Lead (Pb) %	3.29	3.58	5.52	4.02	2.5	4.52	3.6		6.5	2.76	1.52	2.53
Zinc (Zn) %	61.3	61.7	57.8	60.1	58 - 60	58.1	55.5		31.4	30.4	31.1	32
Copper (Cu) %	0.17	0.25	0.26	0.31	0.4	0.36	0.4		0.71	0.41	0.26	0.54
Iron (Fe) %	0.36	0.56	0.81	0.36	1.2	0.77	1.12		1.49	0.73	1.26	1.08
Cobalt (Co) %	<0.02	<0.02	<0.02	<0.02	0.001	<0.001			<0.02	<0.02	<0.02	<0.02
Arsenic (As) %	0.05	0.064	0.082	0.079	0.15	0.05	0.44		0.01	0.045	0.049	0.073
Antimony (Sb) %	0.04	0.065	0.07	0.12	0.14	0.105	-		0.16	0.075	0.067	0.11
Tin (Sn) %	<0.002	<0.002	<0.002	<0.002		<0.01			<0.002	<0.002	<0.002	<0.002
Sulphur (S) %	29.4	29.9	27.8	30		29.8			0.42	0.31	0.39	0.28
Carbon (total) %	0.36	0.38	0.85	0.43		0.51			7.74	6.94	7.5	6.5
Germanium (Ge) g/t	<4	<4	<4	6	2				<4	<4	<4	<4
Selenium (Se) g/t	14	<20	<20	<10		<30			14	28	<20	17
Fluorine (F) %	<0.005	<0.01	0.01	<0.01	0.007	0.028			<0.005	0.02	0.03	0.02
Chlorine (Cl) g/t	144	63	80	57		400			171	101	69	58
Titanium (Ti) g/t	15	48	30	<40	50				260	270	270	300
Calcium (Ca) %	<0.04	0.21	0.11	0.11	0.21	0.23			2.9	2.21	3.69	1.68
Magnesium (Mg) %	0.04	0.066	0.06	0.061	0.02				1.8	1.26	2.19	0.98
Manganese (Mn) g/t	47	60	110	70		60			440	430	530	400
Aluminum ($A_{12}0_3$) %	0.07	0.13	<0.4	0.1		0.098			0.57	0.74	0.8	0.79
Silica ($Si0_2$) %	1.2	1.2	2.53	1.36	1	1.84			8.7	20.6	17.5	23.1
Bismuth (Bi) g/t	<400	<20	<20	<20		<10			<400	<20	<20	<20
Cadmium (Cd) %	0.34	0.24	0.31	0.36	0.5	0.359	0.44		0.15	0.24	0.091	0.22
Mercury (Hg) g/t	2,330	1,520	2,200	2,730	3,500	2,200	1,270		373	20	220	477
Gold (Au) g/t	0.14	0.12	0.37	0.08			0.25		0.07	0.09	0.06	0.03
Silver (Ag) g/t	100	128	143	190	218		168		220	138	19	117

* Estimated concentrations by laboratory.

Table 13.31 SMS analyses of sulphide and oxide zinc flotation concentrates

Concentrate	**Sulphide zinc concentrate**		**Oxide zinc concentrate**
Test program	**KM462**	**LR10916-001**	**LR10916-001**
Head sample	**Comp***	**Comp 2****	**Comp 2 ****
Lead (Pb) %	1.0	2.02	2.71
Zinc (Zn) %	60 -63	59.3	34.1
Copper (Cu) %	0.15	0.14	0.54
Iron (Fe) %	2.0	3.02	1.51
Cobalt (Co) %	< 0.001	<0.02	<0.02
Arsenic (As) %	0.02	0.04	0.06
Antimony (Sb) %	0.10	0.04	0.13
Tin (Sn) %	-	-	<0.002
Sulphur (S) %	-	31.6	0.38
Carbon (total) %	-	0.27	8.20
Germanium (Ge) g/t	40	58	<4
Selenium (Se) g/t	-	<15	<15
Fluorine (F) %	0.009	<0.002	0.02
Chlorine (Cl) g/t	-	118	109
Titanium (Ti) g/t	40	13.0	220
Calcium (Ca) %	0.11	1.9	3.9
Magnesium (Mg) %	0.02	0.07	2.4
Manganese (Mn) g/t	-	56	640
Aluminium ($A_{12}0_3$) %	-	<0.07	0.62
Silica ($Si0_2$) %	0.60	0.90	7.1
Bismuth (Bi) g/t	-	<400	<400
Cadmium (Cd) %	0.20	0.34	0.17
Mercury (Hg) g/t	900	1,740	56
Gold (Au) g/t	-	0.12	0.12
Silver (Ag) g/t	78	83.2	145

* Estimated concentrations by laboratory.
** Blended sample (50% Vein and 50% Stratabound).

13.2.11 Metallurgical performance projection

Based on the historic laboratory test results Tetra Tech WEI Inc. engineering undertook to estimate the metallurgical recovery and grade to the lead and zinc concentrates for pre-feasibility evaluation. Further test work was also recommended to better characterize the mineralization, and for other untested areas of the resource zones. The assumptions and results for their metallurgical projection are provided in Table 13.32.

Table 13.32 Projection of metallurgical performance

Process circuit	Value	Note
MQV (main quartz vein) mineralization		
DMS pre-concentration		
- Mass recovery, %	= 0.8484 x (head grade, % - lead +zinc) + 55.013	< 50% Pb+Zn
	= 98	> 50% Pb+Zn
- Lead recovery, % - sulphide	= 1.293 x ln (head grade, % - sulphide lead) + 95.794	0 - 7.88% Pb-Sulphide
	= 0.0628 x (head grade, % - sulphide lead) + 97.97	7.88 - 16% Pb-Sulphide
	= 99.2	> 16% Pb-Sulphide
- Oxide	= 1.0907 x ln (head grade, % - oxide lead) + 94.632	
- Zinc recovery, % - sulphide	= 0.6549 x ln (head grade, % - sulphide zinc) + 97.107	< 50% Zn-sulphide
	= 99.7	> 50% Zn-sulphide
- Oxide	= 85	< 0.5% Zn-oxide
	= 4.211 x ln (head grade, % - oxide zinc) + 87.579	0.5 – 10.5 Zn-oxide
	= 97.5	> 10.5% Zn-oxide
- Silver recovery, %	= 0.0436 x (mass recovery, %) + 94.508	> 50% mass recovery
	= 90	< 50% mass recovery
Sulphide lead flotation		
- Concentrate grade, % - lead	= 68.5	
- Zinc	= 7.4	
- Recovery, % - lead	= (head grade, % - sulphide lead) / (head grade, % - total lead) x (68.477 x (head grade, % - sulphide lead) $^{0.1253}$)	0 - 17.4% Pb-sulphide
	= (head grade, % - sulphide lead) / (head grade, % - total lead) x 98	> 17.4% Pb-sulphide
- Silver	= 10.117 x (lead recovery to sulphide lead concentrate, %) $^{0.4421}$	
Sulphide zinc flotation		
- Concentrate grade, % - zinc	= 59	
- Lead	= 1.5519 x (lead head, %) $^{0.3717}$	
- Recovery, % - zinc	= (head grade, % - sulphide zinc) / (head grade, % - total zinc) x 50	< 1% Zn-sulphide
	= (head grade, % - sulphide zinc) / (head grade, % - total zinc) x (0.2217 x (head grade, % - sulphide zinc) + 90.135)	1 - 30% Zn-sulphide
	= (head grade, % - sulphide zinc) / (head grade, % - total zinc) x 98	> 30% Zn-sulphide
- Silver	= 0	< 1.5% lead recovery to sulphide zinc concentrate
	= 3.0962 x (lead recovery to sulphide zinc concentrate, %) - 3.7592	1.5 – 7.5% lead recovery to sulphide zinc concentrate
	= 20	> 7.5% lead recovery to sulphide zinc concentrate
Oxide lead flotation		
- Concentrate grade, % - lead	= 48	
- Zinc	= 7.7	
- Recovery, % - lead	= (head grade, % - oxide lead) / (head grade, % - total lead) x 70.5	
- Silver	= 0.4084 x ln (lead recovery to oxide lead concentrate, %) +2.5177	> 2% Pb - oxide
	= 1.0	< 2% Pb - oxide
SMS (stratabound) mineralization		
DMS pre-concentration		

Process circuit	Value	Note
- Mass recovery, %	= 88	
- Lead recovery, % - sulphide	= 98	
- Oxide	= 98	
- Zinc recovery, % - sulphide	= 98	
- Oxide	= 98	
- Silver recovery, %	= 98	
Sulphide lead flotation		
- Concentrate grade, % - lead	= 56	
- Zinc	= 8.6905 x (zinc head / lead head) - 9.3002	Zinc head / lead head <1.35, cap at 2%; if zinc head / lead head >2.0, cap at 8.5%
- Recovery, % - Lead	= 83	
- Silver	= 106.87 x ln(lead recovery to sulphide lead concentrate, %) - 415.34	> 55% lead recovery to sulphide lead concentrate
	= 10	< 55% lead recovery to sulphide lead concentrate
Sulphide zinc flotation		
- Concentrate grade, % - zinc	= 57	
- Lead	= 1.351 x ((lead head, %) x (100-83)/(100- (lead head, %) x 83/56))$^{0.9335}$	Zinc circuit head
- Recovery, % - zinc	= 0.198 x (zinc head, %) + 86.362	
- Silver	= 3.5965 x (lead recovery to sulphide zinc concentrate, %) + 9.2277	Cap at 40%

13.3 2017 test program

In January 2017, a laboratory test program was initiated on samples representing previously untested zones of the Prairie Creek mineral resource. These consisted of deeper zones in the MQV and STK material, obtained from a drill program conducted in 2015. The test work initially followed the earlier developed flowsheet, while focussing on simplifying the flotation procedure, reagent scheme, and optimizing the grind. The related work concluded in May 2017, and SGS issued these results in a report dated 28 July 2017 as project # 16016-001.

The pre-concentration conditions based on historical test work were incorporated into the 2017 program. This consisted of crushing to 12.7 mm (½”) and screening fines at 1.4 mm (12 Tyler mesh). The -12.7 mm +1.4 mm material was subjected to dense media separation (DMS) using a ferrosilicon medium adjusted to SG 2.75. DMS sinks and the screened -1.4 mm fines were combined and forwarded to evaluation using various differential flotation procedures. The 2017 samples did not have sufficient oxide lead or copper present to justify evaluating separate circuits for recovering these two associated minerals. Consequently, the differential flotation procedures focussed on optimizing concentration of 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 tailings.

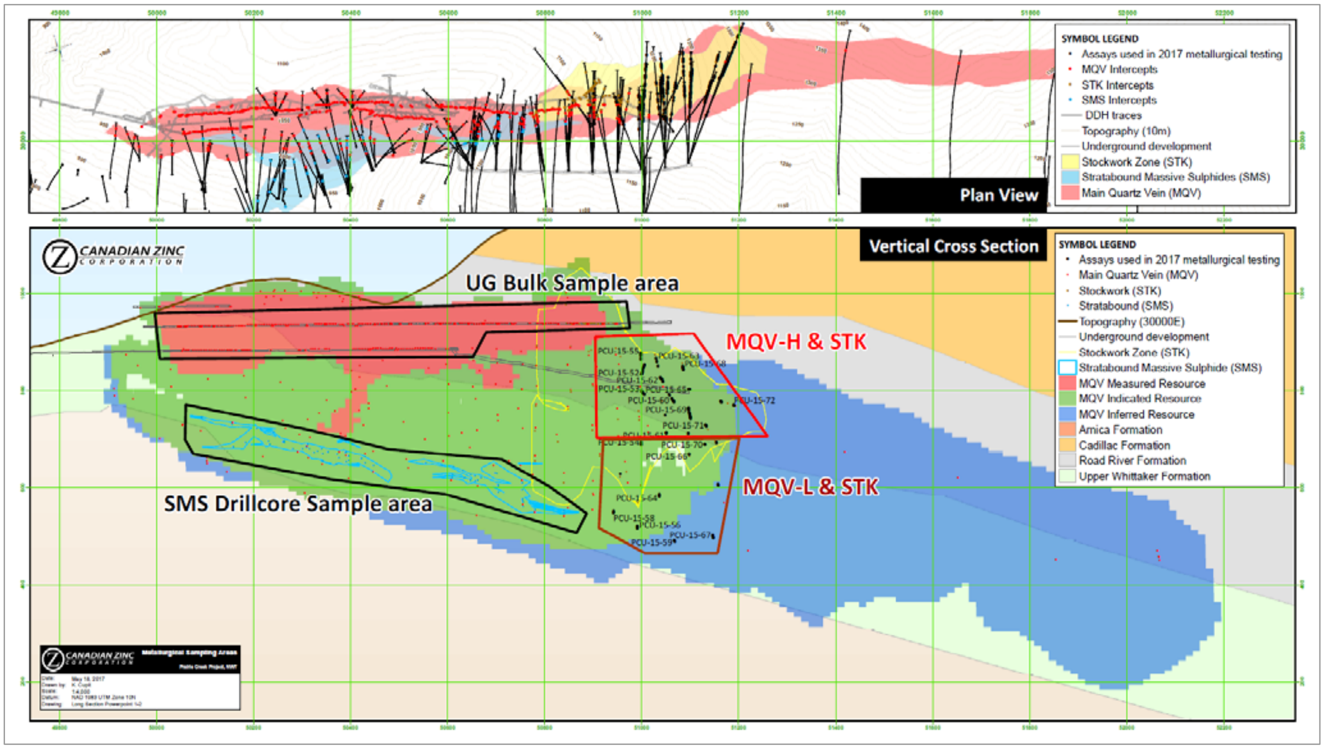
Mineral process testing was performed by SGS Canada Limited, primarily at their laboratory in Burnaby, BC. This included quantitative evaluation of minerals by scanning electron microscopy (QEMSCAN), as well as flotation and related studies. Procedures including DMS piloting and liquid – solid separation,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

13.3.1 Origin of metallurgical composite samples

Three master composites originating from 2015 split drill core were primarily used for testing flotation procedures and flowsheet development. These composite identifications consisted of the STK, and two areas of greater depth in the MQV, broken out initially as upper main quartz vein zone (MQV-H), and lower main quartz vein zone

(MQV-L). The zones from which the samples were obtained are shown in Figure 13.4 below; also shown are the bulk sampling areas of the MQV and SMS that were used in historical test work.

Figure 13.4 Sample locations



The drill hole number and corresponding weights of the intervals used to make up the master composites are provided in Table 13.33, Table 13.34, and Table 13.35.

Table 13.33 Master composite origin – STK

Drill hole ID	From (m)	To (m)	Wt (kg)
PCU-15-52	114.5	115.5	2.67
PCU-15-52	115.5	116.5	2.43
PCU-15-52	117.2	118.3	1.95
PCU-15-52	120.3	121.3	2.71
PCU-15-52	121.3	122.3	2.32
PCU-15-52	129.2	130.2	3.29
PCU-15-52	132.6	133.6	2.56
PCU-15-52	133.6	134.6	2.57
PCU-15-52	135.7	136.6	2.24
PCU-15-52	136.6	137.7	3.14
PCU-15-52	137.7	138.7	2.52
PCU-15-52	149.1	150.6	2.62
PCU-15-52	150.6	152.1	1.82
PCU-15-52	152.1	153.5	1.75
PCU-15-52	173.0	174.0	2.93
PCU-15-52	176.8	177.9	3.06
PCU-15-52	193.8	194.6	2.58
PCU-15-52	194.6	196.0	3.73
PCU-15-52	196.0	197.0	2.71
PCU-15-52	210.0	211.0	2.11
PCU-15-52	214.5	215.5	2.83
PCU-15-53	124.7	126.2	3.49
PCU-15-53	126.2	126.6	1.11
PCU-15-53	130.8	132.3	3.56
PCU-15-53	132.3	133.8	4.25
PCU-15-53	149.2	150.2	2.96
PCU-15-53	150.2	151.5	3.69
PCU-15-53	151.5	152.5	2.36
PCU-15-53	152.5	153.6	2.39
PCU-15-53	153.6	154.7	2.49
PCU-15-53	160.9	161.3	2.33
PCU-15-53	161.3	162.9	2.76
PCU-15-53	162.9	164.3	2.86
PCU-15-53	173.4	174.4	2.09
PCU-15-53	174.4	175.4	3.08
PCU-15-53	175.4	176.4	2.67
PCU-15-53	176.4	178.0	3.96
PCU-15-53	189.0	190.0	2.68
PCU-15-53	190.0	191.5	3.55
PCU-15-53	191.5	193.0	3.72
PCU-15-53	193.0	194.0	2.67
PCU-15-55	126.2	127.9	2.36
PCU-15-55	131.6	132.3	1.47
PCU-15-55	139.9	141.0	2.60
PCU-15-55	141.0	142.0	2.43
PCU-15-55	142.0	143.0	2.88

Drill hole ID	From (m)	To (m)	Wt (kg)
PCU-15-55	144.0	144.8	2.75
PCU-15-55	144.8	146.0	2.50
PCU-15-55	146.0	146.7	0.88
PCU-15-55	146.7	148.0	3.26
PCU-15-55	148.0	149.1	3.09
PCU-15-55	149.1	149.9	2.00
PCU-15-55	149.9	151.1	3.20
PCU-15-55	151.1	152.0	2.47
PCU-15-55	152.0	152.9	2.03
PCU-15-55	152.9	154.1	2.96
PCU-15-55	154.1	155.0	2.36
PCU-15-55	155.0	156.0	2.51
PCU-15-60	138.4	139.4	2.54
PCU-15-60	139.4	140.4	2.88
PCU-15-60	144.4	145.4	3.80
PCU-15-60	145.4	146.4	2.76
PCU-15-60	158.5	159.5	3.10
PCU-15-60	164.6	165.7	3.08
PCU-15-60	165.7	166.7	3.04
PCU-15-60	166.7	167.7	2.80
PCU-15-60	167.7	168.7	2.72
PCU-15-60	168.7	169.7	2.92
PCU-15-60	169.7	170.7	2.78
PCU-15-60	170.7	171.7	2.62
PCU-15-60	171.7	172.7	2.36
PCU-15-60	172.7	173.7	2.46
PCU-15-60	173.7	175.0	3.16
PCU-15-60	176.5	177.5	2.96
PCU-15-61	182.9	183.9	2.56
PCU-15-61	184.6	185.6	2.10
PCU-15-61	185.6	186.5	2.26
PCU-15-61	186.5	187.3	1.92
PCU-15-61	187.3	188.4	2.70
PCU-15-61	188.4	189.2	1.84
PCU-15-61	189.2	189.9	1.60
PCU-15-61	190.6	191.2	1.42
PCU-15-62	146.1	147.1	2.38
PCU-15-62	151.5	152.2	2.20
PCU-15-62	154.1	155.1	2.16
PCU-15-62	157.0	158.0	2.56
PCU-15-62	159.7	160.6	2.38
PCU-15-62	176.0	176.7	1.50
PCU-15-62	176.7	177.5	2.02
PCU-15-62	203.9	205.1	3.28
PCU-15-62	205.1	206.2	2.62
PCU-15-62	206.2	207.2	2.68
PCU-15-62	207.2	208.2	2.94

Drill hole ID	From (m)	To (m)	Wt (kg)
PCU-15-63	143.5	144.5	2.44
PCU-15-63	147.4	148.4	2.54
PCU-15-65	133.3	133.9	1.62
PCU-15-65	133.9	135.0	3.90
PCU-15-65	187.4	188.4	3.00
PCU-15-68	153.6	154.6	2.84
PCU-15-68	154.6	155.6	2.10
PCU-15-68	155.6	156.7	3.10
PCU-15-68	158.0	158.8	2.22
PCU-15-68	159.9	161.1	3.54
PCU-15-68	161.1	162.2	4.12
PCU-15-68	163.3	164.1	2.26
PCU-15-68	164.7	165.8	3.42
PCU-15-68	165.8	166.8	3.12
PCU-15-68	173.4	173.9	1.32
PCU-15-68	173.9	174.9	2.34
PCU-15-69	139.1	140.1	2.58
PCU-15-69	140.1	140.9	2.28
PCU-15-69	140.9	141.7	1.48
PCU-15-69	141.7	143.0	3.18
PCU-15-69	159.0	160.0	2.70
PCU-15-69	160.0	161.0	2.78
PCU-15-69	162.8	163.8	2.96
PCU-15-69	163.8	164.7	2.16
PCU-15-69	166.3	167.3	3.06
PCU-15-69	170.9	171.9	3.10
PCU-15-69	173.5	174.5	2.38
PCU-15-69	182.6	184.1	4.16
PCU-15-69	186.0	187.0	3.00
		Total	**323.8**

Table 13.34 Master composite origin, MQV-H

Drill hole ID	From (m)	To (m)	Wt (kg)
PCU-15-52	104.50	107.50	7.36
PCU-15-53	101.80	106.38	9.84
PCU-15-55	120.61	123.25	6.68
PCU-15-60	96.23	100.28	4.04
PCU-15-61	117.04	118.57	2.02
PCU-15-62	98.50	99.80	2.52
PCU-15-62	179.70	185.65	12.06
PCU-15-63	114.94	117.55	5.44
PCU-15-63	159.85	162.35	5.18
PCU-15-65	114.00	117.04	7.46
PCU-15-65	211.80	214.88	6.26
PCU-15-66	127.41	131.98	8.18
PCU-15-68	144.48	149.05	9.24
PCU-15-69	122.38	133.55	27.98

Drill hole ID	From (m)	To (m)	Wt (kg)
PCU-15-69	190.00	203.06	38.94
PCU-15-71	160.57	169.04	21.64
PCU-15-72	191.40	200.74	31.40
PCU-15-72	263.69	268.68	8.58
		Total	**214.8**

Table 13.35 Master composite origin, MQV-L

Drill hole ID	From (m)	To (m)	Wt (kg)
PCU-15-54	166.04	176.70	11.57
PCU-15-56	326.27	332.71	15.12
PCU-15-58	213.58	214.58	3.21
PCU-15-58	301.45	307.54	15.05
PCU-15-59	367.00	371.60	15.30
PCU-15-64	275.54	284.68	15.04
PCU-15-66	191.00	196.44	10.42
PCU-15-67	349.82	357.53	15.40
PCU-15-70	150.90	155.33	11.08
PCU-15-70	266.09	271.50	10.40
PCU-15-71	248.65	252.02	8.04
		Total	**130.6**

Prior to testing the master composites an initial composite of assay rejects originating from both upper and lower zones of the main quartz vein (MQV-AR) was generated in order to begin flotation scoping studies. Work was undertaken on assay rejects in order to preserve the limited master composite weight and to expedite the testing schedule, while the master composites were subjected to DMS. The origin of the MQV-AR composite sub-samples is provided in Table 13.36.

Table 13.36 Origin of composite MQV-AR

Drill hole ID	From	To	Length (M)	Source zone MQV-	Weight (kg)
PCU-15-53	101.80	106.38	4.58	H	9.84
PCU-15-55	120.61	123.25	2.64	H	6.68
PCU-15-58	213.58	214.58	1.00	L	3.21
PCU-15-60	96.23	100.28	4.05	H	4.04
PCU-15-61	117.04	118.57	1.53	H	2.02
PCU-15-63	114.94	117.55	2.61	H	5.44
PCU-15-63	159.85	162.35	2.50	H	5.18
PCU-15-66	191.00	196.44	5.44	L	10.4
PCU-15-68	144.48	149.05	4.57	H	9.24
PCU-15-69	122.38	133.55	11.17	H	28.0
PCU-15-69	190.00	203.06	13.06	H	19.5
PCU-15-70	150.90	155.33	4.43	L	11.1
PCU-15-70	266.09	271.50	5.41	L	10.4
PCU-15-72	263.69	268.68	4.99	H	8.58
				Total	**133.6**

Once flotation conditions for the master composites were set, further study included the response of three variability composites that were available from archived assay rejects. 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 DMS. The origin of the sub-samples used to make up the variability composites is provided in Table 13.37, Table 13.38, and Table 13.39.

Table 13.37 Sample origin – composite MQV-Fe

CZN sample ID	AGAT[1] lab sample ID	Length (m)	Weight (kg)
A512678	6734433	0.90	2.60
25428	6460962	1.00	3.21
25375	6412567	1.05	3.09
25371	6412563	1.22	2.50
25368	6412560	1.07	2.88
25437	6460971	0.75	2.17
25335	6412527	1.00	2.67
25334	6412526	1.00	3.08
25331	6412523	1.60	2.76
25408	6460942	0.98	2.88
25402	6460936	1.22	3.16
25387	6412579	0.96	2.96
25565	6581914	1.25	3.78
25564	6581912	1.08	3.32
25465	6461000	0.85	2.38
		Total	**43.44**

1. AGAT Laboratories is a laboratory service provider with divisions that offer service to the mining industry.

Table 13.38 Sample origin - Composite MQV-Cu

CZN sample ID	AGAT lab sample ID	Length (m)	Weight (kg)
25424	6460958	1.51	4.46
25425	6460959	1.20	3.15
25423	6460957	0.78	2.39
25422	6460956	1.05	2.48
25421	6460955	0.55	1.95
25434	6460968	1.00	2.82
25467	6461002	1.00	2.95
25608	6461003	0.71	2.14
25464	6460998	0.75	2.70
25463	6460997	1.00	3.59
25461	6460995	0.99	3.12
		Total	**31.75**

Table 13.39 Sample origin - composite MQV-Pbox

CZN sample ID	AGAT lab sample ID	Length (m)	Weight (kg)
25369	6412561	0.95	2.87
25283	6388823	1.40	1.75
25272	6388812	0.94	2.24
25273	6388813	1.06	3.14
25377	6412569	1.15	3.20
25554	6581902	0.86	2.38
25269	6388809	0.97	2.56
25268	6388808	1.00	3.29
25272	6388812	0.94	2.24
25264	6388804	1.00	2.32
25262	6388802	1.00	2.91
25460	6460994	1.00	3.34
		Total	**32.24**

13.3.2 Composite head characterization

The master composites originated from specific spatial zones and had corresponding variations in head grade and mineralogy at greater depth in the resource than those tested historically. The samples also had less sulphide oxidation than those samples that were historically tested. Composite grade variation included following the principal metals of value consisting of lead, zinc, and silver. 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 (TOC). For the master composites these parameters were also re-analyzed following DMS treatment.

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 13.40 below for the three master composites.

Table 13.40 Master composite head analyses

Sample ID	Pb %	Zn %	Pb as Oxide, Pb %	Zn as Oxide, Zn %	Ag g/t	Cu %	Fe %	S %	S= %	Hg g/t	Sb g/t	As g/t
STK	3.86	7.46	0.43	0.029	58.6	0.18	0.956	5.19	4.85	117	680	558
MQV-H	14.7	16	0.39	0.051	222	0.40	0.316	10.1	9.82	436	2050	949
MQV-L	5.9	4.74	0.39	0.017	84	0.18	2.33	5.66	5.42	162	917	552

Similarly head assays for composites generated from assay rejects were analyzed and are provided in Table 13.41.

Table 13.41 Assay reject and variability composites head analyses

Sample ID	Pb %	Zn %	Pb as Oxide, Pb %	Zn as Oxide, Zn %	Ag g/t	Cu %	Fe%	S %	TOC %	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	2.10	0.38	0.008	137	0.37	5.28	7.95	0.15	106	1660	159
MQV-Cu	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	0.40	0.017	133	0.17	0.47	7.3	0.40	158	847	< 70

Mineralogical evaluation representing float feed for two of the master composites (MQV-H and STK) was performed by QEMSCAN. Note that this represents the material following rejection of the majority of gangue minerals to the DMS float. The related information was issued by SGS on 6 March 2017. A synopsis of the results is provided below in Table 13.42 and Table 13.43 respectively for MQV-H and STK.

Table 13.42 Mineral distribution float feed MQV-H

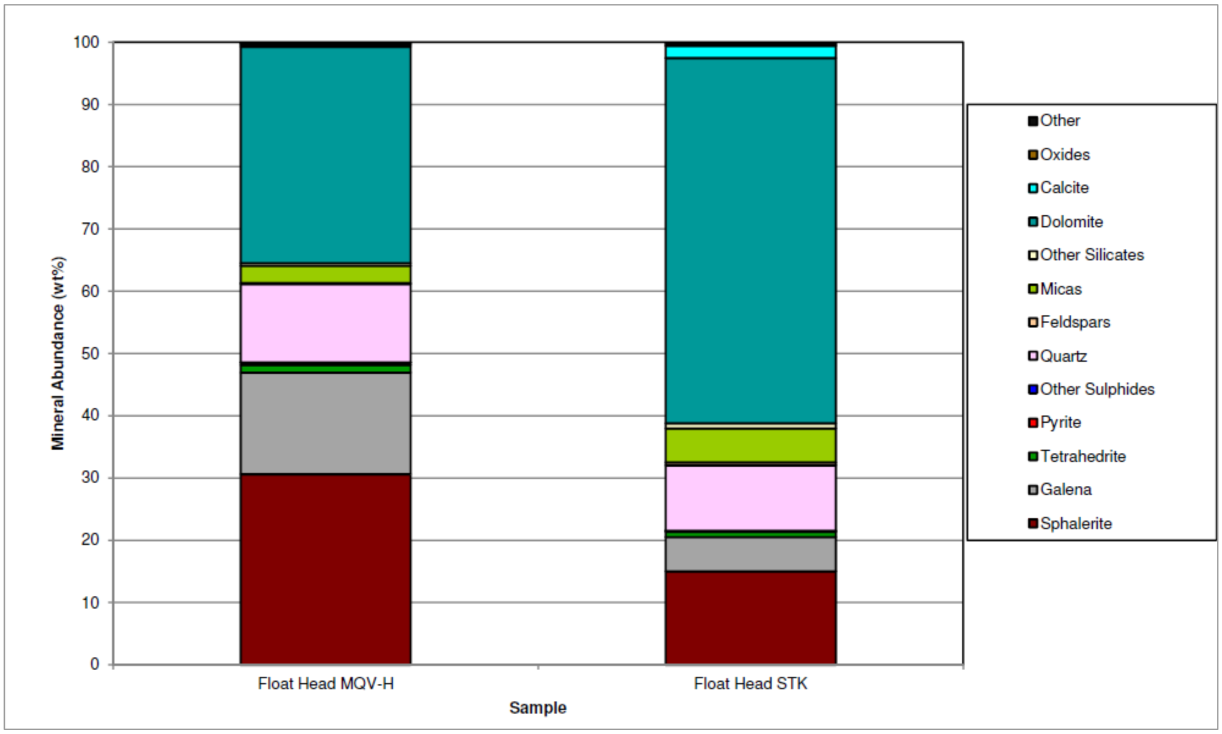
Fraction		Combined	+75 µm		−75 µm	
Mass size distribution (%)			33.7		66.3	
Calc. particle size (µm)		21	85		16	
		Sample	**Sample**	**Fraction**	**Sample**	**Fraction**
Mineral mass (%)	Sphalerite	30.6	12.9	38.2	17.7	26.7
	Galena	16.3	3.40	10.1	12.9	19.5
	Tetrahedrite	1.25	0.37	1.09	0.88	1.33
	Pyrite	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
	Micas	2.77	0.58	1.73	2.19	3.30
	Other Silicates	0.45	0.10	0.30	0.34	0.52
	Dolomite	34.7	11.0	32.7	23.7	35.8
	Calcite	0.39	0.12	0.37	0.26	0.40
	Oxides	0.16	0.01	0.04	0.14	0.22
	Other	0.18	0.02	0.05	0.16	0.24
	Total	100.0	33.7	100.0	66.3	100.0

Table 13.43 Mineral distribution float feed STK

Fraction		Combined	+75 µm		−75 µm	
Mass size distribution (%)			30.8		69.2	
Calc. ESD particle size (µm)		20	86		15	
		Sample	**Sample**	**Fraction**	**Sample**	**Fraction**
Mineral mass (%)	Sphalerite	15.0	6.00	19.5	9.03	13.0
	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	0.00	0.00	0.01	0.02
	Quartz	10.5	3.96	12.9	6.54	9.45
	Feldspars	0.49	0.15	0.48	0.34	0.50
	Micas	5.44	0.98	3.18	4.47	6.45
	Other Silicates	0.81	0.20	0.66	0.60	0.87
	Dolomite	58.7	17.1	55.5	41.6	60.1
	Calcite	1.99	0.68	2.21	1.32	1.90
	Oxides	0.27	0.03	0.09	0.24	0.35
	Other	0.30	0.04	0.13	0.26	0.38
	Total	100.0	30.8	100.0	69.2	100.0

The results confirm that 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 13.5.

Figure 13.5 Modal distribution Float Feed MQV-H and STK



Size distribution of the major minerals is provided in Figure 13.6 and Figure 13.7, respectively for MQV-H and STK.

Figure 13.6 Float Head STK Grain Size

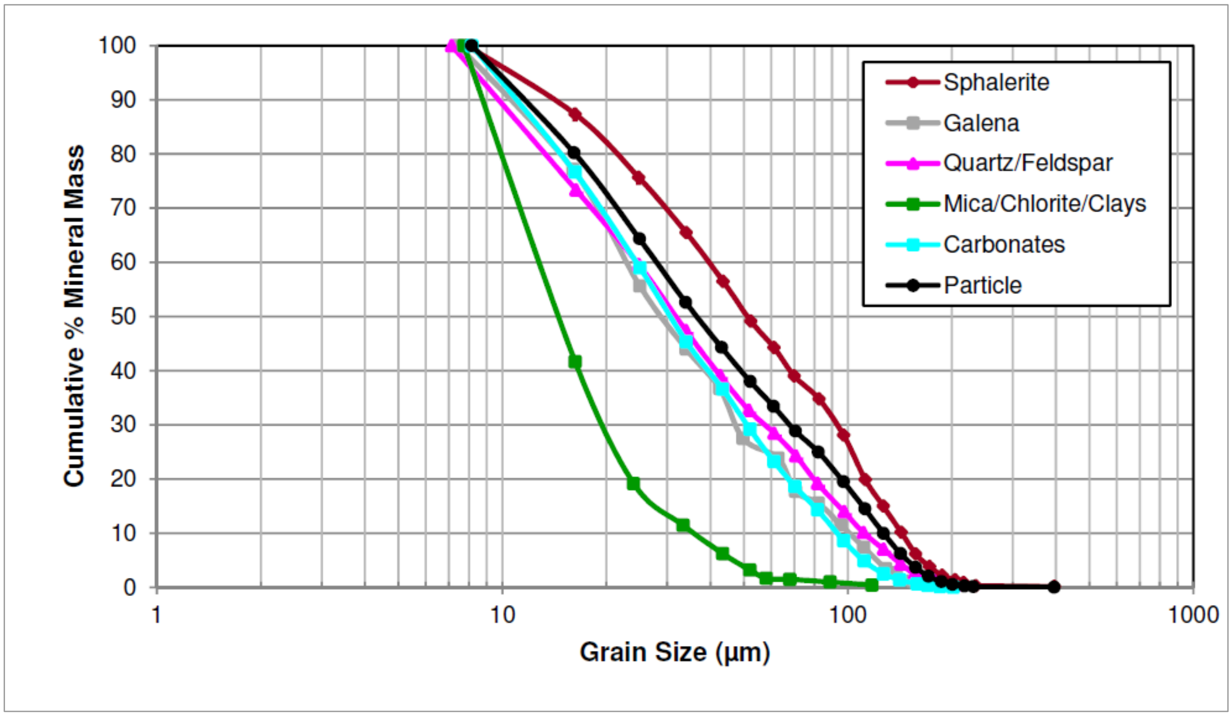
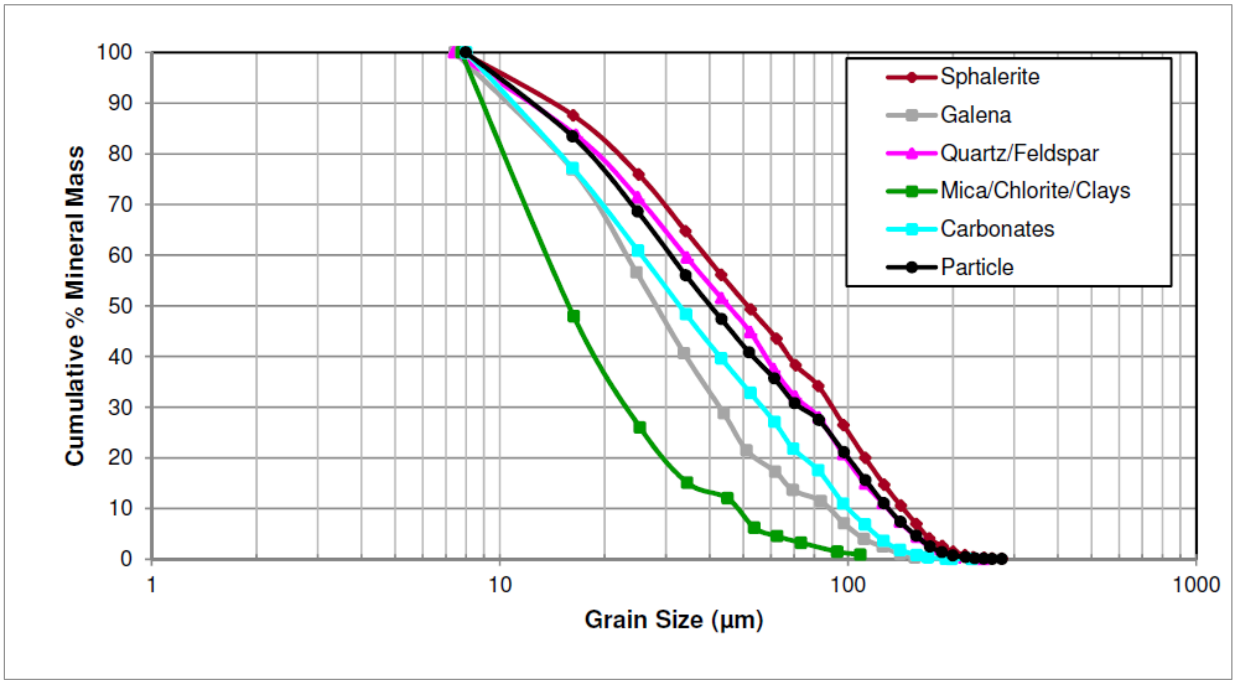


Figure 13.7 Float Head MQV-H Grain Size

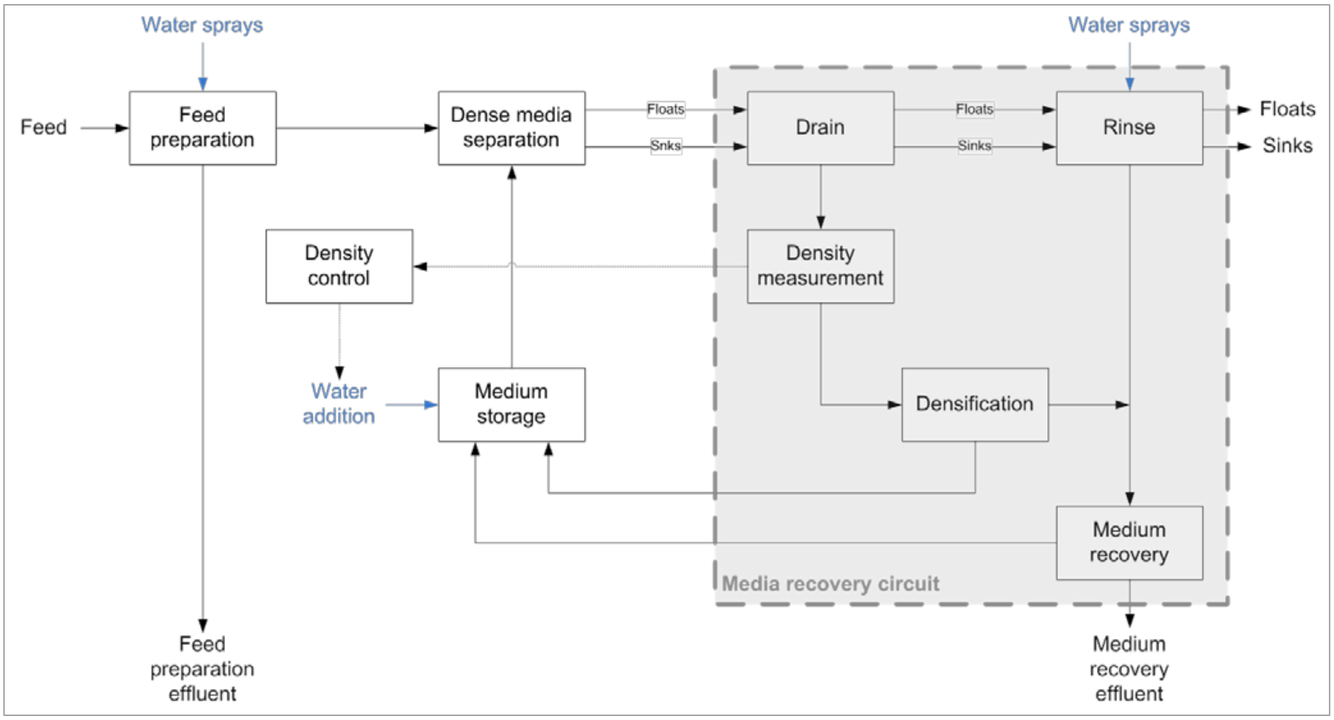


Examination indicated that, for both these master composites, the liberation of galena and sphalerite particles is considered good, and the samples should respond well to standard flotation procedures at moderate grinds.

13.3.3 Dense media separation

DMS was performed on each of the three master composites MQV-H, MQV-L and STK, using procedures that had been developed with the historical metallurgical laboratory test work. Sample preparation and 2017 pilot testing using DMS were performed at the SGS facilities in Lakefield, Ontario. After crushing the split drill core to a nominal 12.7 mm (½”), the fines were removed by screening at 1.4 mm. The minus 12.7 mm, plus 1.4 mm fraction was pumped to the 200 mm Multotec cyclone at feed rates ranging from 185 kg/h to 223 kg/h. The DMS media incorporated ferrosilicon adjusted to SG 2.75. Ferrosilicon was subsequently recovered for reuse by magnetic separation. The DMS floats were removed as reject. The DMS sinks were combined with the fines and forwarded to further mineral testing procedures. A flowsheet of the DMS pilot plant circuit is provided, courtesy of SGS, in Figure 13.8.

Figure 13.8 SGS dense media pilot plant flowsheet



The DMS sink 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 DMS products consisting of the sink, and rejects (float), as well as the fines removed prior to DMS, are provided for each of the master composites in the Table 13.44, Table 13.45, and Table 13.46 below.

Table 13.44 DMS balance on composite MQV-H

Product	Wt (kg)	Wt %	Assay					% distribution				
			% Pb	% Zn	g/t, Ag	% S	% TOC	Pb	Zn	Ag	S	TOC
DMS sink	118.10	62.62	21.1	21.4	250.00	13.1	0.20	88.9	78.1	78.6	79.2	39.2
DMS float	43.20	22.91	0.36	0.49	10.0	0.47	0.43	0.55	0.65	1.15	1.04	30.8
DMS fines	27.30	14.48	10.8	25.2	279	14.1	0.66	10.5	21.3	20.3	19.7	29.9
DMS sink+fines	145.40	77.09	19.2	22.1	255	13.3	0.29	99.4	99.3	98.9	99.0	69.2
Head (calc.)	188.60	100.00	14.9	17.2	199	10.4	0.32	100	100	100	100	100

Table 13.45 DMS balance on composite MQV-L

Product	Wt (kg)	W t%	Assay					% distribution				
			% Pb	% Zn	g/t, Ag	% S	% TOC	Pb	Zn	Ag	S	TOC
DMS sink	67.0	53.3	8.49	6.20	139	7.58	0.11	56.0	59.5	59.1	64.3	34.0
DMS float	31.7	25.2	0.27	0.30	10.0	0.63	0.22	0.84	1.36	2.01	2.53	32.2
DMS fines	27.1	21.5	16.2	10.1	226	9.67	0.27	43.2	39.2	38.9	33.2	33.8
DMS sink+fines	94.1	74.8	10.7	7.32	164	8.18	0.16	99.2	98.6	98.0	97.5	67.8
Head (calc.)	125.8	100.0	8.08	5.55	125	6.28	0.17	100	100	100	100	100

Table 13.46 DMS balance on composite STK

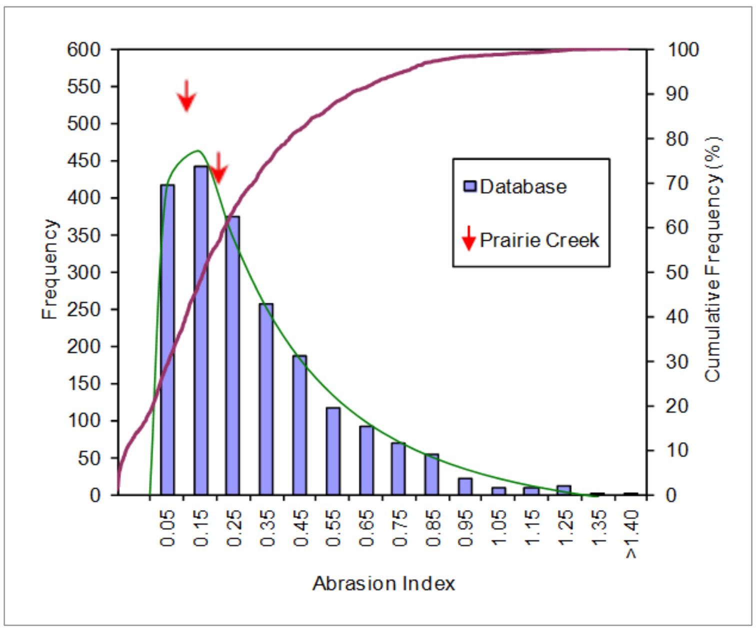
Product	Wt (kg)	Wt %	Assay					% distribution				
			% Pb	% Zn	g/t, Ag	% S	% TOC	Pb	Zn	Ag	S	TOC
DMS sink	177.0	56.5	4.35	9.3	95.4	5.37	0.33	64.4	66.3	63.0	66.4	49.8
DMS float	74.4	23.7	0.16	0.24	5.1	0.22	0.40	1.0	0.7	1.4	1.1	25.4
DMS fines	61.9	19.8	6.67	13.30	154	7.50	0.47	34.6	33.0	35.6	32.4	24.8
DMS sink+fines	238.9	76.3	4.95	10.36	111	5.92	0.37	99.0	99.3	98.6	98.9	74.6
Head (calc.)	313.3	100.0	3.81	7.96	85.5	4.57	0.37	100	100	100	100	100

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

13.3.4 Comminution

Abrasion (Ai) testing, used for calculating grinding media consumption and liner wear rates, was performed on the master composite sinks. The sinks were produced by heavy media separation with a media SG 2.75, on nominal 19 mm (¾") 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 13.9.

Figure 13.9 Abrasion chart



Each of the three master composite DMS sinks were tested for Bond Ball Mill Work Index (BMWi) using a closing mesh size of 105 microns (105 Tyler mesh). Results are provided in Table 13.47 below.

Table 13.47 Bond ball mill grindability test summary

Sample ID	Mesh of grind	F_{80} (µm)	P_{80} (µm)	Gram per revolution	Work index (kWh/t)	Hardness percentile	Category
MQV-H (DMS sink)	150	2,485	81	1.41	13.9	45	Medium
MQV-L (DMS sink)	150	2,294	82	1.45	13.8	44	Medium
STK (DMS sink)	150	2,355	77	1.49	13.0	35	Moderately soft

The work index range of 13 to 14 kWh/tonne 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, and lower extent of sulphide oxidation for the composites that were used in the 2017 test program.

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

13.3.5 Flotation

A series of flotation studies were performed to determine the response of composited mineral samples representing previously untested zones of the resource. Typically the composites had previously been pre-concentrated by the DMS procedures discussed above. The studies consisted of:

Open Cycle Bench (2 kg) Tests: With a goal to simplify the pre-existing flotation procedure including for reagent scheme, coarser grind, and the overall flowsheet. Testing was initially performed on assay reject composite MQV-AR, prior to proceeding to using the DMS products generated from the three master composites. 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.

10 kg Batch Studies: Larger scale 10 kg batch tests were undertaken on a global mix of primarily the master composite DMS sinks and fines. The testing was performed in order 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.

Locked Cycle Testing: The tests were performed to better confirm the optimized open cycle response on the DMS product (sink + fines) of the 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.

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. Generally oxide content was too low to include oxide lead flotation for the 2017 metallurgical composites. Flotation cleaning was accomplished in two to three stages, both with and without use of regrinding in the lead circuit. No regrinding was used prior to cleaning zinc. A more detailed summary of the procedure and corresponding results of each set of tests is provided below.

13.3.5.1 Open cycle flotation

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 less recycle streams. Test conditions initially used the historic 80% passing particle size (K80) primary grind of ~80 µm. Based on the initial eight tests a procedure was then undertaken to test the master composite DMS 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 K80 of 85 microns, 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 test work proceeded at evaluating coarser grind sizes that could more easily be accommodated with modifications to the existing circuit, continuing up to a K80 of 156 microns. The K80 of 156 microns was selected based on modelling by Ausenco, which indicated the existing refurbished mill could accommodate this grind at 1,200 tonnes/day throughput, assuming a BWI of < 12 kWh/tonne. The results of the primary grind are provided in Table 13.48.

Table 13.48 Primary grind (no regrind) vs flotation response

Comp.	Primary grind	Pb Bulk	2nd Pb Conc.		2nd Zn Conc.		Final tail grade	
ID	K80 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

The grind data indicates that a significantly coarser primary grind from that used in the majority of the historical test work 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 13.10 and Figure 13.11, 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 13.10 Grind vs lead recovery and zinc misplacement (MQV-H)

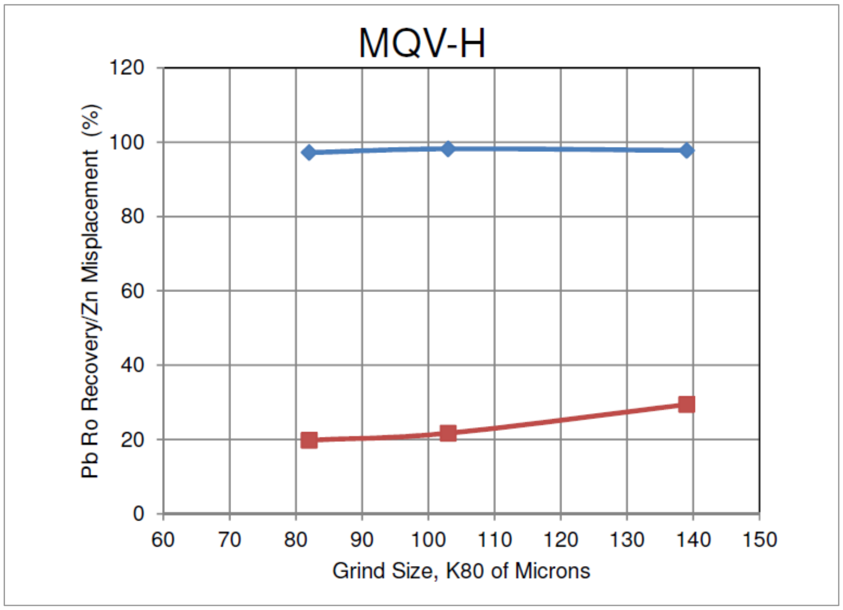


Figure 13.11 Grind vs lead recovery and zinc misplacement (MQV-L)

The data suggested that a coarser grind can be accommodated with minimal effect on flotation response, particularly if regrinding of the lead rougher concentrate was used to assist in improving the final lead concentrate grade. A follow-up examination on a coarser grind of K80 ~156 µm, along with a brief polish regrind, was performed on combined Main Quartz Vein 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 are provided in Table 13.49.

Table 13.49 Primary grind (with regrind) vs flotation response

Comp. ID	Primary grind	Pb bulk	3rd Pb Conc.		2nd Zn Conc.		Zn Ro. tail grade	
	K80 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

If a brief regrind is incorporated with the coarser primary grind of K80 ~156 µm, then the results are similar to the K80 ~110µm grind. Further study showed why this might be expected, with results shown in Table 13.50.

Table 13.50 Comp MQV-HL lead rougher float kinetic response

Stream	Retention	Size	Grade of concentrate (%)			Distribution (%)			
	Minutes	K80 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	8.0	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

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 would 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.

13.3.5.2 Variability flotation testing

Variations to the optimized open cycle float procedure were undertaken to address specific characteristics of each master composite. Among the observations was that 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 that 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 (TOC) analyses, required a more specialized approach.

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 the Table 13.51.

Table 13.51 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

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 was 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 test work, oxide lead flotation is anticipated to be incorporated for feeds earlier in the mine life that may have higher oxide content. While high oxide feed was not available for the 2017 program, the use of soda ash was evaluated as an alternate pH modifier, with results shown in the following tables. Also included in the following tables 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 DMS, 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 test work 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 13.52 and Table 13.53, respectively, for the response on the lead and zinc concentrates.

Table 13.52 Lead flotation variability response on higher iron composites

Comp. ID	Modifier used	pH range		Calc head grade / Wt. Ratio			Final Pb Conc % Rec.			Final Pb Conc grade %		
		Ro. float	Cleaners	% Pb	% Fe	Fe:Pb	Mass	% Pb	% Fe	% Pb	% Zn	% Fe
MQV-HL	Lime	9.0	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	8.0	9-10	12.2	1.8	0.14	16.9	85.5	32.1	61.6	4.40	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	10.0	3.1	0.31	15.1	89.7	18.6	59.2	5.65	3.9
MQV-Fe	Lime	9.0	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	0.90	11.9	88.9	27.9	41.6	2.85	11.4

*Finer primary grind used.
**Collector & $CuSO_4$ dosage decreased.

Table 13.53 Zinc flotation variability response on higher iron composites

Comp. ID	Modifier used	pH range		Calc head grade / Wt. Ratio			Final Zn Conc % Rec.			Final Zn Conc grade %		
		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

*Finer primary grind used.
**Collector & $CuSO_4$ dosage decreased.

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 alternative procedures may need to be investigated. While it is unlikely the high iron to base metal ratios of MQV-Fe head grade (shown in previous table) would be encountered for any extended time during processing, investigation of additional iron depressants and / or decreasing collector dosage and $CuSO_4$ addition can be applied should the galena or sphalerite content decrease in the feed with respect to the iron ratio.

The response of the remaining two variability composites MQV-Cu and MQV-Pbox are shown in Table 13.54 and Table 13.55 for lead and zinc concentrate.

Table 13.54 Lead flotation response variability samples

Comp. ID	Final (3rd Cl) Pb Conc recovery %			Final (3rd Cl) lead Conc grade							
	Lead	Zinc	Mass	% Pb	Ag, g/t	% Zn	% Cu	% Fe	Hg, ppm	Sb, ppm	As, ppm
MQV-Pbox	79.3	1.2	12.2	76.3	898	1.1		0.15	104		
MQV-Cu	88.4	4.6	15.6	62.8		4.4		1.25	306		

Table 13.55 Zinc flotation response variability samples

Comp. ID	Final (2nd Cl) Zn Conc recovery %			Final (2nd Cl) zinc Conc grade							
	Zinc	Lead	Mass	% Zn	Ag, g/t	% Pb	% Cu	% Fe	Hg, ppm	Sb, ppm	As, ppm
MQV-Pbox	80.9	0.5	13.8	66.6	26	0.43		0.27	924		
MQV-Cu	80.8	0.3	18.5	64.6		0.18		1.05	1030		

The MQV-Pbox composite was shown to still have too low an oxide content to sufficiently justify incorporating a separate oxide flotation circuit. Both MQV-Cu and MQV-Pbox responded well, and with a good results comparison to the baseline testing done on the master composite DMS products.

13.3.5.3 10 kg batch flotation

A final set of open cycle tests was undertaken in order to produce sufficient quantities of product to provide concentrate samples to better establish potential smelter terms and to provide slurried tailings 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 needed to be kept wet, the final mass balance (including middling analyses) was not performed, although assay splits were taken for each of the two final concentrates, and the two tailings (Zn 1st Cl scavenger tailing; Zn rougher tailing) produced for each test. The blend for the main feed consisted of master composite DMS 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.

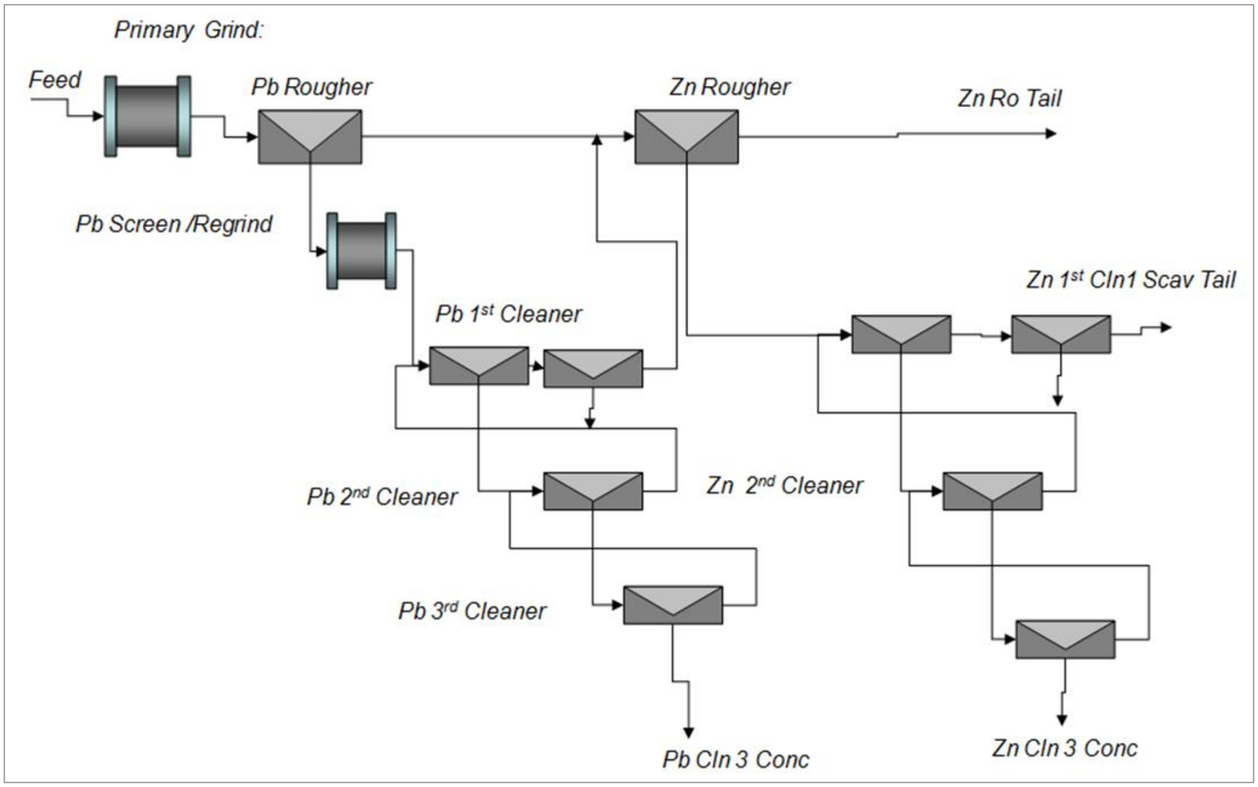
The 10 kg testing used a primary grind K80~135 µm. Lime was used as the pH modifier. Reagent addition initially followed that used in bench scale work (including for locked cycle), but dosage reductions were undertaken as the testing proceeded. The results indicated that the various collector dosages, when compared to the 2 kg open cycle tests, could be reduced by about 30% for average expected feed grades. Addition of the zinc sulfate depressant, and copper sulfate activator were also modestly reduced without apparent negative consequences to the float response. The required float retention time, particularly during lead roughing, was 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.

13.3.5.4 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 (LCT1, LCT2, LCT3) on the master composite DMS sinks plus screen fines, representing flotation feed. The procedure was simplified, as compared to the historic flotation flowsheet, with less scavenging and middling recycling incorporated, as well as reduced reagent requirements. Two of the tests (LCT1 and 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 Main Quartz Vein resource zone, the principal mineralized zone of the project. The remaining locked cycle test (LCT2) was performed on STK material.

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. Due to the low oxide content of the material, a lead oxide float circuit was not incorporated, although it is assumed to be required during initial mining periods when higher oxidized feed may be expected to be encountered. 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, courtesy of SGS, in Figure 13.12 below.

Figure 13.12 Locked cycle flotation flowsheet



The particle size of the locked cycle flotation feed was varied based on start-up of the existing Prairie Creek facility and anticipated expansion of the milling capacity. Consequently, there is more limited grinding capability anticipated at the start of operations. The primary grind product particle size was increased as compared to historical test work. The STK locked cycle test (LCT2) used a primary grind of K80 ~135 µm, which is the primary grind envisioned to be in place later in the mine life, when STK material is expected to be mined. The remaining two locked cycle tests on MQV-HL were performed under similar conditions, with the exception being the grind and cleaner float retention time used. The lower and upper grind range (K80 ~110 to 156 µm) were used to represent potential differences in particle size due to changes in the ball mill work index, as well as the planned addition of primary grinding capacity following the start of operations.

The reagent scheme was developed during the open cycle study. However dosages can be further optimized to suit 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 the STK test, but not for the MQV-HL tests. Any mill feed with an elevated Pb to total organic carbon (TOC) 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 was 9 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. The STK material had lower iron content than the 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 13.56 includes the calculated heads generated from the three tests, which compared well to the assayed head master composites presented previously. Also shown in Table 13.56 are the tailing analyses.

Table 13.56 Locked cycle - calculated head / tailing assay

Comp ID	Test No.	Calc. head grade				Zn 1st Cl tail grade			Zn Ro. tail grade		
		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

The grind and regrind 80% passing particle size as represented by the K80, as well as the mass distribution from the final product streams, are provided in Table 13.57.

Table 13.57 Locked cycle - mass balance

Comp ID	Test No.	Grind ~K80 µm	Regrind ~K80 µm	Mass distribution (%)			
				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

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 Table 13.58.

Table 13.58 Locked cycle - lead concentrate assay and recovery

Comp ID	Test No.	Pb Conc. grade				% recovery	
		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

The first locked cycle performed on MQV-HL material 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 material (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%.

Zinc concentrate grade and recovery were shown to be relatively consistent for the three locked cycle tests as shown in Table 13.59.

Table 13.59 Locked cycle - zinc concentrate analyses and recovery

Comp ID	Test No.	Zn Conc. grade				% recovery	
		Zn, %	Ag, g/t	Pb, %	Fe, %	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

Zinc concentrate could be expected to grade approximately 60% Zn, with about 30-40 g/t Ag. Recovery for MQV-HL material is slightly more variable in the mid- to upper- eighty percent range, depending on the process conditions used. STK testing achieved zinc recoveries approximately 5% higher than the MQV-HL tests.

13.3.6 Flotation product characterization

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 three locked cycle tests were subjected to detailed chemical characterization. The lead and zinc values from the spreadsheet balance of the locked cycle concentrates are provided in Table 13.60 and Table 13.61 respectively, along with an abbreviated summary of the supplemental analyses of final cycle concentrate for both the MQV-HL and STK composites. 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 13.60 Lead concentrate analyses

Element	Comp. ID	MQV-HL	STK	MQV-HL	10 kg blends
	Test ID	LCT 1	LCT 2	LCT 3	F10-1 to 20
Ag	g/t	771	1288	889	939
As	%	0.33	0.67	0.38	0.08
Bi	ppm	<20	<30	<20	<20
C (total)	%	2.31	2.38	1.35	1.96
Cd	ppm	380	271	346	299
Cl	%	<0.005	<0.005	0.005	n/a
Cu	%	1.5	3.7	1.9	2.0
F	%	0.012	0.013	0.007	0.014
Fe	%	2.37	0.25	2.37	1.00
Hg	ppm	-289	-239	-284	258
Pb	%	54.3	61.7	62.9	63.6
S (total)	%	13.6	11.4	14.0	12.8
Sb	%	-0.61	1.27	0.70	1.01
Se	ppm	<30	<30	<30	<30
SiO_2	%	4.65	2.96	3.74	1.21 (%Si)
TOC	%	0.51	0.65	0.54	0.69
Zn	%	5.98	3.05	5.25	4.09

Table 13.61 Zinc concentrate analyses

Element	Comp. ID	MQV-HL	STK	MQV-HL	10 kg blends
	Test ID	LCT 1	LCT 2	LCT 3	F10-1 to 20
Ag	g/t	41	29	39	28
As	ppm	<200	<100	<100	<100
Bi	ppm	<20	<30	<20	<20
C (total)	%	0.38	0.69	0.33	0.48
Cd	ppm	3390	3514	3557	3390
Cl	%	0.008	0.012	0.008	n/a
Cu	ppm	720	681	699	545
F	%	0.003	0.006	0.003	<0.005
Fe	%	1.30	0.38	1.27	0.63
Hg	ppm	-1520	-988	-1530	1400
Pb	%	0.74	1.85	0.62	0.55
S (total)	%	31.7	30.6	32.7	31.3
Sb	ppm	170	<80	99	123
Se	ppm	<30	<30	<30	<30
SiO2	%	1.75	1.74	1.39	0.68 (%Si)
TOC	%	n/a	0.34	<0.05	0.12
Zn	%	60.8	61.4	61.4	63.4

Acid base accounting based on the Solbek 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 together the Zn 1st Cl scavenger tailing and Zn rougher tailing. The bulk SG moisture content was selected based on expected filter cake moisture content, with a 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 are presented in the SGS report: Solid – Liquid Separation and Geotechnical Results, dated 24 May 2017. The physical characteristics of the various products are summarized in Table 13.62.

Table 13.62 Float product physical characteristics

Material	Particle size analysis		Bulk SG	Moisture wt.%	Bulk SG (Kg/L)	Angle of repose
	K80, µm	<20 µm (% vol)	Dry	(assumed)	@ % moisture	Avg @ % moisture
Combined tailing	148	12.3	2.83	13.0	1.627	46 degrees
Zinc concentrate	135	18.9	4.03	5.2	2.083	46 degrees
Lead concentrate	46	62.5	5.75	6.6	2.887	39 degrees

The lead concentrate had the finest particle size, due to the relative softness of galena, as well as the fact that 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.

13.3.7 Settling and filtration studies

The lead concentrate, the zinc concentrate, and the 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: Solid - Liquid Separation and Geotechnical Results, dated 24 May 2017. The combined (final) tailing was produced by blending together the Zn 1st Cl 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 13.63, with abbreviated data of the dynamic thickening provided in Table 13.64.

Table 13.63 Static thickening data

Sample ID	Dosage	Feed1	U/F2	Unit area	ISR3	Supernatant4	TSS5
	Flocc't g/t	% w/w	% w/w	$m^2/(t/day)$	$m^3/m^2/day$	visual	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

All values were calculated without a safety factor.
Common conditions: Raked, ambient temperature.
Magnafloc: BASF Magnafloc 10 flocculants.
[1]Diluted Thickener Feed.
[2]Final Underflow Density.
[3]Initial Settling Rate.
[4]Supernatant Visual Clarity after 30 minutes of elapsed settling time.
[5]Supernatant Total Suspended Solids (TSS) after 30 minutes of elapsed settling time.

Table 13.64 Dynamic settling data

Product conditions	Unit area	Solids loading	Net rise rate	Underflow	O/F - TSS	Residence time (h)
	$m^2/(t/d)$	$t/m^2/h$	$m^3/m^2/d$	wt.% solids	mg/L	
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

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 $m^2/(t/d)$ 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 different filter cloths. The thickened tailing was subjected to vacuum filtration testing with the summarized data presented in Table 13.65.

Table 13.65 Vacuum filtration data

Sample ID	Filter cloth	Operating conditions					Filter options				
		Feed solids %w/w	Vacuum level, inch Hg	Form time, s	Fry time, s	Form / dry ratio	Cake thickness, mm	[2]Throughout put, dry kg/m²·h	Cake moisture, % w/w	Filter TSS	Cake texture
Comb Zn tails	Testori P6583TC	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	[1]DTT
				17	170	0.10	55	1739	12.4	30	DTT

[1]Dry to touch.
[2]Examples of general filter throughout predictions versus test condition 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.

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 13.66.

Table 13.66 Pressure filtration data

Material	Feed	Pressure	Form	Dry	Cake	Throughput	Filtrate	Cake
	wt.% solid	bar	Time (s)	Time (s)	Thickness (mm)	kg/m² h (dry)	TSS (mg/L)	wt.% moist.
Tailing	70	4.1	3	43	30	3,358	197	6.1
	70	6.9	3	47	30	3,684	85	6.1
Zinc	79	4.1	2	42	28	5,488	164	4.3
	79	6.9	1	48	30.5	5,221	180	3.8
Lead	81	4.1	15	44	20	4,322	95	6.7
	81	6.9	15	81	25	3,252	72	6.1

Overall the results show good filter response with the lead concentrate having the finest particle size distribution (see Table 13.62, 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 moisture 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 13.67.

Table 13.67 Moisture saturation and porosity calculation

Sample ID	Solids content (%)	Water content, *w*	Wet bulk density, *y* (t/m^3)	Dry bulk density (t/m^3)	Density of water at 20°C (t/m^3)	Solids specific gravity, G_s	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
	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	3.56	0.998	5.75	0.38	0.61	0.00

Where,
Porosity, *n* = (void space, liquid, and gas) / (total volume including solid)
Void ratio, *e* = (void space, liquid, and gas) / (volume of solid)

13.3.8 Projected recovery

The DMS and flotation recovery projections that were made by Tetratech in 2016 (see Table 13.32) were updated incorporating the more recent test data, as provided in Table 13.68 below.

Table 13.68 Projection of metallurgical performance

Process circuit	Value	Note
MQV (Main Quartz Vein) / STK (stockwork) mineralization		
DMS Pre-concentration		
Mass Recovery, % - MQV	= 0.8484 x (Head Grade, % - Lead +Zinc) + 55.013	< 50% Pb+Zn
	= 98	> 50% Pb+Zn
Mass Recovery, % - STK	= 80	
Lead Recovery, % - Sulphide	= 1.293 x ln (Head Grade, % - Sulphide Lead) + 95.794	0 - 7.88% Pb-Sulphide
	= 0.0628 x (Head Grade, % - Sulphide Lead) + 97.97	7.88 - 16% Pb-Sulphide
	= 99.2	> 16% Pb-Sulphide
Lead Recovery, % - Oxide	= 1.0907 x ln (Head Grade, % - Oxide Lead) + 94.632	
Zinc Recovery, % - Sulphide	= 0.6549 x ln (Head Grade, % - Sulphide Zinc) + 97.107	< 50% Zn-Sulphide
	= 99.7	> 50% Zn-Sulphide
Zinc Recovery, % - Oxide	= 85	< 0.5% Zn-Oxide
	= 4.211 x ln (Head Grade, % - Oxide Zinc) + 87.579	0.5 - 10.5 Zn-Oxide
	= 97.5	> 10.5% Zn-Oxide
Silver Recovery, %	= 0.0436 x (Mass Recovery, %) + 94.508	> 50% Mass Recovery
	= 90	< 50% Mass Recovery
Sulphide lead flotation		
Grade, % - Lead	= 65	
Grade, % - Zinc	= 7	
Recovery, % - Lead	= (Head Grade, % - Sulphide Lead) / (Head Grade, % - Total Lead) x (68.477 x (Head Grade, % - Sulphide Lead)$^{0.1253}$)	0 - 17.4% Pb-Sulphide
	= (Head Grade, % - Sulphide Lead) / (Head Grade, % - Total Lead) x 98	> 17.4% Pb-Sulphide
Recovery, % - Silver	= 10.117 x (Lead Recovery to Sulphide Lead Concentrate, %)$^{0.4421}$	
Sulphide zinc flotation		
Grade, % - Zinc	= 59	
Grade, % - Lead	= 1.5519 x (Lead Head, %)$^{0.3717}$	
Recovery, % - Zinc	= (Head Grade, % - Sulphide Zinc) / (Head Grade, % - Total Zinc) x 50	< 1% Zn-Sulphide

Process circuit	Value	Note
	= (Head Grade, % - Sulphide Zinc) / (Head Grade, % - Total Zinc) x (0.2217 x (Head Grade, % - Sulphide Zinc) + 90.135)	1 - 30% Zn-Sulphide
	= (Head Grade, % - Sulphide Zinc) / (Head Grade, % - Total Zinc) x 98	> 30% Zn-Sulphide
Recovery, % - Silver	= 0	< 1.5% Lead Recovery to Sulphide Zinc Concentrate
	= 3.0962 x (Lead Recovery to Sulphide Zinc Concentrate, %) - 3.7592	1.5 – 7.5% Lead Recovery to Sulphide Zinc Concentrate
	= 20	> 7.5% Lead Recovery to Sulphide Zinc Concentrate
Oxide lead flotation		
Grade, % - Lead	= 48	
Grade, % - Zinc	= 7.7	
Recovery, % - Lead	= (Head Grade, % - Oxide Lead) / (Head Grade, % - Total Lead) x 70.5	
Recovery, % - Silver	= 0.4084 x ln (Lead Recovery to Oxide Lead Concentrate, %) +2.5177	> 2% Pb - Oxide
	= 1.0	< 2% Pb - Oxide
SMS (Stratabound) Mineralization		
DMS Pre-concentration		
Mass Recovery, %	= 88	
Lead Recovery, % - Sulphide	= 98	
Lead Recovery, % - Oxide	= 98	
Zinc Recovery, % - Sulphide	= 98	
Zinc Recovery, % - Oxide	= 98	
Silver Recovery, %	= 98	
Sulphide lead flotation		
Grade, % - Lead	= 56	
Grade, % - Zinc	= 8.6905 x (Zinc Head / Lead Head) - 9.3002	Zinc Head/Lead Head <1.35, Cap at 2%; if Zinc Head/Lead Head >2.0, Cap at 8.5%
Recovery, % - Lead	= 83	
Recovery, % - Silver	= 106.87 x ln(Lead Recovery to Sulphide Lead Concentrate, %) - 415.34	> 55% Lead Recovery to Sulphide Lead Concentrate
	= 10	< 55% Lead Recovery to Sulphide Lead Concentrate
Sulphide zinc flotation		
Grade, % - Zinc	= 57	
Grade, % - Lead	= 1.351 x ((Lead Head, %) X (100-83) / (100- (Lead Head, %) x 83/56))$^{0.9335}$	Zinc Circuit Head
Recovery, % - Zinc	= 0.198 x (Zinc Head, %) + 86.362	
Recovery, % - Silver	= 3.5965 x (Lead Recovery to Sulphide Zinc Concentrate, %) + 9.2277	Cap at 40%
Oxide lead flotation		
Grade, % - Lead	= 48	
Grade, % - Zinc	= 7.7	
Recovery, % - Lead	= (Head Grade, % - Oxide Lead) / (Head Grade, % - Total Lead) x 60	
Recovery, % - Silver	= (Lead Recovery to Oxide Lead Concentrate, %) x 70	

13.4 Recommendations

The 2017 metallurgical test program updated the historic flotation results for additional mineral zones (STK) that had not previously been tested. These samples had a lower extent of sulphide oxidation than those samples used for historical studies. Consequently, the 2017 test work did not require flotation for recovery of oxide minerals, and correspondingly did not require use of sulphidization (sodium sulphide), or soda ash for pH control. A significant portion of the historic test work used higher grades and / or higher oxide content than is expected to be typically experienced in operations. This resulted in a more complicated reagent scheme and flowsheet, which was not found to be necessary with the samples evaluated for the 2017 study. Due to the apparent variability in flotation response between oxidized samples (as represented in historical studies) and sulphide material (as represented by the 2017 samples), every effort should be made to separate (rather than blend) oxidized and sulphide feed material to the mill.

Related concerns that were addressed in 2017 testing were that use of lime might negatively impact oxide lead float recovery, and may need to be replaced with soda ash. Variability testing in the 2017 program replaced lime with soda ash and showed some detrimental impact to the sulphide circuit. Procedures were also modified with regard to pH adjustment with lime on samples with higher iron grades, resulting in some success in reducing iron content to the product concentrates. Further study in iron control to float concentrates could be considered. Confirmation testing is strongly recommended to be performed using the updated treatment procedure on samples with a more moderate degree of sulphide oxidation than those used in the previous test programs. The use of site water should also be tested in final verification testing of the updated flowsheet response.

The metallurgical response of the 2017 composite samples resulted in a zinc concentrate lower in lead and silver values as compared to historical work. The majority of the silver reported to the lead concentrate. Mercury content remained elevated in the zinc concentrate. The use of hydrometallurgy options can be better adapted to treat zinc concentrates with low silver and elevated mercury content than pyro-metallurgy options. Consequently, the use of pressure oxidation treatment might be investigated for the zinc concentrate, as there could be regional refiners in Western Canada willing to provide terms to compare against overseas smelters. A sufficient mine life may also justify further evaluating on-site hydrometallurgical treatment of Prairie Creek zinc concentrate as a future opportunity.

13.5 Summary

A pre-existing mill and concentrator, along with related infrastructure, was constructed at the Prairie Creek site in 1980-2. Construction was never fully completed, nor was the plant ever operated. The existing circuit consists of comminution and differential flotation of galena and sphalerite. Since that time, additional laboratory testing has resulted in developing a modified process flowsheet. Potential modifications include addition of pre-concentration by dense media separation, and scavenging of oxide lead (principally cerussite) as well as associated silver values. Lead oxide would be floated in a separate circuit following the sulphide flotation. Those mineral zones with higher cerussite content would support operating the oxide lead flotation step, which would otherwise be by-passed. Sulphide lead concentrate would be combined with the oxide lead product to produce a combined saleable lead concentrate. During concentration the majority of silver values tend to follow the lead minerals.

The Prairie Creek resource has been shown to be metallurgically complex and variable. There are three principal zones, each with varying mineralogy that will make up process plant feed. These are separated spatially in the resource, although there will be periods of overlap in mining the principal zones, which consist of:

Main Quartz Vein (MQV): forms the main category of mill feed throughout the LOM. Initially the near surface more highly oxidized portion of the zone may provide the majority of the mill feed, and also has the most challenging process response. MQV material was historically tested by various laboratories and by pilot testing samples obtained from existing adits. While the majority of the value minerals are in galena, and sphalerite, this material had up to 30% of the lead mineralization occurring as cerussite, and up to 20% of the zinc occurring as smithsonite. Further, the oxidation of copper mineralization can produce copper sulphate, resulting in premature activation of sphalerite and pyrite during lead flotation. Consequently, metal recovery / concentrate grade tends to be more variable, in large part depending on the extent of oxidation. Down dip, the MQV mineralization becomes harder (higher work index), less oxidized, and generally shows an improved process response. This deeper material was obtained from drill core that was evaluated in the 2017 metallurgical test program as sample composites MQV-H, and MQV-L. MQV-H represented the upper portion, and MQV-L the lower portion of the down-dip MQV resource extension. MQV-H tended to have a higher grade of the value metals, with less iron than MQV-L. Both composites were subsequently combined for final flotation confirmation and locked cycle testing and labelled as MQV-HL, which showed a favourable process response. Those materials classified as sulphide, should be kept separate from the higher oxidized MQV material to as great an extent as possible when feeding the mill.

Stratabound Massive Sulphide (SMS): This zone forms a minor portion of the overall resource, and will principally be mined later in the mine life. It has a lower ball mill grinding work index of approximately 9-10 kWh/tonne, and mineralogically it was described as containing major concentrations (up to 50 wt.%) of iron sulphide, principally pyrite, and marcasite. There was a low extent of sulphide oxidation present, although some mineralogy indicated finer-grained galena and sphalerite as compared to other zones. Metallurgical testing was done historically, but was more limited in scope than for MQV. Generally the mineral processing response showed that higher addition of depressants, including pH modification with lime, produced acceptable lead and zinc products, but at lower concentrate grades and recovery than other un-oxidized zones.

Stockwork (STK): First evaluated in the 2017 metallurgical testing program, the material had a lower mill feed head grade than MQV, but a positive mineral processing response. This was a result of the coarse-grained nature of the mineralization, as well as a low extent of sulphide oxidation of the sulphide minerals, and with lower iron and copper content. A higher graphite to lead ratio is evident that may create potential operating issues. Consequently, the addition of a graphite depressant was evaluated and shown to assist, including for improving lead concentrate grade. The STK also exhibited an increased grinding work index, but is to be developed later in the mine life, when additional grinding capacity is planned.

The 2017 composite samples showed a higher Bond ball mill work index of DMS sinks in the range of 13 to 14 kWh/tonne, as compared to the historical data that gave 8 to 11 kWh/tonne performed on samples taken closer to surface. Due to the changing mineralogy and expectations of harder material being developed, additional grinding capacity is anticipated to be required after start of operations and as development of harder mill feed begins. The updated primary grind requirements were evaluated in context with engineering comminution modeling that initially referenced the existing on-site primary mill and regrind mill, subject to a refurbishing evaluation. With installation of additional primary grinding capacity following start of operations the product particle size to float feed is expected to decrease.

The 2017 metallurgical test program was successful in building on the historical laboratory studies to evaluate samples from previously untested mineralized zones. The test work was also able to establish a more conventional and simplified flotation flowsheet and reagent scheme than what had historically been proposed. The revised flowsheet eliminated a number of scavenging and recycle streams during differential flotation of galena and sphalerite. Due to a lower extent of sulphide oxidation in the 2017 metallurgical samples being tested, there was no requirement for separate oxide flotation, although this is assumed to be retained for mill feed originating from mineral zones closer to surface. An improved reagent scheme resulted in reducing the number of flotation reagents, as well as identifying readily available products that are currently marketed by known suppliers. Alternate methods were also developed for variation in changing mill feed characteristics, particularly with respect to pyrite and graphite content. Altogether, the process procedures that were developed in the 2017 test program allowed for the existing equipment and treatment circuits on the Prairie Creek site to be more fully utilized.

14 Mineral Resource estimates

14.1 Introduction

The current Mineral Resource estimate is an update of the estimate given in a press release of 17 September 2015. The current estimate includes assay data from 21 underground drillholes, (642 samples with an aggregate length of 675 m) and 22 channel samples from the 930 L (50 samples with an aggregate length of 48 m) that have been acquired since the previous estimate. It is the same as the estimate supporting the PFS and reported in the September 2016 Technical Report.

CZN provided wireframes of major lithological units, structures and mineral domains together with drillhole locations, downhole surveys, assays and geology as components of a GEMS (Gemcom resource estimation software) project. Mr. Greg Mosher, P.Geo. (previously of AMC, now with Global Mineral Resource Services Ltd.) completed the Mineral Resource estimate using GEOVIA GEMS™ software from Dassault Systemes.

14.2 Exploration data analysis

14.2.1 Assays

The Mineral Resource estimate is based on assays from all underground channel samples, surface and underground drill core collected by CZN since 1992.

The dataset contains data for 294 surface and underground drillholes of which 219, with an aggregate length of 59,164 m, are located within the three mineral zone domains (MQV, SMS, and STK), and 354 channel samples (1,023 aggregate metres) from the MQV and STK Zones. The 219 drillholes contain 3,129 assays of which 1,944 are contained within the MQV, 539 within the STK and 646 within the SMS domain. The channel samples contain 1,012 assays, of which 938 are within the MQV domain and 74 within the STK. Illustrative views of the MQV, SMS and STK Mineral domains together with the supporting drill and channel sample locations are shown in Figure 14.1 and Figure 14.2. The underground channel sample locations are indicated on Figure 14.1.

Figure 14.1 Illustrative plan view of Mineral domains, drillholes, and channel samples

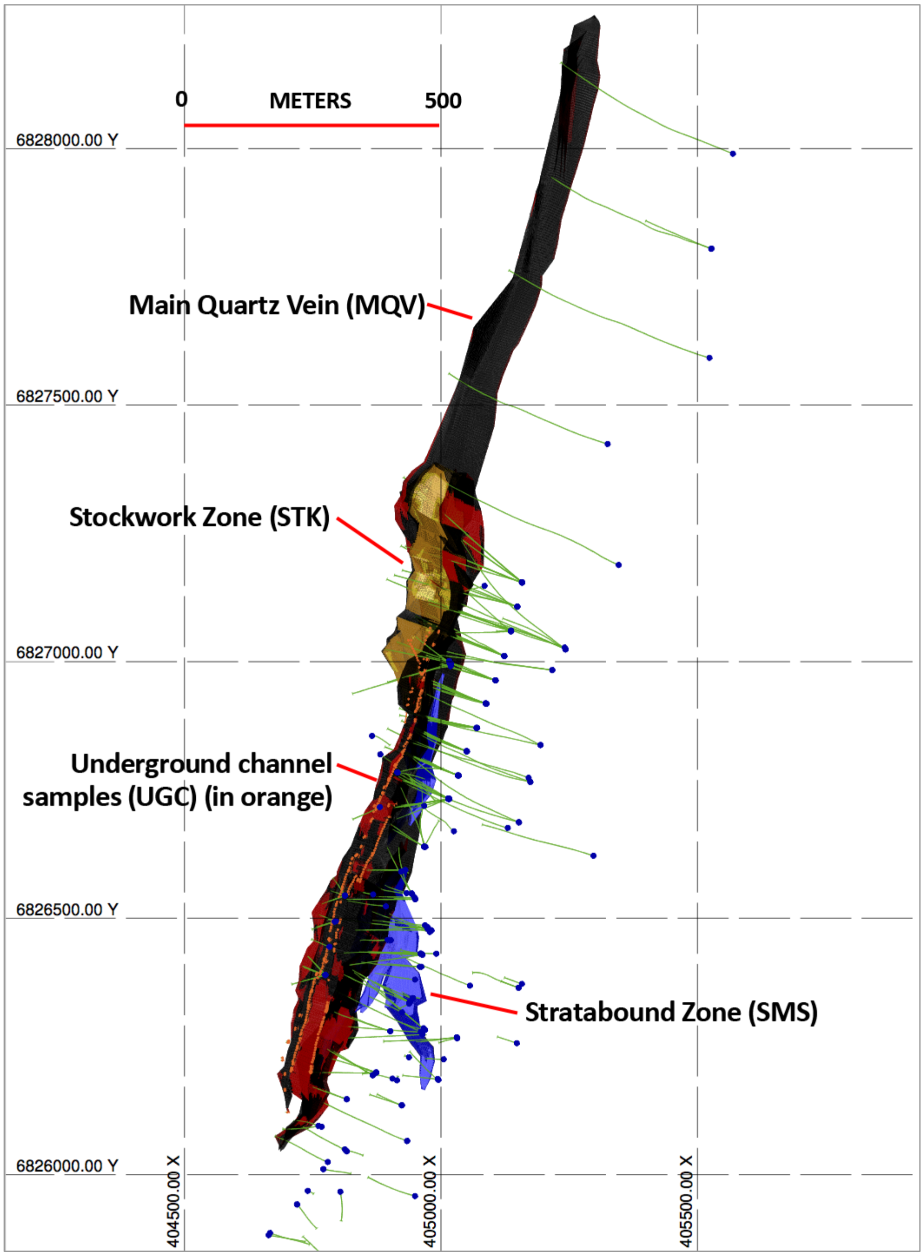
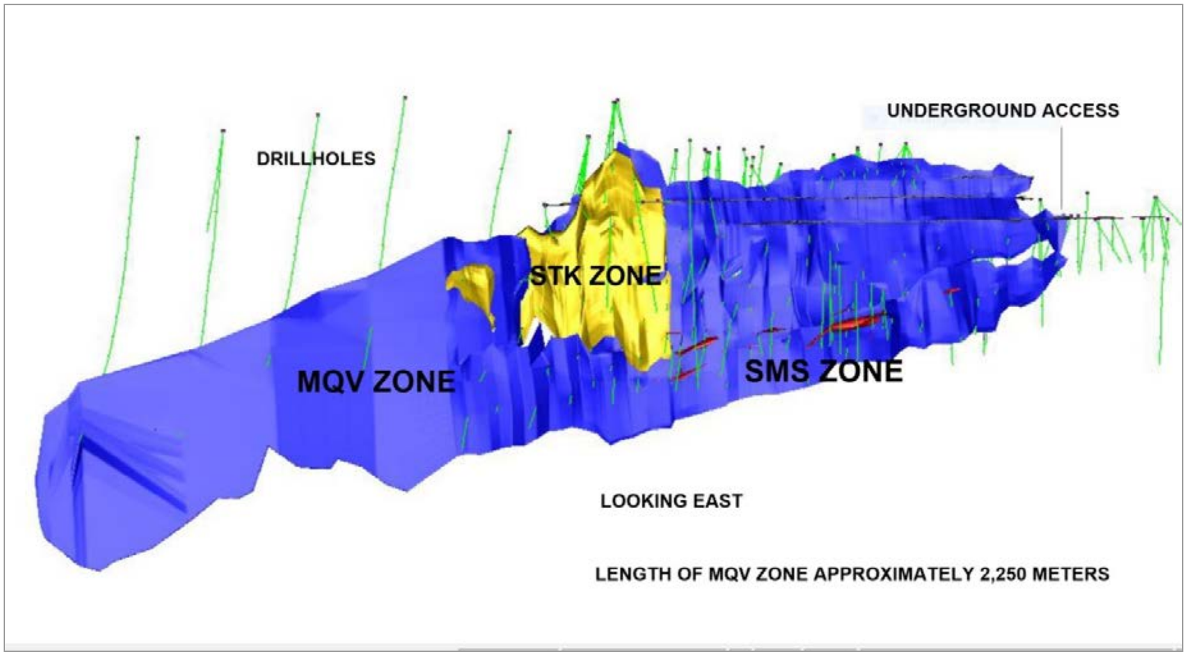


Figure 14.2 Illustrative longitudinal view showing Mineral domains, drillholes, and underground channel samples



The number of assays in the dataset by element / compound and Mineral domain is shown in Table 14.1. As the table clearly shows, not all elements are equally well represented.

Table 14.1 Number of assays by element / compound and domain

Element / compound	MQV DDH	SMS DDH	STK DDH	MQV channel	STK channel
Ag	1722	505	461	889	74
As	984	293	227	229	56
Cd	1191	272	275	334	58
Cu	1379	383	352	760	74
Fe	1762	630	505	213	58
Hg	557	209	90	182	53
Pb	1889	592	486	928	74
PbO	1572	453	323	620	57
Sb	1238	262	316	215	58
Zn	1933	610	493	923	74
ZnO	1601	435	317	622	58

The channel samples are from the MQV and STK Zones and were taken from the three underground levels: 970 mL, 930 mL, and 883 mL.

Descriptive statistics for drillhole and channel sample assays and corresponding composites are presented in Table 14.3.

Table 14.5 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. The corresponding graphs adjacent to the tables demonstrate that although there are differences in magnitude, all domains have generally similar element affinities.

Table 14.2 Mineral domain assay and composite descriptive statistics

MQV DDH assays	Ag (g/t)	Pb (%)	Zn (%)
Mean	107	6.15	6.67
Standard error	4	0.23	0.24
Median	24	0.92	1.76
Mode	1	0.02	0.02
Standard deviation	175	10.08	10.39
Kurtosis	21	6.58	4.44
Skewness	3	2.35	2.14
Range	2,059	69.87	64.11
Minimum	0	0.01	0.01
Maximum	2,059	69.88	64.12
Count	1,722	1,889	1,933

MQV DDH composites	Ag (g/t)	Pb (%)	Zn (%)
Mean	111	6.90	7.54
Standard error	5	0.32	0.32
Median	50	3.07	3.85
Mode	1	0.02	3.80
Standard deviation	138	8.65	8.88
Kurtosis	4	2.75	2.31
Skewness	2	1.67	1.65
Range	821	45.65	44.59
Minimum	1	0.01	0.01
Maximum	821	45.65	44.59
Count	720	745	747

MQV CHANNEL assays	Ag (g/t)	Pb (%)	Zn (%)
Mean	196	10.75	12.25
Standard error	7	0.31	0.35
Median	150	8.97	9.39
Mode	0	0.00	0.00
Standard deviation	213	9.62	10.81
Kurtosis	20	0.86	1.60
Skewness	3	1.03	1.34
Range	2,221	53.72	57.02
Minimum	0	0.00	0.00
Maximum	2,221	53.72	57.02
Count	938	938	938

MQV channel composites	Ag (g/t)	Pb (%)	Zn (%)
Mean	220	11.82	13.21
Standard error	8	0.34	0.43
Median	188	10.60	10.56
Mode	295	15.00	9.56
Standard deviation	162	7.31	9.16
Kurtosis	7	0.42	0.50
Skewness	2	0.74	0.99
Range	1,227	37.50	43.38
Minimum	3	0.11	0.24
Maximum	1,231	37.62	43.62
Count	456	458	454

SMS DDH assays	Ag (g/t)	Pb (%)	Zn (%)
Mean	64	4.89	6.60
Standard error	5	0.31	0.36
Median	28	1.29	1.41
Mode	2	0.02	0.02
Standard deviation	109	7.58	8.99
Kurtosis	16	10.29	2.40
Skewness	4	2.68	1.59
Range	826	63.20	49.08
Minimum	0	0.01	0.01
Maximum	826	63.21	49.09
Count	505	592	610

SMS DDH composites	Ag (g/t)	Pb (%)	Zn (%)
Mean	80	6.54	9.96
Standard error	8	0.50	0.59
Median	50	4.60	8.08
Mode			
Standard deviation	101	6.82	8.04
Kurtosis	10,140	46.53	64.71
Skewness	10	5.00	1.35
Range	609	38.76	44.57
Minimum	2	0.03	0.07
Maximum	612	38.79	44.64
Count	178	186	186

STK DDH assays	Ag (g/t)	Pb (%)	Zn (%)
Mean	82	4.71	5.12
Standard error	8	0.41	0.41
Median	14	0.79	1.11
Mode	1	0.02	0.05
Standard deviation	165	9.12	9.19
Kurtosis	32	8.66	7.73
Skewness	5	2.83	2.71
Range	1,741	58.79	52.75
Minimum	0	0.01	0.01
Maximum	1,741	58.80	52.76
Count	461	486	493

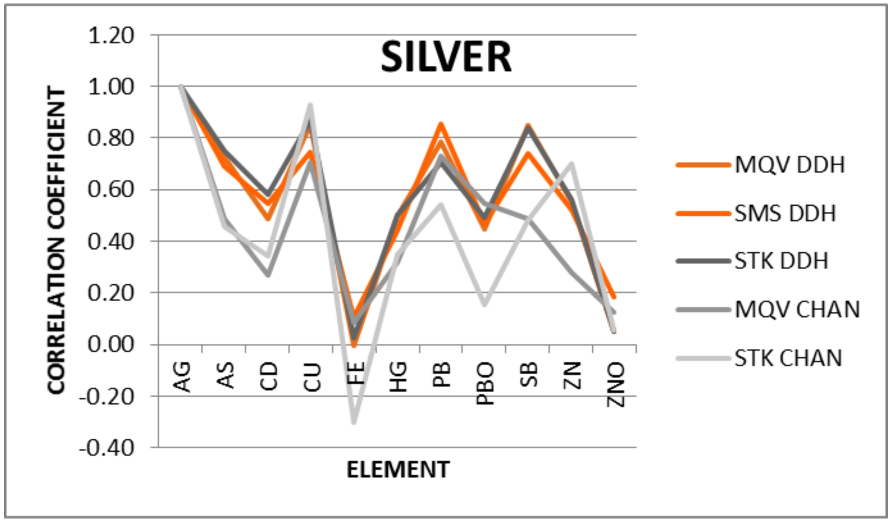
STK DDH composites	Ag (g/t)	Pb (%)	Zn (%)
Mean	50	3.31	3.86
Standard error	6	0.41	0.34
Median	19	1.08	2.37
Mode	4	0.27	
Standard deviation	85	5.66	4.58
Kurtosis	20	11.96	7.77
Skewness	4	3.19	2.47
Range	618	35.91	29.19
Minimum	1	0.01	0.03
Maximum	619	35.92	29.22
Count	183	187	186

STK CHANNEL assays	Ag (g/t)	Pb (%)	Zn (%)
Mean	191	8.90	19.42
Standard error	24	0.76	1.63
Median	153	7.97	17.30
Mode	79	14.80	12.90
Standard deviation	204	6.53	14.04
Kurtosis	14	-0.34	-1.11
Skewness	3	0.67	0.35
Range	1,333	25.91	47.96
Minimum	6	0.26	0.36
Maximum	1,339	26.17	48.32
Count	74	74	74

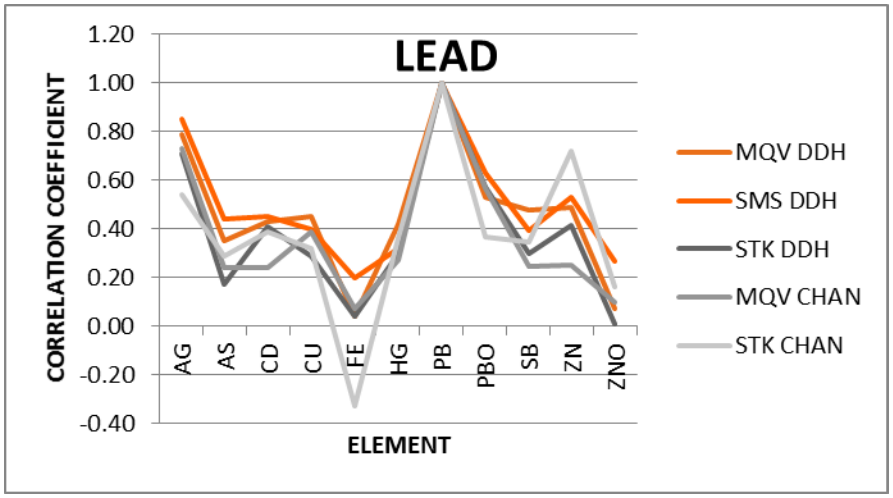
STK channel composites	Ag (g/t)	Pb (%)	Zn (%)
Mean	126	6.16	13.30
Standard error	14	0.55	1.00
Median	112	5.40	12.40
Mode	123	7.41	21.50
Standard deviation	104	3.99	7.31
Kurtosis	13	-0.05	-0.64
Skewness	3	0.61	0.16
Range	654	16.04	30.24
Minimum	3	0.14	0.24
Maximum	658	16.18	30.48
Count	52	53	53

Table 14.3 Correlation coefficients for silver, lead, and zinc

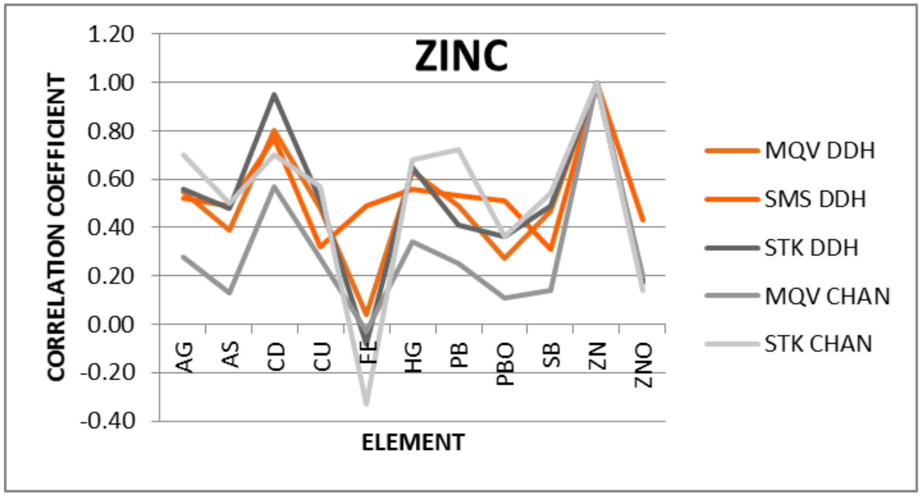
Ag	MQV DDH	SMS DDH	STK DDH	MQV CHAN	STK CHAN
Ag	1.00	1.00	1.00	1.00	1.00
As	0.73	0.69	0.75	0.49	0.46
Cd	0.49	0.55	0.58	0.27	0.34
Cu	0.85	0.74	0.87	0.70	0.93
Fe	0.00	0.11	0.03	0.08	-0.30
Hg	0.50	0.44	0.50	0.32	0.35
Pb	0.79	0.85	0.71	0.73	0.54
PbO	0.45	0.47	0.49	0.55	0.16
Sb	0.85	0.74	0.84	0.49	0.48
Zn	0.55	0.52	0.56	0.28	0.70
ZnO	0.05	0.18	0.05	0.12	0.06



Pb	MQV DDH	SMS DDH	STK DDH	MQV CHAN	STK CHAN
Ag	0.79	0.85	0.71	0.73	0.54
As	0.35	0.44	0.17	0.24	0.29
Cd	0.43	0.45	0.41	0.24	0.39
Cu	0.45	0.40	0.29	0.39	0.32
Fe	0.04	0.20	0.04	0.07	-0.33
Hg	0.42	0.32	0.30	0.27	0.36
Pb	1.00	1.00	1.00	1.00	1.00
PbO	0.53	0.63	0.57	0.57	0.37
Sb	0.48	0.39	0.30	0.25	0.35
Zn	0.49	0.53	0.41	0.25	0.72
ZnO	0.07	0.27	0.01	0.10	0.16



Zn	MQV DDH	SMS DDH	STK DDH	MQV CHAN	STK CHAN
Ag	0.55	0.52	0.56	0.28	0.70
As	0.39	0.49	0.48	0.13	0.50
Cd	0.80	0.77	0.95	0.57	0.70
Cu	0.48	0.32	0.51	0.27	0.57
Fe	0.04	0.49	-0.08	-0.03	-0.33
Hg	0.63	0.56	0.65	0.34	0.68
Pb	0.49	0.53	0.41	0.25	0.72
PbO	0.27	0.51	0.36	0.11	0.36
Sb	0.47	0.31	0.49	0.14	0.54
Zn	1.00	1.00	1.00	1.00	1.00
ZnO	0.18	0.43	0.17	0.20	0.14



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.

14.2.2 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.

14.2.3 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.

14.2.4 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. The estimated values for the MQV were based on zinc and lead grades and had a coefficient of determination (R^2) for the calculated with the measured values of 0.94. For the SMS, the regression was made using lead and iron and the R^2 value was 0.92.

14.3 Geological interpretation

CZN generated wireframe models for the three mineralized domains MQV, STK and SMS. These solids were reviewed by AMC for conformity to the lithological boundaries established by drilling and the wireframes were observed to adhere to the lithological boundaries. AMC used the models as provided. The Mineral domains are illustrated in respective plan and longitudinal vertical views in Figure 14.1 and Figure 14.2.

14.4 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, as well as for bulk density. All models were exponential; variograms were based on two structures and search ellipses on one. Because of the paucity of data, variograms and search ellipses for the SMS domain were constructed using orientations and dimensions appropriate to the orientation and size of the domain rather than by empirical testing. Table 14.5 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.

Table 14.4 Variogram parameters

Domain	MQV														
		C1 rotation				C1 range			C2 rotation				C2 range		
Element	**C0 NUG**	**C1**	**Z**	**Y**	**Z**	**X**	**Y**	**Z**	**C2**	**Z**	**Y**	**Z**	**X**	**Y**	**Z**
Ag	0.68	0.11	-46	-73	-75	9	85	83	0.21	-89	72	143	111	30	314
As	0.57	0.25	113	-56	-14	152	8	220	0.19	-49	-35	43	64	368	400
Cd	0.56	0.22	-26	-28	26	96	73	7	0.22	-31	-31	28	42	280	400
Cu	0.73	0.23	-11	40	89	400	28	105	0.04	-7	-29	20	45	130	400
Fe	0.28	0.41	-10	8	-11	208	6	235	0.31	84	-90	-35	400	400	400
Hg	0.44	0.31	5	29	-52	260	6	134	0.25	-45	-37	44	68	400	400
Pb	0.51	0.20	2	27	53	167	10	39	0.29	-93	76	61	21	31	336
PbO	0.34	0.50	-18	-78	64	309	400	36	0.16	-9	-36	38	278	290	39
Sb	0.75	0.02	-5	39	89	262	31	73	0.23	-28	-30	22	62	221	400
Zn	0.55	0.18	25	1	55	400	59	26	0.27	33	-55	-1	83	313	384
ZnO	0.40	0.31	28	56	-30	268	13	110	0.29	40	25	-9	123	400	400
Domain	**STK**														
		C1 rotation				C1 range			C2 rotation				C2 range		
Element	**C0 NUG**	**C1**	**Z**	**Y**	**Z**	**X**	**Y**	**Z**	**C2**	**Z**	**Y**	**Z**	**X**	**Y**	**Z**
Ag	0.47	0.20	74	-1	51	236	173	21	0.33	-63	30	55	15	108	327
As	0.23	0.49	65	63	113	117	82	6	0.28	-10	88	1	42	400	157
Cd	0.23	0.46	-21	7	83	400	6	77	0.31	22	29	-16	46	385	123
Cu	0.49	0.33	60	74	40	180	49	2	0.18	-91	89	7	35	157	292
Fe	0.17	0.59	29	32	24	169	10	182	0.24	-14	-65	37	186	400	400
Hg	0.18	0.45	69	67	15	57	270	10	0.37	-20	25	20	42	400	400
Pb	0.30	0.20	35	-3	70	400	110	26	0.50	-59	19	42	8	98	219
PbO	0.12	0.33	29	-8	-104	242	219	27	0.55	-82	29	55	8	193	155
Sb	0.18	0.21	118	0	48	264	142	11	0.61	-38	-5	-1	10	49	326
Zn	0.43	0.19	-20	0	80	400	187	16	0.38	-51	26	43	17	400	377
ZnO	0.12	0.55	33	26	26	148	9	128	0.33	23	12	24	66	400	400
Domain	**SMS**														
		C1 rotation				C1 range			C2 rotation				C2 range		
Element	**C0 NUG**	**C1**	**Z**	**Y**	**Z**	**X**	**Y**	**Z**	**C2**	**Z**	**Y**	**Z**	**X**	**Y**	**Z**
ALL	0.25	0.75	-10	90	-10	10	100	50							

Table 14.5 Search ellipse parameters

	Rotation			Range		
Domain	**Z**	**Y**	**Z**	**X**	**Y**	**Z**
MQV	5	75	10	300	400	200
STK	-55	0	0	100	5	200
SMS	0	90	0	50	100	50

14.5 Mineral Resource block model

Block model parameters are summarized in Table 14.9. A range of block sizes was tested to determine, within reasonable limits, whether metal content varies with block dimensions. Although total metal content does vary, the range of variation of total metal content is less than one percent and therefore the impact of block size is immaterial, relative to the influence of other parameters. Regardless, the block size of width 2.5 m (across strike), length 15 m (along strike) and height of 10 m is the configuration that, in AMC's opinion, best represents the total metal content in the Measured and / or Indicated categories for all three mineral domains.

Table 14.6 Block model parameters

Dimension	Number	Size (m)	Coordinates *	UTM NAD 83 ZONE 10V
Columns	200	2.5	X MINIMUM	404450
Rows	170	15	Y MINIMUM	6826000
Levels	110	10	Z MAXIMUM	1123
Rotation	15 (degrees clockwise)			

* Lower left hand corner.

14.6 Interpolation plan

Grades were interpolated into the block model using ordinary kriging (OK) and inverse distance squared (ID^2). OK is, by design, the least biased of estimation methods and, therefore, the outcome is generally perceived as the most reliable and is the one reported for the current estimate. The ID^2 estimate was performed as a check of the reasonableness of the OK estimate.

Grades were interpolated into the block model in a single pass. A minimum of four composites within the volume of the search ellipse was required in order 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. The maximum number of composites was set at 24 (12 drillholes). The estimation was also attempted using a smaller maximum number of samples (12 and 8) but the resultant morphology of Resource classes (see below) was less coherent than when 24 composites were used.

14.7 Zinc equivalency formula

The Mineral Resource is stated using a zinc equivalent grade (ZnEq) as a cut-off that takes into account the economic contribution of silver, lead and zinc. It is also used for display purposes in Figures, for example in Figures 14.3 - 14.5. The equivalency calculation, which expresses the combined value of silver, lead and zinc in terms of percent zinc, was calculated as follows:

*ZnEq% = (Grade of Zn in %) + [(Grade of lead in % * Price of lead in $/lb * 22.046 * Recovery of lead in % * Payable lead in %) + (Grade of silver in g/t* (Price of silver in US$/Troy oz/ 31.10348) * Recovery of silver in % * Payable silver in %)]/(Price of zinc in US$/lb*22.046 * Recovery of zinc in % * Payable zinc in %)*

Metal prices were based on an assessment of three-year market forecasts and considering reasonable prospects for eventual economic extraction. Recoveries and payables were taken from the 2012 AMC Report. Parameters are summarized below in Table 14.10.

Table 14.7 Zinc-equivalency equation parameters

Item	Value
Zn price	1.00
Pb price	1.00
Ag price	20.00
Zn recovery	75%
Pb recovery	88%
Ag recovery	92%
Zn payable	85%
Pb payable	95%
Ag payable	81%

Metal prices in US$.
22.046 = pounds/metric tonne/%.
31.10348 = grams/Troy ounce.

14.8 Mineral Resource classification

The Mineral Resource was classified as Measured, Indicated and Inferred. For a block to be classified as Measured Mineral Resource it was necessary that a minimum of 24 composites be located within the volume of the search ellipse and that the average distance of those composites from the block centroid was 55 m or less.

For a block to be classified as Indicated Mineral Resource within the MQV, it was necessary that a minimum of 10 composites be located within the volume of the search ellipse and that the average distance of those composites from the block centroid was 135 m or less. For a block to be classified as Indicated Mineral Resource within the STK or SMS, a minimum of 10 composites had to be located within the volume of the search ellipse and with an average distance of 100 m of the block centroid.

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

The southern portion of the MQV domain, as well as a portion of the STK that has been explored by underground development, contain Measured Resources; the remainder of the Mineral Resource is classified as Indicated and Inferred.

Figure 14.3 shows a longitudinal vertical view of the zinc-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 Zinc-Equivalent block grade distribution MQV zone

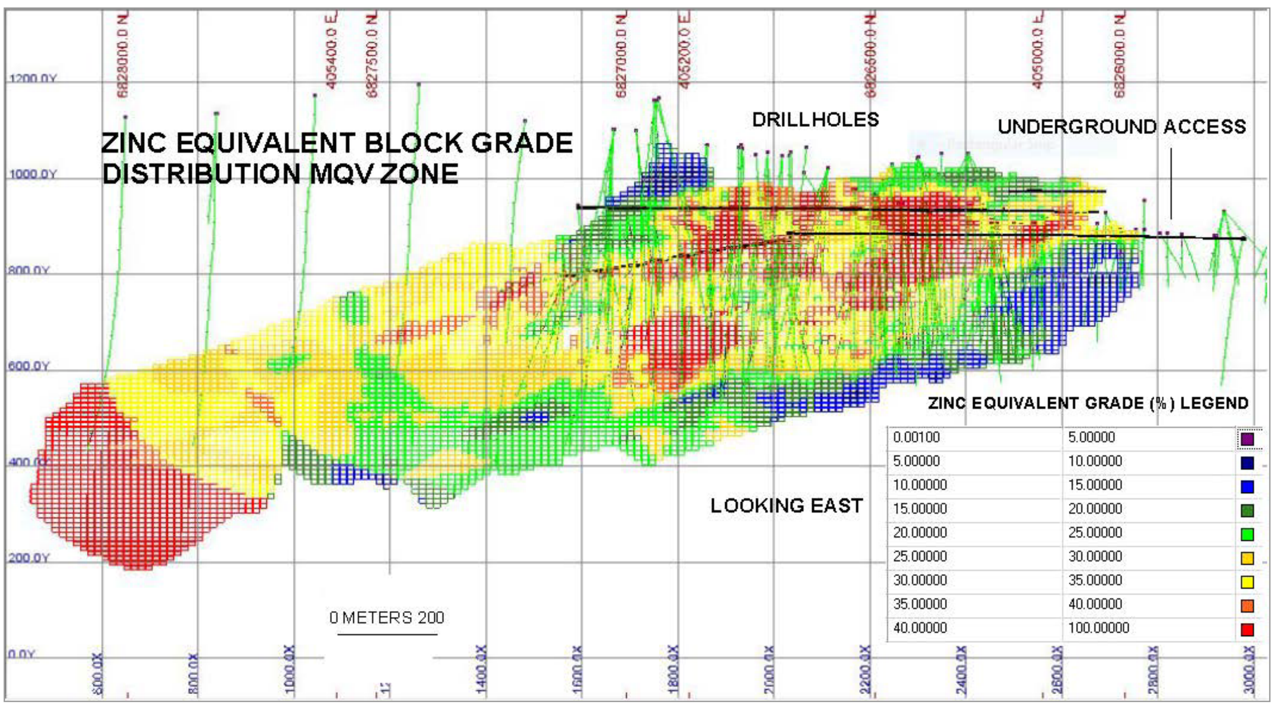


Figure 14.4 Zinc-equivalent block grade distribution SMS zone

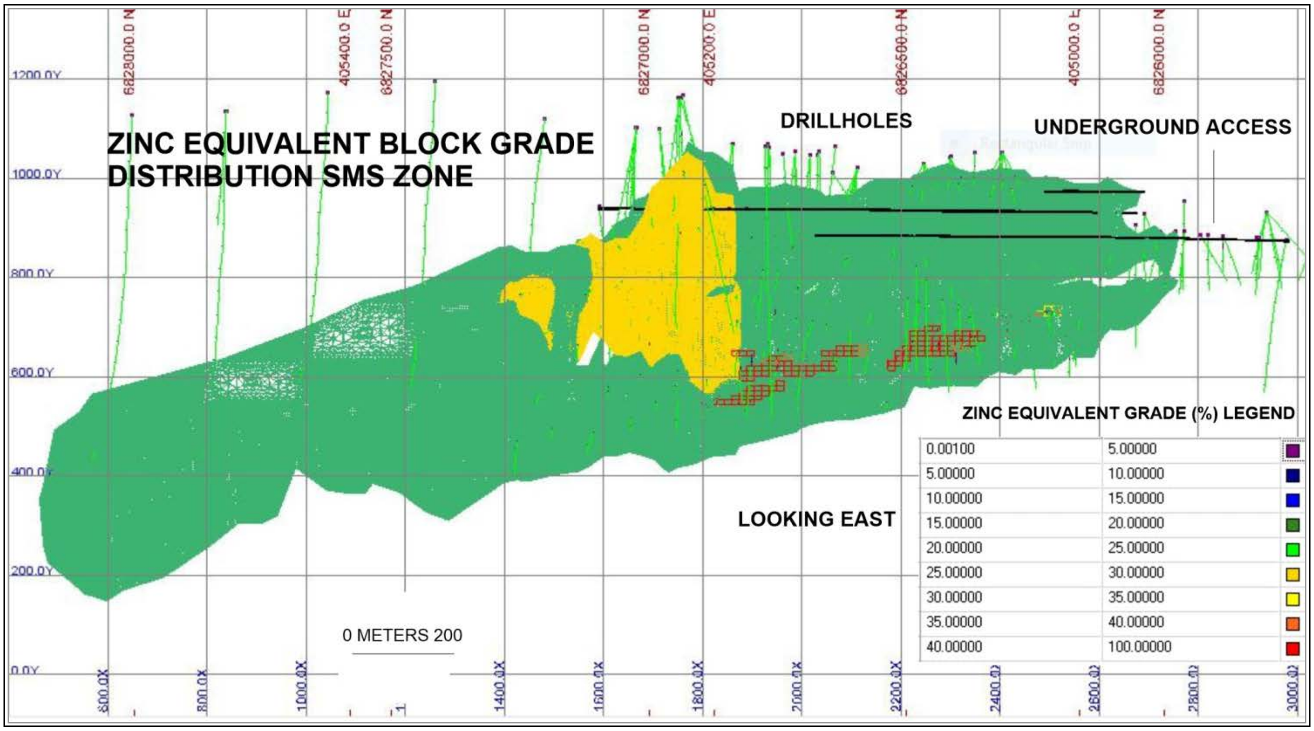


Figure 14.5 Zinc-equivalent block grade distribution STK zone

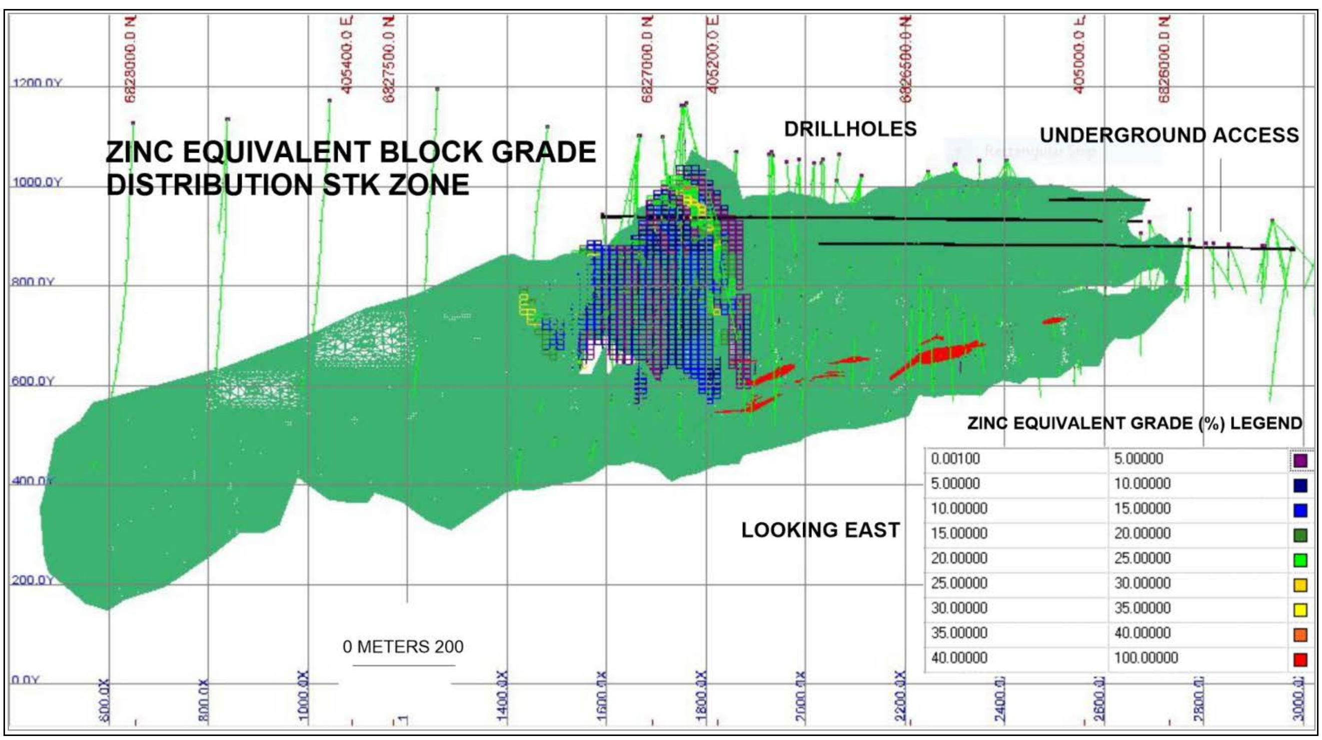
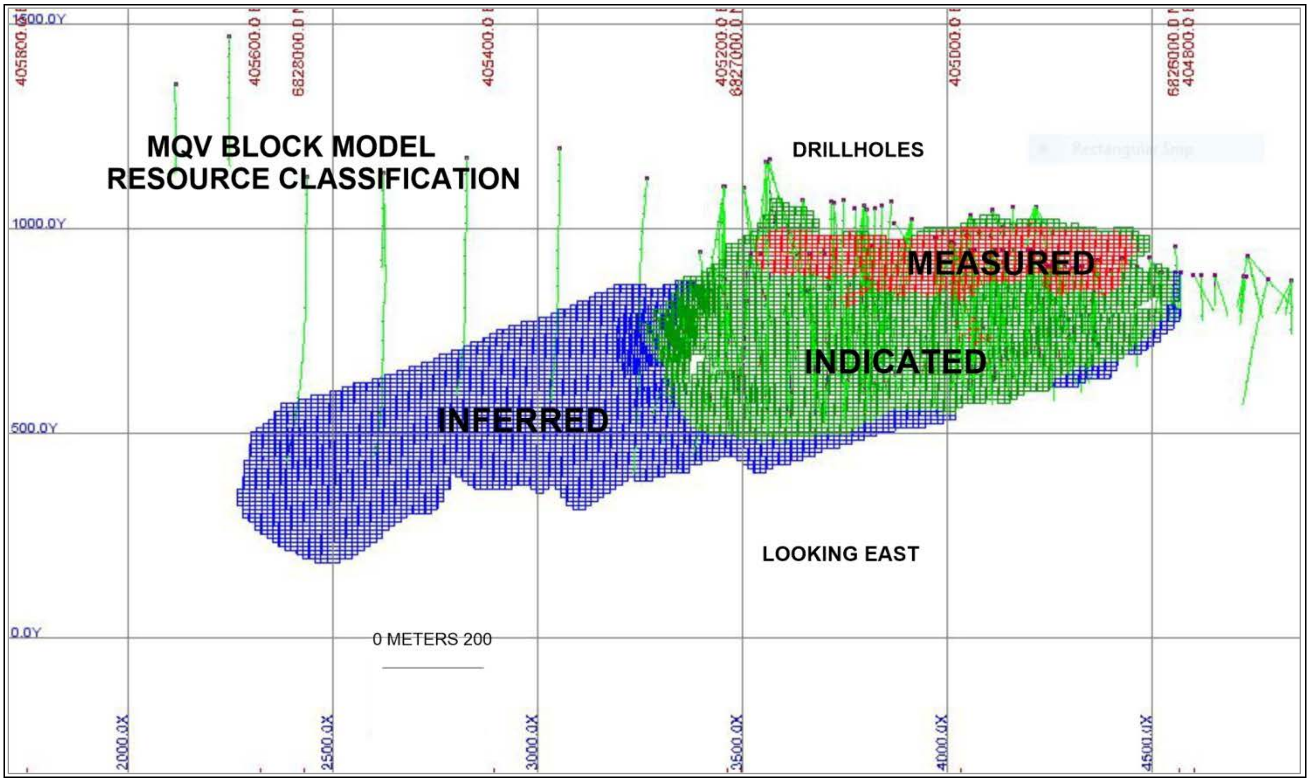


Figure 14.6 Mineral Resource classification MQV domain



14.9 Mineral Resource tabulation

Table 14.10 presents the Mineral Resource estimate for the three mineral zones MQV, STK and SMS, at a ZnEq cut-off of 8%. 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. Tonnes have been rounded to the nearest 1,000, Ag to the nearest g/t, and Pb and Zn to the nearest 0.1%.

Table 14.8 September 2015 Mineral Resource summary at 8% ZnEq grade cut-off

MQV	Tonnes	Ag g/t	Pb %	Zn %
Measured	1,313,000	211	11.5	13.2
Indicated	4,227,000	168	11.6	9.2
Measured & Indicated	5,540,000	178	11.6	10.2
Inferred	5,269,000	199	8.7	12.9
SMS				
Indicated	1,042,000	54	5.2	10.8
Measured & Indicated	1,042,000	54	5.2	10.8
Inferred	170,000	60	6.3	11.2
STK				
Measured	169,000	116	5.3	12.6
Indicated	1,953,000	61	3.5	6.6
Measured & Indicated	2,122,000	66	3.6	7.1
Inferred	1,610,000	70	4.6	6.2
Total				
Measured	1,482,000	200	10.8	13.2
Indicated	7,222,000	123	8.5	8.7
Measured & Indicated	8,704,000	136	8.9	9.5
Inferred	7,049,000	166	7.7	11.3

Mineral Resources are stated as of 10 September 2015.
Mineral Resources include those Resources converted to Mineral Reserves.
Stated at a cut-off grade of 8% ZnEq based on prices of $1.00/lb for both zinc and lead and $20/oz for silver.
Average processing recovery factors of 78% for zinc, 89% for lead, and 93% for silver.
Average payables of 85% for zinc, 95% for lead, and 81% for silver.
ZnEq = (grade of Zn in %) + [(grade of lead in % * price of lead in $/lb * 22.046 * recovery of lead in % * payable lead in %) + (grade of silver in g/t* (price of silver in US$/Troy oz/ 31.10348) * recovery of silver in % * payable silver in %)]/(price of zinc in US$/lb*22.046 * recovery of zinc in % * payable zinc in %).
$ Exchange rate = 1 C/US.
Numbers may not compute exactly due to rounding.

Table 14.9, Table 14.10, and Table 14.11 show the Mineral Resource estimates for the MQV, SMS, and STK zones respectively for a range of ZnEq cut-offs and with the same rounding of tonnes and grades as for Table 14.8. It should be noted that the ZnEq average grade is relatively insensitive to the cut-off grade, with the exception of the STK Indicated and Inferred. At all cut-offs, the ZnEq grade is significantly higher than 8% 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.

Table 14.9 September 2015 Mineral Resource at a range of cut-offs for MQV

Classification	ZnEq cut-off (%)	Tonnes	ZnEq	Ag (g/t)	Pb (%)	Zn (%)
MQV Measured	25	1,128,000	37.9	224	12.3	14.1
	20	1,260,000	36.3	215	11.8	13.5
	15	1,302,000	35.7	212	11.6	13.3
	10	1,310,000	35.6	211	11.5	13.3
	8	1,313,000	35.5	211	11.5	13.2
	6	1,315,000	35.5	211	11.5	13.2
MQV Indicated	25	2,975,000	35.2	193	13.2	11.3
	20	3,465,000	33.4	185	12.7	10.5
	15	3,874,000	31.7	176	12.1	9.8
	10	4,195,000	30.3	169	11.6	9.3
	8	4,227,000	30.1	168	11.6	9.2
	6	4,237,000	30.1	168	11.6	9.2
MQV Inferred	25	3,869,000	35.4	229	9.2	15.6
	20	4,412,000	33.8	217	9.1	14.4
	15	5,107,000	31.6	202	8.8	13.2
	10	5,260,000	31.1	199	8.7	12.9
	8	5,269,000	31.1	199	8.7	12.9
	6	5,269,000	31.1	199	8.7	12.9

Table 14.10 September 2015 Mineral Resource at a range of cut-offs for SMS

Classification	ZnEq cut-off (%)	Tonnes	ZnEq	Ag (g/t)	Pb (%)	Zn (%)
SMS Indicated	25	203,000	27.6	84	7.7	14.6
	20	473,000	24.7	73	6.8	13.3
	15	794,000	21.7	62	5.9	11.9
	10	1,013,000	19.8	55	5.3	10.9
	8	1,042,000	19.5	54	5.2	10.8
	6	1,061,000	19.3	54	5.2	10.7
SMS Inferred	25	39,000	28.6	79	8.8	14.4
	20	106,000	24.6	72	7.6	12.2
	15	151,000	22.6	65	6.7	11.6
	10	168,000	21.6	61	6.3	11.2
	8	170,000	21.4	60	6.3	11.2
	6	174,000	21.2	59	6.2	11.1

Table 14.11 September 2015 Mineral Resource at a range of cut-offs for STK

Classification	ZnEq cut-off (%)	Tonnes	ZnEq	Ag (g/t)	Pb (%)	Zn (%)
STK Measured	25	68,000	27.7	132	6.4	14.8
	20	132,000	25.1	122	5.7	13.5
	15	166,000	23.7	116	5.3	12.8
	10	169,000	23.5	116	5.3	12.6
	8	169,000	23.5	116	5.3	12.6
	6	169,000	23.5	116	5.3	12.6
STK Indicated	25	85,000	29.1	131	6.8	15.7
	20	230,000	24.8	115	6.1	12.8
	15	522,000	20.6	97	5.3	10.4
	10	1,293,000	15.6	72	4.1	7.8
	8	1,953,000	13.3	61	3.5	6.6
	6	2,866,000	11.3	52	3.0	5.6
STK Inferred	25	116,000	31.3	144	8.9	14.8
	20	273,000	26.0	123	8.1	11.2
	15	581,000	21.3	102	6.8	8.9
	10	1,165,000	16.8	81	5.3	7.1
	8	1,610,000	14.7	70	4.6	6.2
	6	2,148,000	12.7	61	4.0	5.4

AMC 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.

14.10 Block model validation

The block model was validated in three ways to ensure that estimated grades honour the raw data and lie within the constraining wireframes: 1) visually, by comparison of drillhole assay grades relative to block grades; 2) numerically by comparison of the OK estimate with ID^2 outcomes; 3) numerically by comparison of composite average grades with corresponding block model average grades.

Visual inspection shows that the block model honours the boundaries of the wireframes and that the block grades correspond well with the relevant assay grades (Figure 14.7).

Figure 14.7 Vertical cross section showing DDH and block model

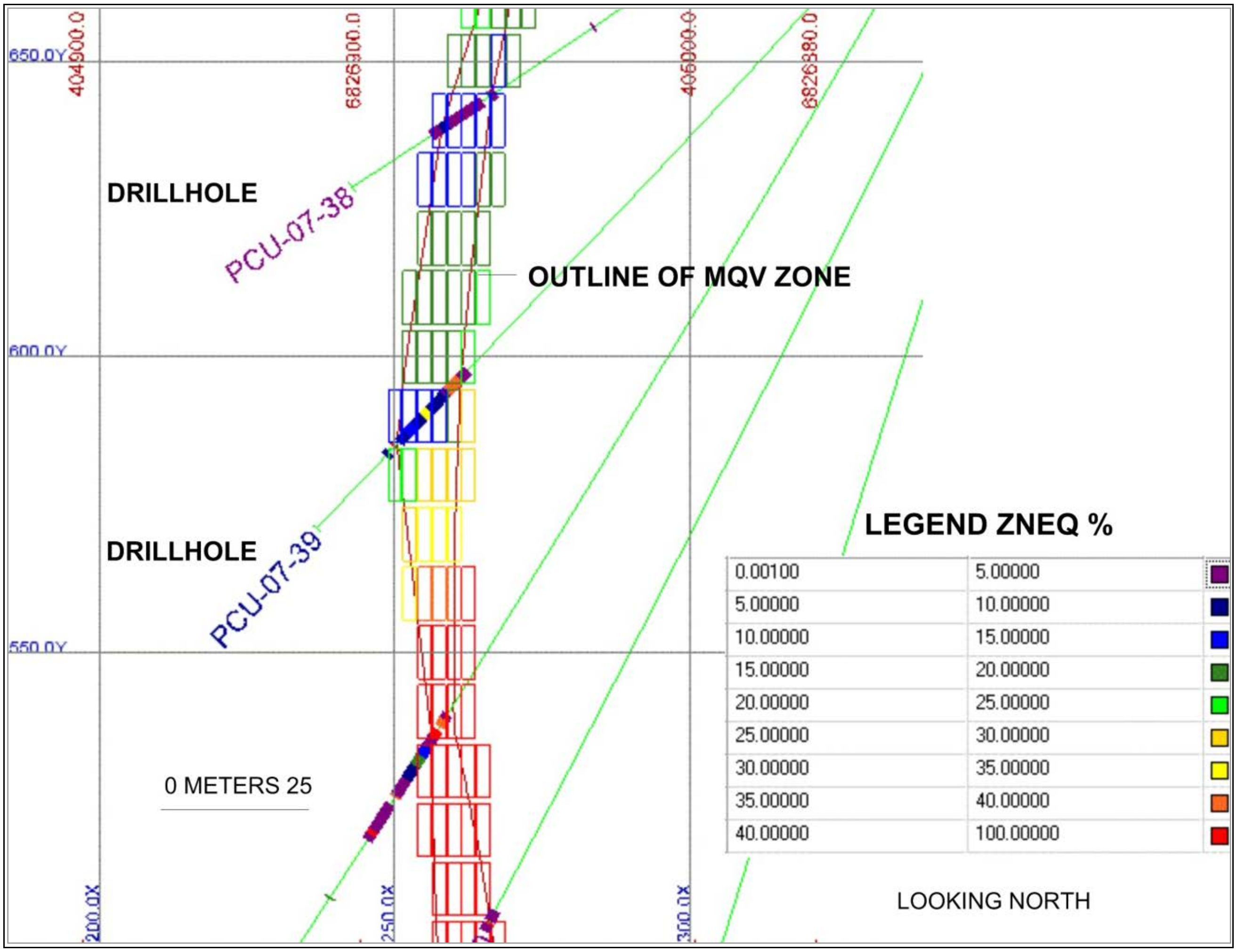


Figure 14.7 shows intercepts within the MQV Zone; mineralized intercepts outside the MQV Zone are located within the STK Zone which is not shown in this figure.

There is good correlation between composite and block model grades, as demonstrated by Table 14.12, which compares composite average grades with block model average grades for silver, copper, lead and zinc for each of the three zones.

Table 14.12 Comparison of assay composites to block model outputs

Zone	Metal	# composites	# composites used	Mean of composites	Mean of estimate
MQV	Ag	880	880	193	195
MQV	Pb	880	880	11.42	11.55
MQV	Zn	880	880	11.42	11.49
SMS	Ag	184	179	48	49
SMS	Pb	184	179	4.98	5.02
SMS	Zn	184	179	10.23	10.26
STK	Ag	860	834	32	32
STK	Pb	860	834	1.86	1.81
STK	Zn	860	834	3.20	3.17

Table 14.13 is a comparison of the ID^2 and kriged estimates. Differences are consistent with the characteristic behavior of the two estimation methods in that OK estimation tends to result in more tonnes at lower grades than does ID^2.

Table 14.13 September 2015 ID^2 and Kriged Mineral Resource estimate summaries at 8% ZnEq cut-off

	ID² resource estimate					Kriged resource estimate				
CLASS	Tonnes	ZnEq (%)	Ag (g/t)	Pb (%)	Zn (%)	Tonnes	ZnEq (%)	Ag (g/t)	Pb (%)	Zn (%)
MQV MEA	1,319,000	36.7	217	12.0	13.5	1,313,000	35.5	211	11.5	13.2
MQV IND	4,153,000	31.7	177	12.1	9.8	4,227,000	30.1	168	11.6	9.2
MQV MEA+IND	5,472,000	32.9	187	12.1	10.7	5,540,000	31.4	178	11.6	10.2
MQV INF	5,181,000	31.9	198	8.7	13.7	5,269,000	31.1	199	8.7	12.9
SMS MEA	-	-	-	-	-	-	-	-	-	-
SMS IND	1,035,000	20.3	57	5.5	11.1	1,042,000	19.5	54	5.2	10.8
SMS MEA+IND	1,035,000	20.3	11	1.0	2.1	1,042,000	19.5	54	5.2	10.8
SMS INF	167,000	22.0	60	6.5	11.5	170,000	21.4	60	6.3	11.2
STK MEA	171,000	26.1	131	5.8	14.1	169,000	23.5	116	5.3	12.6
STK IND	1,887,000	15.0	69	3.9	7.4	1,953,000	13.3	61	3.5	6.6
STK MEA+IND	2,058,000	15.9	74	4.1	8.0	2,122,000	14.1	66	3.6	7.1
STK INF	1,631,000	17.0	81	5.4	7.2	1,610,000	14.7	70	4.6	6.2

14.11 Comparison with March 2015 Mineral Resource Estimate

Table 14.14 shows a comparison of the September 2015 Mineral Resource estimate with the estimate reported in March 2015. The changes reflect the acquisition of additional data, since the March 2015 estimate, for the MQV and STK zones that has generally resulted in the conversion of MQV resources from the Indicated to the Inferred category, and the definition within the STK zone of a Measured Resource.

Table 14.14 Comparison of current and March 2015 Resource Estimates (8% ZnEq Cut-off)

	September 2015 estimate				March 2015 estimate			
MQV	**Tonnes**	**Ag (g/t)**	**Pb (%)**	**Zn (%)**	**Tonnes**	**Ag (g/t)**	**Pb (%)**	**Zn (%)**
Measured	1,313,000	211	11.5	13.2	1,279,000	211	11.6	13.2
Indicated	4,227,000	168	11.6	9.2	2,850,000	193	12.8	10.2
Measured & Ind	5,540,000	178	11.6	10.1	4,129,000	199	12.4	11.2
Inferred	5,269,000	199	8.7	12.9	6,132,000	194	10.4	12.6
SMS								
Measured	-	-	-	-	-	-	-	-
Indicated	1,042,000	54	5.2	10.8	1,059,000	55	5.4	10.8
Mea & Ind	1,042,000	54	5.2	10.8	1,059,000	55	5.4	10.8
Inferred	170,000	60	6.3	11.2	156,000	63	6.6	11.0
STK								
Measured	169,000	116	5.3	12.6	-	-	-	-
Indicated	1,953,000	61	3.5	6.6	1,400,000	63	4.0	7.0
Mea & Ind	2,122,000	66	3.6	7.1	1,400,000	63	4.0	7.0
Inferred	1,610,000	70	4.6	6.2	791,000	61	4.0	4.7
Total								
Measured	1,482,000	200	10.8	13.1	1,279,000	211	11.2	12.4
Indicated	7,222,000	123	8.5	8.7	5,309,000	131	9.0	9.5
Mea & Ind	8,704,000	136	8.9	9.5	6,588,000	147	9.4	10.1
Inferred	7,050,000	166	7.7	11.3	7,078,000	177	9.6	11.7

15 Mineral Reserve estimates

15.1 Introduction

A Mineral Reserve is defined by the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) within the CIM Definition Standards on Mineral Resources and Mineral Reserves, as adopted by CIM Council on 10 May 2014, as follows:

"A Mineral Reserve is the economically mineable part of a Measured and/or Indicated Mineral Resource. It includes diluting materials and allowances for losses, which may occur when the material is mined or extracted and is defined by studies at Preliminary Feasibility or Feasibility level as appropriate that includes application of Modifying Factors. Such studies demonstrate that, at the time of reporting, extraction could be reasonably justified."

CIM guidelines require that only material categorized as Measured or Indicated Resources be considered for potential Mineral Reserves.

15.2 Mineral Reserves estimation

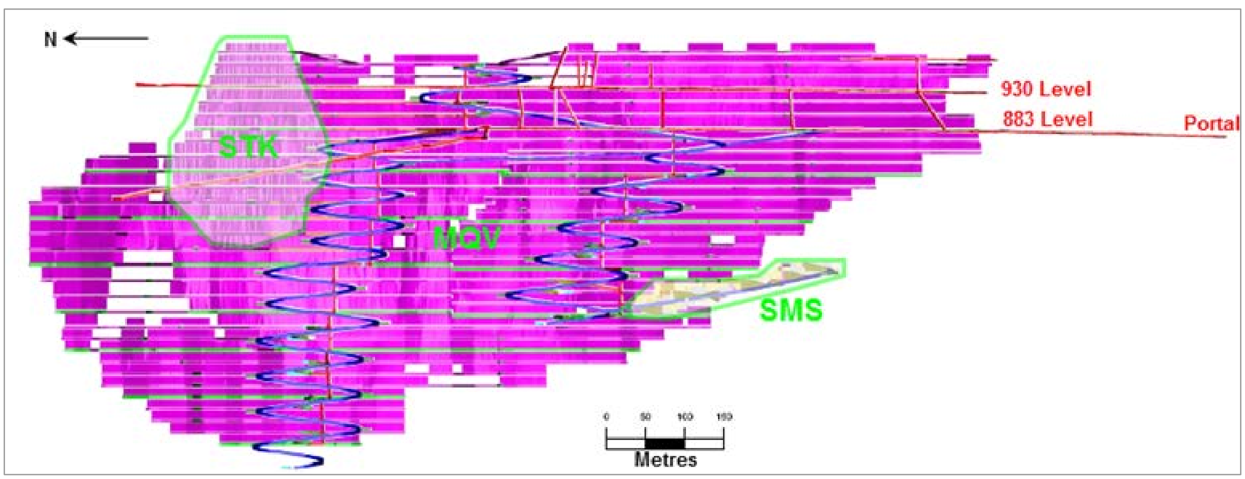
All design and scheduling has been completed to a level of detail appropriate for a feasibility study using a Mineral Resource model generated by AMC on 10 September 2015. All mining at Prairie Creek will be by underground methods.

15.3 Orebody description

The Mineral Resources included in the mine plan occur within three distinct zones: Main Quartz Vein (MQV), Stockwork (STK) and Stratabound sulphides (SMS), as shown in Figure 15.1.

The MQV is steeply dipping (70° to 90°) and generally varies in width from less than 0.1 m and up to 5 m. In the STK zone, the mineralization is also steeply dipping but is distributed over a larger volume and is made up of individual veins or mineralization occupying dilatant fractures. The SMS zone is relatively flat-lying (dipping approximately 15 to 20°), and is generally of the order of 15 m to 20 m in thickness.

Figure 15.1 Vertical longitudinal view of Prairie Creek mining zones



15.4 Preparation of block model for STK design

A single block model was used to estimate Mineral Resources for the MQV, STK, and SMS Zones, as described in Section 14. The size and orientation of the blocks were optimized with respect to the MQV Zone, which accounts for almost 70% of the total Mineral Resource. However, the trend of mineralization in the STK Zone is oblique to the trend of the MQV Zone so that, within the STK Zone, the block size was non-optimal and generally incorporated considerable wall rock dilution on the sides of the main mineralized lens. The presence of these large quantities of wall rock resulted in designed stopes that were non-optimal in terms of shape and dilution. To minimize dilution and provide more realistic shapes on which to carry out stope design, blocks within the STK Zone were reduced in size in an adjusted model. The original blocks were 15 m in strike length, 2.5 m across strike and 10 m in vertical height. In the adjusted model, the strike length of blocks in the STK Zone was reduced to 5 m, which gave a much better spatial fit between the zone interpretation and the filled blocks. The across-strike and height dimensions of the blocks were not changed. The change in block strike dimension had a minimal and non-material impact on the estimated tonnage and grade of the Mineral Resource. Table 15.1 shows a comparison of the reported tonnes and grades using the original block size and the adjusted model blocks with a 5-m strike length.

Table 15.1 Comparison of STK Resources using 15 m and 5 m block lengths

Classification	Tonnes	Ag g/t	Pb %	Zn %	ZnEq %
Original block size 15 x 2.5 x 10 m					
Measured	169,000	116	5.3	12.6	23.5
Indicated	1,953,000	61	3.5	6.6	13.3
Inferred	1,610,000	70	4.6	6.2	14.7
Modified block size 5 x 2.5 x 10 m					
Measured	177,000	119	5.3	12.8	23.8
Indicated	1,935,000	62	3.5	6.7	13.4
Inferred	1,598,000	70	4.6	6.2	14.6
Percent change modified / original block size					
Measured	4.6%	2.6%	0.4%	1.3%	1.2%
Indicated	-0.9%	1.6%	0.3%	0.5%	0.2%
Inferred	-0.8%	0.0%	-0.2%	-0.2%	0.3%

15.5 Cut-off grade

Zinc equivalent (ZnEq) cut-off grades (COG) were calculated using the parameters shown in Table 15.2. Primary ZnEq cut-offs were 11% for longhole open stoping (LHOS) and 11% for drift and fill (DAF). 10% ZnEq was applied as an incremental stoping cut-off and 6% was applied as a cut-off for development ore. These design cut-offs equate to C$303/t (11% Ni), C$267/t (10% Ni) and C$165/t (6% Ni) as in situ values (prior to application of dilution and mining recovery factors) at a zinc price of US$1/lb and exchange rate of C$1 = US$0.8.

Table 15.2 Inputs for zinc equivalent cut-offs

Item	Units	Value
Zinc Price	US$/lb	1.00
Lead Price	US$/lb	1.00
Silver Price	US$/oz	18.00
Zinc Recovery	%	75
Lead Recovery	%	88
Silver Recovery	%	92
Zinc Payable	%	85
Lead Payable	%	95
Silver Payable	%	81
Pounds per Tonne	lb/t	2,204.6
Grams per Troy Ounce	g/oz	31.10348

Exchange rate assumed as C$1 = US$0.8.

The zinc equivalent calculation is as follows:

> *ZnEq% = (Grade of Zn in %) + [(Grade of lead in % * Price of lead in $/lb * 22.046 * Recovery of lead in % * Payable lead in %) + (Grade of silver in g/t* (Price of silver in US$ per Troy oz/ 31.10348) * Recovery of silver in % * Payable silver in %)]/(Price of zinc in US$ per lb*22.046 * Recovery of zinc in % * Payable zinc in %)*

15.6 Mining shapes

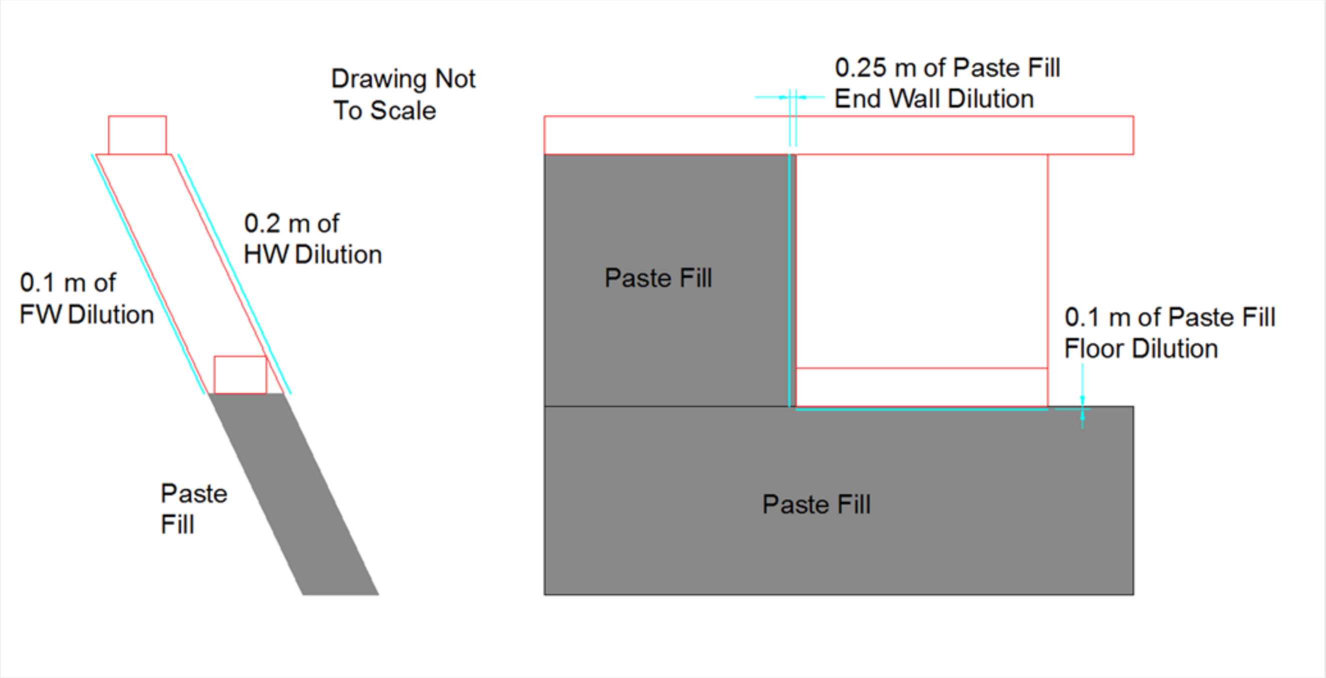
For longhole stope generation in the MQV, AMC used the software Mineable Shape Optimizer (MSO) to produce stope shapes based on the 11% COG and other key design criteria. Shapes were manually combined into full-length stopes of about 30 m along strike. In the STK, MSO was also used to generate LHOS shapes but these were subsequently divided manually into stopes of 7.5 m maximum width. The SMS DAF stopes were generated manually based on 11% COG grade-shells on a 5 m cut interval.

Generally towards the extremities of the ore-body, some marginally economic areas were not included in the mine plan and, therefore, not included in the Mineral Reserves. Capital and operating development cost estimates were used to determine economic viability for these areas.

15.7 Dilution and recovery estimates

The LHOS shapes generated in MSO and then manually edited are inclusive of any planned dilution required to extract the mineralized material. Average unplanned waste dilution thickness along the hangingwall and footwall was estimated to be 0.2 m and 0.1 m, respectively. Average pastefill dilution from the floor was estimated to be 0.1 m and average endwall dilution from an adjacent filled stope was estimated to be 0.25 m. The resultant planned and unplanned dilution values of 13% and 11% respectively were evaluated from the stope shapes. Figure 15.2 represents a single LHOS panel with the unplanned dilution applied. The mining recovery utilized for all LHOS was 95%.

Figure 15.2 Typical LHOS with the unplanned dilution applied



In the DAF stopes, an unplanned dilution factor of 6% and a mining recovery of 98% were used.

15.8 Mineral Reserves

The Prairie Creek Mineral Reserves, in total and broken down by zone, are summarized in the Table 15.3.

Table 15.3 August 2017 Mineral Reserves, Prairie Creek Mine

Mineral zone	Classification	Tonnes (t)	Silver (g/t)	Lead (%)	Zinc (%)	Zinc equivalent
Main Quartz Vein (MQV)	Proven	1,524,171	161.43	8.90	10.22	26.84
	Probable	4,190,187	144.76	9.96	8.20	25.70
	Total	**5,714,358**	**149.21**	**9.67**	**8.74**	**26.00**
Stockwork (STK)	Proven	188,173	108.19	4.84	11.56	21.22
	Probable	1,188,366	63.81	3.54	6.86	13.46
	Total	**1,376,539**	**69.88**	**3.72**	**7.50**	**14.52**
Stratabound (SMS)	Proven	-	-	-	-	-
	Probable	980,566	54.90	5.06	9.64	17.97
	Total	**980,566**	**54.90**	**5.06**	**9.64**	**17.97**
Total	Proven	1,712,344	155.58	8.45	10.36	26.22
	Probable	6,359,119	115.78	8.00	8.17	22.22
	Total	**8,071,463**	**124.22**	**8.10**	**8.64**	**23.07**

2017 Mineral Reserves are as of 2 August 2017 and based on a design cut-off grade of 11% ZnEq for LHOS, 11% ZnEq for DAF, an incremental stoping cut-off grade of 10% ZnEq and 6% ZnEq for development ore.
Cut-off grades are based on a zinc metal price of $1.00/lb, recovery of 75% and payable of 85%, a lead metal price of $1.00/lb, recovery of 88% and payable of 95%, and a silver metal price of $18/oz, recovery of 92% and payable of 81%.
Exchange rate used is C$1.25 = US$1.00.
Average planned dilution, unplanned dilution and mining recovery factors of 13%, 11% and 95%, respectively, for LHOS; and 18%, 6%, and 98%, respectively, for DAF are assumed.

15.9 Conversion of Mineral Resources to Mineral Reserves

The conversion of Measured and Indicated Mineral Resources to Proven and Probable Mineral Reserves is shown for the mine as a whole and for each of the zones in Table 15.4.

In terms of the total Measured and Indicated Resources, the 2017 Mineral Reserves show a 93% tonnage conversion (inclusive of mine design dilution) and an 84% to 85% metal content conversion.

Table 15.4 Conversion of Mineral Resources to Mineral Reserves

		Mineral Resources			Mineral Reserves			Conversion factor*		
		MEA + IND	MEA	IND	ALL	PRV	PRB	ALL	MEA / PRV	IND / PRB
Total										
Tonnes	kt	**8,704,000**	1,482,000	7,222,000	**8,071,463**	1,712,344	6,359,119	93%	116%	88%
Silver	g/t	**136.00**	200.00	123.00	**124.22**	155.58	115.78	85%	90%	83%
Lead	%	**8.90**	10.80	8.50	**8.10**	8.45	8.00	84%	90%	83%
Zinc	%	**9.50**	13.20	8.70	**8.64**	10.36	8.17	84%	91%	83%
MQV										
Tonnes	kt	**5,540,000**	1,313,000	4,227,000	**5,714,358**	1,524,271	4,190,187	103%	116%	99%
Silver	g/t	**178.00**	211.00	168.00	**149.21**	161.43	144.76	86%	89%	85%
Lead	%	**11.60**	11.50	11.60	**9.67**	8.90	9.96	86%	90%	85%
Zinc	%	**10.20**	13.20	9.20	**8.74**	10.22	8.20	88%	90%	88%
STK										
Tonnes	kt	**2,122,000**	169,000	1,953,000	**1,376,539**	188,173	1,188,366	65%	111%	61%
Silver	g/t	**66.00**	116.00	61.00	**69.88**	108.19	63.18	69%	104%	64%
Lead	%	**3.60**	5.30	3.50	**3.72**	4.84	3.54	67%	102%	62%
Zinc	%	**7.10**	12.60	6.60	**7.50**	11.56	6.86	69%	102%	63%
SMS										
Tonnes	kt	**1,042,000**	-	1,042,000	**980,566**	-	980,566	94%		94%
Silver	g/t	**54.00**	-	54.00	**54.90**	-	54.90	96%		96%
Lead	%	**5.20**	-	5.20	**5.06**	-	5.06	92%		92%
Zinc	%	**10.80**	-	10.80	**9.64**	-	9.64	84%		84%

*Metal conversion factors reflect total metal content.
STK Measured/Proven metal ratios reflect re-blocking of resource model for mine design purposes – see Section 15.4.

15.10 Comparison of 2017 and 2016 Mineral Reserves

Table 15.5 compares 2017 Mineral Reserves with those for the previous Mineral Reserve estimation in 2016. The overall increase in Mineral Reserves is due to marginally lower ZnEq cut-off grades - reflecting the final 2016 PFS operating cost estimate, an increase in projected Zn price from $1.00/lb to $1.10/lb, and further optimization of the stoping design. The 2017 Mineral Reserves have slightly lower average metal grades than those estimated in the 2016 PFS, but increased overall metal content.

Table 15.5 Comparison of 2017 and 2016 Mineral Reserves

Mineral Reserves comparison		Mineral Reserve 2017			Mineral Reserve 2016			Tonnes and contained metal difference factor*		
		ALL	PRV	PRB	ALL	PRV	PRB	ALL	PRV / PRV	PRB / PRB
Total										
Tonnes	kt	**8,071,463**	1,712,344	6,359,119	**7,603,590**	1,373,944	6,229,646	106%	125%	102%
Silver	g/t	**124.22**	155.58	115.78	**127.58**	175.7	116.97	103%	110%	101%
Lead	%	**8.10**	8.45	8.00	**8.33**	9.41	8.09	103%	112%	101%
Zinc	%	**8.64**	10.36	8.17	**8.93**	12.02	8.24	103%	107%	101%
MQV										
Tonnes	kt	**5,714,358**	1,524,271	4,190,187	**5,166,136**	1,199,288	3,966,848	111%	127%	106%
Silver	g/t	**149.21**	161.43	144.76	**160.37**	186	152.62	103%	110%	100%
Lead	%	**9.67**	8.90	9.96	**10.42**	10.08	10.52	103%	112%	100%
Zinc	%	**8.74**	10.22	8.20	**9.39**	12.09	8.58	103%	107%	101%
STK										
Tonnes	kt	**1,376,539**	188,173	1,188,366	**1,472,322**	174,656	1,297,665	93%	108%	92%
Silver	g/t	**69.88**	108.19	63.18	**65.97**	105.01	60.72	99%	111%	96%
Lead	%	**3.72**	4.84	3.54	**3.57**	4.80	3.41	97%	109%	95%
Zinc	%	**7.50**	11.56	6.86	**7.22**	11.48	6.64	97%	108%	95%
SMS										
Tonnes	kt	**980,566**	-	980,566	**965,132**	-	965,132	102%	-	102%
Silver	g/t	**54.90**	-	54.90	**46.09**	-	46.09	121%	-	121%
Lead	%	**5.06**	-	5.06	**4.38**	-	4.38	117%	-	117%
Zinc	%	**9.64**	-	9.64	**9.03**	-	9.03	108%	-	108%

For 2017 Mineral Reserves see footnotes to Table 15.3.
2016 Mineral Reserves are as of 31 March, 2016 and based on a design cut-off grade of 12% ZnEq for LHOS, 11% ZnEq for DAF, an incremental stoping cut-off grade of 9.7% ZnEq and 7.1% ZnEq for development ore.
Cut-off grades are based on a zinc metal price of $1.00/lb, recovery of 75% and payable of 85%, a lead metal price of $1.00/lb, recovery of 88% and payable of 95%, and a silver metal price of $17/oz, recovery of 92% and payable of 81%.
Exchange rate used is C$1.25 = US$1.00.
* % comparison 2017 vs 2016.

16 Mining methods

16.1 Introduction

Prairie Creek will be an underground mine extracting the majority of ore from the steeply-dipping, narrow Main Quartz Vein (MQV). Smaller ore quantities will be mined from the Stockwork (STK) and Stratabound Massive Sulphide (SMS) zones, generally later in mine life. Three levels of adits (970 L, 930 L, 883 L) were established previously. Five shrinkage stopes were partly mined above the 930 and 883 levels, giving a stockpile of about 10,000 tonnes of mixed ore and waste that is currently located adjacent to the mill.

The MQV zone area designed for mining in the FS covers a strike distance of about 1,000 m and a vertical distance of about 400 m. Below 883 L, mining levels will be established at generally 60 m intervals with 15 or 20 m sublevels. Initial stoping will start from the 823 L. As mining on the MQV progresses to depth, ore mined will be supplemented by the STK and SMS zones – see Figure 16.2. Lower levels will be developed to depth through ramp access over the first approximately five years of operation.

Mining will be by longhole open stoping (LHOS) in the MQV vein and in the STK area. Drift and fill (DAF) will be employed in the SMS area. An average mining rate of 1,600 tonnes per day of ore is projected. At steady-state, approximately 585,000 tonnes of ore per year will be mined. Mine life is projected to be 14.5 years from start-up of the processing plant for the total Mineral Reserve of 8.1 million tonnes.

MQV material will be the majority of ore production and will be extracted throughout the life of the mine. The vein structure is currently exposed in over 800 m of backs in the existing underground development.

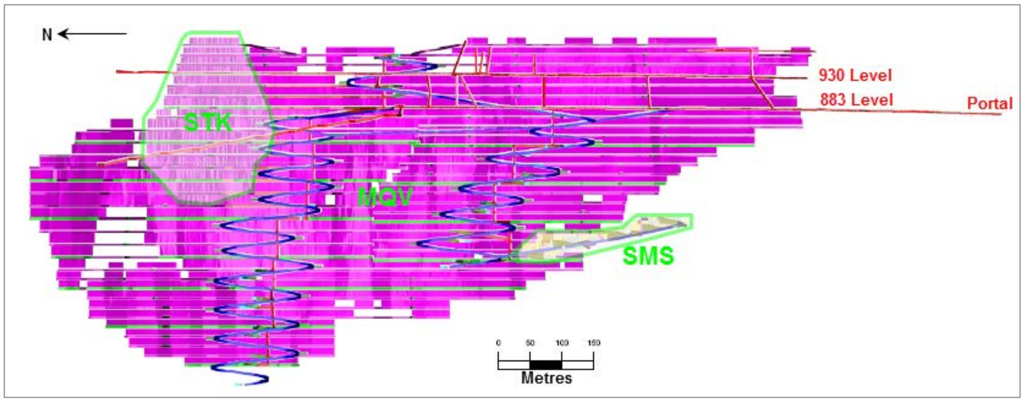
The SMS mineralization occurs approximately 200 m below 883 L and will require significant underground development for access. SMS mining is scheduled to start during the tenth year of the mine life.

LHOS will use electric-hydraulic drill jumbos and diesel-powered scoops for waste development. Ore development will use the same equipment in early years of production but with battery-powered scoops planned to be used in later years. Production drilling will be by conventional longhole drills. The SMS zone will use electric-hydraulic drill jumbos and diesel-powered scoops. Bolting will generally be accomplished by mechanized bolters; ore and waste movement to surface will be via conventional truck haulage.

Main access to the mine will be through the existing 870 portal. It should be noted that, previously, the 883 L has been referred to as both the 870 L and the 880 L. This access will be slashed out to 5.0 m high by 5.0 m wide after removal of the existing track. Underground development currently in place reduces the amount of full-face development needed for mine operation. Access to the ore below the 883 L will be via twin ramps as shown in Figure 16.1. A single ramp will provide access to the ore above 883 L.

Ground conditions in existing development underground are generally good. Current workings have stood unsupported for about 35 years with minimal bolting. Figure 16.1 is a longitudinal view of the underground workings as per the FS mine plan. The existing workings are shown in orange, future ramp development in blue, sublevels and stopes, ultimately backfilled, in purple. Ventilation raises are shown in red.

Figure 16.1 Longitudinal view of underground workings

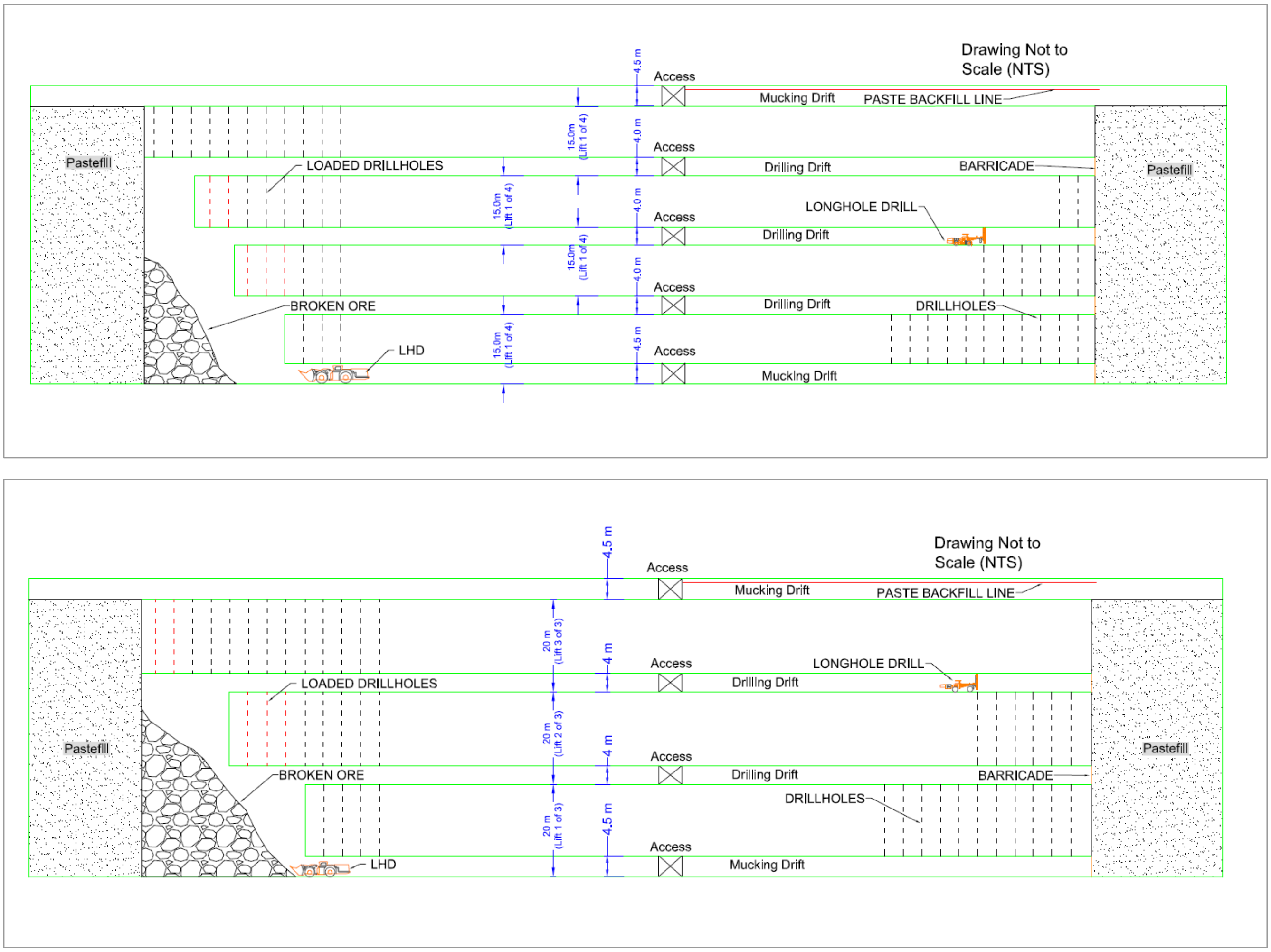


16.2 Mining methods

16.2.1 Longhole Open Stoping (LHOS)

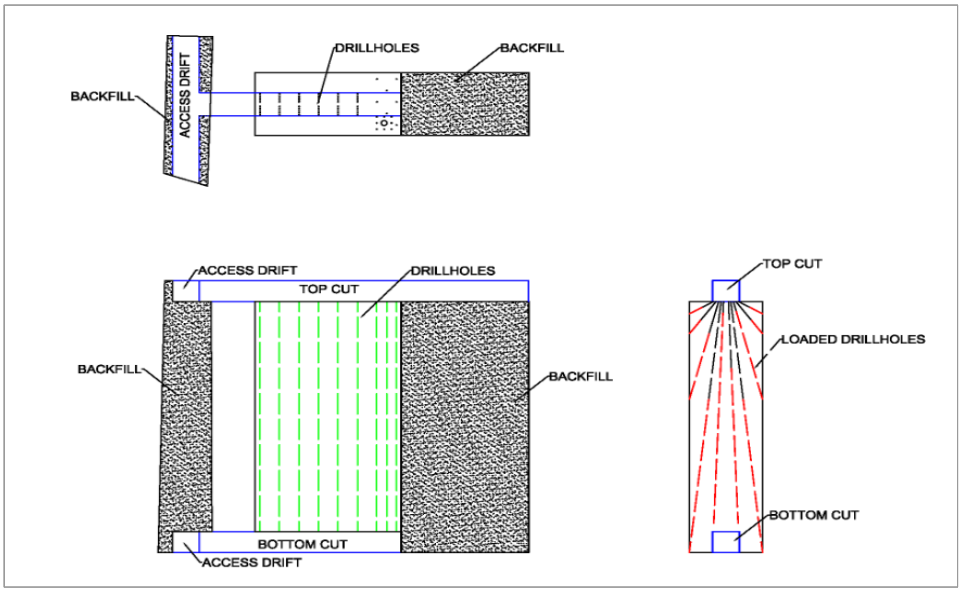
Mining of the MQV will be by LHOS. The mining level in the ore at the 883 m elevation has previously been established along a significant part of its length but will be slashed to 4.5 m W x 4.5 m H to serve as a mucking horizon. The average vein width is less than 4.5 m; where it exceeds 4.5 m, the level and sub-level widths will be adjusted appropriately. Mining below the 883 level will require similar main levels, driven 4.5 m W x 4.5 m H, to be established every 60 m vertically; within this 60 m height there will be a sub-level driven every 15 m or 20 m at 4.0 m x 4.0 m dimensions. Generally, a 15 m sub-level interval is adopted where the vein width is less than 3 m. Access to the sublevels will be gained by ramp and cross-cuts. Ore development in the vein will be for a distance of up to 250 m north and south (each side) of the access point. Slots will then be developed between sub-levels followed by retreat LHOS towards the access in approximately 30 m panels. The lowest elevation sub-level will lead the mining front, as shown in Figure 16.2, with all ore being mucked from the main (mucking) level. Broken ore will be drawn down in a controlled fashion so as to provide interim wall support and assist in minimizing wall dilution. Fill fences will be constructed on all levels other than the uppermost, after the extraction of each panel. The mining cycle will thus involve ore development, followed by drilling, blasting, mucking, and filling.

Figure 16.2 Proposed longhole open stoping method (15 m and 20 m sub-level intervals)



The STK area, and any MQV areas where geometry necessitates less than 60 m high extraction, will generally be mined in similar longhole retreat fashion at individual panel heights up to 20 m. For STK mining, some access development through paste fill is projected for the later stages in the mine life. Figure 16.3 illustrates design and extraction for a single STK panel with access driven through fill.

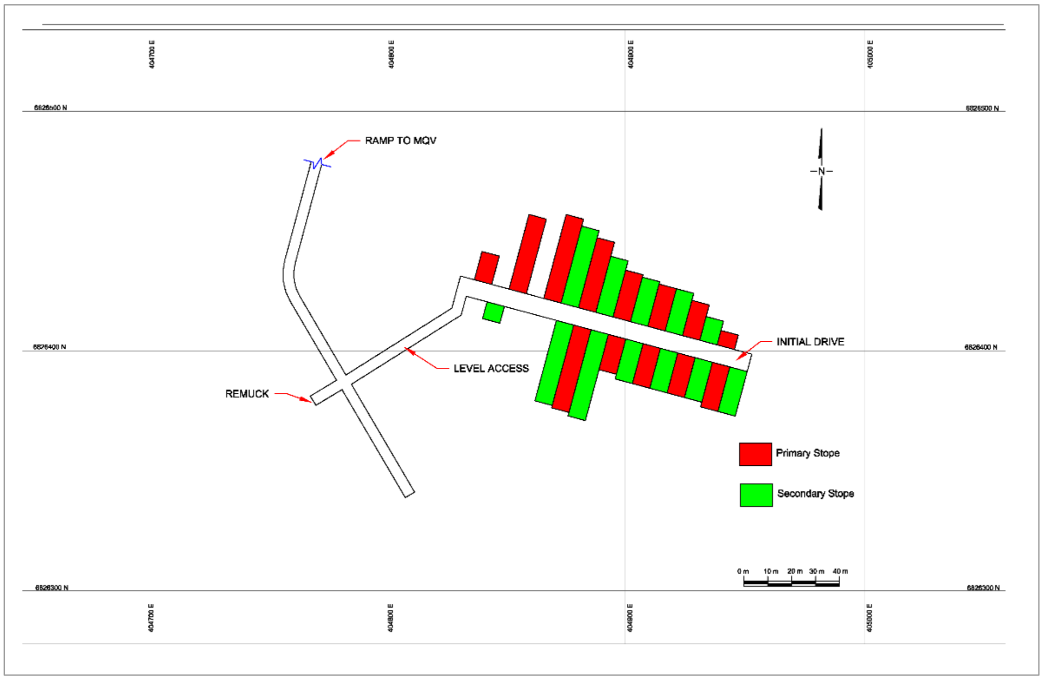
Figure 16.3 STK longhole open stoping, plan and section views showing development access through fill



16.2.2 Drift and Fill (DAF)

The SMS zone is approximately tabular in nature, dipping at about 15 to 20 degrees and with thickness up to, nominally, 20 m. It will be mined using a primary-secondary DAF mining method as shown in Figure 16.4 below. Panels will be driven at 5 m H x 7.5 m W to the extremities of the ore before filling with, largely, paste fill. Primary panels will thus generally have ore on either side, while secondary panels will generally have backfill on either side. Binder concentration in secondaries will generally be significantly lower than that for primaries. Extraction of each 5-m cut on any horizon will be followed by extraction of the cut above in similar fashion. Each access into the zone may be used for up to three cuts by taking down backs in the access after each cut is complete.

Figure 16.4 Proposed drift and fill mining method (plan view)

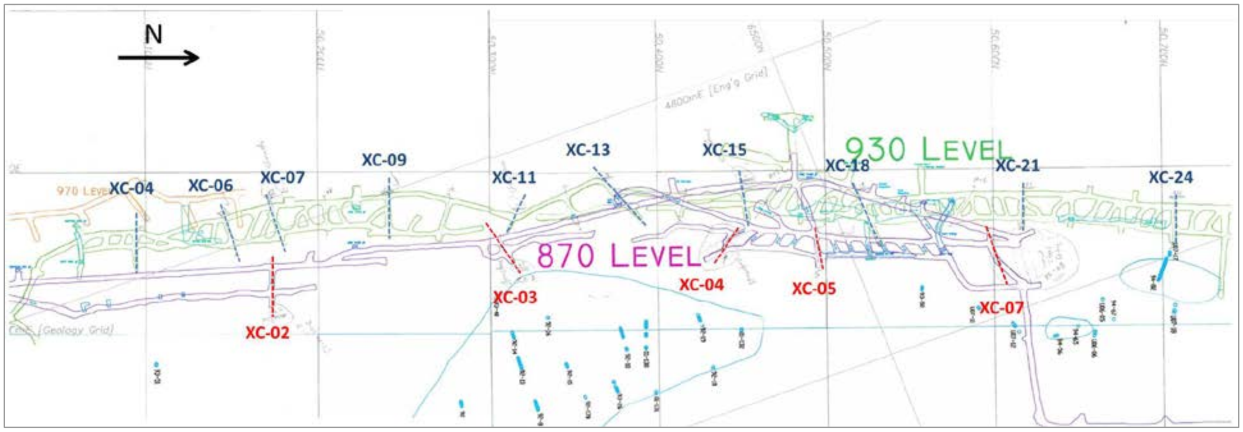


16.3 Geotechnical considerations

Earlier mine plans (AMC, 2012) envisioned mining by mechanized cut-and-fill. In 2013, CZN engaged AMC to evaluate LHOS as an option, with the aim of improving safety and productivity. AMC completed a campaign of geotechnical mapping and review of drill core on site in November 2013 leading to underground mine design criteria for stope stability and ground support (AMC, 2014). Subsequent assessment was undertaken as part of the PFS and FS work. This more recent work included particular consideration of the LHOS mining method with blasted ore offering wall support for much of a stope panel life prior to final mucking and filling with paste fill.

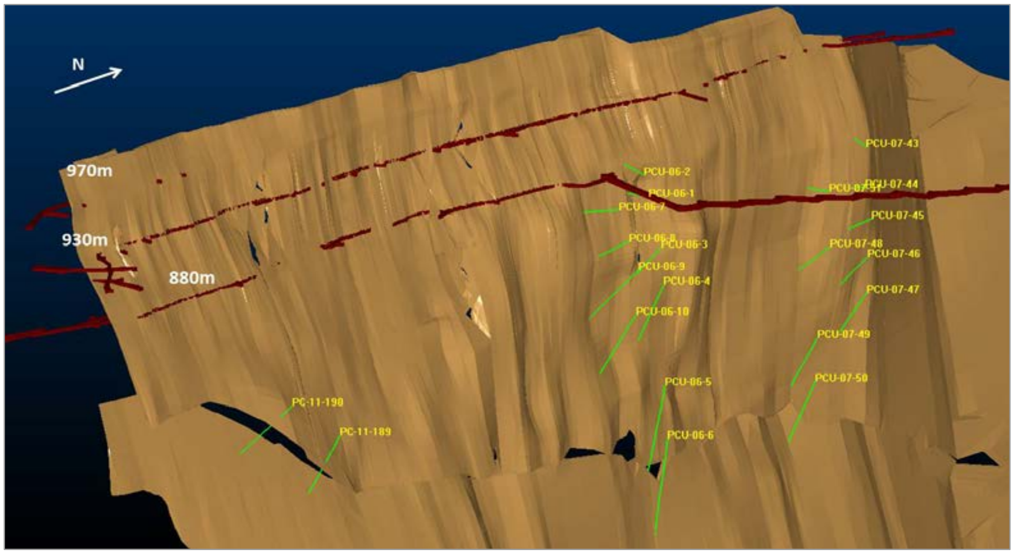
A snapshot of 2013 mapping locations and the levels accessed is shown in Figure 16.5. Geotechnical mapping data and geological information for each mapping section were provided in an AMC geotechnical report (2014).

Figure 16.5 Geotechnical mapping sections



Due to the uneven distribution of geotechnical information relative to the overall geometry of the Mineral reserves (significant data for the upper part of the mine and much less at depth), and to assist with analysis, core photographs from exploration holes drilled in 2006, 2007, and 2008 were also utilized. In spatial terms this information generally covered the hangingwall (HW) area, within 30 m of the ore zone contact. The location of these holes relative to the ore-zone wireframe is shown in Figure 16.6.

Figure 16.6 Exploration drillhole traces within 30 m of the ore contact, used for geotechnical assessment

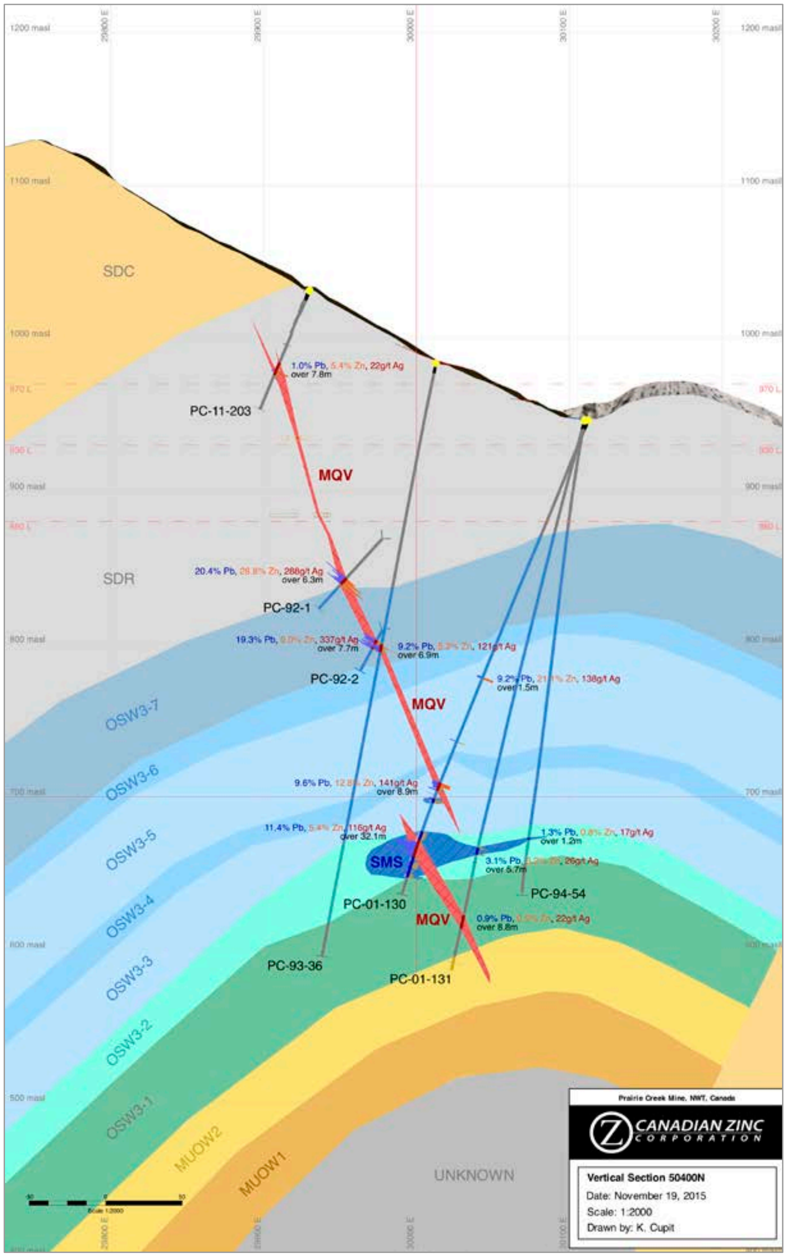


In general, the core photograph assessment indicated the HW lithologies with RQD values typically ranging from 75% to 90% (rock units OSW3-4, OSW3-5, and OSW3-6). The orebody and footwall (FW) lithologies have a wide range of RQD values, typically from Very Poor to Good. Rock unit OSW3-1 RQD values show a typical range of 10% to 25%.

Rock mass quality in drift areas mapped is generally seen to be variable, with transition zones between the main lithological units and the veined ore zone. In some areas a Poor-quality shear zone has been identified in the hangingwall. The HW rocks were noted to be usually of poorer quality than the ore zone or FW rocks.

The various sub-units provide a means of assessing rock mass parameters with depth. For example the Whittaker Formation OSW3-7 sub-unit overlies the OSW3-6 sub-unit, which in turn overlies the OSW3-5 sub-unit, etc. For this reason the assessment of appropriate geotechnical parameters for design has used the sub-units to group the various parameters assessed. Figure 16.7 is a representative cross-section through the main mine area showing both stratigraphy and mineralization.

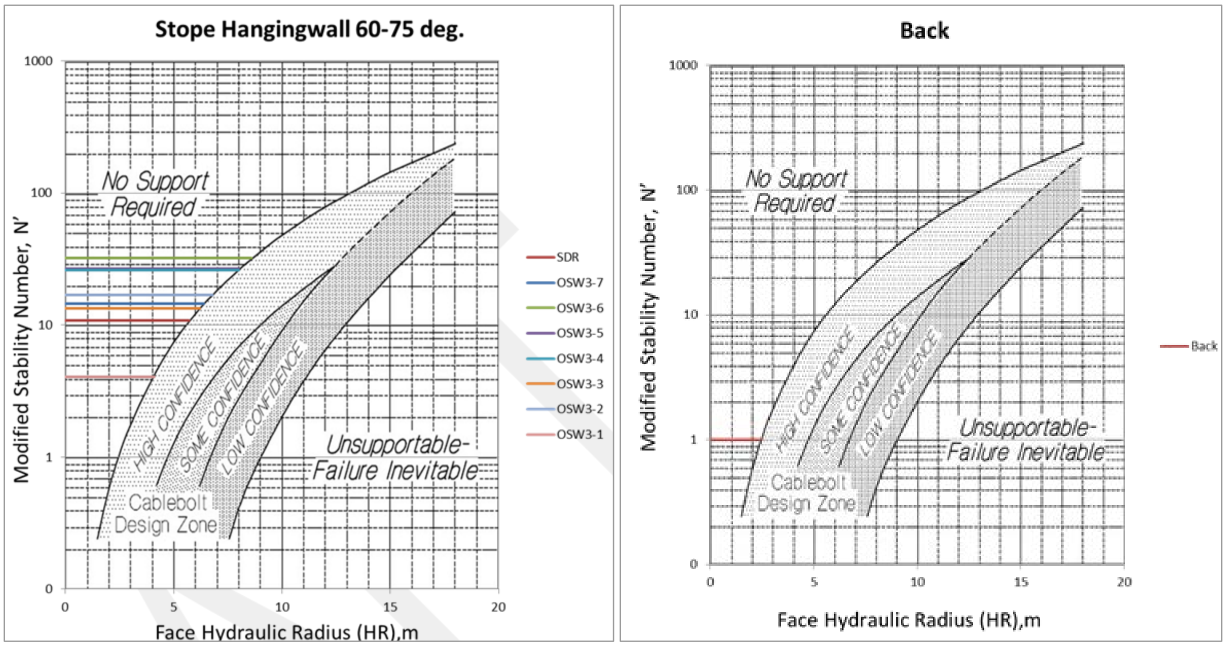
Figure 16.7 Simplified vertical cross section of Prairie Creek Main Zone



Based on analysis of the available data, shear zones may occur within the HW sequence at varying distances from the HW contact. In general, the shear zone can be seen from at the HW contact to approximately 30 m from the HW contact. In some areas the HW shear zone has been noted to be absent. The HW shear zone consists of varying degrees of fragmented and sheared dolomite. It is commonly a single zone varying in width from 0.1 m to 2 m. In some areas, however, it occurs as two or more zones of poor-quality rock separated by narrow zones of intact rock.

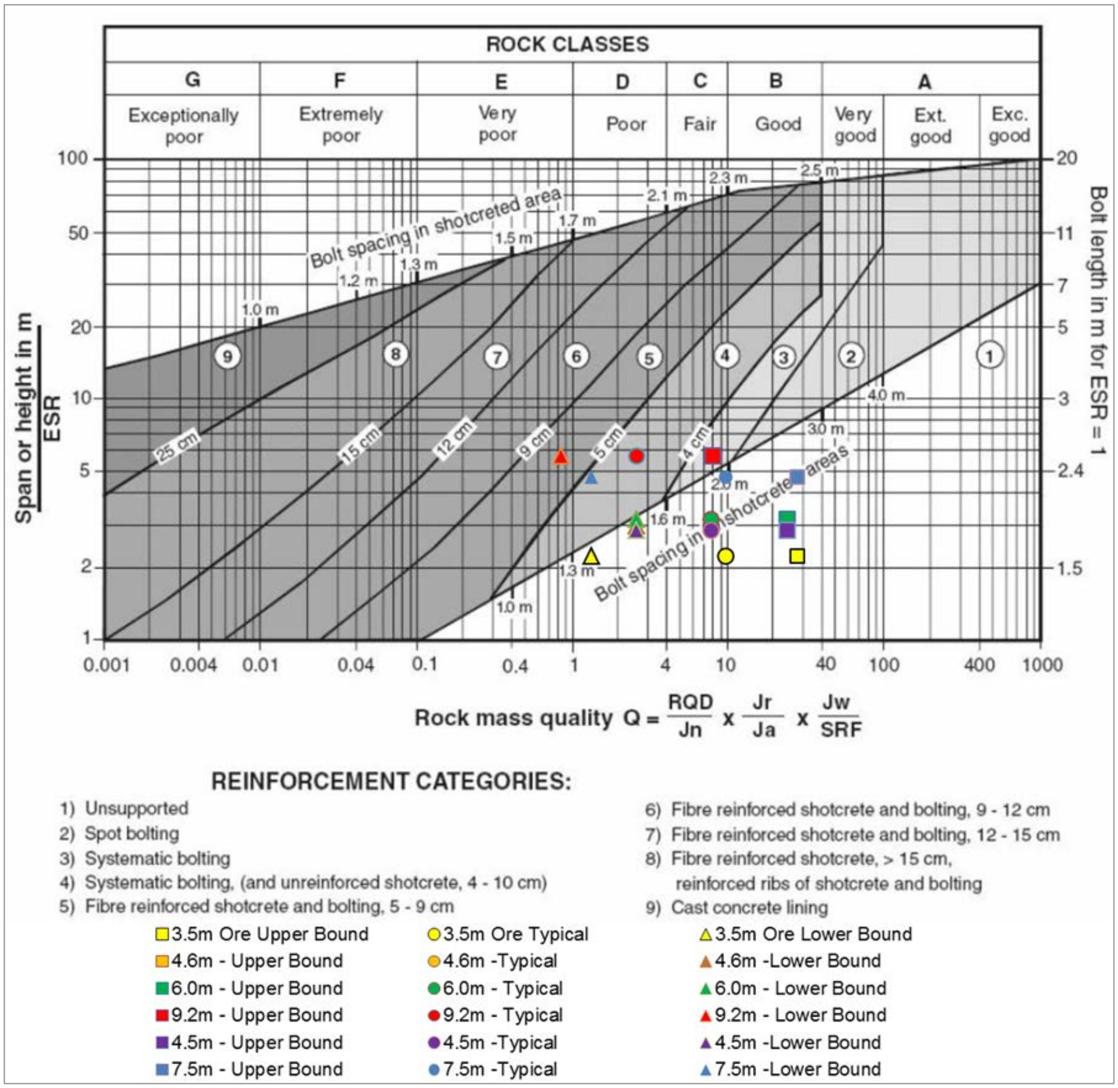
The stability number (N') for rock mass conditions of each lithological sub-unit is plotted on the stability graphs shown in Figure 16.8. These graphs are representative of the stope walls and stope back, respectively. They illustrate the correlation between N' and the excavation surface hydraulic radius (HR). Some dilution should be expected as the final muck from a stope is drawn down; it is, however, difficult to quantify this aspect without additional structural information and prior to actual stope excavation experience. Timely removal of final stope ore and prompt placement of paste fill in stopes will impact stope wall stability.

Figure 16.8 Stability graph results for stope walls (typical average dip 60–75 deg.) and stope backs



Preliminary empirical support requirements are presented in Figure 16.9. The results indicate that typical (Fair) to upper bound (Good) ground conditions plot within the 'unsupported' category. For safety and stability, however, systematic bolting and mesh placement would be required for rock catchment in areas where personnel may be exposed. Excavations within the lower bound (Very Poor to Poor) ground conditions plot within support categories 4 to 5. These ground conditions would require systematic bolting as well as shotcrete support for stability. Preliminary ground support standards are outlined in Figure 16.8.

Figure 16.9 Support chart from the Q System



It should be noted that Figure 16.9 also indicates that, in Poor to Fair ground conditions, additional secondary ground support (cable bolts) will be required in wide spans such as excavation intersections. All excavations should have weldmesh screen placed to cover the back and haunches of the excavation to within 1.5 m of the floor. For the shorter-life needs of ore development, the option exists to use split-sets or Swellex. For permanent mine development, #6 (5/8 inch) or #7 (7/8 inch) resin-grouted rebar should be used.

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

Table 16.1 Ground support recommendations

		Bolt length minimum (m)	Bolt spacing (m)	Bolt type	Fibre reinforced shotcrete (mm)	Cable length minimum (m)	Cable spacing (m)
Ore development 4.0m × 4.0 m)	4.0 m Ore Upper Bound	1.8	1.8	Swellex or Split set	-	-	-
	4.0 m Ore Typical	1.8	1.5	Swellex or Split set	-	-	-
	4.0 m Ore Lower Bound	1.8	1.5	Swellex or Split set	50	-	-
Main development	4.6 m - Upper Bound	1.8	1.8	Resin Rebar	-	-	-
(Waste) (4.6 m × 4.6 m)	4.6 m -Typical	1.8	1.5	Resin Rebar	-	-	-
	4.6 m - Lower Bound	2.4	1.2	Resin Rebar	50	-	-
Main development	5.0 m - Upper Bound	2.4	1.8	Resin Rebar	-	-	-
(Waste) (5.0 m × 5.0 m)	5.0 m - Typical	2.4	1.5	Resin Rebar	-	-	-
	5.0 m - Lower Bound	2.4	1.2	Resin Rebar	50	-	-
Main development	9.2 m - Upper Bound	2.4	1.8	Resin Rebar	-	6	2
Intersections	9.2 m -Typical	2.4	1.5	Resin Rebar	50	6	2
(9.2 m span)	9.2 m -Lower Bound	2.4	1.2	Resin Rebar	70	6	2
Mucking Drive	4.5 m - Upper Bound	1.8	1.8	Resin Rebar	-	-	-
(Waste) (4.5 m × 4.5 m)	4.5 m -Typical	1.8	1.5	Resin Rebar	-	-	-
	4.5 m - Lower Bound	2.4	1.2	Resin Rebar	50	-	-
Drift and Fill	7.5 m - Upper Bound	1.8	1.8	Swellex or Split set	-	-	-
(Ore) (7.5 m × 5 m)	7.5 m -Typical	1.8	1.5	Swellex or Split set	-	-	-
	7.5 m - Lower Bound	1.8	1.5	Swellex or Split set	50	-	-

A better understanding of the factors affecting stope stability and the proposed mining methods would be gained from additional data collection, interpretation, and analysis, including the following:

- Development of a series of 3D models that includes lithology, alteration and major structure, and including the interpreted location and thickness of the HW shear zones.
- Using data from these models to develop a 3D geotechnical model.
- Further detailed geotechnical data collection, including oriented core and laboratory testing.
- Hydrogeological characterization of the site.
- 2D-3D modelling with updated parameters to assess stope and crown pillar stability.
- Further assessment of ground support requirement.
- As the mine is developed to depth, in situ stress testing may be appropriate. This can be carried out during mine operations.

16.4 Mine design

The existing access via the 883L adit will be enlarged to 5.0 m H x 5.0 m W to provide appropriate main access from surface for personnel, equipment, fresh air and materials handling. The main access will connect to two internal ramps driven at 4.6 m H x 4.6 m W; one at the north end to service the MQV and STK zones between elevations 965 and 478, and a south ramp, between elevations 883 and 643, to service the generally narrower areas of the MQV in that area and the SMS zone. The ramps have been designed at a maximum +/-15% gradient with a minimum 20 m turning radius and remucks at 150 m intervals. Ore remucks and truck loading areas will be sited at every level access to the MQV. Within the vein, the maximum distance between remucks is approximately 200 m.

16.4.1 Lateral and vertical development design

Sublevels will be accessed from the ramps on a 15 m or 20 m vertical interval defined by the planned stoping heights. Ramp development will be set back typically 40 m (minimum 25 m) from the ore contact. This arrangement recognizes long-term geotechnical stability and provides adequate space for the placement of a return air raise and other services such as sumps, remucks, transformer bays and portable refuge locations.

The mine design includes raises for return air. Generally, raises from level to level (nominally 60 m) will be excavated by Alimak and will be outfitted to provide a means of second egress. One existing raise to surface will be rehabilitated to exhaust return air from the mine. The 930L will be an additional exhaust airway.

Ore drives in the MQV zone will typically be for a distance of up to 250 m each side of the access cross-cut from the ramp. In the STK zone, ore drives on a level will be driven in accordance with the geometry and mining sequence. The SMS zone will be accessed by a secondary ramp driven up from the south ramp at a gradient of 15%, with entry to the zone by a series of cross-cuts.

In the development of drives to support stoping, low-grade ore will be determined as that with a cut-off grade of 6% zinc equivalent (ZnEq); such material is projected to be approximately 3% of mined ore tonnes. Any development material below the marginal cut-off grade will be considered waste and will generally be placed on the surface waste stockpile; small amounts of waste may be placed in stopes as fill.

Development heading design considers equipment needs, mine services and regulatory requirements. Ore drives at the base of a mining block will be used for mucking access and will be driven at 4.5 m x 4.5 m; sub-level ore drives will be driven at 4.0 m x 4.0 m. Figure 16.10 and Figure 16.11 show typical ore drift cross-sections, considering the dimensions of mobile equipment and ventilation ducting. Figure 16.12 shows a typical level development in plan view.

Figure 16.10 4.0 m by 4.0 m drift for MT2000 truck

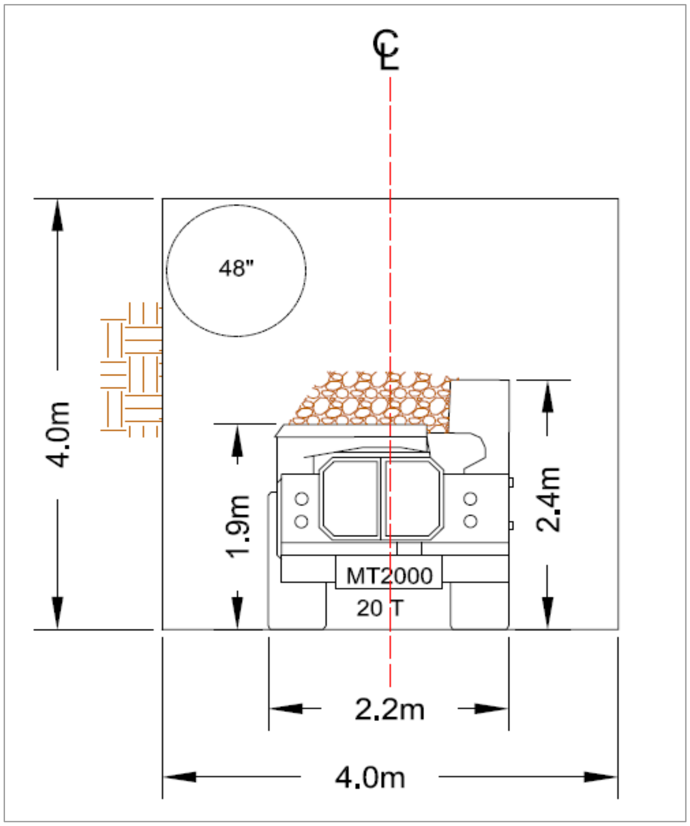


Figure 16.11 4.5 m by 4.5 m drift for D40 truck

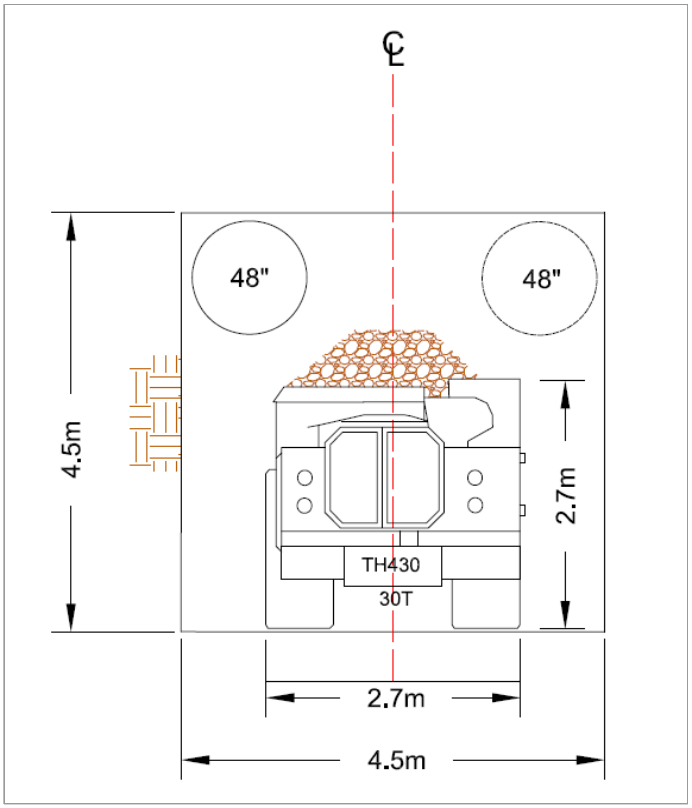
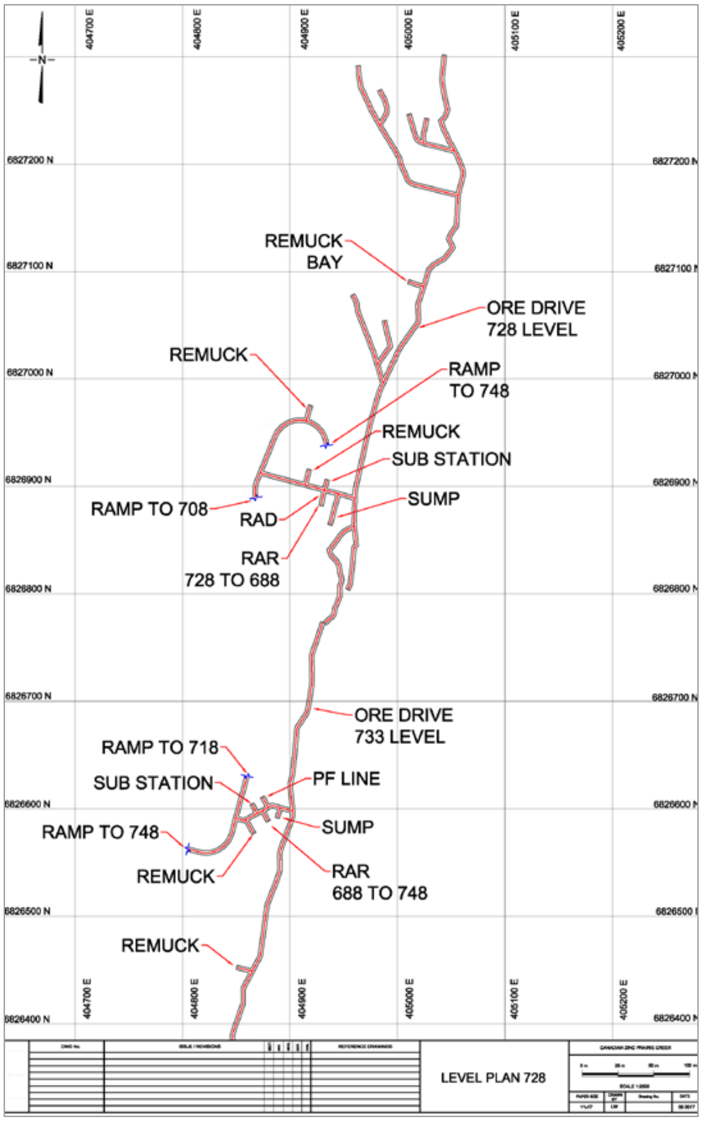


Figure 16.12 Representative sublevel arrangement in plan



*RAR – return air raise; RAD – return air drift; PF – paste fill.

16.4.2 Stope design

AMC used the Mineable Shape Optimizer (MSO) to produce conceptual stope shapes. Key parameters used in the MSO process for LHOS are summarized in Table 16.2. The conceptual stope shapes were refined as necessary and combined into stopes that fit the LHOS length and mine sequencing requirements. In addition, the stope shapes were assessed relative to development economics. A listing of key design parameters for each mining zone and method is shown in Table 16.2

The MQV mineralization widths generally vary between less than 0.1 m and up to 7.0 m. A minimum mining width of 2 m has been adopted, but with most of the MQV width considered for mine design in the range of 3.0 m to 7.0 m. In the STK zone, the stopes have been divided into 7.5 m wide panels after consideration of geotechnical aspects. The MQV and STK veins generally dip at 60 to 70°, but are near vertical in some areas. The SMS zone is largely tabular with an approximately 15 to 20° dip.

The strike length of the orebody varies by elevation, but is roughly 925 m on average above the 807 L and 650 m on average below.

Table 16.2 MSO parameters for LHOS stope shape optimization

Parameters	Field	Default	Units
Density	DENSITY	2.78	t/m^3
Optimization field	NSR	0	$/t
Cut-off grade	NSR	157*	$/t
Slice interval		0.5	m
Default dip		90	degrees
Strike azimuth		15	degrees
Sub-blocking		No	
Optimization length		3	m
Minimum mining width		2	m
Hangingwall dilution		0**	m
Footwall dilution		0**	m
Minimum hangingwall angle		55	degrees
Minimum footwall angle		55	degrees
Maximum strike variation		45	degrees
Maximum strike change		20	degrees
Stope maximum side-length ratio		2.25	*ratio*

*Average NSR cut-off for 20 m and 15 m stope heights.
**Planned dilution is within stope shapes. For unplanned dilution estimates see Table 16.3.

Table 16.3 Key design parameters by zone and mining method

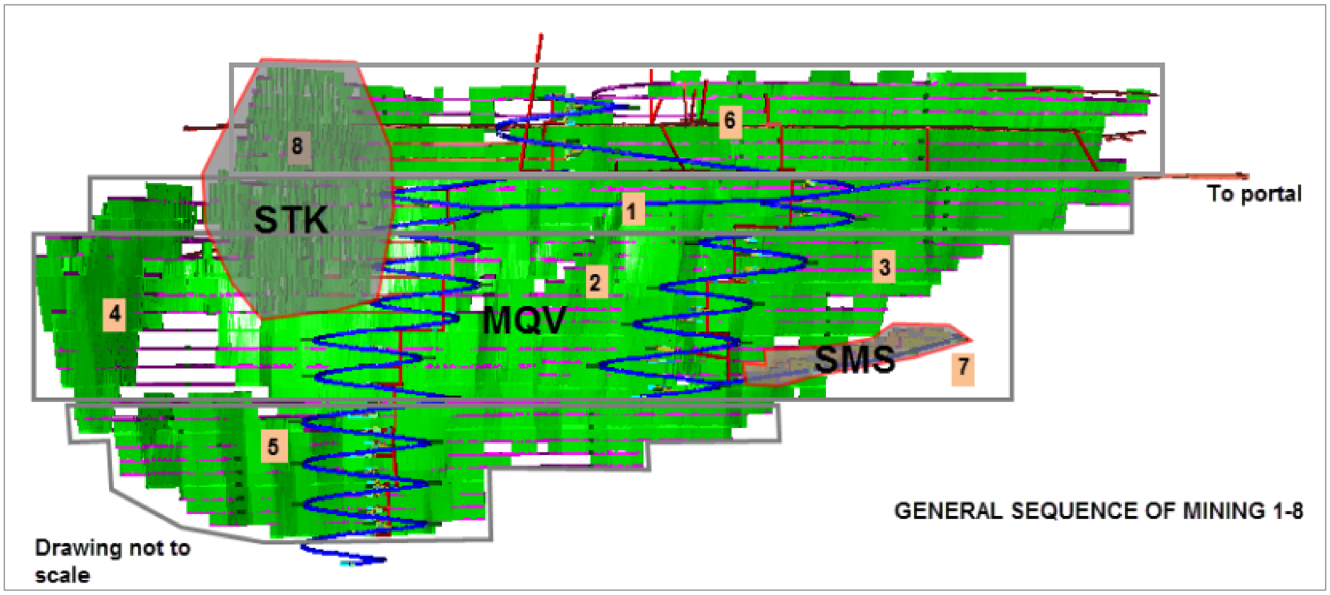
	Units	MQV LHOS 15	MQV LHOS 20	STK LHOS 15	STK LHOS 20	SMS DAF
Cut-off zinc equivalent	%	12	11	12	11	11
Minimum mining width	m	3	3	3	3	7.5
Mining height	m	15	20	15	20	5
Mining length	m	30	30	30	30	n/a
Minimum HW angle	º	55	55	55	55	90
Minimum FW angle	º	55	55	55	55	90
HW unplanned dilution	m	0.2	0.2	0.2	0.2	0.2
FW unplanned dilution	m	0.1	0.1	0.1	0.1	0.2
Floor unplanned dilution	m	0.1	0.1	0.1	0.1	0.1
LHOS endwall unplanned dilution	m	0.25	0.25	0.25	0.25	n/a
Unplanned dilution*	%	11	11	11	11	5
Mining recovery	%	95	95	95	95	98

* For LH stopes unplanned dilution % is average estimate for 60 m high block.

16.4.3 Zone and mining block definition

Figure 16.12 is a longitudinal view of the orebody looking east, showing the location of the MQV, STK, and SMS zones, and the split of the orebody into mining blocks. The general sequence of mining is also shown in Figure 16.12, although mining of the different zones will overlap significantly. Major goals in sequencing the blocks as shown are to access higher grade sulphide ore as early as practicable while minimizing development costs as much as possible.

Figure 16.13 Longitudinal view of zones and mining blocks looking east



16.4.4 Stope cycle and sequence

LHOS shapes were created on both 15 m (stope width < 3m) and 20 m (stope width > 3 m) sublevel intervals and grouped vertically in sets of three or four, to a maximum 60 m stoping height. The mining sequence in the MQV zone involves 30 m long panels being mined and filled, retreating towards the central access. The STK zone stopes will also be taken on retreat, with primary access in the later stages by drifting through backfill along the previously mined MQV.

From each side of the central access, ore drives up to about 250 m long for drilling and mucking will be developed to the strike extents of the ore block. Forced ventilation through ducting will provide fresh air from the ramp to the ore drive face and exhaust out of the ore drives into the return air system connected every two levels at the level access. To begin production in each LHOS, a slot raise will be excavated to provide a free face, followed by production ring drilling and blasting. Mucking will be from the lowest level of the stope set, with that level leading the excavation of the block as a whole, as shown in Figure 16.12 above. For the MQV and STK LHOS typical mining sequence, an entire mining block will be exhausted before moving to the one above; the exception will be where a designed cemented sill is located at the bottom of an upper block before the lower block is mined.

After a block of stopes is exhausted of ore and ancillary activities such as cavity measurement are completed, fill fences will be constructed on each sub-level to allow placement of paste fill, and waste rock as appropriate, into the stope. Table 16.4 shows the total aggregate LHOS stoping rate (drill, blast, muck, cavity monitoring, erect fill fences) per 60 m high x 30 m long panel, together with the backfill and curing rates.

Table 16.4 LHOS production / backfill rates for 60 m stoping block

Activities	Units	Number of lifts
		3 or 4
DBM Rate (drill*, blast, muck, CMS, fill fence)	t/d	700
Fill	m^3/d	1000
Cure	days	21

*Drilling off-cycle after first stope.

Longitudinal LHOS is a non-entry method requiring remote mucking, due to the scoop operator potentially being exposed to the open stope and / or uncontrolled sloughing of ore at the stope brow. As an added safety measure that is now standard industry practice, AMC advises the use of remote mucking stands for operators.

For drift and fill (DAF) mining in the SMS zone, panels will be mined in a primary-secondary sequence to the economic margins of the mineralization, with primaries being filled with paste fill before mining of the secondaries begins. Secondaries will be filled with paste fill although waste rock may also be used.

The DAF will be completed for each level before moving onto the next level in a bottom-up progression. Cuts of 7.5 m width and 5 m height will be utilized in the DAF.

16.5 Backfilling and waste management

16.5.1 Backfill system description

Paste fill will generally be the material used as backfill within the mine. A newly installed paste backfill plant will return 100% of the flotation tailings to underground. The planned Dense Media Separation (DMS) plant at the front end of the mill will remove a significant amount of waste rock from the mill feed, reducing the production of tailings. As the resource is a high-grade base metal deposit, the high concentrate mass pull will further reduce the amount of tailings produced. All the aforementioned factors contribute to allowing full disposal underground of the tailings produced at the mine as paste backfill, thus negating the need for a permanent surface tailings facility.

The paste plant will be located immediately south of the mill. Paste will be produced from dewatered tailings mixed with cement binder and make-up water to the target density. The ratio of binders will be varied to produce high-strength paste for sill pillars, normal-strength for retreating stopes and primaries in DAF, and low-strength for bulk fill and secondaries in DAF. Binder addition rates will typically average 3.5%, varying between 2% (where only low-strength fill is required) and 6% (sills only), dependent upon the required strength and slump (the paste density is typically measured by its slump, a term commonly used in the cement industry - the higher the slump of the paste, the lower its density). Laboratory-scale test work has been carried out to determine the paste recipes required.

16.5.2 Paste fill production and delivery

The tailings management system is designed so that the mill can operate continuously, whether the backfill system is operating or not. Processing of ore at 1,600 tpd will produce, on average, approximately 410 tpd of heavy media reject aggregate, approximately 380 tpd of mineral concentrates and approximately 810 tpd of flotation tailings. Through a combination of direct tailings use, stockpiling and reclaiming of tailings, the paste fill system will produce 55 m^3/hr of paste fill from tailings, cement binder and make-up water at a utilization rate of approximately 40%, to provide backfill for underground stopes.

The paste fill system has been designed to produce approximately 186,000 m^3 of paste fill per year, which is matched to the steady state yearly underground production void. Development waste rock will be predominantly transported to surface for disposal. If required for short-term filling purposes, development waste rock and, continuously available DMS reject material are additional sources of backfill. A schematic of the paste fill system is shown in Figure 16.14. An isometric view of the paste plant is shown in Figure 16.14.

At the completion of production in each longhole stope panel, structural shotcrete barricades will be built at the draw-point stope access and at each of the sub-level accesses above to retain the paste fill. Reticulation pipes will be extended into the highest level opening for the placement of fill.

For DAF in the SMS zone, waste rock barriers will be placed at the entrance to a mined-out panel with paste fill delivered to the stope through a fill pipe placed along the stope back.

Figure 16.14 Prairie Creek paste system schematic

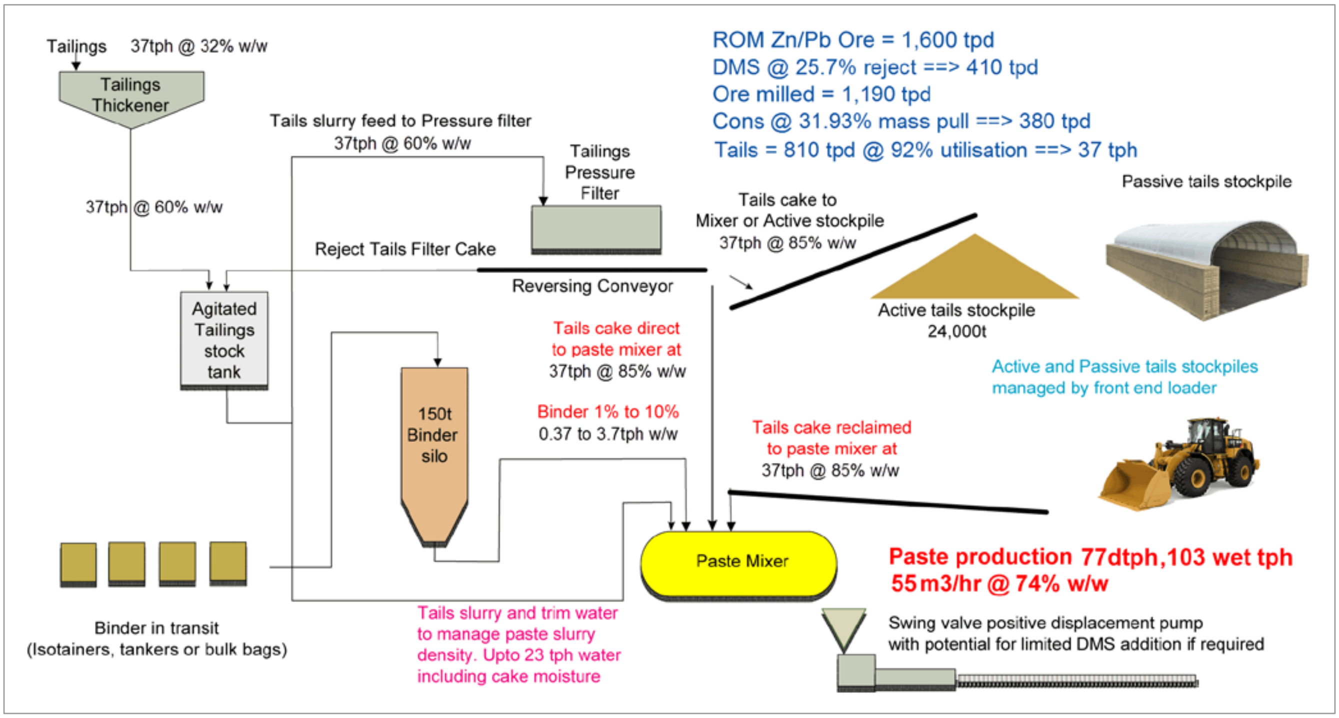
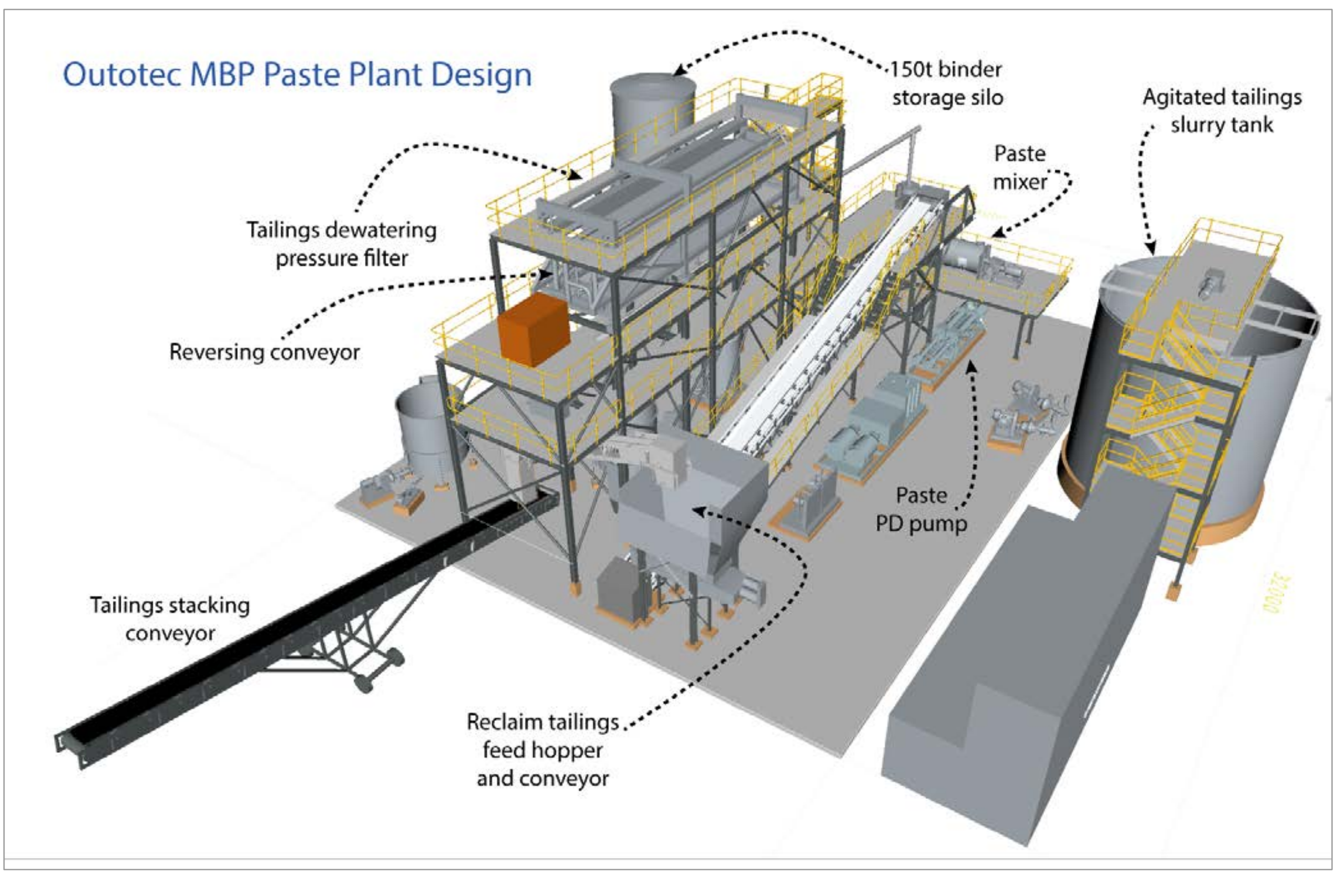


Figure 16.15 Paste Plant Isometric view



The pressure filters will operate in batch mode, dewatering the thickened tailings slurry to form a moist cake with moisture content between 10% and 15% at a rate of 37 tph. When backfill is not required, this cake will be conveyed to the active tailings stockpile. This conveyor will be reversible and any out-of-specification cake will be returned to the tailings filter stock tank. A front-end loader will manage the stockpile. The active filter cake stockpile will be in a building that can be heated in winter. Excess filter cake will be stored out-doors between the plant and the water storage pond; details of the storage arrangement will be finalized in the next project phase.

When backfill is required underground, the tailings cake from the pressure filters will be routed directly to the mixer feed hopper. At the same time, the loader will deliver tails from the active stockpile into the adjacent mixer feed hopper at a rate of 37 tph. This will result in a feed rate of 74 tph of tailings solids to the mixer.

The paste plant operator will select the required fill recipe, specifying density, cement dosing and delivery rate, and will start up the paste mixing plant. The tailings cake, cement and process water will be mixed in the continuous mixer to produce a cemented paste fill. At an average dosing rate of 3.5%, cement will be added at 2.7 tph with make-up water to produce 55 m^3/hr of cemented paste fill for delivery underground by a high pressure positive displacement pump.

The paste fill will be pumped underground along the 883L using 150 mm nominal bore high-pressure pipelines to a pair of near-vertical boreholes, approximately 900 m from the paste plant. From the top of the boreholes the paste will then be delivered through internal boreholes and pipelines to the stopes to be filled.

At each sub-level where paste is required, steel pipes will be installed from the borehole to close to the point of discharge. Because of lower paste pressure at the discharge end of the pipe, the final 50 to 100 m of pipeline can be HDPE to simplify handling and installation.

Stopes in the northern area will be supplied with fill by an additional borehole; this will enable quick sequential filling of both northern and southern stopes. A number of stopes above the 883L will require uphill pumping of paste through a single vertical borehole and / or combination of pipes up the decline. Filling the highest stopes will add 60 m of static head to the pump duty and will require a separate scuttling and flushing arrangement at the base of the piping servicing those stopes.

Filling will continue in each stope until the paste reaches the required elevation. Typically for LHOS, filling will stop about 0.5 m below the floor elevation of the top drive. The line will be flushed clear of paste and the paste plant will be prepared for the next fill run. Waste rock will then be pushed on to the top surface of the paste to complete filling and to provide a traction surface for mucking and access as required for the next stage of production.

16.5.3 Fill test work

Test work was carried out at McGill University (McGill) using metallurgical testing tailings. Material characterisation, rheology and strength testing was carried out for a range of binder types, dosing rates and curing times. The test program is described fully in a separate report; key summary results are shown in Table 16.5 and Table 16.6 below.

Table 16.5 Rheology summary

	Tailings (%)	RGUL binder	Solids concentration at 100 Pa yield stress	Solids concentration at 250 Pa yield stress	Solids concentration at 350 Pa yield stress Pa
No binder	100%	0%	78.5%	80.1%	80.5%
5% binder	95%	5%	77.0%	79.0%	80.0%

The test work report notes that settlement and some water bleed were observed in rheology samples. Additional test work is recommended during the next phase to finalise recipes and reticulation design.

Table 16.6 shows the summary strength results for the best performing binder trialled (Lafarge RGUL – Richmond, BC produced general usage cement with up to 15% ground limestone). All samples show a delayed onset of binder hydration (curing), particularly at lower dosing rates, followed by rapid increase in strength between 14 to 21 days. AMC considers this behaviour is probably caused by an initial retardation effect of zinc oxides in the tailings followed by acceleration once hydration has commenced. The required curing cycle time of 21 days falls within the acceleration phase of curing.

Table 16.6 Fill strength test work summary

Curing (days)	2% RGUL (kPa)	3% RGUL (kPa)	4% RGUL (kPa)	5% RGUL (kPa)	8% RGUL (kPa)	11% RGUL (kPa)
7	-	-	-	-	1,080	1,370
14	-	-	130	540	1,560	1,960
21	-	120	440	760	1,850	2,980
28	-	150	520	890	2,020	3,550

16.5.4 Fill design strengths and recipes

AMC has designed fill strengths based on the geometries of stopes and the dimensions of the subsequent fill exposures. The majority of fill exposures will be narrow vertical walls on retreating stopes, typically 60 m by up to 7.5 m wide. The lower half of each stope will have a fill strength of approximately 300 kPa and the upper half approximately 200 kPa. For a stope cycle time of 21 days for curing, 4.0% RGUL binder is projected to be required in the lower half and 3.0% in the upper half. The average value for economic purposes is 3.5%.

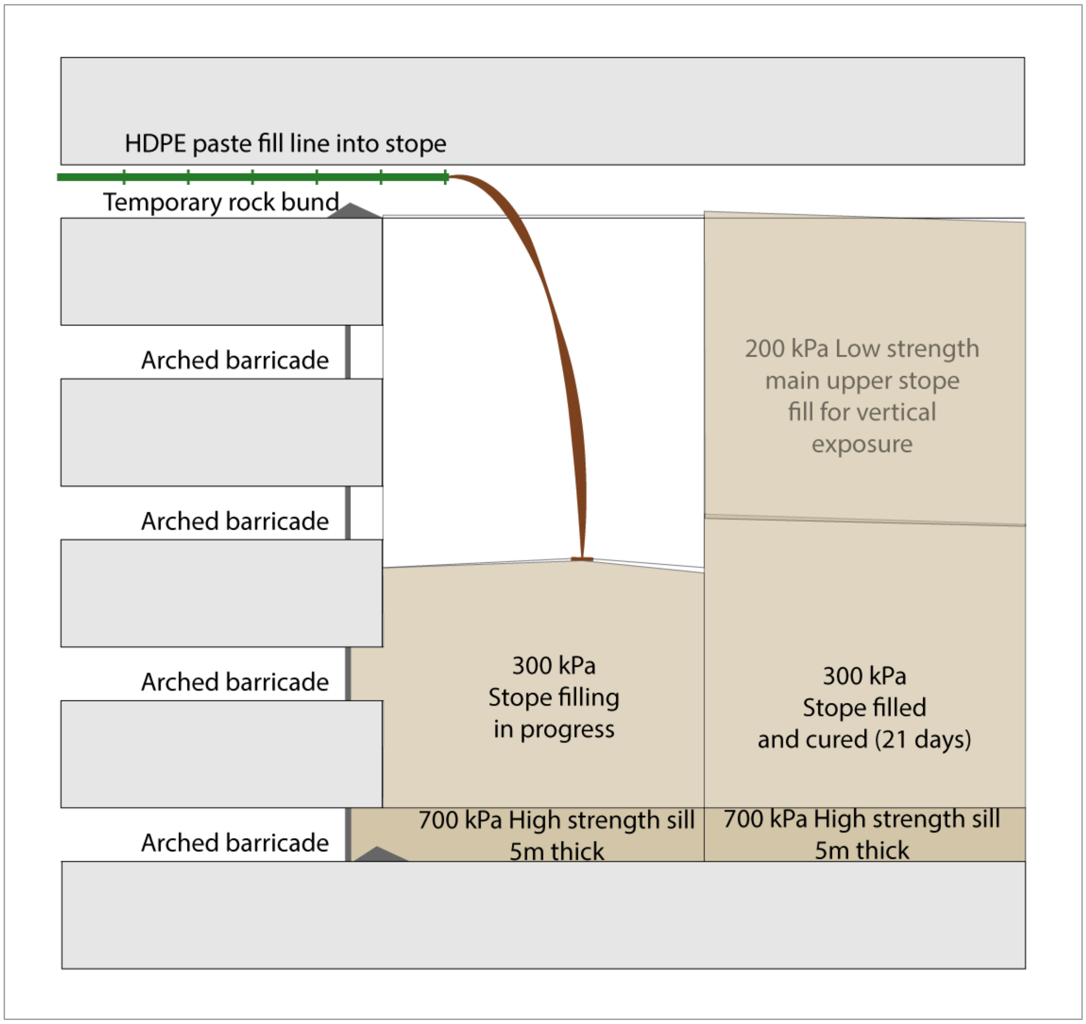
At the base of any production block under which a stope will be extracted in future, a higher strength fill sill is needed. 700 kPa strength is deemed to be required over 5 m thickness; at 28 days curing (the curing time will invariably be much longer in practice) 4.0% binder will be sufficient. Stope fill strength values are summarized in Table 16.7.

Table 16.7 Fill design strengths and recipes

Application	Design Strength (kPa)	Binder recipe at 21 Days	Binder recipe at 28 Days	Comments
Undercut sills to 7.5 m wide	700		4.0%	Usually much longer curing time available
Vertical stope faces, 7.5 m wide, lower half	300	4.0%	3.0%	Zoning the fill strength to optimize binder content
Vertical stope faces, 7.5 m wide, upper half	200	3.0%		Zoning the fill strength to optimize binder content
Long term average binder		3.5%		Use in economic model

Figure 16.16 illustrates filling activities for longhole stopes including barricade construction and fill pipe installation (sill pour shown at base of stope assumes stope to be mined immediately beneath at future date).

Figure 16.16 Stope filling showing strength requirements



16.5.5 Waste management

The backfill system design has been based on the requirement to place all flotation tailings underground by the end of the life of the mine. Averaged across the projected daily ore production rate of 1,600 tpd there is an estimated net shortfall of approximately 25 m^3/day of available paste backfill. This small volume will generally be made up from DMS reject, including that required for top dressing of paste fill in the longhole stopes. The heavy media rejects and remaining development waste rock (which could also be available for backfill if needed) will be stored permanently on surface in an engineered stockpile, located in a ravine west of Harrison Creek.

16.5.6 Tailings management

Under the Prairie Creek operating licence, storage of tailings in the water storage pond is limited to 50,000 t at any one time. The proposed backfill design is intended to minimize the need to use this storage facility and, instead, manage active and passive tailings material as part of the backfill processing system.

When the backfill system is not running, an active storage of tailings will be banked-up by stacking moist filtered tailings cake in a shed with heating capability. The volume of the tailings stored here will be approximately 16,000 m^3 (about 24,000 t of tailings solids). When the backfill plant is running, this stockpile will be drawn down to residual levels as the tailings are delivered underground as fill. The stockpile will be fed from an overhead conveyor and managed with a front-end loader.

If backfill is not placed for about 30 days at full production rate, excess tailings will be stacked adjacent to the water storage pond. When backfilling, the live and active storage areas will be managed so as to minimize out-door storage volumes.

In the event of extended downtime of the backfill system, tailings may be temporarily stockpiled in the water storage pond area, with its capacity being limited to 50,000 tonnes by the operating licence. At the limit, this will provide approximately 60 days of additional storage capacity at average steady state production rates.

One of the objectives of the mine plan, and of the backfill system design, is to achieve rapid filling of the stoped voids when they become available. The surface system will achieve this by stockpiling sufficient tailings to supplement a continuous pouring campaign for each individual stoping unit.

16.5.7 Development waste rock and heavy media rejects

The projected development waste, ore, dense media reject, and backfill schedules are detailed in Table 16.8. Development waste will be hauled to surface and disposed of in the surface waste stockpile. Approximately 1.1 Mt of development waste and 2.1 Mt of DMS reject (average 27% reject rate to ROM ore produced) are anticipated to be generated over the LOM. The DMS material will also be hauled to the waste stockpile, which will grow to about 3.2 Mt over the LOM.

Table 16.8 Ore, waste, and backfill material movements

Year	Total ore tonnes	Total waste rock tonnes produced	Total pastefill volume (m^3)	Waste stockpile on surface tonnes	Total DMS tonnes	Total DMS stockpile tonnes
Y-01	14,607	94,399		94,399		
Y01	230,789	208,044	16,666	302,443	41,430	41,430
Y02	552,666	248,028	164,530	550,471	161,818	203,248
Y03	584,466	196,686	171,557	747,157	171,977	375,225
Y04	587,360	56,677	144,540	803,834	173,726	548,951
Y05	584,217	28,601	166,537	832,436	172,004	720,955
Y06	589,218	11,040	183,879	843,476	180,467	901,422
Y07	575,511	82,522	186,077	925,998	174,996	1,076,418
Y08	596,248	34,053	191,639	960,051	173,150	1,249,569
Y09	578,926	48,161	176,436	1,008,211	163,971	1,413,540
Y10	591,111	36,880	207,033	1,045,091	132,927	1,546,467
Y11	588,204	19,755	233,291	1,064,846	133,884	1,680,351
Y12	570,119	24,046	215,936	1,088,892	120,509	1,800,860
Y13	599,133	17,090	190,661	1,105,982	85,424	1,886,284
Y14	545,808	2,882	214,207	1,108,864	103,076	1,989,360
Y15	273,936		131,874	1,108,864	79,848	2,069,207
Y16	9,144		1,301	1,108,864	1,829	2,071,036
Total	**8,071,463**	**1,108,864**	**2,596,165**	**1,108,864**	**2,071,036**	**2,071,036**

16.6 Ventilation

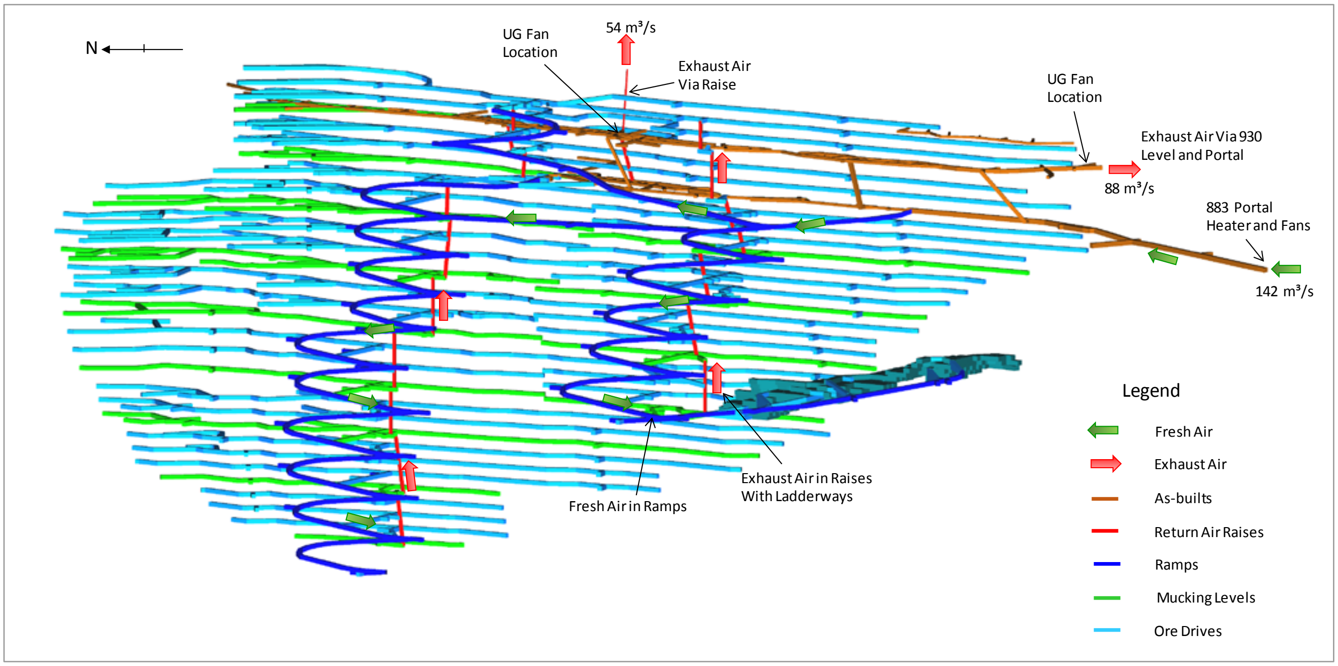
The function of the ventilation system is to dilute/remove airborne dust, diesel emissions, blasting smoke and other contaminants and to maintain temperatures at levels appropriate for safe production throughout the life of the mine. The ventilation system for Prairie Creek was designed in accordance with the “NWT and Nunavut Mine Health and Safety Regulations - 2016.”

The mine will be ventilated by a “pull” or exhausting type ventilation system. That is, the primary mine ventilation fans (with Variable Frequency Drives) will be located in the primary exhaust airways of the mine and will develop sufficient pressure to ensure that all work places are supplied with the required fresh air from the intake portal (883 L), and that contaminants are removed to the exhaust air system and ultimately to the surface via the 930 L portal and an exhaust raise.

In consideration of the equipment list required to maintain a 1600 tpd production rate, and also through a separate ventilation analysis of concurrent mine activities, the planned mine airflow is 142 m³/s. This includes accounting for the diesel equipment fleet, future battery powered scoops, infrastructure and personnel, such that legislated requirements are met. A 15% contingency is applied to the calculated air volume to account for leakage and system inefficiencies.

The overall layout of the ventilation system is shown in Figure 16.17.

Figure 16.17 Ventilation layout



16.6.1 Lead exposure considerations

Regulations and current practices with respect to lead contamination and worker exposure were reviewed and mitigation options for both underground and on surface were investigated. Exposure of underground workers to lead and the impact upon the health of the underground workers has been identified at other mine operations. Monitoring and establishment of worker exposure limits to lead is regulated at a provincial level. The Northwest Territories and Nunavut do not outline removal or action levels. However, the Occupational Health and Safety Regulations[1] outline the requirement for employers to develop work procedures and processes to protect workers from chemical and biological substances, of which lead is identified as a 'Designated Chemical and Biological Substance' requiring an employer to:

- Provide adequate engineering controls to prevent, to the extent that is reasonably possible, the release of the substance into the work site; and
- Take other measures and provide personal protective equipment that meets the requirements of Part 7 [Personal Protective Equipment] to prevent, to the extent that is reasonably possible, exposure of workers to the substance.

[1] NWT and Nunavut Occupational Health & Safety Regulations (2016).

Further, the Workers' Safety and Compensation Commission has recently issued a Code of Practice[2] for working with lead.

As a response to the above, it is recommended that the "Working with Lead Guideline Code of Practice" be reviewed and a lead exposure control plan be developed and followed.

16.6.2 Fresh air circuit

Level distribution is designed so that fresh air will be sourced from the portal at 883L and then distributed through the ramps in the north and south of the orebody. The ramps will thus be in fresh air, a consideration in the event that mine rescue operations become necessary. Fresh air will be delivered along each ore drive by auxiliary fan and duct installations. Figure 16.14 above shows positioning of auxiliary ducting in development drives. Intake air will be heated during the winter by a duplex propane- or LNG-fired heater at the 883 portal to prevent ice build-up in the adit.

16.6.3 Return air circuit

Contaminated air from development and production activities will exhaust to the 3 m x 3 m Alimak-driven return air raises, which break through the level accesses at twenty or thirty metre intervals and ultimately exhaust to the surface. The 930 L will serve as the primary exhaust route with exhaust fans located at the ventilation transfer drift adjacent to the 930 L portal and at the base of the existing raise from 930 L to surface.

16.6.4 Second egress

The exhaust air raises will be fitted with ladderways to serve as the second means of egress through to the 930 L, where egress will be through the 930 L portal. The existing ladderway in the return air raise from 930 L to surface will be removed and the raise used for ventilation only. Further details can be found in the emergency preparedness Section 16.10.

16.6.5 Ventilation power

A natural ventilation pressure exists in the mine, which is upcast in winter and weak or downcast in summer. The power requirement for the primary fans to enable a maximum airflow of 142 m^3/s in summer is 230 kW in total. This is achieved by the two main fans each having a 200 hp (150 kW) motor, pulling the exhaust air through the return air raise and 930 level exhaust to the surface. During the winter season, Natural Ventilation Pressure (NVP) will assist the air movement such that the power consumed will be reduced.

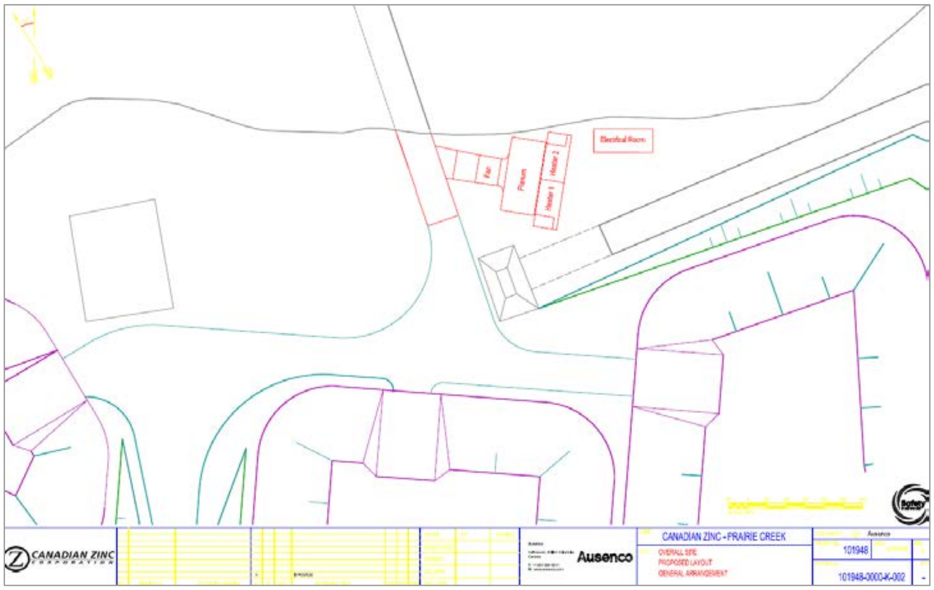
During the development of the ramps, 150 hp (112 kW) auxiliary fans will provide fresh airflow through twin 48" (1219 mm) ducts. For level distribution of auxiliary air, development and subsequent activities will require 100 hp (75 kW) auxiliary fans with 48" (1219 mm) ducting to provide fresh airflow, noting that smaller battery-operated LHDs are projected to be ultimately utilized in these drifts.

16.6.6 Mine air heating

The source of fresh air is exclusively the 883 L portal. Owing to the below-freezing air temperature during the winter months, this air must be heated before being introduced into the side of the portal through a fan / heater arrangement. A direct-fired propane or LNG heating system will be sited at the 883 L portal to heat the ambient air to a temperature of 1° C to ensure that access ways do not ice up in winter conditions, and to prevent service water pipes from freezing. Considering that the portal will be the sole access point for truck haulage and all other mobile equipment, the intent is for the portal fans to deliver slightly more air than the exhaust fan capacity. The objective is to ensure that the portal structure outcasts this excess of air to the atmosphere, therefore avoiding the need for an airlock ventilation door arrangement. It is planned that an extra 10 m^3/s should be introduced for a total of 152 m^3/s being delivered by the portal fans. A schematic of the portal fan set-up is shown in Figure 16.14

[2] Workers' Safety and Compensation Commission. (2017) Working with Lead Guideline.

Figure 16.18 Portal area schematic showing vent fan and heater set-up



16.7 Drill and blast

16.7.1 Explosives selection

The primary options for explosives in production (LHOS and DAF) and development blasting are: Emulsions (bulk and packaged), and Blasting Agents (bulk and packaged) – ANFO.

Bulk emulsion can be used in development and production blasting. It is very safe to handle, can be used in both up- and down-holes, is highly water resistant and produces low nitrate levels in mine water. In bulk form it is less costly and can be left in blastholes for up to four weeks before detonation. Emulsions typically have a shelf life of one year and the explosive must be warmed to > 0º C before use or the product may misfire due to reduced sensitivity. Extremely cold temperatures are expected at the Prairie Creek mine and therefore any emulsion product brought to site must be transferred underground and allowed to warm up to > 0º C before use.

In early mine development using a contractor, implementation of bulk emulsion infrastructure and bulk emulsion loading equipment may not be cost effective or practical. Therefore, packaged emulsion explosives should be considered for both early development and stoping.

Planned Prairie Creek dewatering offers the potential for ANFO use in production blasting. It is generally the cheapest explosive option, but should only be used in dry ground, avoiding possible high nitrate levels in mine water. Also, use in areas with very high concentrations of sulphide ore should be done in close consultation with the explosives supplier as it can react exothermically with sulphides, which, potentially, can lead to spontaneous detonation.

The production blast design used in this study assumed that ANFO would be purchased in standard 25 kg poly bags and pneumatically loaded. If required for wall control purposes, blastholes could be gravity loaded to achieve a reduction in energy (gravity-loaded ANFO has 19% less energy, on a volume basis, than pneumatically-loaded ANFO). Future consideration may be given to using ANFO in the larger tote bags as ANFO explosive is less expensive when ordered in bulk.

For this Feasibility Study, AMC has assumed that packaged emulsion explosives will be the primary product used for lateral development blasting and bulk ANFO explosive will be used for production blasting at the Prairie Creek mine.

16.7.1.1 Boosters, detonators, and stemming

It is recommended that boosters and detonators be used in the production blastholes to ensure reliable and effective initiation of the pneumatically-loaded ANFO explosive.

Electric detonators have been assumed for development and stope blasting, but AMC recommends that future consideration be given to the use of electronic detonators, which are higher cost but offer greater safety, programmability of blasting and greater blast control, and the opportunity for centralized blasting.

All downhole slot raise rounds will be stemmed before blasting to maintain the integrity of the unblasted portion of the blasthole.

16.7.2 Longitudinal downhole stope design

Longhole open stoping (LHOS) and drift and fill (DAF) are the primary Prairie Creek mining methods. DAF mining uses standard development practices, described in Section 16.2.2. The following parameters, assumptions, and constraints have been used in the preliminary LHOS drill and blast design:

- Applicable explosives products have been identified (e.g. ANFO, detonating cord, boosters, electric and non-electric detonators).
- Downhole stope sizes are:
  - 15 m or 20 m high (floor to floor)
  - 4 m wide typically (varies with vein width)
  - 30 m long
- Stopes are mined as primary / secondary and will be filled with cemented pastefill (no rock).
- Top and bottom sills are mucking drifts and are 4.5 m wide (typically) x 4.5 m high.
- Drilling drifts are 4.0 m wide x 4.0 m high.
- Blasthole diameter is 76 mm.
- A Boart Longyear Stopemaster drill or equivalent, suitable for accurate drilling of 76 mm production blastholes and for reaming to 114.3 mm for relief (unloaded) holes in slot raises, is assumed to be used.
- Holes are assumed to be drilled to maximize accuracy, minimize loading and blasting problems, minimize dilution, and optimize fragmentation.
- No advice on standards / procedures and / or safety issues given as yet, although these issues must be addressed before stoping begins.

16.7.3 Stope layout and design

Figure 16.19 shows a longitudinal section of a longhole open stope.

Figure 16.19 Longitudinal section of a longhole open stope showing four lifts

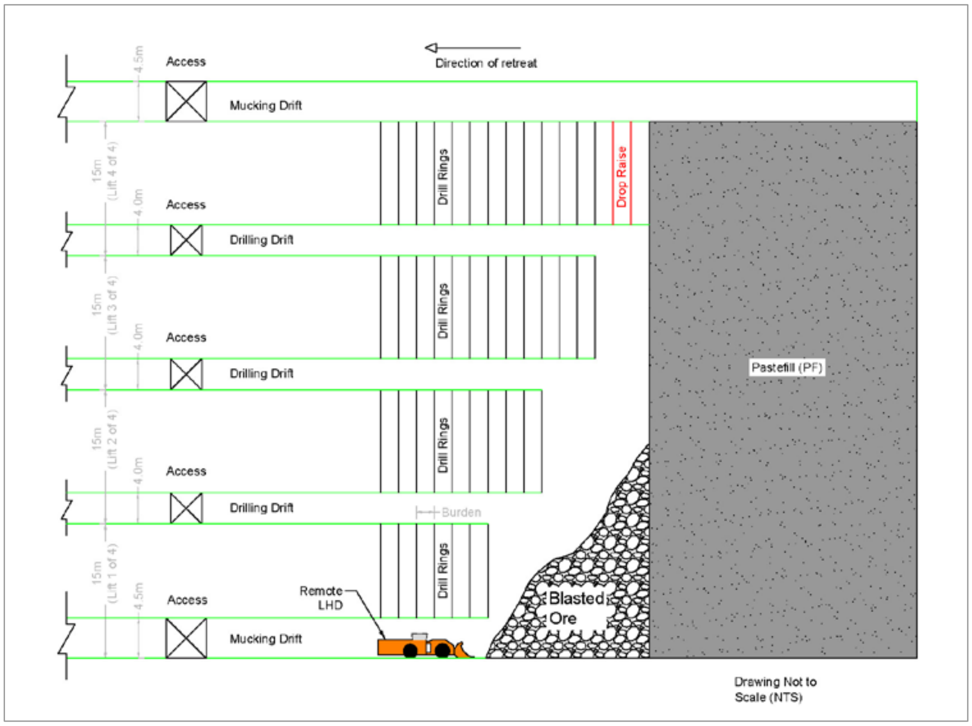


Figure 16.20, Figure 16.21, Figure 16.22, and Figure 16.23 illustrate a typical drilling plan and sections based on 76 mm diameter downholes with a 2.2 m ring burden and maximum 2.0 m toe spacing. Production stopes have an average powder factor of 0.49 kg/tonne. The choice of 76 mm diameter is based on effective explosives energy distribution, drilling deviation control, and ease of cleaning blastholes between blasts during development of the slot raise.

Figure 16.20 Longhole stope cross-section showing a slot raise ring

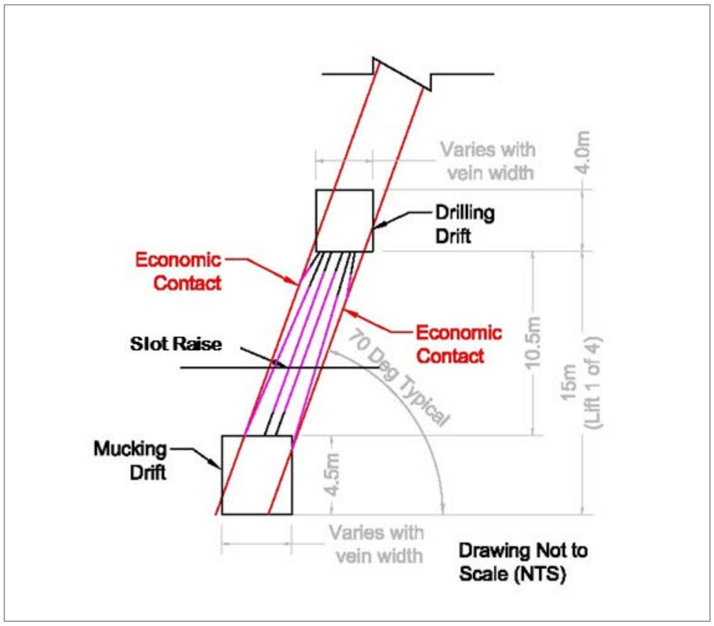


Figure 16.21 Longhole stope cross-section showing a production ring

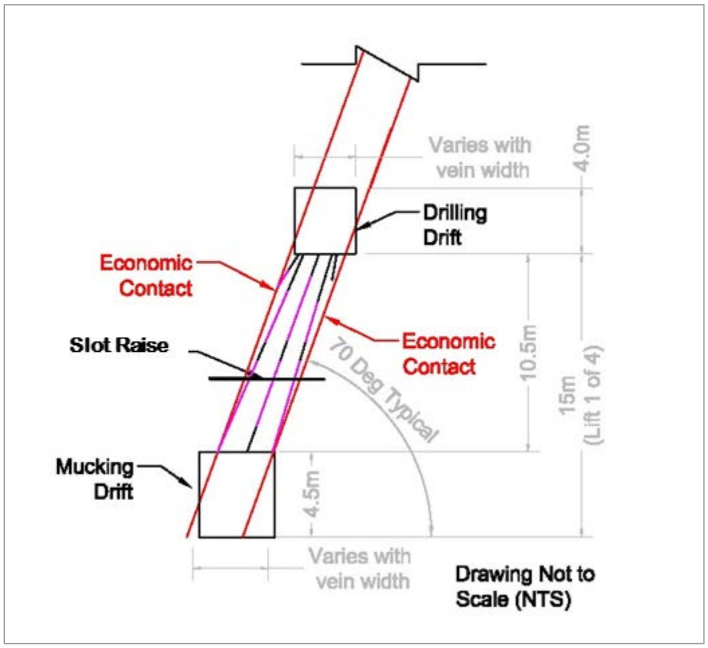


Figure 16.22 Longhole open stope (LHOS) - plan view

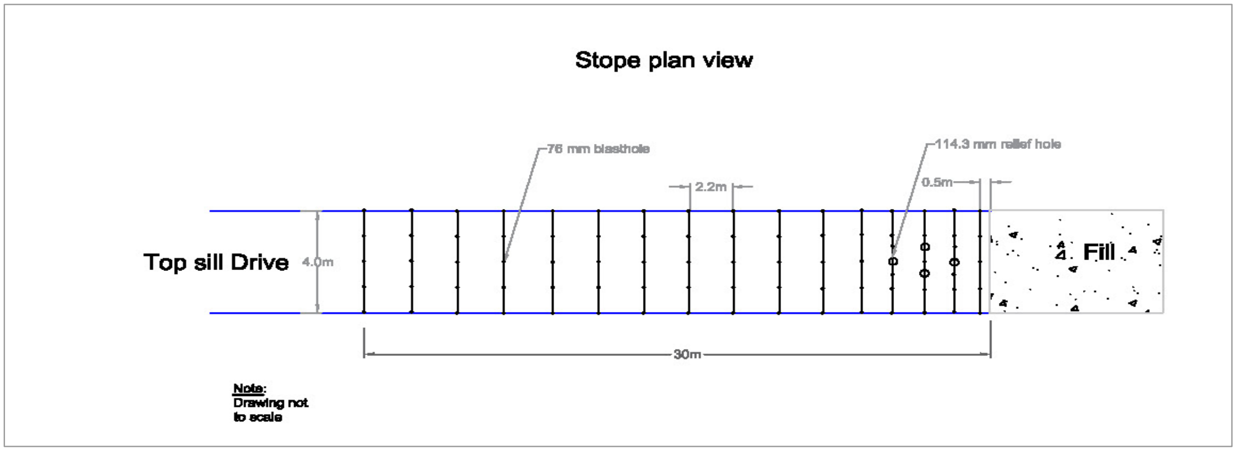


Figure 16.23 Longhole open stope (LHOS) - long section showing four blasts per lift

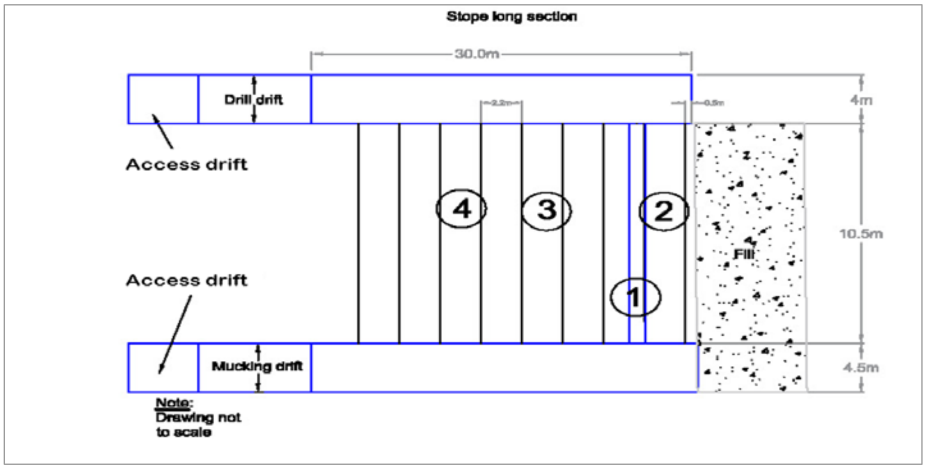


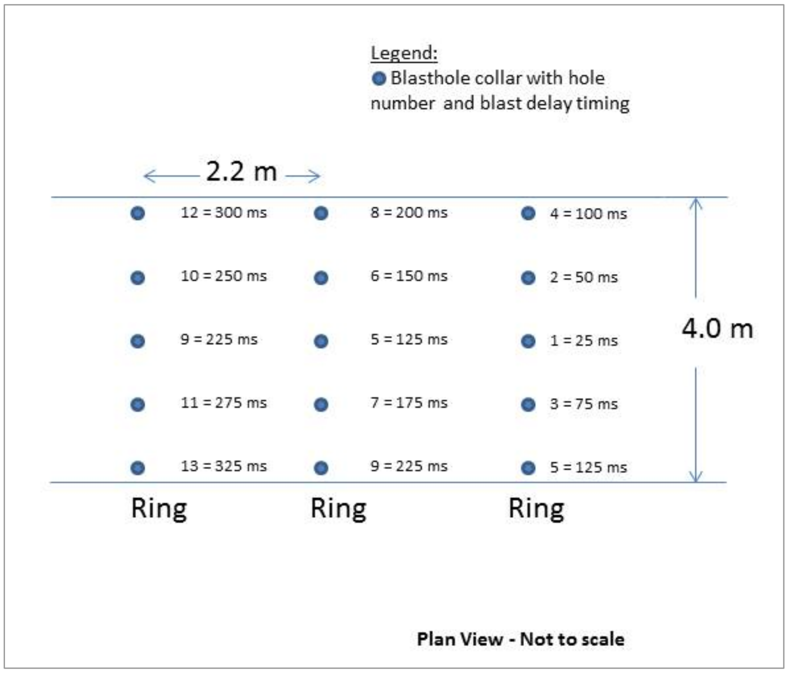
Table 16.9 Typical blast sizes in a stope lift

	Volume (cubic metres)*	Tonnes*
Blast #1 (slot raise)	35	113
Blast #2 (slot raise)	143	459
Blast #3 (production rings)	527	1,687
Blast #4 (production rings)	554	1,774
Total	**1,260**	**4,032**

* Totals are subject to rounding.

Blasts #3 and #4 are the main production blasts - see Figure 16.23. Figure 16.24 shows the delay timing for the blastholes in the first three rings of blast #3.

Figure 16.24 Plan view of select production rings and blast timing



16.7.4 Lateral development blast design

Assumption and constraints for development drill and blast design parameters include:

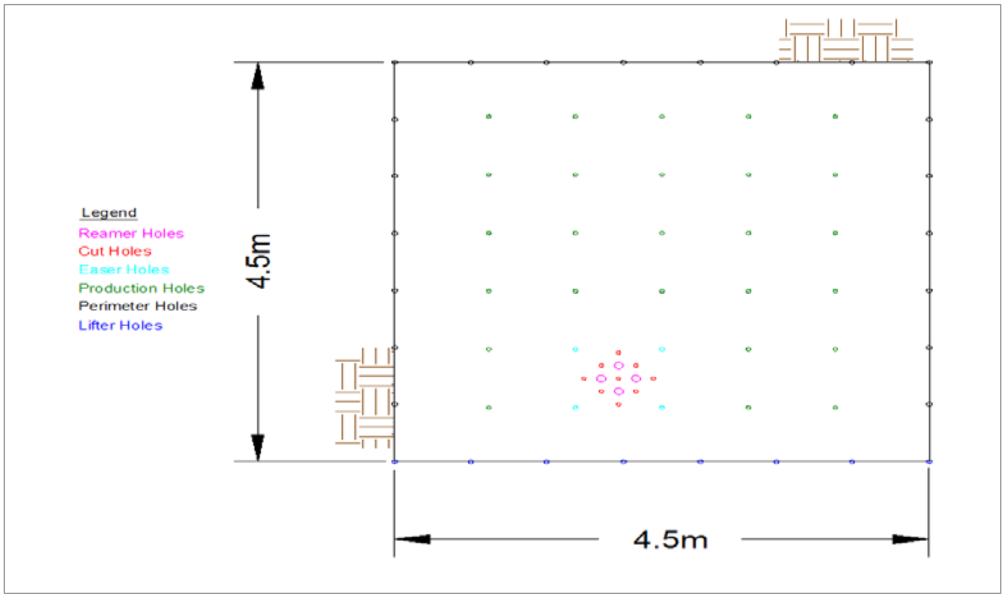
- Choice of packaged emulsion explosive and detonators.
- Typical development round dimensions of 4.5 m high and 4.5 m wide with a flat back, but ranging from 4.0 m x 4.0 m (ore drives above the bottom panel) to 4.6 m x 4.6 m for main ramps.
- Drills capable of reaming drilling holes to a diameter up to 102 mm.
- All holes are assumed to be drilled to maximize drilling accuracy, minimize hole-loading and blasting problems, minimize dilution, and optimize fragmentation.
- Atlas Copco Boomer S2 twin-boom jumbo or equivalent.

16.7.4.1 Lateral development – drill layout

Figure 16.26 shows the drill layout for a typical lateral stope access or mucking drive development round measuring 4.5 m (H) x 4.5 m (W) x 4.3 m deep. The drill holes for this design include:

- 4 reamer holes (102 mm)
- 9 cut holes (45 mm)
- 30 body holes (45 mm)
- 8 lifter holes (45 mm)
- 20 perimeter holes (45 mm)

Figure 16.25 Typical lateral development – drill layout



The overall powder factor using packaged emulsion explosives for lateral development design is 1.02 kg/t.

16.7.4.2 Blast initiation

Blasting during the contractor phase of operation with most activities relatively close to surface is anticipated to be on an as-required basis, with initiation from surface. In later phases of operation, AMC recommends that a dedicated central blasting system from surface be considered. The design and use of a central blasting system would necessarily be in-line with all pertinent NWT regulations.

16.7.5 Explosives management and logistics

16.7.5.1 Explosives consumption

The Prairie Creek mine is projected to consume approximately 18 tonnes of production explosives (ANFO) per month at peak production consumption in Year 10 and 37 tonnes of development explosives (packaged emulsion) per month at peak development consumption in Year 3. Blast design powder factors for production blasting and development blasting are shown in Table 16.10. Drift and fill stopes will use development blasting techniques and their powder factor will be the same as for development blasting.

Table 16.10 Blast design powder factors for production blasting and development

Description	Powder factor (kg/t)
Production blasting (including slot raises)	0.49
Development blasting	1.02

16.7.5.2 Underground transportation and loading

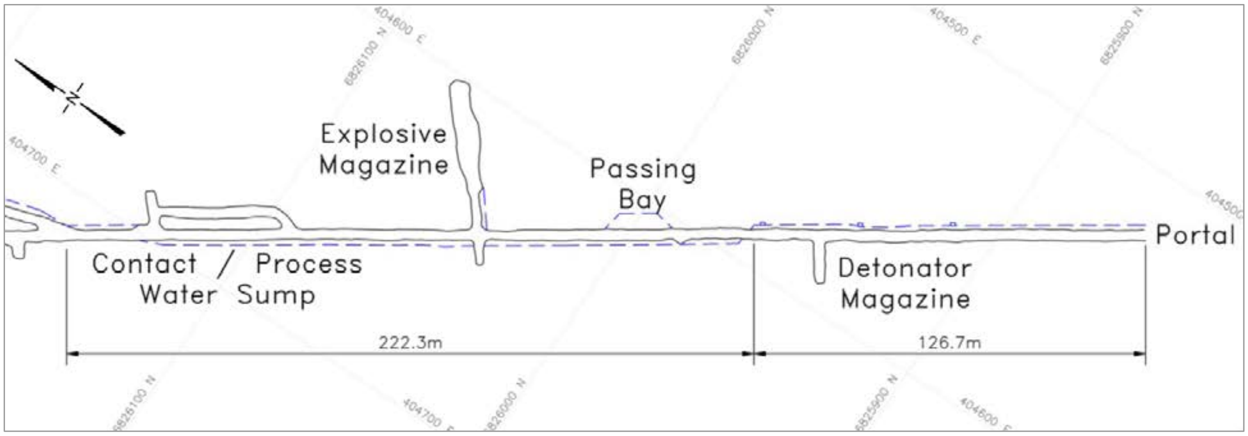
Equipment recommended for explosives transportation and loading include the MineCat carrier (or equivalent) with regular cab and rear enclosure, which can be used for distributing explosives underground to both stopes and development headings.

The MineCat MC100F carrier is well suited to loading ANFO in small drift mines. The 2000 lb. ANFO holding capacity means more efficient loading and less downtime to replenish the ANFO tanks. It can also be retro-fitted with an emulsion tank and pump and front-end forks for moving pallets of explosives.

16.7.5.3 Explosives delivery and storage

Packaged emulsion explosives and ANFO blasting agents will be delivered to the mine portal by the explosives supplier. Explosives will be received by mine personnel and promptly transported to the underground explosives magazine. Figure 16.26 shows the location of the main magazine in the 883 L access drift.

Figure 16.26 Magazine location



Blasting accessories (e.g., detonators, boosters, detonating cord) will also be delivered to the portal by the explosives supplier, usually in shipments that are separate from the delivery of packaged explosives. Blasting accessories will be received by mine personnel and promptly transported to the underground detonator storage location, sited approximately 110 m from the main magazine and towards the 883 L portal.

Packaged explosives and blasting accessories are usually transported in cardboard cases, or plastic bags in the case of ANFO, that have been shrink-wrapped and placed on wooden pallets. Off-loading deliveries at the portal will be handled by forklift, tele-handler or an LHD fitted with forks.

If required, the mine's local explosives suppliers can provide on-site representatives to assist with blast design, explosives inventory management and explosives loading operations. Typically, a supply contract is negotiated with the supplier and the product and service costs, roles and responsibilities, and terms and conditions will be contained in the supply contract.

16.8 Development and production schedule

16.8.1 Production rate

Stope sequence scheduling indicates that 700 tpd can be produced from an active stope. Therefore, to achieve 1600 tpd, three stopes have to be in production. Incorporating filling and stope development, the aggregate production rate for a single block of 60 m high longhole stopes is estimated at 284 tpd. Steady state ore production of 1,600 tpd will therefore be achieved by having a minimum of six active stoping areas. AMC's mine scheduling has demonstrated that 1,600 tpd is readily sustainable at steady state.

16.8.2 Pre-stoping development

During the period prior to the commencement of stoping, the focus will be on driving ramps and accesses towards the first mining blocks and developing the ore drives to enable extraction of the initial stopes. This timeframe will also include completion and commissioning of mill and paste plant infrastructure so that shortly after that commissioning, open stopes will be available in which to pour paste fill. The development is also planned so that ramp-up to full mine production is expedited. It is projected that a waste development crew will achieve 120 m/month of lateral advance with a single face, and an ore development crew will achieve 160 m/month with two active faces.

16.8.3 Sustaining development

To sustain steady state production with an optimum grade profile, access to targeted mining areas is made priority. The total project development is 52 km, of which approximately 65% will be in ore development headings. This development will be achieved with two development crews.

The projected LOM development schedule is shown in Table 16.11.

Table 16.11 Projected development schedule over the LOM

		Total	Y-02	Y-01	Y01	Y02	Y03	Y04	Y05	Y06	Y07	Y08	Y09	Y10	Y11	Y12	Y13	Y14
Portal Slashing Metres	km	**0.54**		0.54														
Oredrive Slashing Metres	km	**1.01**		0.18	0.23						0.12	0.47						
Total Slashing	km	**1.54**		0.72	0.23						0.12	0.47						
Access Metres	km	**4.07**		0.16	0.77	0.91	1.04	0.12	0.09	0.01	0.13		0.23	0.27	0.04	0.11	0.12	0.07
Through-Fill Drive Large Metres	km	**0.35**												0.04	0.12	0.20		
Through-Fill Drive Small Metres	km	**1.08**											0.35	0.22	0.23	0.25	0.03	
Ore Drifting Large Metres	km	**7.92**		0.15	0.93	1.06	1.30	1.13	0.61	1.02	0.40	0.56	0.01	0.29	0.23	0.03	0.20	
Ore Drifting Small Metres	km	**21.24**		-	2.07	1.69	1.93	2.42	2.71	2.15	2.30	2.12	2.46	0.59	0.37	0.01	0.43	
Pastefill Drive Metres	km	**0.21**		0.01	0.06	0.06	0.05	0.02			0.01							
Return Air Drive Metres	km	**1.20**		0.07	0.51	0.34	0.13	0.07			0.02	0.03	0.02					
Remuck Metres	km	**2.17**		0.07	0.39	0.49	0.38	0.13	0.09	0.09	0.16	0.10	0.16	0.07	0.01			
Ramp Metres	km	**6.57**		0.80	1.69	2.35	1.21				0.17		0.21	0.14				
Sump Metres	km	**0.55**		0.02	0.12	0.12	0.24	0.02	0.01		0.02							
Electrical Substation Metres	km	**0.47**		0.02	0.11	0.08	0.19	0.02	0.01		0.02		0.01	0.01				
Waste Drifting Large Metres 4.5x4.5	km	**1.09**		0.01	0.06	0.03	0.15	0.03	0.21		0.32	0.16	0.03	0.04	0.01		0.04	
Waste Drifting Small Metres 4.0x4.0	km	**3.54**			0.02	0.12	0.26	0.69	0.28	0.04	0.83	0.77	0.25	0.12	0.08		0.08	
Total Dev Lateral Metres	km	**52.01**		2.04	6.96	7.26	6.89	4.65	4.00	3.31	4.52	4.21	3.74	1.78	1.09	0.59	0.89	0.07
Total Dev Vertical Metres	km	**0.80**		0.03	0.21	0.28	0.14	0.08			0.03	0.03						

16.8.4 Life of Mine production schedule

Steady-state production of 1,600 tpd in the FS mine plan is projected to be achieved part-way through 2020 and sustained through to Project Year 15, scheduled as 2032, with a ramp-down to mining completion in 2034. An ore stockpile of about 100,000 t is generated in the first year of ore development and prior to the mill becoming available. Table 16.12 shows the projected life of mine production schedule and metal grades.

Table 16.12 LOM production tonnes and grade

LOM ore production schedule					
Project year	**Tonnes**	**ZnEq**	**%Zn**	**%Pb**	**g/t Ag**
Y-01	14,607	16.06	6.46	5.29	86.82
Y01	230,789	25.66	9.54	8.93	144.02
Y02	552,666	28.55	11.24	9.48	158.81
Y03	584,466	24.68	9.1	8.66	137.68
Y04	587,360	24.41	7.87	9.37	138.9
Y05	584,217	26.7	8.92	9.92	155.18
Y06	589,218	24.72	6.95	10.19	143.38
Y07	575,511	23.44	7.18	9.35	130.21
Y08	596,248	23.42	8.63	8.27	128.67
Y09	578,926	25.35	7.99	9.9	142.74
Y10	591,111	22.53	8.01	8.33	117.33
Y11	588,204	19.7	9.28	5.7	96.11
Y12	570,119	23.91	10	7.66	126.09
Y13	599,133	19.58	9.6	5.71	81.06
Y14	545,808	15.25	7.4	4.26	73.94
Y15	273,936	16.3	8.41	4.18	78.41
Y16	9,144	15.38	7.9	3.84	79.87
Total	**8,071,463**	**23.07**	**8.64**	**8.1**	**124.22**

Figure 16.27 shows the LOM production profile for each zone and for the Mine as a whole.

Figure 16.27 LOM production profile

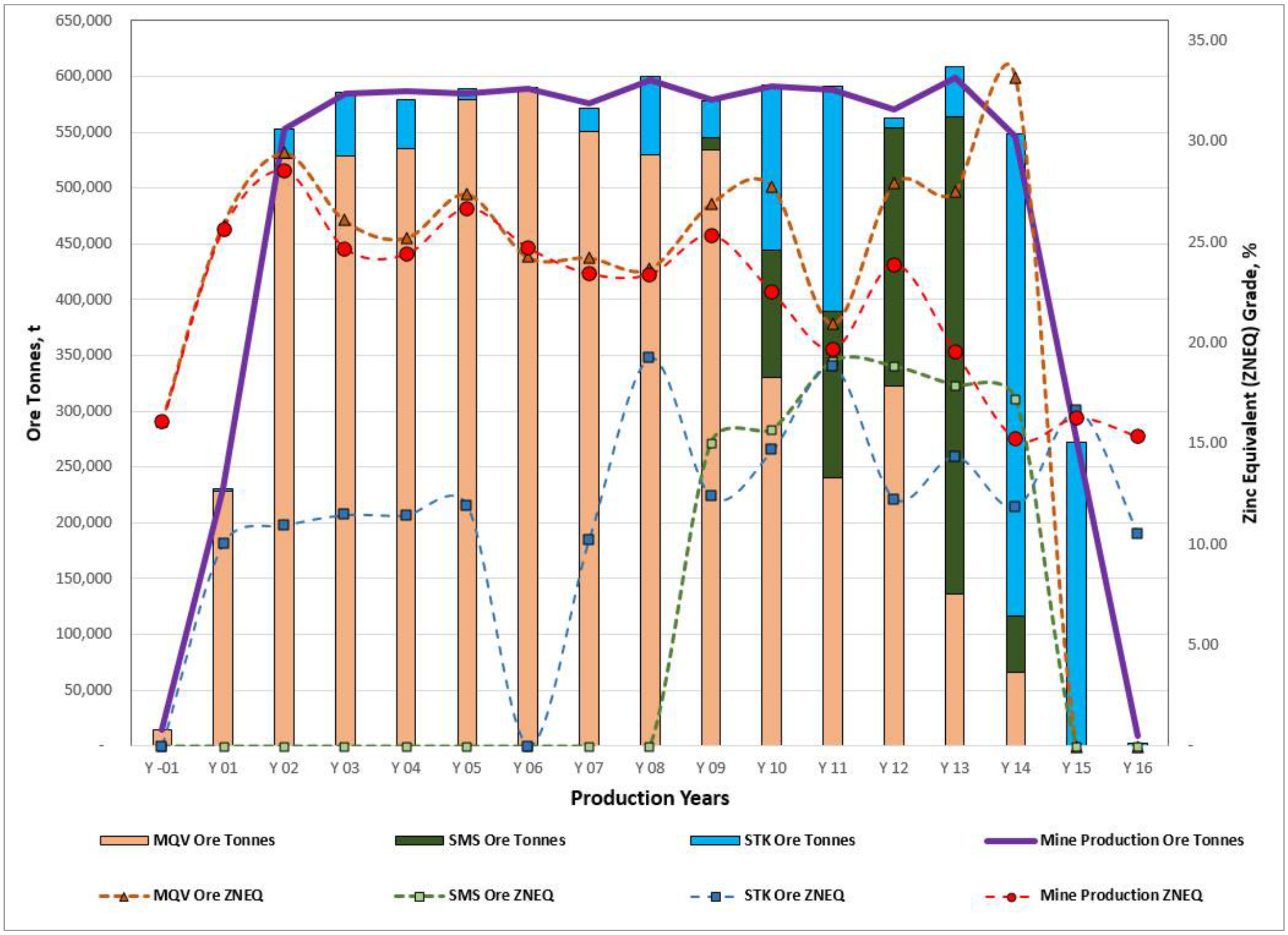
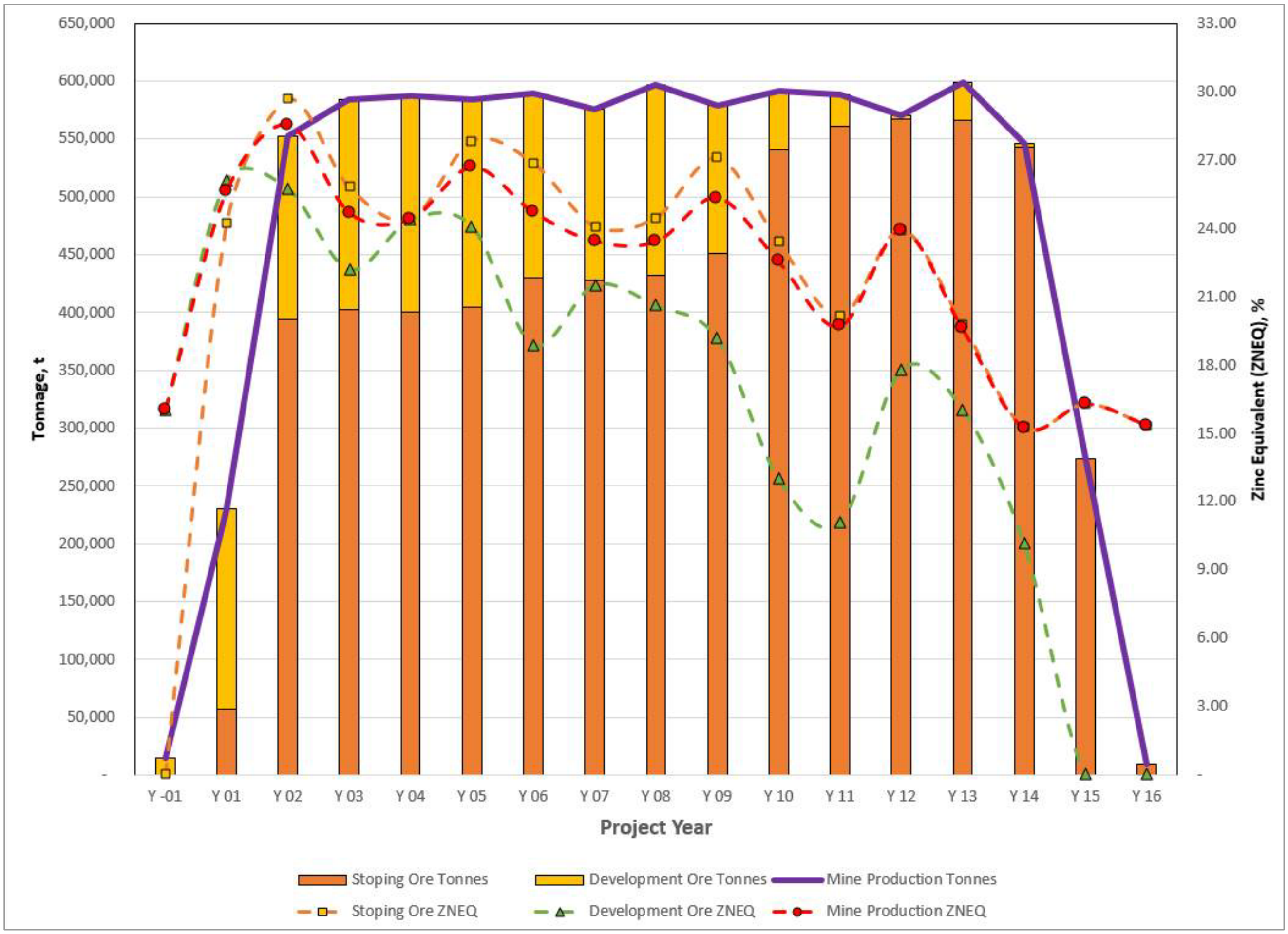


Figure 16.28 shows the split between stoping ore and development ore over the LOM.

Figure 16.28 Split of LOM ore tonnes and grade by stoping and development



16.9 Mobile equipment requirements

For the period through to achievement of steady state production and for approximately two years thereafter, a contractor will supply the personnel and mobile equipment to execute the mine plan, inclusive of all capital and development activities. At the completion of the contract, the ownership of the contractor-supplied mobile equipment is planned to be transferred to the Owner. To continue to meet the steady state development and production schedule over the LOM, procurement of additional equipment will be required at particular intervals. Table 16.13 shows the total steady state mobile equipment numbers and types that are projected to be required to meet the development and production schedule for the LOM plan after the mine switches to Owner operation.

Table 16.13 Mine equipment requirements

Item	Description	Purpose	Total
Jumbo - 2 boom jumbo	2 boom jumbo	waste development	2
Jumbo - 2 boom jumbo	2 boom jumbo	Ore / waste mucking	2
LHD	Muckmaster 600EB	Ore / waste mucking	2
LHD	Sandvik LH410	Ramp Development	2
Haulage Truck*	Sandvik TH430	Ore and waste haulage	4
Long hole drill 4 x 4 Heading	Stopemate	Production drilling	1
Long hole drill	Stopemaster	Production drilling	1
Bolter 4 x 4 Heading	Small section bolter	Bolting headings	1
Shotcrete Sprayer	Reed Lova / Aliva arm	Shotcrete as required	1
Scissor Lift	Maclean type	Bolting & services	3
Telehandler	Cat TL943	Consumable transport	1
Personnel Carrier	Landcruisers	Crew transport	6
Boom Truck	Maclean BT3	Consumable transport	1
Lube / Fuel Truck	Maclean FL3	Transport fuel and lube	1
Water Truck			1
Grader	Case 845B	Grade decline	1
Tractor	Kubota or similar	Crew and materials transport	2

Emergency preparedness

In developing the ventilation strategy for Prairie Creek, and with due regard to other operational issues, consideration has been given to the potential for mine emergencies. As such, the following criteria have been established:

- In general, ramps will be in fresh air once developed.
- Escape can be either to a ramp or to the escape ladderway in the return air raise.
- In each ramp, escape may either be up the ramp or down the ramp to a safe area.
- Portable refuge chambers are recommended for flexibility of location at appropriate points in the mine.
- Whilst the primary means of communication will be by radio, a stench system will be in place for introduction of ethyl mercaptan into the fresh air at the portal in the event of fire.

A variety of incidents may trigger the emergency response plan and / or evacuation plan. Such events may be fire, rock fall, injured personnel or major ventilation equipment breakdown.

In the event that the primary egress (883 L portal) is unavailable, a secondary means of egress from the Mine must be available to allow evacuation of all persons from underground when it is safe to do so. The secondary egress to the surface is via the 930 L portal.

For the production stoping blocks, a ladderway will be installed in the return air raise connected to the main ramp. The raise will be sized to afford easy passageway. Where the main ramp is unavailable for travel, the route of escape for personnel will be to use the ladderway to reach the 883L and then walk to the 883 L or 930 L portals. The primary exhaust raise from 883 L to surface is for ventilation only and will not be used as a second means of egress.

16.10 Refuge stations

Personnel not readily and safely able to get to surface in an emergency situation, generally the production, development and service crews, will be provided refuge by means of 12-person mobile, self-sufficient refuge chambers. These will have an independent oxygen supply, with other appropriate provisions for safe refuge. They will be located close to active working areas and in areas where secondary egress is not, or has not yet been established, or is not able to be safely accessed. They will be within the average walking pace duration of a personal self-rescuer device.

Each refuge chamber will include the following:

- Standard occupancy 12-person chamber for stand-alone operating duration of 36 hours.
- Oxygen Supply – Primary source of breathing air/life support used to replenish oxygen and flush toxic gases from within the refuge chamber. Air supply to be filtered and regulated to .09 m^3/minute (3 CFM) per occupant, with ability to isolate the system during emergencies.
- Secondary Oxygen Supply – Medical grade oxygen cache, sized to provide oxygen at a minimum rate of 0.5 L per occupant, per minute, for the duration of the use of the refuge.
- Third source of breathable air – sodium chlorate O_2 candle.
- Carbon Dioxide Removal – CO_2 removal system capable of removing no less than 24 L CO_2 per person, per hour, for the duration of the refuge, in order to maintain levels at less than 1%.
- Carbon Monoxide Removal – CO removal system capable of maintaining levels below the maximum exposure limit of 25 ppm.
- Cooling and Dehumidifying – A cooling system with nominal capacity of 130 Watts per person, to mitigate heat loads of occupants and additional heat sources, with the ability to dehumidify the chamber interior.
- Atmospheric Monitoring – Ability to monitor levels of oxygen, carbon dioxide, and carbon monoxide within the chamber and outside the chamber during emergency.
- Emergency food and water rations.

16.11 Mine dewatering

The dewatering system for the Prairie Creek mine is comprised of three sub-systems for: contact water, non-contact water, and process water. The flows from the contact water and non-contact water are kept separate in order to minimize water treatment needs on surface, with infrastructure and operating requirements for each category being different.

Contact water has been exposed to the environment of a working stope or development and may contain contaminants from blasting, combustion, or lubrication. While the aim is to recycle, as process water, as much of this water as is practicable, a small quantity will be discharged to surface (predominantly in the summer months) and will be required to be treated.

Non-contact water consists of water that has been intercepted by drill-holes before reaching the mine openings. This water will have few contaminants. Due to the flow variability of the nearby natural streams, different amounts of water can be discharged into the environment in different seasons, and any surplus during a low-flow season must be able to be stored until conditions allow discharge.

Process water will be recycled from contact water and will be used for flushing drill cuttings, washing the face, and washing equipment. Contaminants in the water will be kept at levels that are not harmful to equipment or people, but this will require monitoring. Since the process water is recycled into the contact water system, contaminants can build up (particularly during winter months) and some amount of dilution from the non-contact stream may be required to keep concentrations within acceptable limits.

16.11.1 Design criteria

The design criteria for the dewatering system is based upon work undertaken by Robertson GeoConsultants Inc. (Robertson). Robertson prepared a simulation of the mine environment and calibrated it against field test-work at the mine site. AMC provided preliminary mine plans to support the work (similar to the final design) and these were used to facilitate estimation of water inflows as the mine is developed.

Given that the dewatering strategy is to intercept the mine water at various mining levels prior to vein drifting and stoping, Robertson was asked to estimate the inflows based upon a series of dewatering horizons that would lower the water table in the ore zones in advance of mining activity. Summary results of the inflow modelling are given in Table 16.14 below.

Table 16.14 Projected Inflows to Non-Contact Water Sumps

SUMP elevation	Average rate (l/s)	Peak rate (l/s)
792	58.0	71.7
728	59.0	133
613	65.0	212
448	81.4	212

The water is planned to be taken from drill holes extended into the ore body from the sub-level below the intended mining horizon and, after a possible initial inflow of solids particles when the holes are initially placed into service, the water is anticipated to be clean with a low level of contaminants.

The non-contact water pump stations will be designed to handle the average rates indicated in Table 16.14 on an ongoing basis. Energy forecasts in the FS have considered average projected flow rates. The peak rate will be the upper limit of the pumping system capacity.

Contact water quantities were estimated to be approximately 4.5 L/s from mining activities such as drilling, muck-pile washing, products of combustion, condensation from air, and some inflows from ground water. As per the Robertson report, ground water inflows are expected to be seasonal and at a rate driven by direct vertical flow into the mine openings.

Contaminants in the contact water sumps are expected to be small particles of ore and waste rock, dissolved metals, lubricants, products of combustion, products of blasting, lubricants and diesel fuel. The concentrations of each contaminant will be relatively low and the pumps and line sizes will be chosen to reflect estimated quantities and contaminant concentration levels. Due to the requirement to maintain solids in suspension the capacity of the system will be larger than the average expected inflow rate.

16.11.2 Contact water

16.11.2.1 Overview

The contact water system is characterized by relatively low flows, handling of solids, and a rapid evolution dependent on mining activities. A practical system is envisaged that will use existing technology, be inexpensive to construct, and adapt readily to the changing mine. Staged submersible pumps in small sumps near the level access will intercept water from the stopes and declines.

Small settling sumps for each pump will settle out the larger particles of rock. Silt will be transferred along with the water to the next sump above. Each sump will be staged to the next until the system reaches the 883L adit. Water is then discharged into the existing settling sump on this level.

16.11.2.2 Design considerations

The design of the contact dewatering system is informed by considerations that include:

- Proposed paste backfill system
- Wear on pump impellers and piping
- Handling of suspended solids
- Process water sources

Paste backfill systems typically release little water or fine solids. A key aspect of the system is to trap solids and water in backfill permanently. An allowance for discharge from the paste backfill system was included in the design criteria.

Mine process water (for drilling and dust suppression) will be provided by recycled contact mine water. Information from vendors and past practice have indicated that the use of recycled water is not harmful to drills, provided excess solids are removed and no contaminants that promote corrosion are present. Contact water will be collected in individual sumps before being pumped to a settling sump on the 883 L; here remaining solids will be removed to a level suitable for the water to be used as process water. Further discussion of the process water system is provided in Section 16.2.4.

16.11.2.3 Contact water system description

The contact dewatering system consists of a series of small sumps along each decline connected by 75 mm DR 17 HDPE lines and style 905 Victaulic couplings. This type of pipe can also be supplied in continuous lengths via a reel and the style 905 Victaulic couplings require no edge preparation, therefore being easier to install.

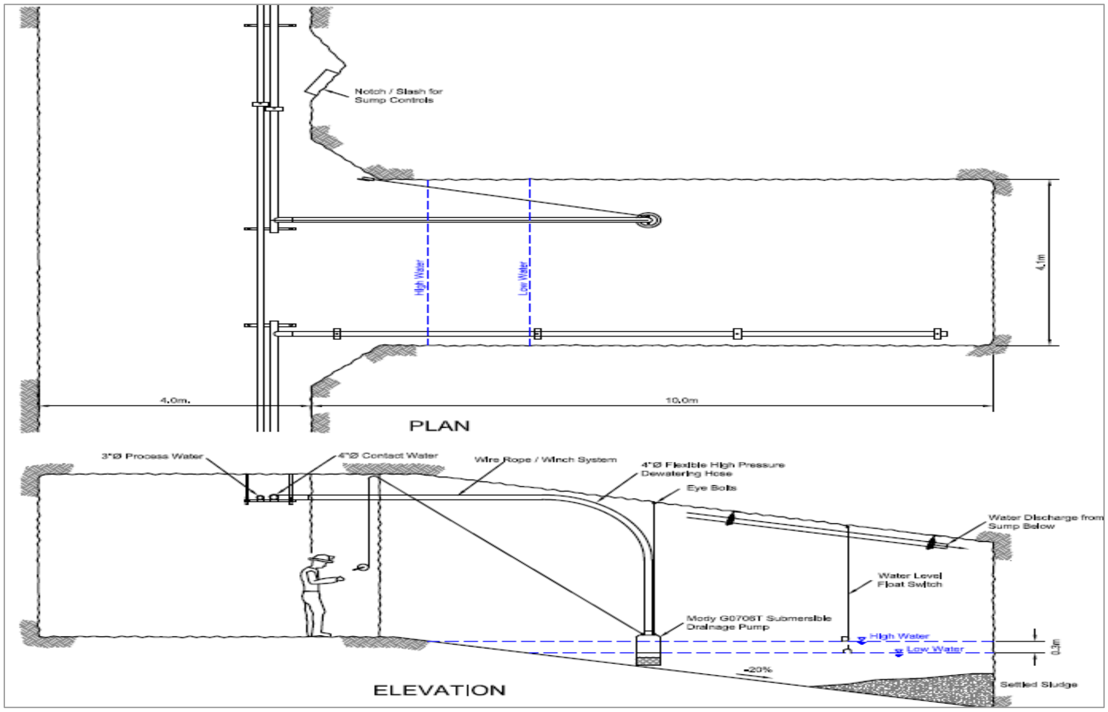
In each sump a submersible pump will be used to transfer the contact water (and some suspended solids) to the sump above. The submersible pump will have a wear resistant impeller. The pump will be hung from the back and can be recovered using a hand winch mounted near the mouth of the sump. Approximately 20 pumps will be required over the life of the mine.

Each decline's contact water system will discharge into the existing settling sump on the 883 L so that the water can be clarified before being recycled as process water.

From the 883 L clarifying sump, clear water will be sent back into the mine as process water, or discharged to the surface water treatment plant.

Figure 16.29 shows a typical sump arrangement.

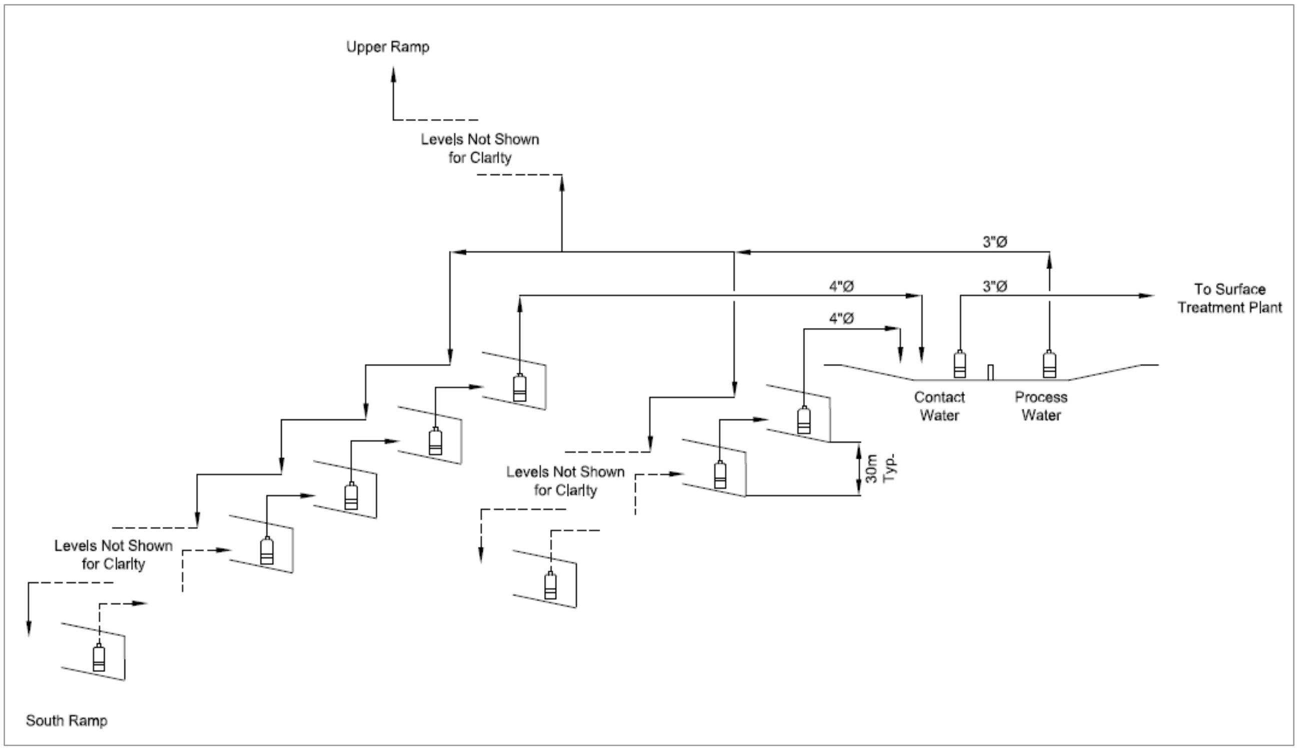
Figure 16.29 Typical contact water sump



Anticipated flow rates indicate one pump per sump and pumps can be changed out without sump overflow.

A schematic of the contact water system is shown in Figure 16.30.

Figure 16.30 Contact water schematic



16.11.3 Non-contact water

16.11.3.1 Overview

The non-contact water system is intended to intercept water in the ground around the orebody before it enters the mine workings. This minimizes discharge stream contaminants and facilitates surface storage and treatment. Non-contact dewatering sumps will be in place and operating before mining begins on the levels above.

The dewatering sumps will be established below (nominally one sub-level down), and prior to, active mining levels, and drainage holes will be drilled into the vein so that they span the length of the area to be mined. The intent is to lower the water table in the MQV zone so that mining can proceed in relatively dry conditions.

16.11.3.2 Design considerations

Design considerations for the non-contact water system include:

- Control of ground water captured via drill-holes.
- High volumes of water in upset conditions and during initial dewatering.
- High pressures due to a single stage to surface strategy.
- Flexibility of pumps during the development of the mine.
- Storage for water inflows to allow for motor cycling times.

Based upon the modelling performed by Robertson, a number of drill-holes will be required for intercepting water at each dewatering level. The longest drill-holes may need to be over 240 m long and must to be drilled with appropriate care and accuracy. Each hole will be drilled through a casing and blow-out preventer so that once the MQV zone is intersected the drill steel and bit can be withdrawn while controlling the water. The pressure rating of the casing, valves, and intermediate piping will need to be able to withstand the hydrostatic pressure of the current water table.

Each of the drill holes will be fitted with a case, piping, and an accessible valve that allows discharge of water into the sump. During initial dewatering the inflow of water must be controlled so that the pumps are not overloaded. As the water table is drawn down the flow rate will decrease and more drillhole valves will be opened to maintain flow rates. During the steady state phase the valves will all be open and the pumps will remove as much water as is required to maintain the water table a nominal distance above the drill holes.

Water will be discharged up the ramps to surface in 200-mm steel pipes. In the latter stages of the mine life, this system will develop relatively high pressures of the order of 6,800 kPa at the maximum predicted inflow rates, requiring heavy-wall steel lines. In addition, the high power required encourages the use of a high efficiency system; thus, horizontal, multiple-stage pumps will be used.

The pump design is based on preparation of separate system curves for each sump and operating condition. This confirmed the number of pumps to be operating under each condition and the number of pipelines required. Friction for the lines was calculated from the Hazen-Williams equation using a C factor of 100, which is conservative for long-service steel pipe.

16.11.3.3 Non-contact water system description

The mine will be provided with four non-contact dewatering sumps, each installed separately as the mine is developed. As ramp development progresses, each sump will be installed on the sub-level below the next active mining horizon. Boreholes will be drilled out to intercept the orebody along the strike length of the area to be mined. The pumps will be operated to dewater the ore body before mining sill drives are driven along the lowest mining level.

Enough pumps will be available at the mine so that a new sump can be constructed and commissioned below the current operating sump. When a new sump is put into service the pump(s) from the previous sump can then be leap-frogged down to the next dewatering level.

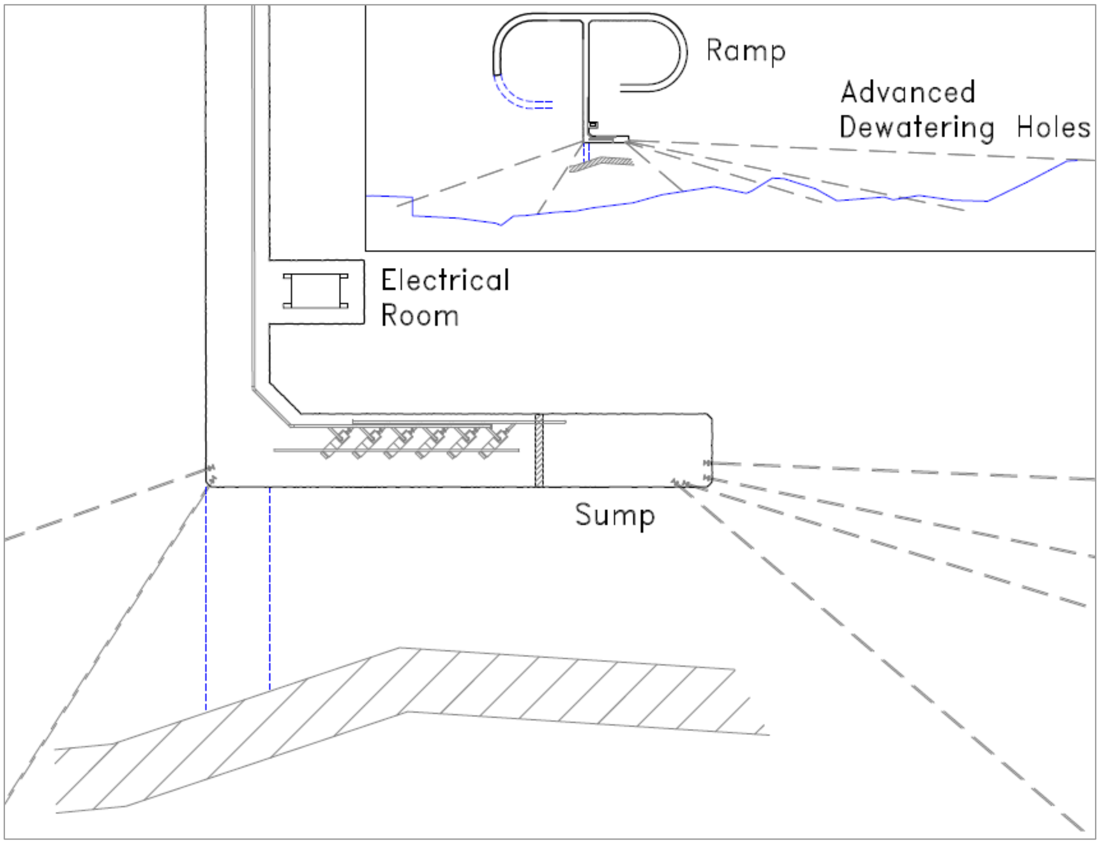
The drill holes will be driven out from the sumps (or near them) at a slight incline and through the orebody. The holes will be laid out so that intersections with the ore are evenly spaced. Each hole will be collared with pipe and routed to the sump. The borehole valves will allow manual control of the rate of water discharge into the sump. The boreholes may be up to 240 m long and are to be 150 mm in diameter.

During the commissioning of a sump, the dewatering hole valves will be opened until pumps are operating at maximum capacity. As the rate at which a dewatering drill hole produces water drops, more drill holes will be brought into commission.

The non-contact sumps will consist of a chamber sized to provide at least 15 minutes of storage at maximum design flow rate. They will be constructed by driving a 4.5 m by 4.5 m drift and constructing a concrete dam across the open end. The dam will be 300 mm thick and 3.5 m high, and mechanically keyed into the sill and the walls of the excavation.

Figure 16.31 shows a typical drill hole layout and a sump layout.

Figure 16.31 Typical drill hole and sump layout



The pumps will consist of multi-stage horizontal stacked impeller sections. The pumps will be driven by synchronous motors at 1800 rpm using 4160 volt power. Each pump may have stages inserted or removed to adapt a standard frame to each of the sump levels in the mine. A typical pump for this purpose would be a Weir PS 150 J with a maximum of six stages. Table 16.15 lists the location and number of pumps required to handle the non-contact water.

Table 16.15 Summary of non-contact pumps

Level	Pumps	Stages	Lines
792	2	2	1
728	3	3	1
613	3	4	2
448	3	6	2

The pipeline system must be designed for a long service life and a maximum pressure of the order of 6,800 kPa. A standard weight steel line with a nominal diameter of 200 mm installed in the decline will provide enough area and flow resistance to allow the pumps to operate for the upper sumps. An orifice restriction plate will provide enough resistance for the uppermost sump. For the lower sumps this will be adapted to include a second line routed via ventilation raises.

Pumping systems will be equipped with isolating valves and back-flow prevention devices. Strainers will be situated on pump inlets and will be equipped with blocked flow indicators. Strainers will also be provided with a pressure (or manually) activated bypass system to prevent cavitation.

The non-contact pumps will be driven by 4,160 volt power. No variable speed drives will be installed. Local sub-stations and switchgear will be located so that if the associated sump floods, the water will run down a decline before impacting the switchgear.

16.11.4 Process water

16.11.4.1 Overview

The process water system recycles contact water from the settling sump on the 883 L. Water is decanted from the sump at the clear end and distributed back into the mine via a small pumping system, with excess water being pumped to surface. Pressure is provided by the pumps to service levels above 883 L, and pressure reducing valves staged through small head tanks will provide process water to the levels below.

16.11.4.2 Design considerations

The design of the process water system is relatively simple with few constraints. The operating pressure at the discharge will be between 270 kPa and 410 kPa and flow rates are anticipated to be low. A 75 mm HDPE line will both meet the pressure requirements and keep the system flexible and simple to install. Average flow rates are anticipated to be about 3.1 L/s, although peak flow rates could be up to 50 L/s during wash-down service.

16.11.4.3 System description

The distribution pipe will be 75 mm HDPE DR 17 (the same as the contact dewatering system). The lines will be interconnected with Style 905 Victaulic couplings or strung intact from hose reels as long lengths of this pipe can be supplied in this manner. The lines will run down both ramps and out on the levels in order to support drilling activities.

Head tanks will be installed on 90 m intervals, with pressure reducing valves to lower the pressure onto each of the levels. This approach ensures that the static pressure cannot rise above the pressure rating of the pipe or pressure reducing valves.

16.11.5 Construction

16.11.5.1 Contact

The contact water system will be constructed as the decline and ramps are advanced. As each sump is completed a submersible pump will be installed. Water from the temporary face pumps is pumped to the first available ramp sump.

16.11.5.2 Non-contact

The first two sumps (792 and 728) are on the critical path for mine operations as the mine dewatering must lower the water table before stoping development in ore (first stoping is below 883 L) can begin. The sump must be developed, drill holes installed, and pumps commissioned before dewatering of the ore zone can begin.

The construction timing for the lower two non-contact sumps in the mine is not critical in that the development of the ramp will reach these sumps well in advance of the mining operations related to them.

One sump will remain in operation while the next lower sump is constructed. As the lower sump is commissioned the upper sump will see less water report to it and will ultimately be dry.

A short section of 200 mm line will be installed in the south ramp to service the 792 sump. This will not be needed for the full LOM and, as the sump is abandoned, the line can be re-used. Lines installed in the north ramp are required for the duration of the mine life.

Two sumps will be in service at any one time and thus six pumps will need to be available. Each set will be leapfrogged to the next lower sump as it is put into operation.

16.11.6 Non-contact sump operation

When the non-contact sump is commissioned the pumps will be ready to operate to surface. A valve from one of the pre-drainage drill holes will be opened to allow water into the sump and the pumps will begin to discharge into the handling system on surface. Flow rates must be carefully monitored and as the water table is reduced and the ore zone is drained the flow rate will reduce. As this happens additional drill holes must be opened until they are all fully engaged.

Static pressure gauges mounted on the drill holes will provide feedback as to the level of the water table above. Once pressures have dropped to the point that the water table is below the mining level, development can begin on the ore sill drives.

The upper two sumps will be serviced by a single 200 mm line. As the same pumps will be used in all sumps, pumps in the upper two sumps will have stages removed to prevent cavitation resulting from low-flow conditions.

The lower two sumps are serviced by two 200 mm lines; one line routed up the decline and a second line up the ventilation raises. The peak dewatering flows require both lines. During normal flows a single line can be used.

During regular operations the mine will only need three pumps to be on-line (two operating and one on-line spare). If the lower sump is unable to keep up with an unexpected inflow, or in the unlikely event of damage to all the pumps on a level, pumps from the preceding (upper) sump will be available as spares.

As the mine is developed and after the installation of the first two non-contact sumps, knowledge of actual pumping requirements will facilitate the detailed design of the lower sumps.

16.12 Compressed air

In the first three years of operations the contractor will provide a stationary diesel-powered compressor for all the underground requirements. For owner operations, mobile / portable compressors have been specified and costed.

During owner operation and considering cost and efficiency issues with mine-wide compressed air systems, compressed air will be supplied by local portable electric compressors, which will use the same jumbo plugs and jumbo boxes as other mobile equipment. Portable compressors will be required for the following demands:

- Elevated pressure requirements for ITH drill activities
- Powering mechanized raise climbers
- Miscellaneous activities such as spot bolting with jacklegs, powering air tools, etc.

All mobile drilling equipment, including jumbos, long-hole drills and bolters will be equipped with on-board compressors. Typical compressors that could provide air for production drilling are 470 L/s electrically driven rotary screw compressors. These would be rubber-tire mounted and moved within the production levels to follow the drills.

16.13 Underground power distribution

16.13.1 Power requirements and electrical distribution

Infrastructure and equipment that will be serviced by the mine electrical distribution system include:

- Main mine ventilation fans and mine air heaters.
- Underground dewatering systems.
- Paste distribution pump for the stopes above 883 L.
- Underground mining mobile equipment.
- Other loads such as lighting, fuel transfer, and refuge stations.

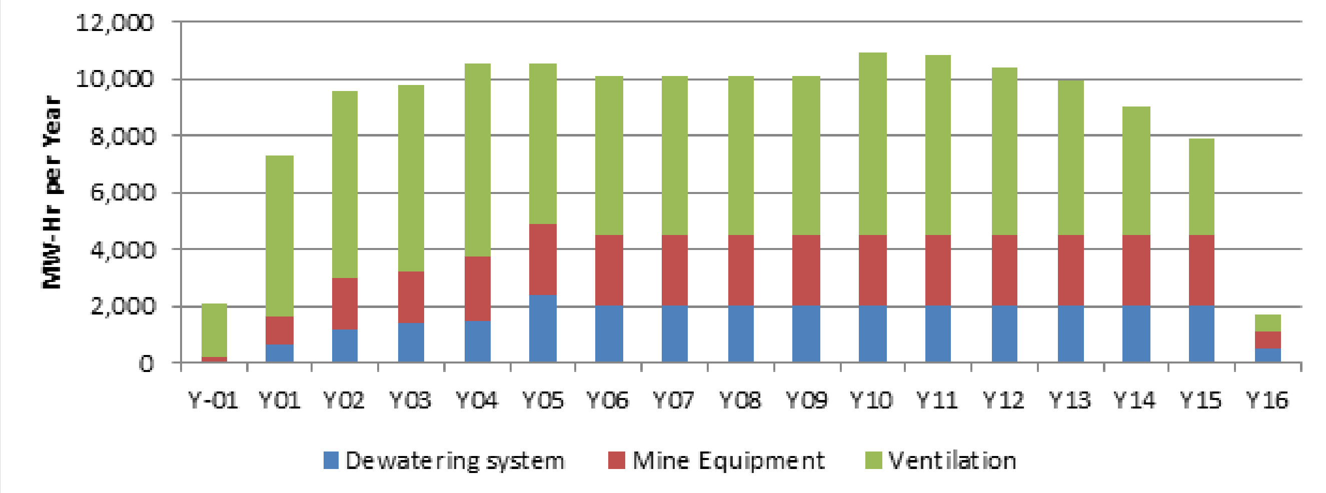
An estimated load list is summarized in Table 16.16.

Table 16.16 Underground electrical load list

Description	No.	Power (kW)	Load factor	Demand factor	Diversity factor	Running load (kW)	Demand load (kW)	Connect load (kW)
Dewatering								
792 pump station	2	671	0.294	0.328	1	129	395	1,342
728 pump station	3	671	0.433	0.359	0.667	209	581	2,013
613 pump station	3	671	0.828	0.643	0.333	357	555	2,013
448 pump station	3	671	0.884	0.92	0.333	545	593	2,013
Contact water stages	20	19	0.76	0.167	1	48	289	380
						593	**881**	**2,393**
Ventilation								
Ramp development fans	8	112	0.8	1	1	717	717	896
870 portal (with heaters)	2	75	0.8	1	1	120	120	150
930 portal exhaust	1	150	0.765	0.328	1	38	115	150
930 raise exhaust	1	150	0.765	0.359	0.667	27	77	150
Level fans	20	56	0.8	0.75	1	672	672	1,120
						857	**983**	**1,570**
Equipment								
Electric scoop chargers	3	95	0.8	0.5	0.67	76	152	285
Jumbos	4	120	0.85	0.25	1	102	408	480
Mobile air compressors	2	150	0.85	0.25	1	64	255	300
Lighting, shops, tools	1	100	1	0.8	0.5	40	50	100
						282	**865**	**1,165**
Total						**1,732**	**2,730**	**5,128**

The estimated life of mine power consumption is shown in Figure 16.32.

Figure 16.32 Estimated annual LOM underground power demand



16.13.2 Overview

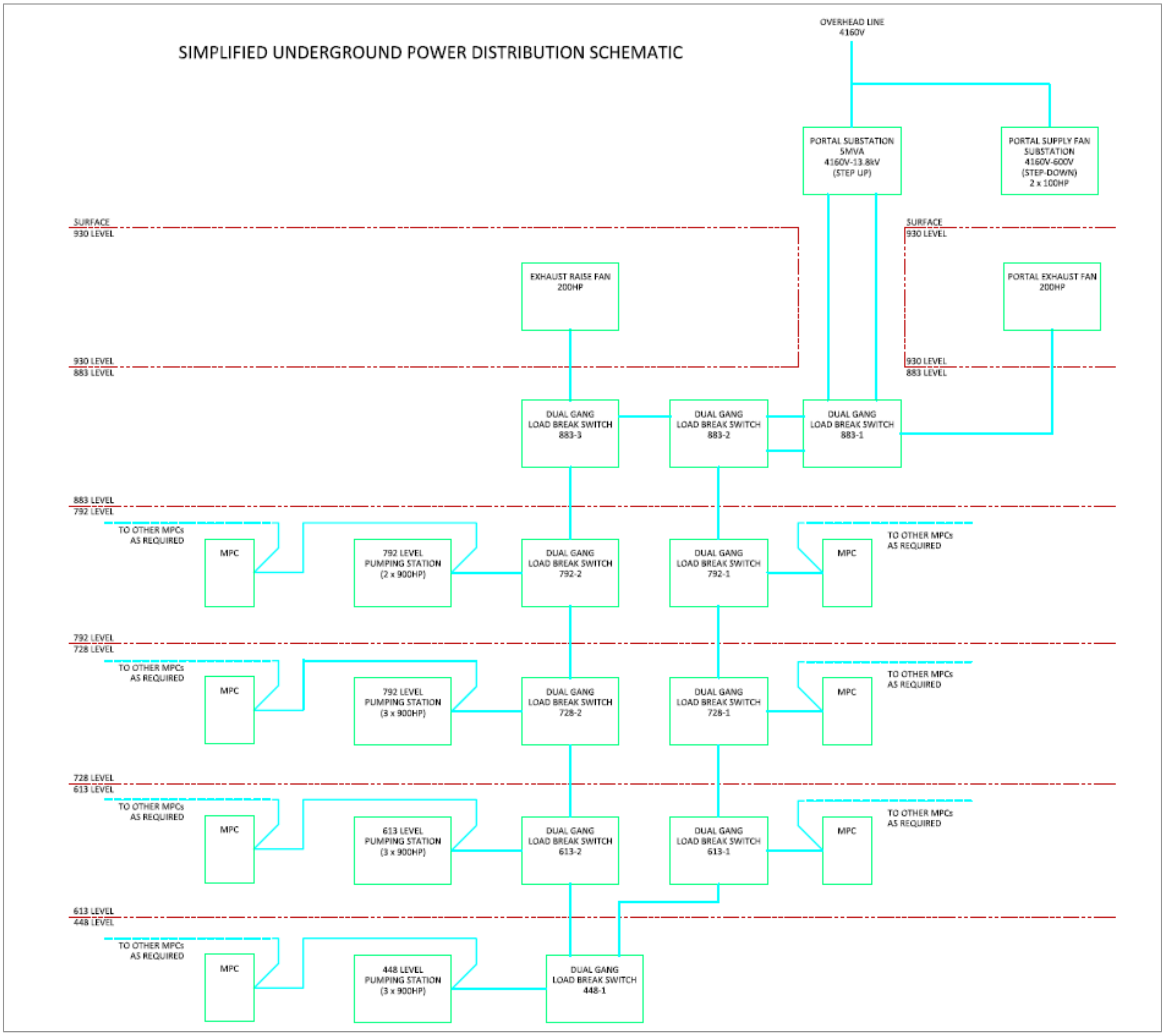
The electrical distribution system for the mine will consist of two main feeders brought down the 883 portal and access. The simplified electrical schematic for the underground power distribution system is shown in Figure 16.33.

An overhead powerline will be constructed to feed the portal substation. The incoming line voltage will be 4160 V. A tap in the overhead line will be made prior to the portal substation to supply power to the portal supply fans substation. The portal supply fans substation will step the voltage down from 4160 V to 600 V to feed two 100 HP supply fans and a 600 V-120/240 V 30 kVA transformer to supply auxiliary loads for the substation.

The portal substation will have an incoming breaker protecting the 5 MVA step-up transformer, which will raise the voltage to 13.8 kV to feed the underground distribution. A 15 kV switchgear line-up will contain two breakers that will feed one cable each to underground, thus providing a redundant feed system.

Two feeder cables from the substation will enter the portal at the 883 level and connect to a dual gang load break switch 883-1. A third switch bussed to the dual gang switch will feed a cable up a ventilation raise to the 930 L to feed the 200 hp portal exhaust fan. Initial slashing of the 883 level access will be supported by diesel generators brought in by the contractor.

Figure 16.33 Schematic of underground power distribution



16.13.3 Underground power distribution

Eight mine power centres (MPC) will provide local electrical power to levels 930, 792, 728, 613, and 448.

From the dual gang load break switch 883-1, the two 15 kV feeder cables will connect to a second intermediately located dual gang load break switch 883-2. From this switch the two cables will take alternate routes to create a redundant feed in the mine.

The first cable leaving 883-2 will be routed down a ventilation raise and a ramp to the 792 L to connect to a dual gang load break switch 792-1. From the 792-1 switch, one switch will feed down a ventilation raise and ramp to the 728 L to connect to the 728-1 load break switch. The other fused switch will feed mine power centres on the 792 L. Additional MPCs can be added via the ability of MPCs to daisy chain together as the development of the level expands.

The second cable will continue along the development at the 883 L to the third dual gang load break switch 883-3. From the 883-3 switch, one fused switch will feed a cable up a ventilation raise to supply power to the exhaust raise fan located at the 930 L. The second load break switch feeds a cable further along the 883 L to a ventilation raise down to the 792 L to a dual gang load break switch 792-2.

Levels 728, 613, and 448 (bottom pump station level) will be supplied in similar fashion. If required, additional MPCs can be added via the ability of MPCs to daisy chain together as the development of any particular level expands.

16.13.4 Backup power

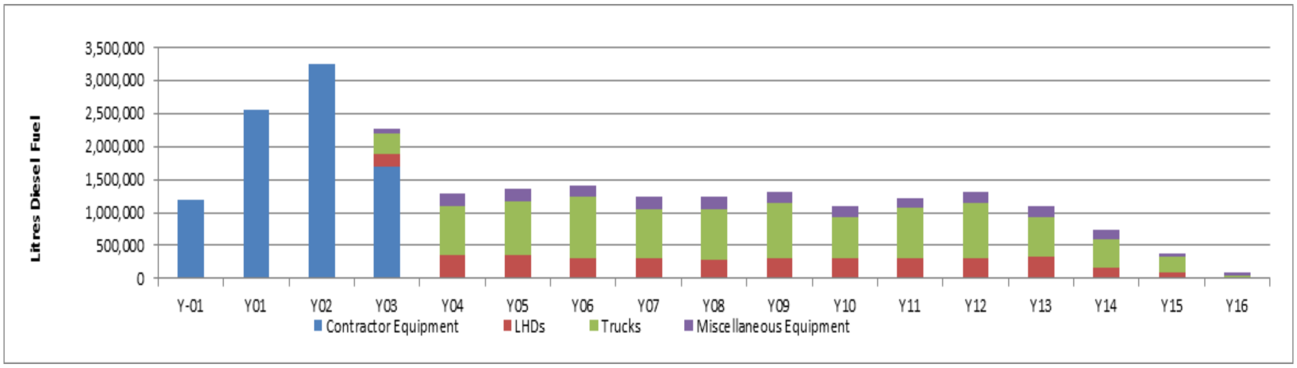
No backup power is included in the design of the underground mine's electrical system. Long term asset preservation power, will be provided by the surface generator redundancy system.

16.14 Fuel supply

Estimated annual underground fuel consumption is shown in Figure 16.34. The estimated underground peak fuel consumption is 3.2 M litres per year, with mobile equipment being the biggest user. The elevated contractor fuel consumption profile includes diesel fuel required for temporary electrical generators until the main mine power generation facility is operable in August 2020.

Vehicles that come to surface regularly will also re-fuel there. For underground diesel storage and dispensing, a 5,000 L portable 'SatStat' fuel tank will be located off the main ramp near active working horizons to re-fuel vehicles. The fuel tank will be self-bunded and fitted with a fire suppression system and self-closing fire doors. These units incorporate safety valves, dry disconnect fittings, door lock release latch and an emergency lever. The tank will be refilled as required from a fuel supply truck that will source fuel from the main surface fuel facility.

Figure 16.34 Underground diesel fuel consumption profile



16.15 Workshop facility

The main equipment maintenance area will be located on surface. All major scheduled preventive maintenance and rebuilds will take place in this surface shop. Two small service bays will be located underground, centrally located on each ramp. Equipment requiring significant work will be taken directly to the surface shop for maintenance. The underground service bays will have a finished concrete floor, monorail hoist, tire storage, lubricant storage and the capacity to make hydraulic hoses.

The service area will be equipped with a stationary compressor and airlines to power air tools and provide compressed air as needed. A welding plug will also be sited in this area.

16.16 Underground communications

Radio communications will be established underground by means of Leaky Feeder and handheld VHF radios. The Leaky Feeder system head-end unit will be installed at a suitable location near the 883 portal. The Leaky Feeder cables will run the length of the declines and also to a surface antenna. In the mine, VHF amplifiers will be spaced between Leaky Feeder VHF coax cable segments at no more than 500 m intervals. Leaky feeder cables will also branch out to all active mining levels with “end-of-line” termination antennas, as required.

16.17 Underground mine personnel requirements

In the first 39 months the mine will be operated by a contractor with oversight by the owner. Personnel will be scheduled on a regular, fly-in-fly-out rotation of two-weeks-in and two-weeks-out during operations. Most positions in operations will require a day and night shift, while technical positions typically only require a day shift. Table 16.17 lists the steady state underground personnel requirements of mine operations. At steady state 139 people (total on payroll) will be employed within the mining technical, and production departments. Some redundancy has been built into the personnel requirements to account for training, sickness and absenteeism.

Table 16.17 Underground mine steady state personnel requirements

Position	Total
Chief Engineer	1
Mine Planning / Ventilation / Ground Control	4
Mine Technologist / Surveyor	2
Chief Geologist	1
Sr Geologist	1
Grade Control / Beat	2
Mine Superintendent	1
General Foreman	1
UG Supervisors	3
Safety / Training Co-Ordinator	2
Sub-total, mining supervision, and technical	**18**
Miners	32
Longhole drillers	5
Longhole blasters	8
Services crew	8
LHD - production	8
Haul truck	12
Backfill - surface paste plant (costed with surface crew)	8
Construction / Fill Barricades / Pumping	8
Labour	4
Sub-total, mining	**93**
Maintenance Superintendent	1
Chief Mechanic	1

Position	Total
Chief Electrician	2
Maintenance Planner	2
Welder	4
Electricians	6
Diesel Mechanics	8
Plumber / Pipefitter	4
Sub-total, underground maintenance*	**28**
Total	**139**

*Some maintenance resources will be shared between underground and surface.

17 Recovery methods

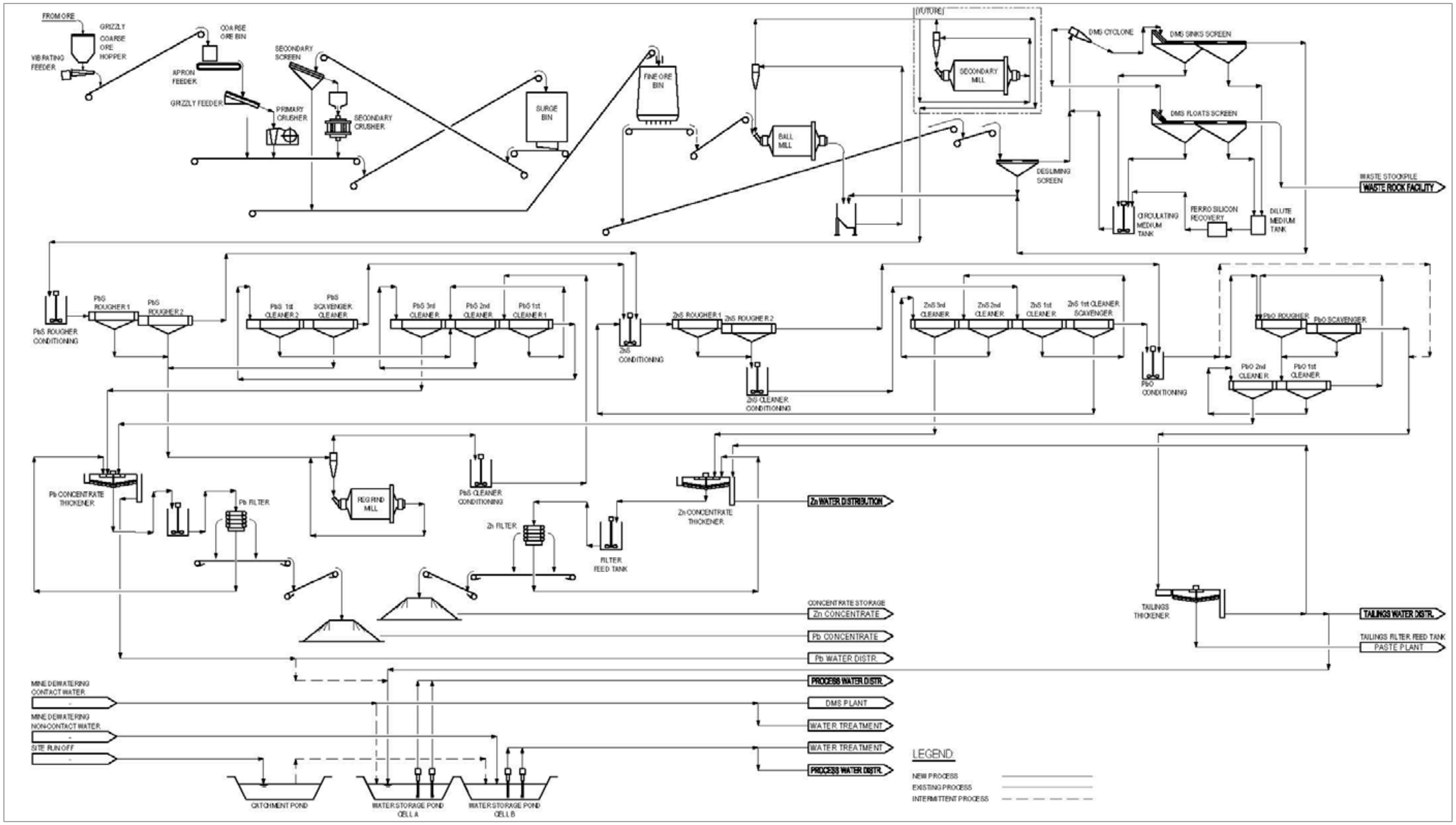
Ausenco has produced a process design for Prairie Creek that relies on information, data and analysis prepared by the Qualified Person for Section 13 of this report. As referenced in Section 13, metallurgical tests indicate that the Prairie Creek mineralization is amenable to a combined process of pre-concentration by dense media separation (DMS) and sequential flotation circuits for lead sulphide, zinc sulphide, and lead oxide concentrates.

The process design is based mainly on the results from the 2017 metallurgical test programs, including flotation, heavy liquid separation, ore hardness, and dewatering tests. However the ore hardness or Bond Work Index (BWI) has also considered the 75th percentile (100 micron) of the 1992 to 2017 BWI tests, which included a total of 12 tests with three (3) tests conducted in 2017; the BWI results ranged from 9 to 14 kWh/t. The plant is expected to treat softer ores during initial years of production and under these circumstances (BWI < 12) the milling circuit (post DMS) will achieve 1,200 t/d (nominal). As the ore becomes harder (BWI >12) an allowance has been made for the addition of a 200 kW secondary mill.

The current process design incorporates the design of the existing processing plant, which was moved from another mine and installed at Prairie Creek in 1981-2. Most of the processing circuits, including crushing, grinding, and dewatering are unchanged but will require refurbishment. The lead sulphide and zinc sulphide flotation circuits will be refurbished and upgraded to meet the required throughput; the lead oxide circuit will be a new circuit and the reagent preparation system will be completed to modern standards for lead and zinc concentrate production. The flowsheet update also incorporates DMS pre-concentration to reject gangue material prior to the grinding circuit.

Figure 17.1 shows the simplified process flowsheet.

Figure 17.1 Simplified process flowsheet



17.1 Major design criteria

The main processing design criteria are outlined in Table 17.1, which also references data presented in Sections 13 and 16.

Table 17.1 Main processing design criteria

Criteria	Unit	Value
Annual Throughput (Nominal)	tpa	584,000
Operating Days per Year	d	365
Operating Availability – Crushing	%	70
Operating Availability - DMS Plant	%	91.7
Operating Availability - Grinding and Flotation	%	91.7
Operating Availability - Concentrate and Tailings filtration	%	75
Operating Availability - Paste Plant	%	95
Nominal Rate – Crushing	tph (dry)	87
Nominal Rate - DMS Plant	tph (dry)	67
Nominal Rate - Milling and Flotation Rate	tph (dry)	50
Nominal Rate - Pb Concentrate Filtration Rate	tph (dry)	12.9
Nominal Rate - Zn Concentrate Filtration Rate	tph (dry)	12.3
Nominal Rate - Paste Plant	tph (dry)	74.0
Crushing Feed Size, 100% Passing	mm	300
Crushing Product Size, 80% Passing	mm	10.3
Ball Mill Product Size, 80% Passing	µm	156
Ball Mill Circulating Load	%	250
Bond Ball Mill Work Index	kWh/t	11-13
Bond Abrasion Index	g	0.205
ROM Head Grades Pb (Average)	%	8.33
ROM Head Grades Zn (Average)	%	8.93
ROM Head Grades Ag (Average)	g/t	127.58
Metal Recovery Method		DMS & polymetallic sequential flotation
Mass Recovery – DMS	%	75
Lead Recovery to Sulphide Lead Concentrate	%	90
Sulphide Lead Concentrate Grade	%, Pb	65
Lead Recovery to Oxide Lead Concentrate	%	9.2
Oxide Lead Concentrate Grade	%, Pb	48
Silver Recovery to Lead Concentrate	%, Ag	86
Zinc Recovery to Sulphide Zinc Concentrate	%	90
Sulphide Zinc Concentrate Grade	%, Zn	59
Silver Recovery To Zinc Concentrate	%, Ag	2.8

17.2 Process plant description

The processing plant consists of crushing, DMS pre-concentration, grinding, sulphide and oxide polymetal sequential flotation, concentrate dewatering, and tailings dewatering / paste preparation units.

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 ROM ore to a particle size of 80% passing 10 mm.

The major equipment and facilities in this area include:

- ROM ore dump pocket (40 t live capacity) with a fixed grizzly and a vibrating feeder.
- Coarse ore surge bin (136 t) with an apron feeder fitted with grizzly bars.
- Kue-Ken 36'' x 24'' jaw crusher.
- Secondary crushing feed surge bin (45 t) with a belt feeder.
- Double deck screen with apertures of 25 mm and 15 mm.
- Symons Nordberg 5.5' shorthead cone crusher.
- Conveyors including a metal detector and a magnetic separator.
- Fine ore bin (1,800 t) with a reversible belt feeder.
- Dust collection systems.

The feed tonnage will be controlled by adjusting the speed of the fine ore bin discharge conveyor belt. Operators will have the ability to set an optimal feed rate based on DMS plant on-stream analyser information.

17.2.2 DMS plant

The DMS 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 crushed ore is deslimed, whereby material finer than 1.4 mm is removed and is followed by a dense media (ferrosilicon) cyclone separation at a proposed separation SG of 2.8.

The rejects from the DMS plant will be conveyed to a temporary 200 t stockpile (uncovered) and will be loaded onto the waste ore haul trucks by a front-end loader for transport to the waste rock storage facility.

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

- Desliming screen with apertures of 1.4 mm.
- Heavy media cyclone.
- Sieve bends.
- Drain and rinse screens.
- Heavy media preparation system.
- Circulating heavy media handling system.
- Dilute heavy medium handling system including a wet magnetic separator.
- Tanks, pumps, and conveyors.

The sink fraction will be discharged into the deslime pump box where the fines and sink will both be pumped to the primary ball mill grinding circuit. The DMS circuit is designed to be by-passed, whereby the feed will be directed to the grind circuit via the feeder under the fine ore bin.

17.2.3 Grinding and classification

The grinding circuit will consist of a ball mill in closed circuit with classifying hydrocyclones located within the existing mill building.

Major equipment and facilities in this area include:

- Existing refurbished 10' diameter x 14' wet ball mill with a 700 horsepower motor.
- New classifying hydrocyclone pack.
- Existing refurbished ball mill discharge pump box.
- Existing refurbished hydrocyclone feed pumps.
- New on-stream particle size analyzer.

- Existing ball mill feed conveyor.
- Ancillary equipment including a steel ball storage bin and a ball bucket.

The ROM ore is expected to be harder as the mine develops deeper and an increase in the Bond Work Index is anticipated. As such, a new secondary ball mill (200 kW tyre mill) with associated cyclones (closed circuit), pumps and pump boxes is planned for installation, costed as sustaining capital after 5 years of operation. Precise timing of this additional milling power will need to be optimized based on the work index progression over time and other economic factors.

17.2.4 Flotation concentration

Polymetallic, sequential flotation will be employed to separate lead and zinc sulphide and lead oxide. Three separate flotation circuits will be installed in order: sulphide lead, sulphide zinc, and oxide lead flotation. All existing refurbished flotation cells will be utilized for sulphide flotation. The existing regrind ball mill will be refurbished and utilized to further grind the lead sulphide rougher flotation concentrate in order to maximize grade of the final lead sulphide concentrate.

Lead sulphide flotation

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

- One new rougher conditioning tank, equipped with a mechanical agitator.
- Refurbished existing rougher flotation trough cells (ten (10) of 2.8 m^3 capacity) with the existing concentrate diverter plates.
- New rougher flotation trough cells (seven (7) of 5 m^3).
- Two rows of new 4 x 5 m^3 1st cleaner, 1st cleaner scavenger, and 2nd cleaner flotation trough cells.
- Refurbished existing 3rd cleaner trough flotation cells (six (6) of 1.4 m^3 capacity).
- Two (2) existing refurbished cleaner conditioning tanks, equipped with mechanical agitators.
- Ancillary equipment including pump boxes, sump pumps, and samplers.
- Refurbished existing regrind ball mill with cyclones, pumpbox, and pumps.

Zinc sulphide flotation

The zinc sulphide flotation circuit consists of rougher flotation followed by three stages of cleaner flotation.

The major equipment for the zinc sulphide flotation circuit includes:

- Two (2) existing rougher conditioning tanks, each equipped with mechanical agitators.
- Refurbished existing trough rougher flotation cells (ten (10) of 2.8 m^3 capacity).
- Refurbished existing trough 1st, 2nd, 3rd cleaner flotation cells and 1st scavenger cleaner flotation cells (twelve (12) of 1.4 m^3).
- Ancillary equipment including pump boxes, pumps, and samplers.

Lead oxide flotation

The lead oxide flotation circuit consists of new equipment including rougher / scavenger flotation and two stages of cleaner flotation cells located within a new building connected to the mill building via an arctic corridor. The lead oxide flotation circuit design is based on historical testwork (pre 2017) results. New lead oxide flotation floatation test work is underway and the new lead oxide flotation circuit will be optimized when the test results are available.

The major equipment proposed for the lead oxide flotation includes:

- Two (2) new rougher conditioning tanks equipped with mechanical agitators.
- Nine (9) 5 m^3 new rougher / scavenger trough flotation cells.
- Five (5) 5 m^3 new 1st and 2nd cleaner trough flotation cells.

- New building.
- Glycol heating system.
- Ancillary equipment including pump boxes, pumps, and samplers.

17.2.5 Concentrate dewatering and load out systems

The concentrate from the three flotation circuits will be dewatered by thickening and pressure filtration. The lead oxide and lead sulphide concentrates are combined in the lead thickener, resulting in a blended lead concentrate. The filtered lead and zinc concentrates will be stored separately on site in a temporary stockpile before being loaded into 20 t purpose built concentrate containers. Both the existing concentrate thickeners and pressure filters will be refurbished and upgraded for use.

Lead concentrate dewatering and load-out system

The existing 10.7 m (35') diameter lead sulphide and oxide concentrates thickener will be upgraded by installing a new high-rate feed well. The existing centrifugal thickener underflow pumps will be refurbished and the underflow will be pumped to the 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 five (5) 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 the additional capacity in the existing 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% by the existing refurbished Larox pressure filter. The dewatering filtrate will return to the lead concentrate thickener. The concentrate will then be conveyed a lead concentrate stockpile with one (1) day of storage capacity and loaded into purpose built 20' concentrate containers complete with removable lids. Additional buffer storage to account for closure of the transport route (up to one (1) week) will be provided by using additional bulk containers, which will be stored on site in a newly constructed container storage area near the air strip.

The containers will be loaded using a front end loader and a weigh scale with a digital readout to assist the operators to load to the desired payload. Reach stacker container handlers will be used to manoeuver the containers in and out of the building and to relocate containers to/from the container storage area.

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 refurbished 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.
- Ancillary equipment including pump boxes and pumps.

The following equipment will be stored between the lead and zinc concentrate systems:

- Concentrate containers
- Container weigh scale
- Reach stacker container handler
- Vacuum clean-up system
- New storage building

- Glycol heating system
- Dust collection system

Zinc concentrate dewatering and load-out system

The existing 10.7 m diameter zinc sulphide concentrate thickener will be upgraded by installing a new high-rate feed well. The existing centrifugal thickener underflow pumps will be refurbished and the underflow will be pumped to the zinc concentrate surge tank (equipped with a mechanical agitator) at a solids density of approximately 65%. 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 five (5) 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% by the refurbished existing Larox pressure filter. The dewatering filtrate will return to the lead concentrate thickener. The zinc concentrate will then be conveyed to a dedicated zinc concentrate stockpile (adjacent to the lead concentrate stockpile) with a temporary storage capacity of one (1) day of concentrate production. The zinc concentrate is loaded into 20 t containers prior to shipping, as noted above.

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

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

17.2.6 Tailings handling

The rougher / scavenger tailings from the lead oxide flotation circuit will be pumped to the tailings thickener and then to the backfill plant to produce paste for backfilling the underground slopes. The flotation tailings will be pumped to a new high-rate tailings thickener where a solids density of approximately 60% will be achieved. Flocculant will be added to the thickener to facilitate the thickening process. The thickened tailings will be pumped to the paste filter feed tank (equipped with an agitator) for feed to the paste plant discussed in Section 16.5.2. The overflow from the tailings thickener will be pumped to Cell A of the water storage pond, which is the process water storage compartment as discussed in Section 18.24.

The major equipment in the tailings handling area includes:

- New 10 m diameter high-rate tailings thickener.
- New ancillary equipment including pump boxes and pumps.

17.2.7 Tailings paste plant

There will be a new paste plant and paste delivery system as discussed in Section 16.5.2.

17.2.8 Reagent preparation and delivery

Various chemical reagents will be added to the flotation circuits to facilitate lead, zinc, and silver recovery. Specific reagent requirements for the Prairie Creek 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
- 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 the 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 in a section of the lead oxide flotation building, which will be connected to the mill building to provide access for operators without going outside.

17.2.9 Assay and metallurgical laboratory

The metallurgical laboratory will be located in the existing refurbished office rooms in the mill building complete with HVAC and safety station. This laboratory will undertake test work to monitor metallurgical performance and facilitate improvement of process unit operations and efficiencies. The metallurgical lab will be equipped with equipment that is relatively insensitive to vibration and dust to perform tests such as flotation tests, size analysis, grinding tests and prepare shift samples for assays.

The metallurgical laboratory equipment will include:

- Laboratory crusher
- Laboratory ball mills
- Sample pulveriser
- Splitter
- Ro-tap sieve size analyser
- Laboratory flotation test cells
- Laboratory vacuum filters
- pH meters
- Weighing scale
- Hot plate
- Work bench
- Drying oven

A new stand-alone assay and water treatment laboratory with HVAC and safety stations will be housed in a free-standing pre-engineered building remote from vibration and dust caused by operating machinery. It will be equipped to conduct all routine assays for the mine, concentrator and environmental department.

The assay and water treatment laboratory equipment will include:

- Laboratory crusher
- Sample pulveriser
- Splitter
- Microwave plasma-atomic emission spectrometer (MP-AES)
- Graphite atomic absorption spectrophotometers (AAS)
- X-ray fluorescence spectrometer (XRF)
- UV / VIS spectrophotometer
- Drying oven
- Pressed pellet
- Chloride ISE kit
- Laboratory-size pressure filter
- Fusion furnace

- Cupelling furnace
- Hot plate
- Weighing scale
- Work bench
- pH meters

17.2.10 Mill water supply and distribution

Fresh water

Fresh water will be supplied from Cell B of the Water Storage Pond, which is supplied with mine dewatering non-contact water; treated sewage treatment plant effluent (liquid) and site run off will be the main sources.

Fresh water will be used primarily for:

- Fire water for emergency use.
- Gland services for the slurry pumps.
- Reagent make-up.

Fire water and potable water are discussed further in Section 18.2.

Process water

The concentrate thickener overflows will be pumped back to the respective flotation circuits and re-used. The tailings 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 reagents are allowed to degrade for approximately six (6) months. No treatment of the process water is required.

Process water is supplied mainly to the DMS plant and the grinding circuit through a ring main process water pipe arrangement.

17.2.11 Compressed air supply

The two (2) 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 to remove moisture 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 (2) existing blowers will be refurbished and supply air for all flotation cells.

17.2.12 On-stream sample analyzers

For process control, the processing plant will be equipped with an on-stream analyzer capable of analyzing nine (9) process slurry streams. On-stream analysers will aid operators to optimize each process circuit with real time information on the DMS, grinding, and flotation circuits. Shift samples will also be taken for metallurgical accounting purposes and will include feeds to the DMS plant, three flotation circuits, the final tailings, and the final concentrates. The shift samples will be assayed in the site assay laboratory.

An on-stream particle size analyzer will be included in the grinding circuit to measure particle size of hydrocyclone overflow for controlling and monitoring circuit grinding performance.

17.3 Process plant instrumentation and controls

17.3.1 Plant control

The refurbished process equipment will be manually controlled with aids of a programmable logic controller (PLC)-based process monitoring. The system will generate production reports and provide data and malfunction analyses, as well as a log of all process upsets. All process alarms and events will be also logged into the historian database.

Secondary local interface (or control panels) will be provided for the following areas:

- DMS plant
- Backfill paste plant
- Water treatment plant
- Concentrate filters
- Power generation plant

New intelligent-type motor control centres (MCCs) will be located in electrical rooms throughout the plant. A digital interface to the control system will facilitate MCC remote operation and monitoring.

For site-wide infrastructure (i.e. telephone, internet, security, fire alarm, and control systems), a fiber-optic backbone will be installed.

17.3.2 Control philosophy

To control and monitor all mill building processes, three (3) PC work stations will be installed in the refurbished central control room located within the mill building. The following will be controlled and monitored:

- Underground production, primary crushing and secondary screening.
- Dense media separation circuit.
- Grinding feed conveyors (zero speed switches, side travel switches, emergency pull cords, and plugged chute detection).
- Ball mill (mill speed, bearing temperatures, lubrication systems, clutch, motor, and feed rates).
- Pump boxes, tanks, and bin levels.
- Variable speed pumps.
- Hydrocyclone feed density controls.
- Thickeners (drives, slurry interface levels, underflow density, and flocculent addition).
- Flotation cells (level controls, reagent addition, and airflow rates).
- Samplers (for flotation optimization).
- Concentrate filters, and load out.
- Reagent handling and distribution systems.
- Tailings disposal to paste backfill or tailings storage.
- Water treatment, storage, reclamation, and distribution, including tank level automatic control.
- Air compressors.
- Fuel storage.

An automatic sampling system will collect samples from various process streams for online analysis and daily metallurgical balance accounting.

17.4 Annual production estimate

Annual metal production shown in Table 17.2 was developed by John Huang from TetraTech and is projected from the mine production schedule shown in Section 16 and the metallurgical performance projection outlined in Section 13. Based on this for the LOM annual average, the process plant is estimated to produce approximately

64,000 t of lead concentrate with an average grade of 61.9% lead and 67,000 t of zinc concentrate with an average grade of 58.7% zinc. Average silver grades are projected at approximately 800 g/t for lead concentrate and 140 g/t for zinc concentrate.

Arsenic, mercury and antimony contents in lead concentrate and mercury contents in zinc concentrate may be higher than the penalty thresholds given by most smelters. For lead concentrate, on a yearly basis, average main impurity contents are anticipated to vary from 0.1 to 0.4% for arsenic, 0.2 to 1.1% for antimony and 0.006 to 0.03% for mercury. For zinc concentrate, on a yearly basis, average mercury contents are estimated to fluctuate from 0.05 to 0.20%. Further review of smelting terms for the Project should be conducted, including for all the impurities that may attract penalties.

Table 17.2 Projected lead and zinc concentrate production

Year	01	02	30	04	05	06	07	08	09	10	11	12	13	14	15	16	Total / LOM average
Mill feed tonnage / grade																	
Tonnage, kt	135.5	596.9	607.3	602.6	606.3	594.7	575.5	594.6	578.9	570.5	571.9	557.1	523.4	541.0	406.1	9.1	8,071.5
Grade																	
- Pb, %	8.4	9.7	8.9	9.2	9.7	10.1	9.3	8.3	9.9	8.5	5.7	7.7	6.0	4.4	4.0	3.8	8.1
- Zn, %	8.8	11.4	9.3	7.8	8.8	7.0	7.2	8.6	8.0	8.0	9.3	10.0	9.9	7.6	8.0	7.9	8.7
- Ag, g/t	139	161	141	137	152	143	130	129	143	119	97	127	85	74	73	80	124
Lead concentrate tonnage / grade																	
Tonnage, kt	15.4	80.3	75.5	78.5	83.5	85.9	76.3	68.5	81.8	70.5	45.5	62.2	47.6	31.4	20.8	0.4	924.2
Grade																	
- Pb, %	60.08	61.08	62.74	63.46	63.48	63.33	63.37	63.00	63.26	62.52	59.82	57.97	56.43	59.32	62.08	62.42	61.92
- Zn, %	7.22	7.18	7.11	7.08	7.08	7.09	7.09	7.10	7.10	7.12	7.38	7.53	7.73	7.19	7.20	7.12	7.19
- Ag, g/t	884.5	881.2	847.8	790.2	832.9	746.8	740.1	833.6	762.7	718.5	864.5	802.3	621.8	881.3	1,012.2	1,194.9	799.5
Zinc concentrate tonnage / grade																	
Tonnage, kt	13.5	82.9	76.6	67.4	78.2	58.7	60.8	75.1	66.5	68.0	74.0	72.0	72.8	58.9	48.8	1.1	975.4
Grade																	
- Pb, %	3.89	4.03	3.93	4.00	4.07	4.17	4.03	3.80	4.04	3.32	2.69	2.78	2.03	2.58	2.73	2.75	3.47
- Zn, %	59.00	59.00	59.00	59.00	59.00	59.00	59.00	59.00	58.95	58.56	58.41	58.04	57.50	58.53	58.89	59.00	58.70
- Ag, g/t	151.2	164.8	152.2	139.1	153.3	128.6	130.7	142.0	135.4	121.3	117.1	136.4	113.1	130.3	117.9	135.8	135.8

17.5 Stockpile

The ROM area will store up to approximately 100,000 t of coarse ore during the life of the mine and the stockpile will be used to supplement mine production, as necessary, against mill capacity (refer to Figure 18.3 for location). A coarse ore stockpile of approximately 10,000 t currently exists outside the mill building and will be relocated to the ROM stockpile. The ROM area will also include a 1,500 t stockpile for temporary storage of waste rock, which will be loaded into the haul trucks via a front end loader.

18 Project infrastructure

The Prairie Creek Mine is a remote, isolated site, with infrastructure that requires upgrade, expansion or replacement where necessary. Figure 18.1 shows a photograph of the site from 2015 and is representative of current day site conditions.

Figure 18.1 The present-day Prairie Creek Mine site infrastructure

The mine site lies on the flood plain of Prairie Creek and engineered dykes and berms were built in 1980-82 adjacent to Prairie Creek to prevent flooding of the site. The dykes next to the impoundment pond are cored with clay material, and are lined with coarse rip-rap armour at their base against the Prairie Creek flow. Since this infrastructure was built the site has not flooded, in spite of a number of significant flood events during that time. The dyke / berm system can be seen at the base of the above photograph with the coarse rip-rap armour showing up as the light bleached colour.

Figure 18.2 is a plan view of the existing site layout. Figure 18.3 shows the proposed general arrangement for modified infrastructure at the site.

Figure 18.2 Existing site layout of the Prairie Creek Mine infrastructure

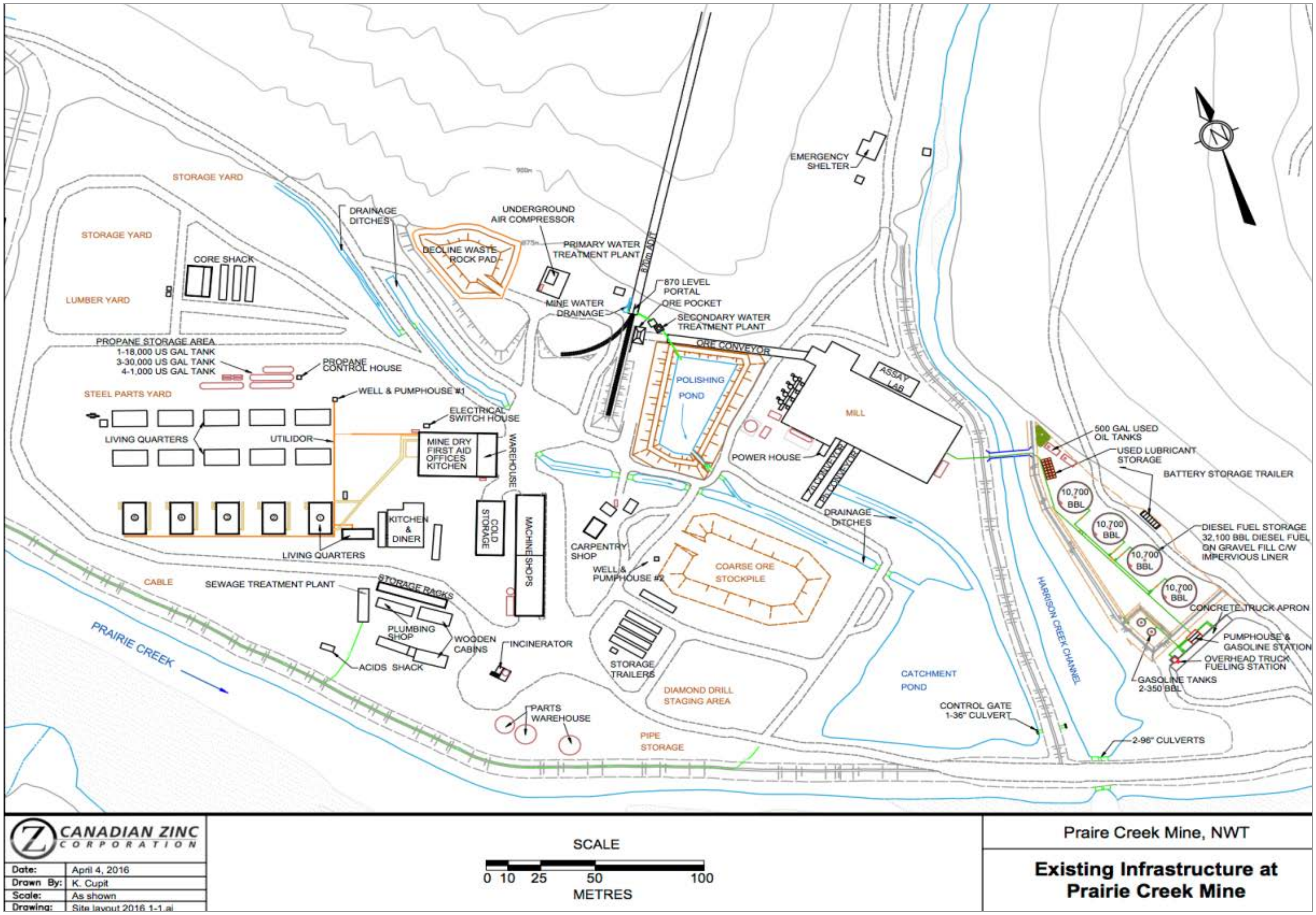
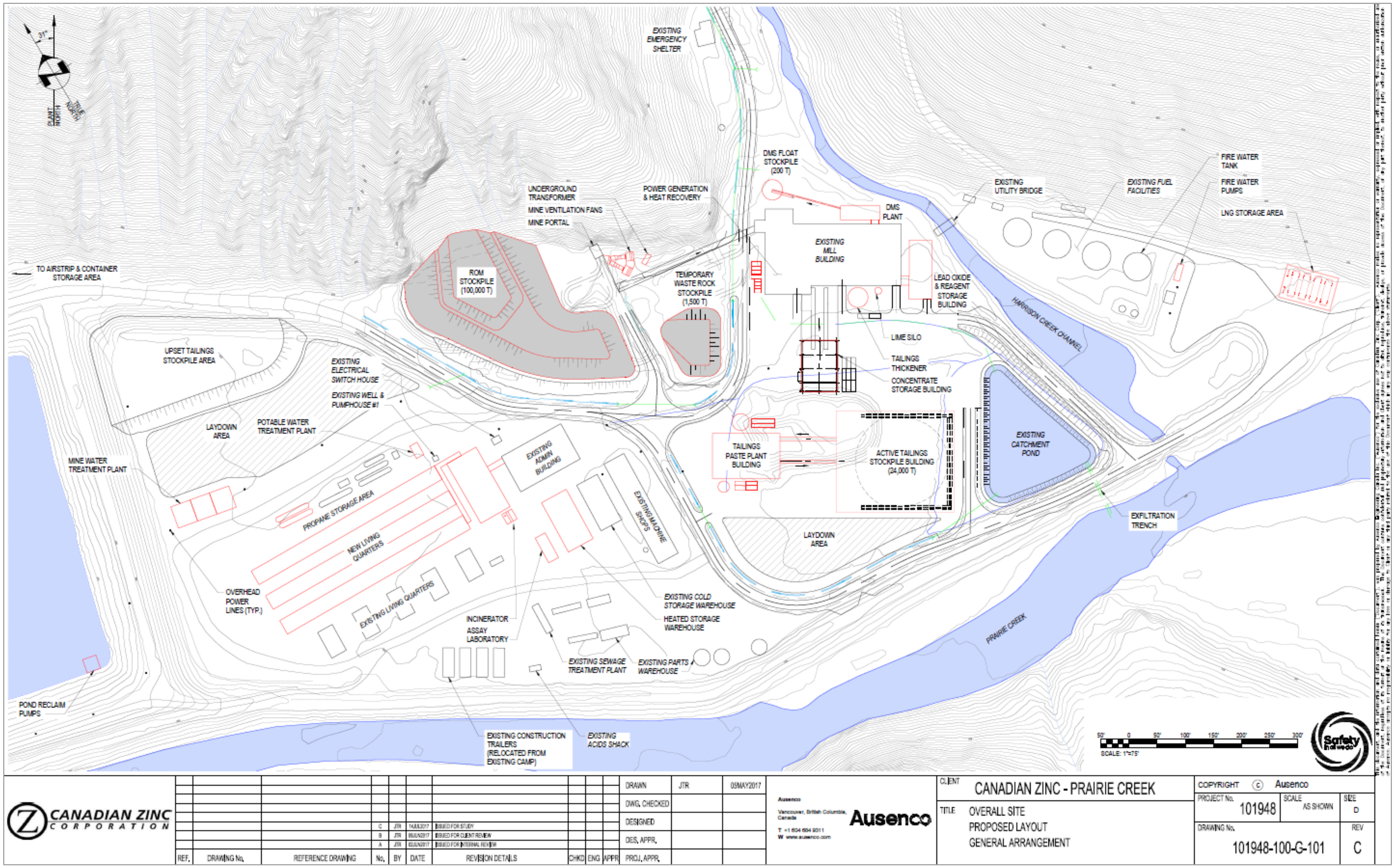


Figure 18.3 Proposed general arrangement of the modified Prairie Creek Mine site infrastructure



18.1 Camp

The five modules of the current camp accommodation (with orange stripes in the foreground of Figure 18.4) will be refurbished and are adequate for up to 50 people, with operational cookhouse facilities in the administration building.

The ten modules of the current camp accommodation (with yellow stripe in the background of Figure 18.4) are in various stages of deterioration. Some will be repurposed for use during construction such as warehouse, storage and offices and the others will be demolished.

A self-contained, modular camp (second-hand) with accommodation for an additional 170 people will be constructed to support the construction and operations activities throughout the mine life. The current 50-person accommodation (the five trailer units with orange stripes shown in the foreground of Figure 18.4), will be used for short-stay and overspill accommodation. The new kitchen will be sized for 250 people.

Figure 18.4 Prairie Creek present-day accommodations

18.2 Water

18.2.1 Domestic water

Domestic water will be pumped from an existing well, which was tested in July 2014 and showed no draw-down after four hours at a pumping rate of 46 litres per minute. This is sufficient to supply 300 litres per person per day to 200 people. A new potable water treatment plant will be provided for treatment of the well water for domestic use.

18.2.2 Fire water

There will be a minimum of two (2) hours of dedicated fire water supply stored in an above ground fresh / fire water tank (minimum 680,000 L), and a fire water pump will deliver fire water through a distribution network to each of the protected areas. One (1) of the existing diesel fuel tanks will be refurbished and repurposed for use as the fresh / fire water tank.

Fire water will be supplied from Cell B of the WSP and distributed using electric pumps (main pump with jockey pump) and a diesel fired fire water pump will be provided for automatic start in the event of a failure of the prime electric fire pump.

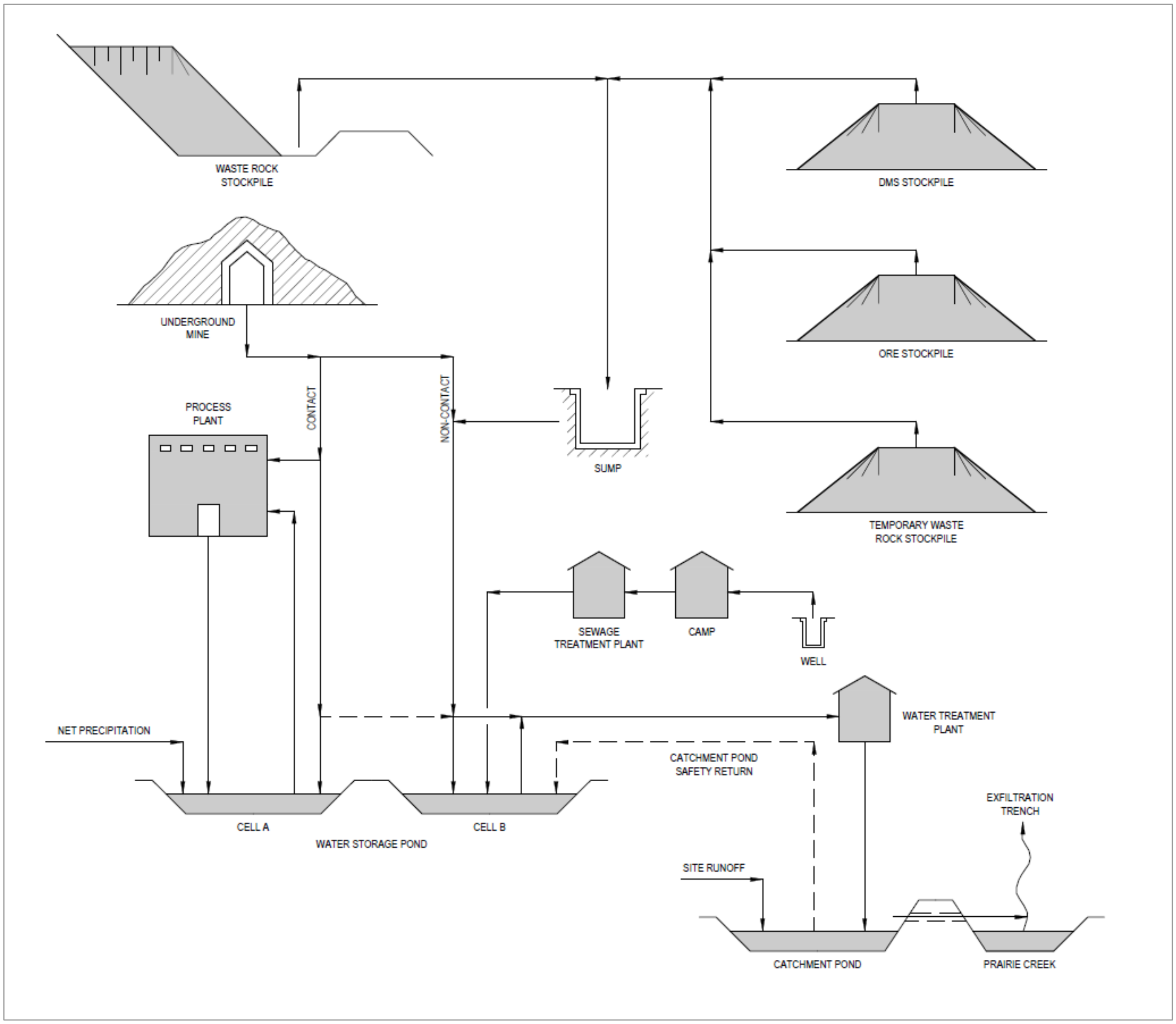
18.2.3 Site water management

Figure 18.5 shows the general site-wide water balance. The main inflows to the site water balance are from the mine and recycled process water from the mill, with minor inputs from local runoff and treated sewage effluent.

The existing water treatment plant treats only the mine water that flows from the underground workings during the warm season. This plant consists of a primary mixing tank where the main reagents (sodium sulphide and / or soda / lime) are added to the mine water followed by addition of some flocculants before the mine water flows into the polishing pond. The polishing pond is a three-celled pond, where most of the dissolved zinc precipitates out of the water before it flows into the catchment pond and thence into Harrison Creek through a controlled culvert. This water treatment plant will be dismantled and replaced by a new plant; the polishing pond will be converted into the temporary waste rock stockpile.

Discharge from the storage pond will consist of treated water from the WTP and natural non-contact site run off during the runoff season. A new culvert will be installed in the catchment pond that receives the two streams and discharges to the exfiltration trench. In winter, the culvert will only receive treated mine water. The culvert inlet will include a recycle option in the event that the discharge does not meet water quality objectives, in which case the water will be pumped back to the WSP. The exfiltration trench will contain two perforated pipes of different lengths. The longer pipe will be used in summer when the channel is wider, and the shorter pipe will be used in winter. Having two pipes also provides redundancy in the event that one pipe is unusable for any reason.

Figure 18.5 Prairie Creek proposed new site water management



Two main sources of water will need to be managed during mine operations. These are:

- Inflows to the mine
- Process water from the mill

Both water sources will contain metals in varying amounts. Process water is expected to contain much higher concentrations of most metals plus residues from flotation chemicals.

A large pond was originally built on site with dykes and a clay lining, which was originally intended for tailings disposal. This pond will be re-engineered (including the installation of a new synthetic liner) as a Water Storage Pond (WSP). The WSP will consist of two cells:

- Cell A – process water and contact mine water
- Cell B – non-contact mine water and site drainage

Water levels in the WSP will fluctuate seasonally, increasing in the winter as water is accumulated in storage, and decreasing in the summer during the main water treatment period.

Water will be managed during operations according to metals content. Cell A of the WSP, with a capacity of 264,000 m^3, will temporarily store mill process water for several months in order to allow recycling to the mill. Losses to solids (DMS, tailings, concentrates) will be made-up by use of mine contact water. Any excess contact water would be treated for discharge in the WTP.

Cell B of the WSP, with a capacity of 402,000 m^3, will temporarily store mine non-contact water and runoff from stockpiles. Mine non-contact water flow is expected to peak at 65 L/s. Flow from the stockpiles is estimated to average less than 2 L/s. The WTP will have an initial capacity of 75 L/s. Because of the expected low metals content in Cell B water, not all of the water is expected to require treatment for discharge. However, in the event that mine non-contact water flow is greater than expected, and there is a greater water treatment demand, the WTP will be expanded. Non-contact water flows will be monitored so that the required treatment capacity can be available before it is required.

During operations, process water will be pumped from Cell A of the WSP after any residual reagents are allowed to degrade in the WSP for approximately 6 months in lieu of treatment in the WTP. The majority of groundwater will be intercepted before it flows into the Mine, thus preventing its contamination with oil, mud and blasting residues resulting from mining operations (refer to Section 16.11). This non-contact water will be sent to Cell B of the WSP. Water that flows into the Mine and becomes contaminated will be sent directly to the DMS plant as feed water, with any excess being sent to the Mine Water Treatment Plant (WTP) and to Cell A of the WSP if the WTP is not operational. The majority of mine water reporting to the WSP will be treated year-round in the WTP, although some may not require treatment to meet discharge criteria.

Process water will be contained within a closed loop (Cell A and the process plant) with make-up water added from Cell B to make up for water losses in the concentrate and tailings paste streams. The process water will not require treatment in the WTP.

Any water that will be released to Prairie Creek will meet the water quality protocol described below.

The treatment and release rates of both mine water and process water will vary depending on flows in Prairie Creek at the time of discharge in order to meet in-stream water quality objectives and minimize fluctuations in receiving water metal concentrations. A discharge schedule has been developed based on a detailed water balance to guide how water will be stored, managed and treated seasonally. The schedule will be adjusted based on the magnitude of mine flows that actually occur, as well as the magnitude of flows in Prairie Creek.

Effluent discharge in the NWT is regulated by a Water Licence that specifies end-of-pipe concentrations, and in some cases, volume restrictions. During the EA and permitting period, CZN demonstrated that this approach would be impractical for the Prairie Creek Project because of the variable quality and flow rate of the combined effluent stream seasonally, as well as the substantial difference in seasonal creek flows. CZN developed a variable load discharge (VLD) approach whereby the parameter loads in effluent are varied according to the creek flow rate in order to consistently achieve downstream water quality objectives. The site Water Licence includes the downstream objectives as a compliance point, rather than the usual end-of-pipe concentration approach.

For the construction period prior to mill operation, mine water discharge will be regulated by fixed end-of-pipe concentrations. For operations, the Water Licence includes fixed Effluent Quality Criteria as a temporary measure of regulating discharge in the early phases of the project while VLD parameters are being further developed, and before adopting VLD as the main method for discharge.

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 Prairie Creek. This exfiltration system will promote mixing of the effluent with the receiving waters, thus reducing metal concentrations in the ‘dilution zone’.

Site run of that has the potential or contamination (such as stockpile pad run off) will report to a collection sump and pumped to Cell A. The un-contaminated site run off will report to the new water treatment plant if there is a need for treatment, prior to discharge to the environment.

18.3 Medical facilities

The location of the Prairie Creek site and the possibility of air access being interrupted by adverse weather will require that more enhanced medical facilities be provided and staffed on site than would be the case at a less remote site. Minimum standards are set by regulations, but CZN will access the experience of similarly-situated mines.

Medically-trained staffs are listed in the organization chart and will comprise a Health, Safety and Training Superintendent on rotation with a Health, Safety and Training Supervisor, and one full-time paramedic with no other duties; the warehouse persons will be first aid trained. These employees will also be responsible for emergency response.

CZN will seek to enter into mine rescue mutual aid agreements with such other mines as may be operating in the area.

18.4 Telecommunications

In order to support the number of personnel expected to be accommodated at Prairie Creek, upgrades to the existing telecommunications infrastructure will be needed to allow for effective and reliable emergency, recreational, and administrative use.

Two technologies are commonly used for providing internet and phone service to remote sites: satellite and microwave. Microwave technology relies on repeater installations that may be problematic to install due to steep terrain and permitting and environmental considerations in the surrounding area. Satellite technology is therefore preferred as it relies only on line-of-sight to an orbiting geostationary satellite; CZN has had satisfactory experience with this technology.

Phone and internet equipment to be installed at Prairie Creek for a managed network, including modems, routers, phone equipment and satellite dish, will provide a throughput of 50 Mbit/sec downstream and 20 Mbit/sec upstream, with 950 GB of allowed traffic per month.

The targeted monthly service availability is rated at 99.97%; handheld satellite phones will be available as back-up. An additional satellite link using the existing C-Band satellite dish may also be set up for redundant fail-over communications.

18.5 Administration building

The Administration Building is an existing two storey steel clad building that includes the Mine dry, a warehouse, offices and training rooms as shown in Figure 18.6. This building has been maintained as the site operations base and will retain this function. The building will need some basic maintenance and refurbishing to bring it up to current standards including roof repairs, window replacement and repairs to plumbing fixtures.

Figure 18.6 The two storey steel clad Administration Building at the Prairie Creek site

18.6 Warehousing

Supplies and spare parts are currently stored in several different small buildings around the site and are in generally poor condition. A new 30 m x 15 m fabric-structure warehouse (heated) will be erected on a compacted gravel pad adjacent to the existing administration building. The existing cold storage building will undergo basic maintenance and refurbishment and will continue to be used.

18.7 Workshops

The existing heavy-equipment workshop will undergo basic maintenance & refurbishment and will be used for the maintenance of both surface and underground mobile equipment (see Figure 18.7) and to refurbish and maintain process plant components.

Figure 18.7 Interior of the workshop at the Prairie Creek site

18.8 Air strip

The site is serviced by a 1,000 m gravel airstrip approximately 1 km from the camp and is registered with Navigation Canada as CBH4. The airstrip is beside Prairie Creek at the bottom of a narrow, sinuous canyon with obstructed approaches. Passenger aircraft up to DHC-7 size can use the strip; this does not limit crew movements for the forecast employee numbers. The current maximum size of freight aircraft capable of using the strip, however, is a DHC-5 Buffalo; the site does not permit a sufficient runway extension to accommodate a bigger and more economical freight aircraft, such as a Hercules.

Presently a visual approach is mandatory and the tops of the surrounding mountains must be clear of cloud to permit safe operations. Access may be interrupted in poor weather conditions. Beacons and additional navigation aids may be added to further facilitate safety and more extended operation.

Figure 18.8 is a photograph showing the Prairie Creek air strip looking south.

Figure 18.8 The 1,000 m gravel airstrip (CBH4) at Prairie Creek site

18.9 Fuel storage

Four 1.7 million litre diesel fuel tanks exist on the site, as shown below, complete with dispensing equipment, with a combined capacity of 6.8 million litres, all within an engineered clay-lined berm containment system. The nearest (white) tank in Figure 18.9 is presently in service.

Figure 18.9 Diesel tank farm at Prairie Creek within a clay-lined berm impoundment structure

An inspection by Roosdahl Engineering Enterprises in 2011 showed that minor repairs are needed to restore all four fuel tanks to serviceable condition and the fuel farm containment system meets the required Environment Canada regulations. Based on the API 653 tank inspections conducted on 20 September 2008, the structural integrity of the diesel fuel bulk storage and dispensing facilities was considered to be good and suitable for continued operation, with routine inspection and maintenance for the next 19 to 20 years with the approval of the Authority Having Jurisdiction (AHJ). The next inspection by the AHJ is scheduled for 2018 to confirm the corrosion rate before the inspection interval is extended to 20 years.

As the current CZN operating plan is for access by all season road, one tank will suffice for ongoing site fuel storage needs and another will be re-purposed for the fire water tank as mentioned in Section 18.2. During construction, it is expected that an additional two tanks will be utilized.

18.10 Sewage treatment

The existing Sewage Treatment Plant (STP) is a secondary-level, extended aeration treatment plant as shown in Figure 18.10; the plant will be reactivated.

Figure 18.10 Sewage Treatment Plant

Sewage treatment in the plant is based on aerobic biological digestion of the sewage with the addition of air. The sewage is kept in an aerated tank for 24 hours during which oxidization of the solids takes place. After the solids settle, the effluent is pumped out and irradiated with a UV system. Alum will also be added to control phosphate concentrations. The effluent will be pumped to the Water Storage Pond Cell B. Settled solids will be returned to the aeration tank if needed.

Sewage will be piped within each building and pumped to the STP from strategically located lift stations through force mains in the utilidors. Any sewage generated in outlying areas will be collected in local holding tanks and removed by means of a tanker truck for treatment in the STP.

The treatment of the raw sewage is based on a biological oxygen demand (BOD) of 220 to 300 mg/L. The flow rate per person per day of 300 litres is estimated to have a loading of 220 to 300 mg/L of total suspended solids (TSS). The design parameters for treated effluent quality are BOD : <20 mg/L, and TSS: <20 mg/L.

The combined discharge to the environment will include the following limits: ammonia 1.5 ppm as N, nitrate 6 ppm as N, total phosphorous 0.15 ppm. In addition to this there will be receiving water targets downstream.

18.11 Garbage incineration

Suitably trained members of the site work force will collect garbage from bins at the work sites and deal with it as follows:

- Food waste — Incinerate, ash to waste rock pile
- Combustible scrap — Incinerate, ash to waste rock pile
- Non-combustible scrap — Bundle and back haul to Fort Nelson for recycling or sale as scrap
- Hazardous waste — Stored in designated containers, back haul to Fort Nelson for disposal

The incinerator will be located near the kitchen to facilitate the transfer of the main source of waste for incineration. Combustible wastes from other locations will be transported by truck.

The incinerator will also generate energy which will be used to supplement the camp heating system.

18.12 Electrical system

The original owner intended to provide 2,400 volt site power by means of four Bessemer-Cooper diesel generators, which are currently installed in part of the mill building. These have been deemed inefficient and beyond reasonable repair and will be replaced, as proposed by the Northwest Territories Power Corporation with four (n-1 configuration) 2.77 Megawatt low-speed diesel / LNG dual-blend generators to provide the required 10.8 MW (at 4,160 volts) of installed power. The new generators will be located where existing diesel generators are installed to allow for reuse of the building, cabling and trays where possible. The existing, installed and operating power generation capability on site, totalling 1.075 MW, will be used to supply essential power to process and to supplement the power system at camp and administration area.

The new plant will maximize heat recovery from the coolant circuits and from the generator exhaust by means of glycol loops. This heat will be used to heat the process plant buildings as discussed in Section 18.16.

Some of the existing electrical cabling and switchgear does not conform to current standards or has deteriorated due to weathering and will be replaced.

The electrical system design includes the following:

- New electrical equipment. Where possible, some existing electrical equipment (such as MCC, transformers, motors, local start / stop stations) may be reused after testing and evaluation. The electrical equipment will be located where existing equipment is to maximize the reuse of cable and tray.
- Existing electrical rooms will be reused.
- Existing cabling and trays will be reused where possible and supplemented with new materials. Some additional existing electrical materials will be reused after testing to confirm the condition.

A portion of the existing overhead line from the mill building to the administration and camp will be demolished and re-routed to make room for the ROM pad. A new line will be installed to supply the camp and new water treatment plant to account for the higher electrical load and an extension of the line will be installed to feed the water pond reclaim pumps. The existing overhead line to the tank farm will be extended to the catchment pond to supply power to the exfiltration equipment.

Temporary construction power for the mine, process plant and surface infrastructure will be provided by diesel generators, on surface near the 883 L portal. Permanent power during operations will be provided by the new generators when they become available.

Electric power will be used for underground fans, pumps, electric-hydraulic jumbos, longhole drills, skid-mounted mobile air compressors and local permanent lighting. CZN plans to use electric power for battery-powered scooptrams to muck the majority of ore and waste, but envisages starting operations with contractor-supplied diesel equipment.

The camp will be installed with auto-start emergency power supply.

Load-sharing and load-shedding protocols will be a part of powerhouse operating procedures.

Table 18.1 shows the estimated life-of-mine average connected and running loads.

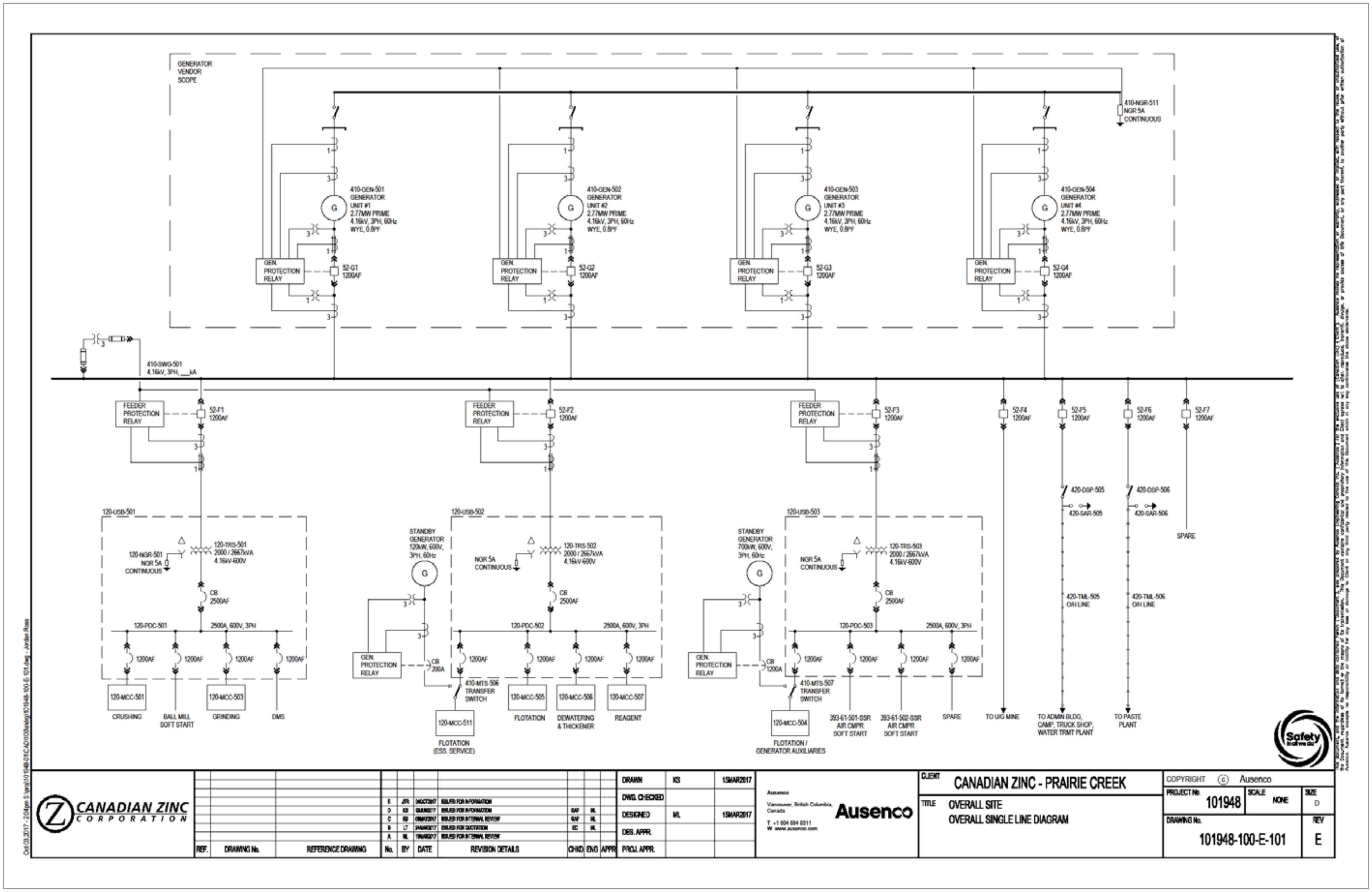
Figure 18.11 is single-line electrical drawing for the site.

Table 18.1 LOM power demand

Area	Connected load (kW)	Running load (kW)
Mine		
Ventilation	1,570	854
Dewatering	2,393	593
Mobile electric-hydraulic eqpt	945	242
Infrastructure	100	40
Sub-total mine	**5,235**	**1,978**
Mill		
Mill building	130	91
Crushing	381	290
DMS plant	334	236
Grinding	796	491
Lead-copper flotation	636	459
Lead oxide flotation	286	218
Zinc sulphide flotation	252	192
Dewatering	264	201
Concentrate handling	15	11
Paste / backfill plant	686	481
Reagent handling	76	58
Metallurgical laboratory	11	8
Tailings thickening	45	34
Instrument air	447	170
Fresh water	147	38
Process water	265	119
Sub-total mill	**4,771**	**3,097**
Plant		
Power generation service	112	85
Mill services (lighting, heat tracing, misc. power)	855	599
Water treatment	236	166
Raw sewage	15	11
Administration building	150	105
Camp	1,100	500
Sub-total plant	**2,468**	**1,466**
Total site	**12,474**	**6,541**

“Connected Load” is the total of all electrical devices connected to a power supply.
“Running Load” is the long-term average power draw.

Figure 18.11 Overall site single line electrical drawing



18.13 LNG power generation

Generators used to supply the site operating power needs will be capable of using a mixture of liquefied natural gas (LNG) and diesel as the fuel source. As indicated in the Northwest Territories Power Corporation's proposal, the generators could normally utilize an LNG/Diesel mixture ratio of 9:1 but the generator units could also be adjusted to run on mostly diesel if LNG supply runs low and there is adequate existing diesel storage available.

Initially the generators will operate on diesel only. The LNG fuel storage system will then be constructed and will include four LNG storage tanks, a LNG vaporizer, a LNG unloading pump and a LNG transfer pump located within a bunded area. LNG storage for 10 days of mine operations and diesel fuel available for a minimum of 20 days is envisaged. The LNG fuel storage could be located close to the mill or in the proximity of the existing tank farm depending on configuration of delivery trucks.

18.14 Plant control

Instrumentation and control technology has made major advances since the Mine was originally built, offering significant improvements in economy and efficiency. An entirely new instrumentation and control system will be designed and installed.

18.15 Fire detection and suppression systems

The fire protection system will be based upon the National Building Code of Canada Codes, as well as specific National Fire Prevention Association (NFPA) standards.

The fire protection system will consist of new equipment and materials and will make use of existing hardware where possible upon further testing and inspection.

The distribution network will be maintained under a constant pressure with a jockey pump, and will be looped and sectionalized to minimize loss of fire protection during maintenance. Where run outside buildings, fire water piping will run above ground and be heat traced and insulated.

Yard hydrants will be limited to the fuel storage tank area. Wall hydrants will be used in lieu of yard hydrants, and these will be located on the outside walls of the buildings in heated cabinets.

Fire protection within buildings will include standpipe systems, sprinkler systems and portable fire extinguishers. Standpipe systems will be provided in structures that exceed 14 m in height and additionally where required by Code, local authorities or the Insurance Underwriter.

Sprinklers will be provided in the following locations (or to protect the following items):

- Truck shops
- Assay laboratory
- Over hydraulic or lube packs that contain more than 454 litres of fluid
- Lube storage rooms
- Any conveyor belts that are within tunnels or other enclosed spaces which would be hazardous to fight manually.
- Transformers
- Heated warehouse
- Cold Storage

Camp modules will be purchased with fire detection; fire rated walls and will utilize separation as a means of fire protection. Handheld extinguishers will be located throughout the buildings.

Fire protection of the generators will be provided by a water mist system. Gas detection will be provided to detect high levels of LNG gas within the generator room.

18.16 Heating, Ventilation, and Air Conditioning (HVAC)

The heating system at Prairie Creek will involve several sources of heat, including:

- Heat recovered from the on-site generators – using circulated hot glycol as the heat transfer medium.
- Latent heat from drive motors in the concentrator, such as air compressors and mills.
- Propane gas from on-site storage tanks.
- LNG from on-site storage tanks

The primary source for heating the following areas will be high grade heat recovery from the power generators:

- Mill Building
- Lead Oxide Building
- DMS Plant
- Tailings Paste Plant Building
- Active Tailing Stockpile Building
- Concentrate Storage Building

The primary source for heating the following areas will be propane gas or LNG:

- Administration Building
- Camp
- Workshop
- Assay Lab
- Heated Warehouse
- Mine Water Treatment Plant

The generators will include a heat recovery module on each generator skid which will transfer heat from the jacket water and the exhaust boiler through a plate and frame heat exchanger. The generators will also include a liquid to air radiator, which will be capable of removing all of the heat that is generated by the generators when there is no requirement for any heat recovery.

A hydraulically separated glycol system will be connected to the heat exchangers and be pumped around the plant as a high grade primary loop. The hydraulic separation ensures that any issues with either circuit will not impact the other circuit. This loop will include an expansion tank, a waste glycol tank, a clean glycol fill tank, a make-up glycol pump, and primary glycol pumps.

The glycol will be a factory mixture comprising 60% ethylene glycol and 40% demineralized water, and the distribution piping located outside buildings will be insulated to minimize heat loss.

Secondary high-grade heating loops will be connected to the primary loop to serve the buildings that are outside the Mill Building.

“Process” type buildings will be heated in winter to achieve a minimum indoor air temperature of +5° C at the design outdoor air temperature of -47° C. Glycol unit heaters will provide heating to the perimeter of buildings and air handling units with glycol coils will provide either make-up air or ventilation air.

“Occupied” buildings will be heated in winter to achieve an indoor air temperature of no less than +18° C at the design outdoor air temperature.

The propane system will comprise a storage tank with a minimum capacity of 7 days, a vaporizer and a primary pressure reduction valve (PRV) to distribute the propane gas to the end users at 10 psig. Secondary PRVs will be located at each building.

The heating of mine air and the ventilation systems for the underground mine are discussed in Section 16.

18.17 Mine Water Treatment Plant

The Mine Water Treatment Plant (WTP) will treat excess water in Cell B of the WSP prior to discharge to the environment (refer to Figure 18.5).

The design of the treatment plant was based upon test work by SGS-CEMI; the primary conclusions of the test work indicated that mine water treated with hydrated lime to a pH>9 will be sufficient to meet the effluent quality.

The treated water will then be pumped to a reactor clarifier for the addition of flocculant to aid solids settling. (Space has been reserved in the layout for additional treatment trains if the mine water treatment rate increases and additional equipment is required).

The capacity of the mine water treatment plant is 75 litres per second based on the best estimate of probable inflows underground and with addition of a contingency. However, the plant will be modular and can be expanded to increase this capacity if necessary, which would be sufficient to manage the projected upper-bound of possible inflows. Inflows will be monitored during the early years of mine development, allowing the model to be calibrated and any necessary treatment circuit changes to be anticipated.

CZN's current mine plan envisages pre-drainage of mining areas so as to discharge ground water direct to surface as non-contact water, avoiding contamination with metals, sludge, oil, and ammonia residues. This will minimize demand on the WTP.

18.18 Explosives

Refer to Section 16.7.

18.19 Paste Plant

Refer to Section 16.5

18.20 Mine services – compressed air and communications

Refer to Section 16.12 and 16.16.

18.21 Dewatering

Refer to Section 16.11.

18.22 Underground refuge station and emergency egress

Refer to Section 16.10 and 16.6.4.

18.23 Surface mobile equipment

CZN has a fleet of mobile equipment on site (refer to Figure 18.13), a portion of which, upon refurbishing, would be capable of supporting operating requirements.

Figure 18.12 Surface mobile equipment at the Prairie Creek site

The mobile equipment fleet required for the operation and maintenance of all the surface facility areas (including the roadways and airstrip) is listed in Table 18.2.

Table 18.2 Surface mobile equipment required

Equipment	Specifications	Qty
D6T dozer	Refurbished equipment	1
Forklift for maintenance	New equipment DP50N1 – 5 t capacity	1
Skid steer loader (Bobcat)	New equipment	1
Mobile rock breaker for ROM	New equipment CAT 304.5E2 XTC - rock breaker (hammer) mounted on CAT excavator	1
Front end loader for ROM	Refurbished equipment CAT 950/960	1
Front end loader for waste rock	Refurbished equipment CAT 950/960	1
Front end loader for concentrate	New equipment CAT 950/960	1
Front end loader for paste plant / dms plant	New equipment CAT 950/960	1
Ambulance 4x4	New equipment	1
Fire truck	New equipment	1
Reach stacker for concentrate containers @ site	New equipment Konecranes SMV 2115 TB3	2
Fuel tanker / lube day truck	New equipment	1
Waste rock dump truck	Refurbished equipment 20 t	2
Grader	Refurbished equipment	2
Pick-up trucks	Refurbished equipment	2
Mini-bus	Refurbished equipment	1
Mobile crane - pick and carry	New equipment AT20-3 Terex model, 20 t capacity	1
Telescopic handler (telehandler)	Refurbished equipment 3 t, 11m lift	1
Mechanics truck	New equipment	1

18.24 Water storage pond

The water storage structure will be the key facility of CZN's water management plan. Three types of water streams are to be managed at the Prairie Creek Mine site: run-off from snowmelt and rainfall, water from the mine, and water from the mill. Runoff management will be essentially the same as current operations using installed components; ditches along with routing all site contact runoff into a water storage pond (the catchment pond). Currently, water from the pond discharges to Harrison Creek through a culvert with a gate. During operations, this culvert will be retained for possible emergency use, but normal discharge to Prairie Creek will be through an exfiltration trench, the details are discussed below.

The site is protected from flooding associated with Prairie Creek by a berm creating one side of the WSP and a flood protection berm that protects the mine, which is connected to the pond berm. Both structures are armoured with rip-rap to prevent erosion during large flood events. These structures are inspected annually by a geotechnical engineer, and were re-assessed recently by a hydraulic engineer and were confirmed to be of suitable design to withstand the probable maximum flood.

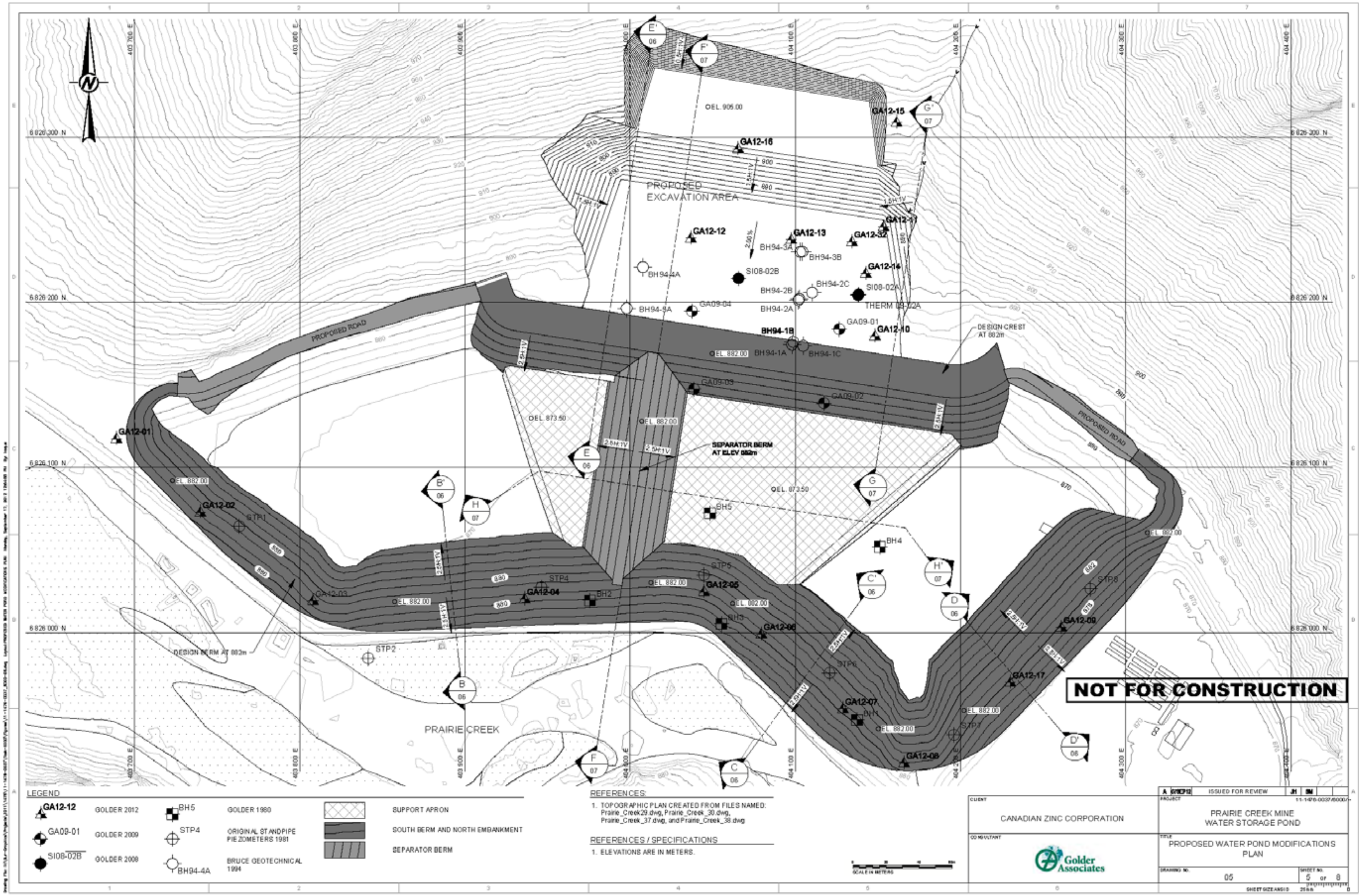
The large pond located northwest of the plant and offices (see Figure 18.13) was originally intended for the disposal of tailings, although none were placed as the mill was not commissioned. Soon after construction, a section of the pond's back-slope slumped, most likely due to a combination of permafrost thaw and slope movement along a weak zone in the underlying in-situ clay layer. Recently slope inclinometers show the slope is creeping at a rate of several millimetres per year. In addition, there have been a number of small failures along the inside of the pond berm due to improper compaction, materials being placed while frozen, poor design and construction of the interior slope and subsequent freeze thaw cycles. Although the failures have affected the engineered interior slopes of the pond, the impoundment can be remediated to provide a stable water impoundment facility.

All contact water streams will be sent to the pond. In addition, the pond will supply water to the mill for processing, and to the water treatment plant (WTP) for treatment and discharge to the environment. In addition, the ROM and stock pile areas will be lined and drain to a collection sump to collect and discharge water to WSP. There will also be a new unlined ditch close to camp site to collect and discharge surface run-off from those areas into the catchment pond.

The plan is to convert the pond into two cells by the construction of a divider berm as shown in Figure 18.14. The divider berm will also buttress the slumped section of the back-slope along with other additional remediation measures based on both the geotechnical field investigation, laboratory program, and the engineering stabilization program discussed in more detail below.

Figure 18.13 Tailings impoundment facility - to be converted into a Water Storage Pond

Figure 18.14 Proposed water storage pond layout



There is some uncertainty regarding the magnitude of mine flows that will occur. For this reason, CZN has endeavoured to maximize storage capacity in the WSP. CZN recently commissioned an increase in the existing storage capacity of the WSP, along with the remediation program. Total storage capacity will be increased from 590,000 m^3 (original design) to 666,000 m^3 and live storage from 410,000 m^3 (original design) to 426,000 m^3 (refer to Table 18.3).

Table 18.3 Storage capacity of water storage pond

Cell	Live storage volume (m^3)	Total storage volume (m^3)
Cell A	158,000	264,000
Cell B	268,000	402,000
Total	426,000	666,000

As described in Section 18.2.3, process water from the mill will be sent to Cell A, and all mine and runoff contact water to Cell B. CZN produced a Water Quality and Effluent Management report during the permitting process which contains relevant background information on water management during operations, including WSP water balances for various mine drainage scenarios and the associated water treatment and discharge schemes.

The proposed remediation program to convert the pond into the storage water pond is as follows:

- Preparation of the base on pond to an elevation of 870 m (AMSL), which will be the foundation for the separation berm and stabilization apron.
- Construction of the separation berm with 2:1 (H:V) slopes to elevation 882 m and stabilization aprons of variable thickness to stabilize the back-slope movement.
- Construction of an upstream containment berm raise with 2.5:1 (H:V) slopes to stabilize the interior of the old berm to an elevation of 882 m.
- Excavation of the northern slope east of the separation berm removing any significant overburden load acting on the in-situ clay.
- Installation of a reinforced polypropylene liner in both cells of the pond.
- Construction of a diversion channel along the back of the WSP lined with geomembrane to prevent further saturation of the northern slope along with an energy dissipator structure at the outlet.

Stability analyses of the pond remediation program of the back-slope and the containment berm were performed in accordance with dam safety guidelines for the Northwest Territories. The configuration of the storage water pond utilized limit 2D equilibrium analysis using SLOPE/W from Geostudio 2007. Material properties for the pond are based on a geotechnical field investigation (including test pits, boreholes, and dynamic cone penetrometer programs) and laboratory testing program. The results of the stability analyses for the remediation program for the WSP showed factors of safety that exceeded the prescribed requirements (1.5 static and 1.1 pseudo-static) and therefore the feasibility design is considered stable. However, the northern slope still needs to be monitored to determine if the remediation measurements above completely arrest the creep or if additional remediation needs to be performed during the mine life. There may be opportunities during construction to utilize silty gravel and sandy gravel for construction materials to partially offload the over burden within the zone of movement, further aiding in the stabilization of this zone.

18.25 Waste rock facility

A new waste rock disposal facility (WRF) will be constructed to store approximately 3.2 Mt of combined development waste rock and dense media separation float rock, along with 35,000 m^3 of solid waste. The solid waste is expected to consist of non-hazardous wood, metal, paper, etc. The site of the new waste rock facility is located in a ravine approximately 1 km north of the plant site (refer to Figure 18.15). The capacity of the WRF is open-ended since the volume of waste can migrate upslope within the ravine as more waste is added.

A geotechnical program was performed to determine subsurface conditions for the development of the WRF, which included 4 boreholes and 8 test pits. Toward the bottom of the WRF and collection pond area, the overburden consists of sand and gravel and the upper area (above 975 m) consists of thin overburden overlying shale bedrock. There was no evidence of permafrost in this area. The overburden between 925 and 975 m (AMSL) will be removed down to bedrock for the installation of both the seepage collection pond and the toe section of the WRF. Above 990 m, within the WRF footprint, only the organic and deleterious materials need to be removed along with any permafrost, if found. This will provide suitable foundations for both the WRF and the seepage collection pond.

The toe of the WRF will be at an elevation of 937 m and proceed up the valley with an overall external slope of 2:1 (H:V) to an elevation of 1,047 m. The exterior slope will have a bench mid-slope to capture contact surface runoff above it and divert it to the collection pond. The rock portion of the WRF will be developed from the bottom up to provide a stable platform and placed in 6 to 7 m high benches (refer to Figure 18.16).

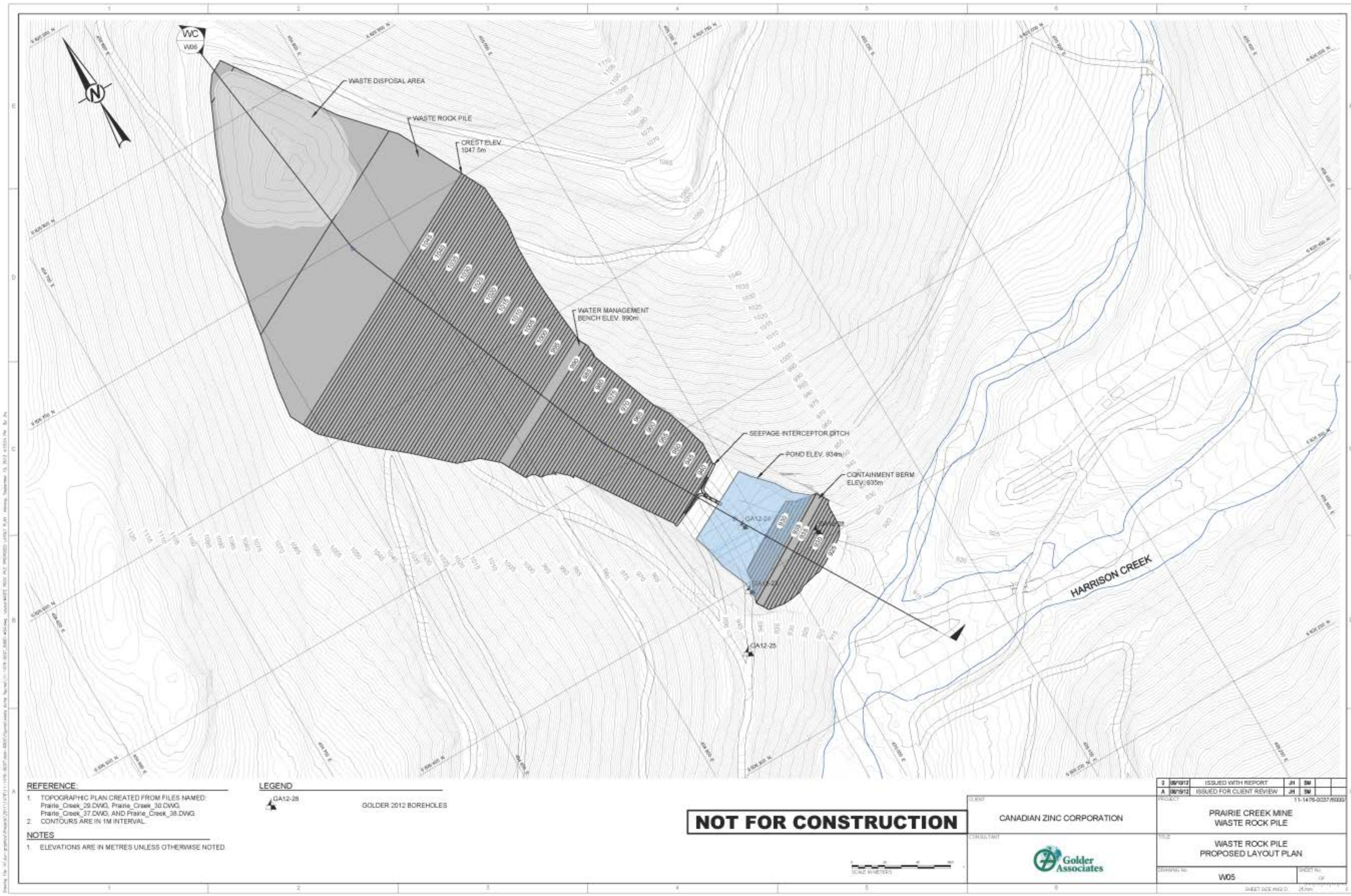
The WRF has been designed with a water management system. A seepage collection pond will be constructed below the WRF. The pond will be constructed by placing a berm (2H:1V slopes) across the small valley and the pond will be lined with geomembrane along the upstream side. The collection pond was designed for the 100-year return runoff flow, based on the ultimate WRF footprint. The pond has been sized to store 7,000 m^3 of contact water. Contact water collected in this pond will report to the WSP via a pipeline along the haul road to a collection sump adjacent to the ROM stockpile. The seepage collection includes an emergency spillway located on the southwest abutment. At the toe of the WRF a shallow interception ditch will be installed into the bedrock to capture seepage from the base of the WRF. Water collected in this structure will also report to the seepage collection pond. To control and separate surface non-contact and contact water, diversion channels around the east and west sides of the waste rock facility will carry non-contact water to Harrison Creek. The diversion channels are designed for a 100-year storm event.

Slope stability analyses were performed for both the WRF and the seepage collection pond berm. Limit 2-D equilibrium analyses were performed using SLOPE/W from Geostudio 2007. The results of the analyses showed factors of safety that exceeded the prescribed requirements (1.5 static and 1.1 pseudo-static) and therefore the feasibility design is considered stable.

A haulage road connecting the WRF to the existing road at the mill, a distance of approximately 950 m, has been designed to reduce existing road gradients to less than 11%.

Figure 18.15 Proposed site of Waste Rock Pile Storage facility

Figure 18.16 Proposed Waste Rock Storage facility layout



18.26 Laydown areas

The Prairie Creek Mine occupies a constricted site in the bottom of the Prairie Creek valley. The available space will have to be carefully managed for use as a laydown area for inbound freight and outbound material.

A portion of land at the North-West corner of the site will be used for this purpose. A container storage area on compacted gravel will also be provided on the northern side of the air strip to provide short-term storage of concentrate and consumables containers as they are readied for transport to and from site.

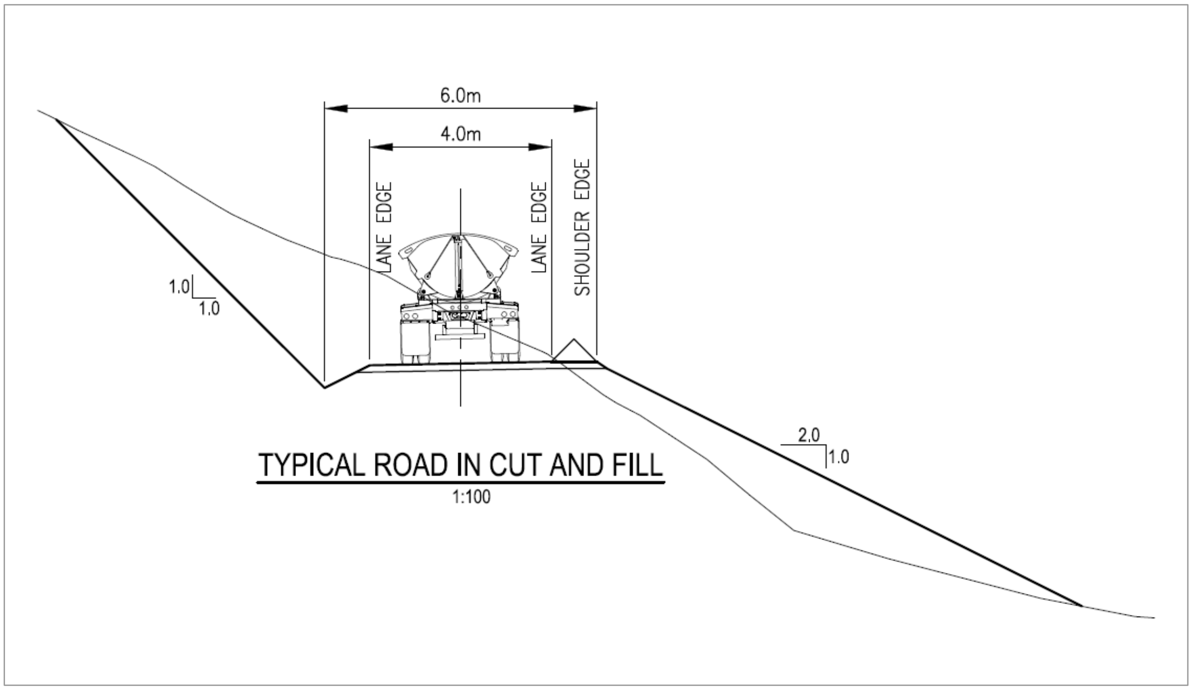
18.27 Transportation

18.27.1 Site roads

The existing site contains a number of roads connecting the various facilities, the longest being a 1 km length to the airstrip. These roads are in good shape and will require on-going maintenance for use throughout the year. A road to the new WRF will be constructed to accommodate the waste rock haul trucks as shown in Figure 18.17.

The newly designed haul road from the mill to the WRF consists of a 4 m wide gravel road with a maximum sustained grade of less than 11%. The vehicular traffic load is expected in the order of 40 to 50 tonne gross vehicle weight (GVW).

Figure 18.17 Proposed waste rock haul road typical cross section



18.27.2 All season access road

The existing mine plant was hauled in over a winter road and the mine was fully permitted to operate on a winter road basis in 1982. When CZN obtained new operating permits in 2013, this included a winter road permit. Access limited to winter roads, however, would cause some constraining issues, namely:

- Large working capital needed to support concentrate sales once a year.
- Need to forecast materials and equipment 18-24 months in advance.
- Risk of late freeze / early thaw, compromising both inbound and outbound freight campaigns. This is especially significant in view of the airstrip being too short for large freight aircraft.
- Competition with other winter road users for crews and equipment.

Accordingly, CZN applied for permits to construct an all season road, substantially following the winter road alignment as shown in Figure 18.18.

Figure 18.18 Proposed route of the all season access road into Prairie Creek Mine

The all season access road will be purpose built to support the transportation needs of the mine to transport consumables in and salable product out of the site.

The total length of the road is 179.5 km commencing from Highway 7 (approximately 131 km north from BC border) through to the mine site. The road has been designed to accommodate conventional commercial transport traffic, based on the low traffic volumes of up to 20 trucks per day. The design of the road incorporates localized cut and fill construction and the use of local borrow.

The road traverses mixed low valley bottom terrain, higher elevation plateaus, and rugged mountainous valleys and passes. The route includes a total of 19 major watercourse crossings which will include nine bridge structures, nine large / multi large culverts, and a combination ice bridge / barge crossing over the Liard River. The crossings have been designed to meet the life cycle of the mine with the consideration of the total number of loads and the ability to plan preventative maintenance operations.

Construction is projected to start in 2018 and to continue through to the fall of 2020. The construction program has been developed to have an operational road for the winter of 2019 / 2020 with the final all season road operational for summer 2020.

Allnorth has developed a proposed execution plan for the construction of the all season road, integrated with the construction of winter roads and taking seasonal conditions into account, comprising the following main elements:

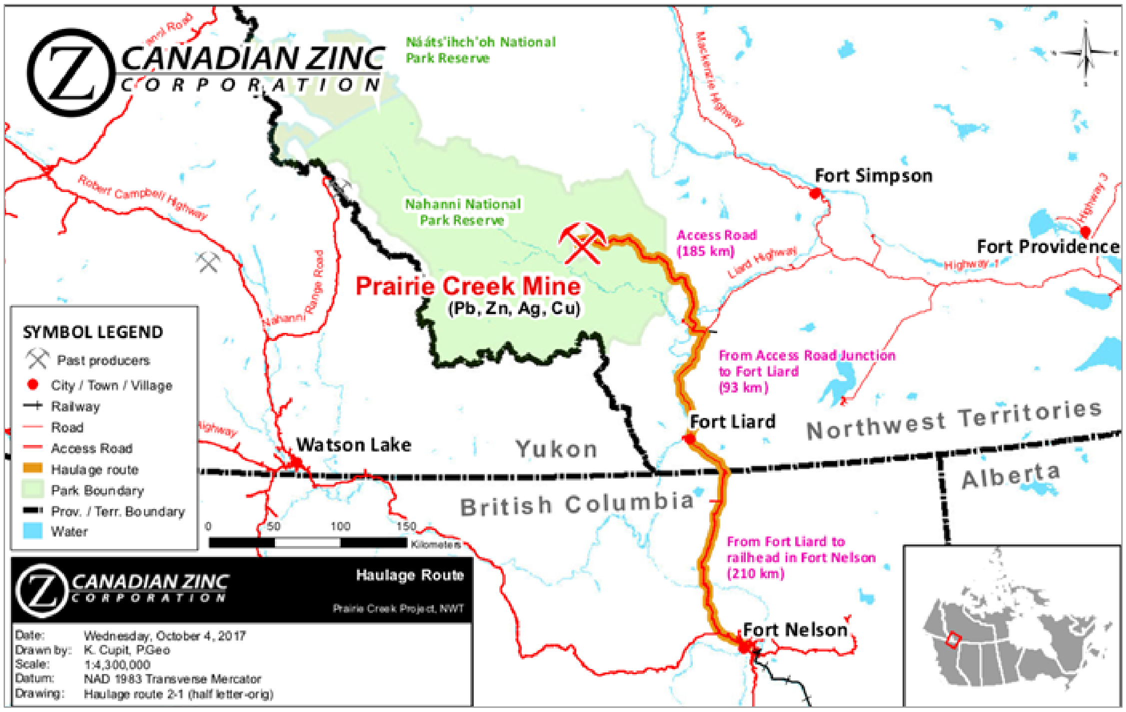
- Final field investigations in 2018 for the purposes of both engineering and permitting;
- A winter road in early 2018 to support a logistics campaign for site work in the summer of 2018;
- A winter road in early 2019 to support construction work on the mine site during the summer of 2019;
- The majority of the construction work scheduled for 2019;
- Limited road access throughout the Project in mid-2019 with air support for consumables and personnel;

- Sub-grade completion in late 2019 with final surfacing in early 2020;
- Road fully operational in July, 2020 to support a start of mine production in August 2020.

Allnorth has developed a Class 3 cost estimate for this work.

Figure 18.19 shows the total proposed trucking route from the Mine to Fort Nelson.

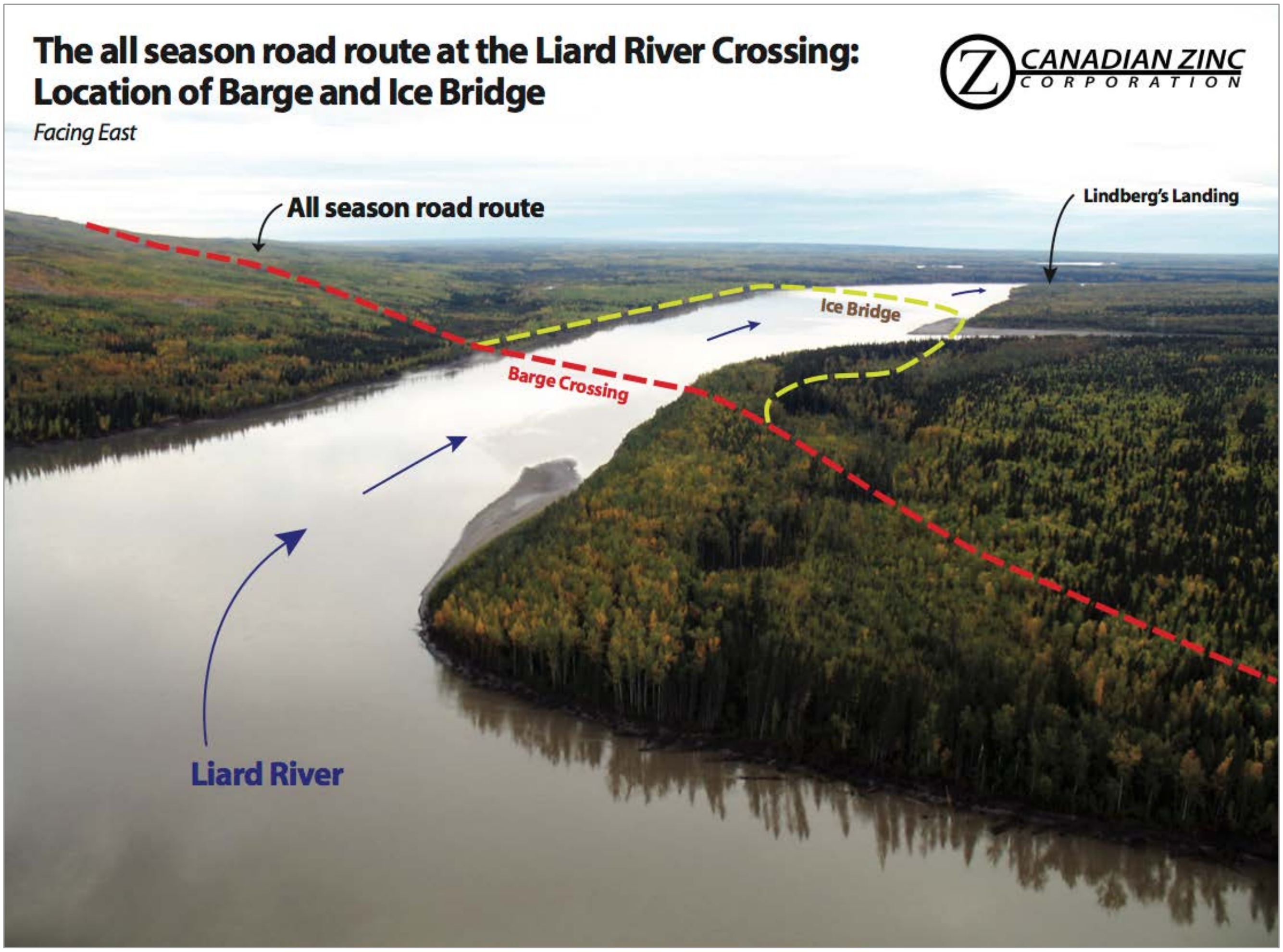
Figure 18.19 Proposed trucking route Prairie Creek Mine to Fort Nelson



The Liard River Crossing

The Liard River Crossing is approximately 850 m wide and represents a significant challenge and limitation to the operational transportation infrastructure. The 2016 PFS Update envisioned a barge crossing in summer and an ice bridge crossing in winter, as shown in Figure 18.20 below. There remained, however, periods of 2-3 months during freeze-up and break-up when neither would be available.

Figure 18.20 Proposed route of the all season access road at the Liard River Crossing



A barge craft referred to as a Hoverbarge offers a potential solution to this problem. A Hoverbarge can float on water like a barge, but can also float on an air cushion similar to a Hovercraft, allowing it to slide over unobstructed, flat, unpaved surfaces, including water, level ground, beaches, marshland, sand and gravel or ice. Hoverbarges may be self-propelled like a Hovercraft but can also be cable-driven across rivers and lakes. The following photograph shows a Hoverbarge used on the Yukon River in the mid-1970s.

Figure 18.21 Example of a Hoverbarge used on the Yukon River Crossing

The use of a Hoverbarge will potentially shorten the all season road closure from the current two two-month periods. The use of a Hoverbarge offers the following benefits to Prairie Creek mine:

- Reduction of site storage from two months to a prudent minimum of 7-10 days for all materials and commodities used on the site and, also, for outbound concentrate;
- Reduction of site building space needed for storage;
- Improvement of cash flow;
- Increased use of LNG owing to reduction in costly site storage;
- Elimination of cost of building and maintaining an ice bridge on the Liard River;
- Elimination of a barge usable for part of the year only.

Costs incurred by the use of a Hoverbarge will include:

- Capital and operating cost of the Hoverbarge itself;
- Installation of a haulage cable across the Liard River;
- Availability of barge master and crew training.

Initial correspondence with Hoverbarge Freight Ltd., based out of the United Kingdom, has resulted in a comprehensive proposal to utilize this technology at the Liard River Crossing. Hoverbarge Freight Ltd. offered a year-round charter rate plus fuel and crew consisting of a Barge Master, mechanic and two deck hands. The initial price given for the Hoverbarge is for a unit capable of a payload from 80 t or 150 t depending on traffic. Lease provisions are incorporated into this 2017 FS to accommodate associated capital costs and, in addition, operating costs, maintenance and shipping/assembly costs are applied.

A further factor potentially inhibiting logistics operations is the poor condition of some sections NWT Highway 7 during the spring thaw period. CZN is presently in discussions with the GNWT to implement a plan allowing un-interrupted road use with provisions for maintenance and road upgrades/repairs so as to reduce the effect of this condition.

CZN plans a staging area at the Liard Crossing area consisting of a cleared level area for truck operations to exchange trailers, perform maintenance and temporarily park loads. Highway 7 in the Northwest Territories crosses into British Columbia where it becomes Highway 77 to the Alaska Highway near Fort Nelson. The highway is entirely chip sealed in British Columbia and also for 21 km north of the border in the Northwest Territories.

18.28 Logistics

The Prairie Creek Mine location will require significant logistics management for the efficient movement of people to and from the site, supplies inbound, and concentrates outbound.

18.28.1 Operations movement

The site workforce will work on a regular fly-in-fly-out rotation (two weeks on and two weeks off). Fort Nelson and Yellowknife are the nearest communities served by scheduled air services with large aircraft. Additional movements per year may be anticipated for visitors and senior management.

CZN will charter flights from one or both of Fort Nelson and Yellowknife, depending on the availability and reliability of scheduled services. Yellowknife is farther from the site than Fort Nelson, but is more easily accessible for a workforce that may be recruited from all parts of Canada and offers a wider range of charter aircraft.

The Prairie Creek airstrip is usable by DHC-7 aircraft, which will suffice for all foreseeable passenger movement needs. Flights will be restricted to day visual flight rules conditions; some weather delays may be anticipated.

18.28.2 Inbound freight

Construction of the project will be serviced by winter roads for initially moving construction freight to the site and to mobilize equipment for the construction of the all season road. The all season road will be used to move freight over the life of the mine during operations. With the assistance of a Hoverbarge to facilitate movements across the Liard River and the removal of seasonal load limitations on any existing roadways through negotiations with the Northwest Territories Department of Highways, the maximum interruption anticipated is one (1) week per year.

The bulk of the inbound freight will consist of food, diesel fuel, equipment, spare parts, mining consumables, mill reagents, and general supplies to support the operation.

18.28.3 Outbound concentrate

The Pre-Feasibility Study envisaged moving approximately 120,000 t of concentrates per year out over a winter road. This entailed a large storage building at the site and two transfer facilities, one on each side of the Liard River. More detailed analysis showed that moving a year’s concentrate production to market during the two to three months of winter road availability would impose significant costs for transfer facilities and rehandling as well as substantial funds tied up in concentrate inventory.

The current logistics system envisages moving concentrate over an all season road to market in 20 tonne bulk containers comprising the following general operating areas and transport route segments:

- Bulk handling and buffer storage (one day) of lead and zinc concentrate at the mine site.
- Loading of open top bulk containers (with lids) and temporary storage at the mine site for both loaded and empty containers.
- Transfer of loaded containers to B-train container haul trucks (incl. removal of empties).
- Trucking from the mine site to the Liard Highway 7 (NWT) on an all season resource road, which includes crossing the Liard River using either a Hoverbarge or a bridge structure as potential future development.
- Trucking from Liard Highway 7 (NWT) to Fort Nelson, British Columbia (BC) where containers will be staged at an intermodal storage yard facility.
- Transfer of containers onto flat deck rail cars for movement to the Kinder Morgan terminal facility, North Vancouver, BC, where the product will be discharged to storage using a rotary spreader unit; empty containers will be returned to site by train along the same route.
- Concentrate will be subsequently loaded from the Kinder Morgan bulk storage areas onto bulk carrier vessels for overseas export.

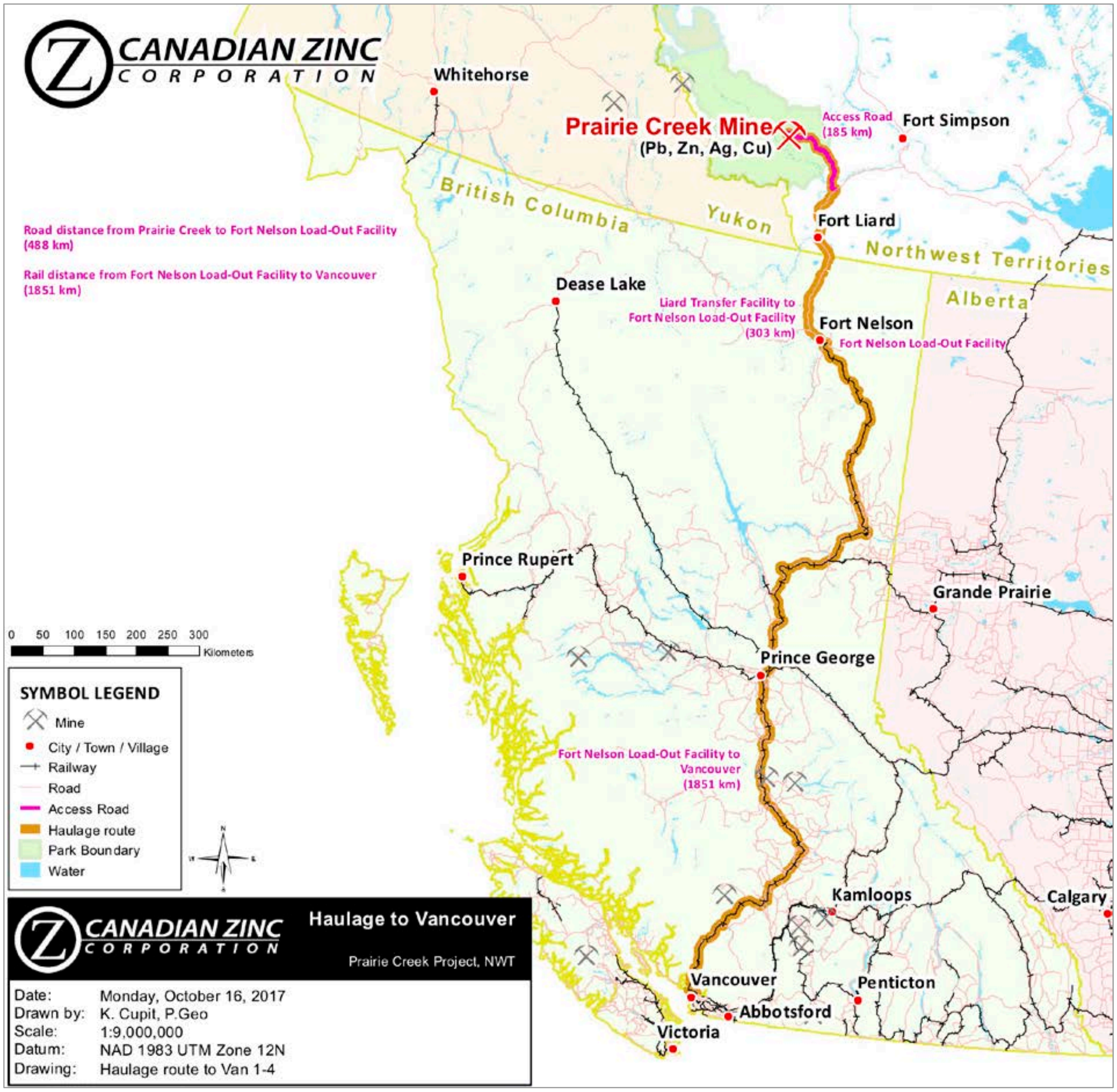
A transportation plan was developed, which investigated alternative transport routes and ports of export. It was determined that the most viable route was to truck to Fort Nelson and rail to Vancouver. Figure 18.22 shows a typical 20 tonne bulk container. Figure 18.23 shows a container haulage train-truck.

Figure 18.22 Typical 20 tonne bulk container for metal concentrate

Figure 18.23 Reverse Super B-Train truck for container haulage to Fort Nelson

The concentrate is transported by containers to the Port of Vancouver over the proposed route shown in Figure 18.24 and empty containers will follow the similar route back to the mine site. There are some synergies that can be exploited in the use of these empty concentrate containers for movement of consumables for the mine, such as grinding media and reagents. When in use, LNG will be moved using LNG tank containers with an ISO footprint, hauled from Fort Nelson to the mine using the same container truck fleet. The proposed truck fleet for hauling concentrate will provide sufficient space for up to a 5,000 litre fuel tank behind the cab or on the bridge of the trailer. With this configuration the mine can be re-supplied with fuel up to a capacity of 65,000 litres/day during hauling operations.

Figure 18.24 Total haulage route to Vancouver



CZN engaged Vancouver-based shipping brokers, logistics consultants and port operators to produce plans and cost estimates for receiving and shipping concentrate onward from the CN Rail terminus in Vancouver to prospective ports and overseas destinations, and alternatively to smelters in Canada. CZN has estimated transportation costs accordingly.

19 Market studies and contracts

19.1 Market studies

The Prairie Creek mineral deposit contains various minerals of economic interest (Pb, Zn, Ag, Cu in particular) which, depending on the milling process, have the potential for production of a number of different combinations and volumes of mineral concentrates. The Prairie Creek mill has the capability of producing various combinations of concentrates of lead sulphide, lead oxide, zinc sulphide, zinc oxide, and copper sulphide from the defined Mineral Resource.

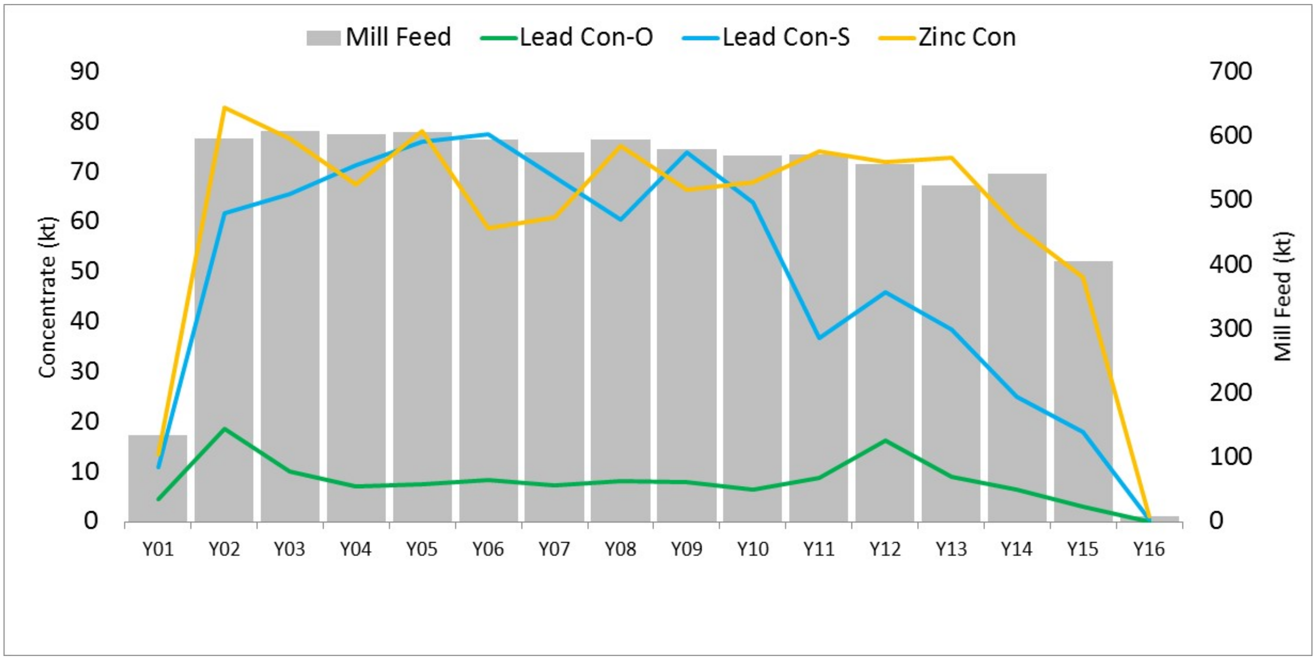
Bench scale locked cycle tests produced samples and specifications of each of five different concentrates which included copper sulphide, lead sulphide, lead oxide, zinc sulphide, and zinc oxide concentrates. In 2014, the Company engaged Cliveden Trading AG, an international metal trading and advisory company based in Switzerland (Cliveden), to formulate a comprehensive market assessment and marketing strategy for all possible concentrate production and advise CZN on commercial and marketing matters. Cliveden identified multiple smelter destinations and developed a marketing strategy. It was determined, with regards to expected near-term market conditions, to proceed with a milling operation that produces a lead sulphide and lead oxide or blended lead concentrate and a zinc sulphide concentrate, with the option of further customizing milling facilities to produce other types of concentrate if deemed economic in the future.

19.1.1 Concentrate volumes and quality

The September 2017 Mineral Reserve estimation resulted in a moderate increase in LOM Mineral Reserves to 8.1 Mt compared to the 7.6 Mt in the PFS; however at increased mine production rates, from 1,350 tpd to 1,600 tpd. This had a net effect of reducing the LOM from 17 to 15 years (16 years of ore production, 15 years of concentrate production).

Figure 19.1 shows the projected volumes of the three concentrates that will be produced at the Prairie Creek Mine over the LOM.

Figure 19.1 Predicted LOM concentrate production



While the mill will utilize separate flotation circuits for lead sulphide and lead oxide, at this time it is anticipated that the two types of lead concentrate will be blended before shipping. If however, markets indicate a preference, separate lead oxide and lead sulphide concentrates could be produced.

The yearly variations in the volume of concentrate production shown in Figure 19.1 are caused by a number of factors that include:

- Source of mill feed: MQV, STK and SMS ores have differing grades and compositions.
- Grinding / flotation capacity of the mill.
- Latter years are running lower on defined MQV Mineral Reserves.

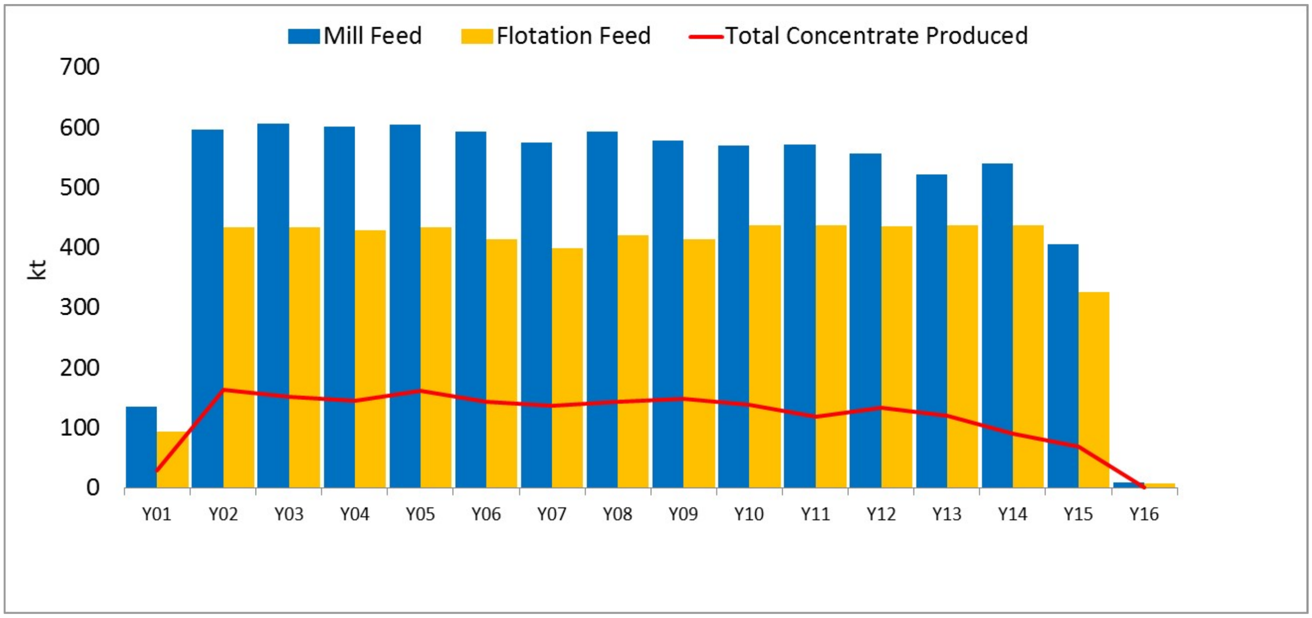
In the initial years, sufficient mining areas will be accessed and developed to reach targeted steady state production in Production Year 3. The present mine plan is designed for the first nine years to access mainly the high grade MQV-type ore. Years 10-14 are mining a blend of MQV and SMS (lower lead content), which is followed by the blending in subsequent years of STK-type ore, which is also lower in lead content.

It is also noted in Figure 19.1 that the volume of concentrate production drops off after Year 13, which is a reflection of depleted Mineral Reserves. However, it is anticipated that, during future operations, further exploration and definition drilling of the presently defined 5.0 million tonne MQV Inferred Resource would be undertaken in tandem, with a view to potentially maintaining the MQV Mineral Reserves ahead of mining; and that this effort may extend the LOM and prolong the higher level of concentrate volumes.

While the Prairie Creek mill has the capacity to crush 1,750 tpd, the actual mining and processing rates, at this time, are determined by the volume the mill can process in its currently planned configuration. The envisaged mill grinding and flotation circuits are restricted in capacity by the maximum capacity of the grinding (ball mill) and maximum saturation of grade in the flotation tanks and dewatering processes. This limitation dictates the amount of ore and the grade that the mill can accept from the mine and, therefore, places a constraint on the mine plan. In addition, the mine has a greater oxide component in its higher elevation levels and this is taken into account in feeding a minor proportion of oxides to the mill in order to maximize recoveries associated with sulphide type ores. The planned addition of a Dense Media Plant on the front end of the milling circuit provides capacity to mine at rates of 1,600 tpd which, as indicated, is predetermined by the mill capacity. The mill process (grinding / flotation) nominal capacity of 1,200 tpd post-DMS determines the volume of concentrate produced, which results in a LOM average of 32% of the flotation feed reporting to the concentrates or 24% of the mill feed.

Figure 19.2 shows the currently planned mill feed, milling rates and concentrate production over the LOM.

Figure 19.2 Mill feed, milling rates, and concentrate production



The Prairie Creek concentrates, to varying degrees, are expected to contain levels of impurities that will have some smelter penalty implications. The main impurity of interest is mercury.

Table 19.1 shows the anticipated average concentrate specifications.

Table 19.1 Average concentrate specifications

Projected average LOM concentrate specifications - Prairie Creek Mine				
Element	**Zinc sulphide**	**Lead sulphide**	**Lead oxide**	**Blended lead (S+Ox)**
Lead (Pb) %	3	67	48	65
Zinc (Zn) %	59	7.5	7	7.4
Copper (Cu) %	0.2	1.5	1.2	1.4
Iron (Fe) %	2	1	0.2	
Cobalt (Co) %	<0.02	<0.02	<0.02	<0.02
Arsenic (As) %	0.02	0.2	0.2	0.2
Antimony (Sb) %	0.07	0.6	0.5	0.6
Tin (Sn) %	<0.01	<0.002	<0.002	<0.002
Sulphur (S) %	~30	15	<5	12
Carbon (C, total) %	<1	0.6	4	1
Germanium (Ge) g/t	<100	<10	<10	<10
Selenium (Se) g/t	<30	<30	<30	<30
Fluorine (F) %	<0.01	~0.005	~0.005	~0.005
Chlorine (Cl) g/t	100	<500	<500	<500
Titanium (Ti) g/t	<100	~50	<300	<300
Calcium (Ca) %	1	<0.5	1	<1
Magnesium (Mg) %	<0.5	<0.6	1	<1
Manganese (Mn) g/t	<100	<50	~200	~75
Aluminum Oxide (Al2O3) %	<0.5	<0.5	<0.5	<0.5
Silica (SiO2) %	1	<2	~10	<2
Bismuth (Bi) g/t	<400	<400	<400	<400
Cadmium (Cd) %	0.3	~0.04	~0.03	~0.04
Mercury (Hg) g/t1	1700	200	100	200
Gold (Au) g/t	<1	<1	<1	<1
Silver (Ag) g/t	136	877	~500	824

[1] See discussion on mercury specifications below.

Variations in the grade of concentrate and grade of deleterious elements occur throughout the projected LOM, depending on the composition of the feed product from the mine and the stockpile. Table 19.2 shows this variability over time for the four types of concentrates produced in relation to the key major and minor elements. The variations reflect the type of ore being mined (MQV, STK, SMS), its associated grades, and the final delivery of feed to the mill.

Table 19.2 Predicted variations in major and minor elements in concentrates over LOM

Concentrate production year	Y01	Y02	Y03	Y04	Y05	Y06	Y07	Y08	Y09	Y10	Y11	Y12	Y13	Y14	Y15	Y16
Mill feed (kt)	135.5	596.9	607.3	602.6	606.3	594.7	575.5	594.6	578.9	570.5	571.9	557.1	523.4	541.0	406.1	9.1
Flotation feed (kt after DMS)	94.1	435.0	435.3	428.9	434.3	414.2	400.5	421.4	415.0	437.6	438.0	436.6	438.0	437.9	326.2	7.3
Lead sulphide grade (kt)	10.9	61.8	65.5	71.4	76.0	77.4	69.0	60.5	73.8	64.0	36.7	45.9	38.6	24.9	17.9	0.4
Pb, %	65.00	65.00	65.00	65.00	65.00	65.00	65.00	65.00	64.92	64.00	62.66	61.50	58.42	62.28	64.38	65.00
Zn, %	7.02	7.02	7.02	7.02	7.02	7.02	7.02	7.02	7.04	7.06	7.30	7.46	7.74	7.06	7.12	7.02
Ag, g/t	1,069.3	1,009.5	897.9	811.6	854.8	771.4	762.6	872.6	787.3	736.7	954.4	925.2	663.3	976.8	1,078.4	1,286.2
Cu, %	1.79	1.91	1.63	1.47	1.46	1.29	1.25	1.51	1.24	1.09	1.54	1.49	0.87	1.62	1.95	2.85
As, %	0.12	0.20	0.21	0.25	0.24	0.22	0.23	0.25	0.15	0.17	0.18	0.14	0.14	0.17	0.28	0.39
Sb, %	0.56	0.72	0.70	0.67	0.65	0.60	0.59	0.65	0.51	0.50	0.44	0.32	0.22	0.59	0.91	1.09
Hg, %	0.01	0.02	0.01	0.02	0.02	0.02	0.02	0.02	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01
Cd, %	0.03	0.04	0.04	0.04	0.04	0.04	0.04	0.04	0.04	0.04	0.03	0.03	0.02	0.03	0.04	0.04
Lead oxide grade (kt)	4.4	18.5	10.1	7.1	7.4	8.4	7.3	8.1	8.0	6.5	8.8	16.3	9.1	6.5	2.9	0.1
Pb, %	48.00	48.00	48.00	48.00	48.00	48.00	48.00	48.00	48.00	48.00	48.00	48.00	48.00	48.00	48.00	48.00
Zn, %	7.70	7.70	7.70	7.70	7.70	7.70	7.70	7.70	7.70	7.70	7.70	7.70	7.70	7.70	7.70	7.70
Ag, g/t	430.6	452.7	521.0	574.3	608.7	521.1	526.5	540.1	536.2	539.0	490.3	455.6	445.4	516.4	607.5	684.9
Cu, %	0.58	0.83	1.38	1.93	1.94	1.55	1.55	1.49	1.50	1.41	0.85	0.55	0.49	0.83	1.61	2.13
As, %	0.04	0.09	0.18	0.33	0.32	0.26	0.29	0.25	0.18	0.22	0.10	0.06	0.09	0.09	0.23	0.30
Sb, %	0.18	0.31	0.59	0.88	0.87	0.72	0.73	0.64	0.62	0.65	0.24	0.12	0.12	0.31	0.75	0.81
Hg, %	0.01	0.01	0.01	0.02	0.01	0.02	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.00	0.01	0.01
Cd, %	0.02	0.02	0.02	0.03	0.03	0.04	0.04	0.03	0.03	0.03	0.02	0.02	0.02	0.02	0.03	0.03
Lead blended grade (S+Ox)(kt)	15.4	80.3	75.5	78.5	83.5	85.9	76.3	68.5	81.8	70.5	45.5	62.2	47.6	31.4	20.8	0.4
Pb, %	60.08	61.08	62.74	63.46	63.48	63.33	63.37	63.00	63.26	62.52	59.82	57.97	56.43	59.32	62.08	62.42
Zn, %	7.22	7.18	7.11	7.08	7.08	7.09	7.09	7.10	7.10	7.12	7.38	7.53	7.73	7.19	7.20	7.12
Ag, g/t	884.5	881.2	847.8	790.2	832.9	746.8	740.1	833.6	762.7	718.5	864.5	802.3	621.8	881.3	1,012.2	1,194.9
Cu, %	1.44	1.66	1.59	1.51	1.50	1.32	1.28	1.51	1.27	1.12	1.41	1.24	0.80	1.45	1.90	2.74
As, %	0.10	0.17	0.21	0.25	0.25	0.22	0.23	0.25	0.15	0.17	0.17	0.12	0.13	0.15	0.27	0.38
Sb, %	0.45	0.62	0.68	0.69	0.67	0.61	0.60	0.65	0.52	0.52	0.40	0.26	0.20	0.53	0.89	1.05
Hg, %	0.01	0.01	0.01	0.02	0.02	0.02	0.02	0.02	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01
Cd, %	0.03	0.03	0.04	0.04	0.04	0.04	0.04	0.04	0.04	0.04	0.03	0.02	0.02	0.03	0.04	0.04
Zinc sulphide grade (kt)	13.5	82.9	76.6	67.4	78.2	58.7	60.8	75.1	66.5	68.0	74.0	72.0	72.8	58.9	48.8	1.1
Pb, %	3.89	4.03	3.93	4.00	4.07	4.17	4.03	3.80	4.04	3.32	2.69	2.78	2.03	2.58	2.73	2.75
Zn, %	59.00	59.00	59.00	59.00	59.00	59.00	59.00	59.00	58.95	58.56	58.41	58.04	57.50	58.53	58.89	59.00
Ag, g/t	151.2	164.8	152.2	139.1	153.3	128.6	130.7	142.0	135.4	121.3	117.1	136.4	113.1	130.3	117.9	135.8
Cu, %	0.19	0.22	0.20	0.23	0.21	0.24	0.18	0.16	0.19	0.13	0.09	0.14	0.07	0.07	0.06	0.10
As, %	0.01	0.02	0.03	0.04	0.03	0.04	0.03	0.03	0.02	0.02	0.01	0.01	0.01	0.01	0.01	0.01
Sb, %	0.06	0.08	0.09	0.10	0.09	0.11	0.09	0.07	0.08	0.06	0.02	0.03	0.02	0.02	0.03	0.04
Hg, %	0.08	0.13	0.12	0.20	0.15	0.18	0.15	0.15	0.09	0.11	0.08	0.08	0.09	0.06	0.06	0.06
Cd, %	0.27	0.31	0.32	0.37	0.34	0.37	0.37	0.33	0.31	0.32	0.24	0.20	0.17	0.26	0.32	0.33

Canadian Zinc has signed MOUs with Korea Zinc and Boliden for the sale of zinc and lead concentrates. The MOUs set out the intentions of Canadian Zinc and each of Korea Zinc and Boliden to enter into concentrate sales agreements for the concentrates to be produced from the Prairie Creek Mine on the general terms set out in the MOUs, including commercial terms, which are to be kept confidential.

Canadian Zinc has entered into a Memorandum of Understanding with Korea Zinc for the sale of approximately 20,000 to 30,000 wet metric tonnes of zinc sulphide concentrates, approximately 15,000 to 20,000 wet metric tonnes of lead sulphide concentrates and approximately 5,000 tonnes of lead oxide concentrates, per year, for a minimum period of five years from the date of start-up of the Prairie Creek Mine, with exact annual quantities to be mutually agreed.

Canadian Zinc has also entered into a Memorandum of Understanding with Boliden for the sale of a minimum of 20,000 dry metric tonnes and up to 40,000 dry metric tonnes of zinc sulphide concentrates, per year, for a minimum of five years from the start of regular deliveries, with exact annual quantities to be mutually agreed.

These offtake arrangements with two of the pre-eminent smelting companies in the world, confirm the marketability of Prairie Creek's zinc and lead concentrates**.** The sale agreements will account for all of the planned production of zinc concentrate and about half of the planned production of lead concentrate for the first five years of operation at the Prairie Creek Mine.

Korea Zinc is a Korea-based, globally known, general non-ferrous metal smelting company principally engaged in the manufacture and marketing of non-ferrous metal products. Korea Zinc owns and operates zinc smelters in Korea and Australia and a lead smelter in Korea, and its metal products consist of zinc products, including zinc slab ingots, zinc alloy jumbo blocks, zinc anode ingots and zinc die-casting ingots, and precious metal products, including gold and silver products. Korea Zinc is leading the world resource market in terms of zinc production and market share.

Boliden is a metals company with a commitment to sustainable development. The company's core competence is in the fields of exploration, mining, smelting and metals recycling. Boliden is one of the world's largest zinc mining and smelting companies, owning and operating zinc smelters in Norway and Finland, and is Europe's leading copper and nickel company. Boliden is the world's fifth largest zinc mining company and the sixth largest zinc smelting company. Boliden is also the eleventh largest lead mining company in the world and a medium-sized lead smelting company in terms of primary lead.

The sales agreements will provide that treatment charges be set annually at the annual benchmark treatment charges and scales, as agreed between major smelters and major miners. Payables, penalties and quotational periods will be negotiated in good faith annually during the fourth quarter of the preceding year, including industry standard penalties based on indicative terms and agreed limits specified in each MOU.

It is expected that shipments will be made from the Port of Vancouver with the exact shipping schedule and lot sizes in each delivery to be mutually agreed within the project's shipping season.

19.1.2 Contracts

As part of Canadian Zinc's ongoing socio-economic commitment to the region and local stakeholders, it is the Company's preference to award contracts to local businesses as well as other businesses in the Dehcho region. Canadian Zinc will continue to focus on opportunities for the residents and businesses of the Northwest Territories to participate in the Project through existing impact benefits agreements and socio-economic agreements in further support of sustainable economic development of the region.

No contractual arrangements for concentrate trucking, port usage, shipping, smelting or refining currently exist. Although the Company has signed MOUs with each of Korea Zinc and Boliden for the sale of concentrates, no binding contractual arrangements have been made for the sale of zinc or lead concentrates at this time.

19.2 Metal prices

There were no independent market studies completed specifically for the Prairie Creek Project in support of the 2017 Feasibility Study.

A number of sources were reviewed to determine a market consensus for the long-term prices of lead, zinc and silver, including Energy & Metals Consensus Forecasts published quarterly by Consensus Economics Inc., an independent macroeconomic survey firm that prepares compilations of metal prices using more than 40 analysts' commodity price forecasts.

Consensus price forecasts obtained from Consensus Economics Inc. (Consensus) were reviewed to assist in arriving at pricing for the 2017 Feasibility Study Base Case. The Consensus Long-term Price Forecasts (2022-2026) for each of the metals to be produced at the Prairie Creek Mine are: lead US$2,040/tonne (US$0.93/lb); zinc US$2,530/tonne (US$1.15/lb) and silver US$19.46/oz. Other market forecast information, current metal prices, rolling three-year averages, and prices used in recent similar mining project studies were also considered for the Prairie Creek economic evaluation.

As discussed below, the long-term outlook for both lead and zinc is considered very positive. These fundamentals, along with the continuingly favourable Canadian-US dollar exchange rate, which improves metal prices expressed in Canadian dollars, impact positively on the Prairie Creek Project. Long-term base case metal prices used in the financial analysis in the 2017 Feasibility Study are shown in Table 19.3. The selected prices used for zinc and silver of US$1.10/lb and US$19.00/oz respectively are lower than the long term Consensus prices of US$1.15/lb and US$19.46/oz, respectively. In the case of lead, the price used of US$1.00/lb is 7% higher than the Consensus long term price of US$0.93/lb but 15% lower than recent current spot prices. China is indicated to be the leading producer and consumer of lead, accounting for 50% of the world's mined production, and 42% of the refined lead production and usage in 2016. Published data shows that mine lead output has been falling in China as more stringent environmental controls are applied and many closures have been enforced, while to date in 2017, the lead price is reported to have outperformed many metals, reaching a recent 6-year high of US$1.15 per lb. as LME warehouse stock levels have fallen. Based on the evaluation of Consensus forecasts and the other factors referenced above, the long-term metal prices selected for the base case economics of the Prairie Creek Project are considered to be reasonable.

Table 19.3 Prices of zinc, lead, and silver used in financial analysis

Metal	Forecast
Zinc ($/lb)	$1.10
Lead ($/lb)	$1.00
Silver ($/oz)	$19.00

Prices above are in U.S. dollars

19.2.1 Macroeconomic outlook

The following discussion on the long-term outlook for metal prices draws heavily on published public information extracted from various sources including: International Lead and Zinc Study Group; World Bank Group; Consensus Economics Inc.; CRU; Wood Mackenzie; Metals Bulletin Research; Boliden 2016 Annual Report; Teck Corporation 2016 Annual Report. None of this information was prepared for, or on behalf of, Canadian Zinc Corporation.

Economic growth rates, particularly industrial production growth forecasts, impact the demand and behaviour of metals prices. Both worldwide real GDP growth and worldwide industrial production growth were down significantly in 2015 and 2016, as compared to the previous two years. However, in 2017 to date, GDP growth has recovered in most global economies and is expected to continue into 2018. US growth has been soft thus far in 2017 but is picking up pace and is expected to strengthen well into 2018. China, led by manufacturing is continuing strong growth and while the percentage growth has dropped as compared to earlier double digit growth numbers, the historic growth volumetrically has made the current growth numbers relatively large. Political risks in Europe appeared to have calmed and most economists have positively upgraded growth forecasts. Japan has exceeded expectations to date despite lingering fears about weak inflation.

The Economist Base Metals Index (which weights the prices of major base metals according to their share of world trade in 2005) and the Reserve Bank of Australia (RBA) Non-Rural Commodity Price Index (which weights the prices of major minerals according to their share of Australian exports in 2015/16) both show sustained decline to the end of 2015 with subsequent moderate growth since, but nowhere yet near the highs experienced in 2011. Most financial analysts appear to agree that moderate growth will continue to be experienced over subsequent years.

The prices of lead and zinc, expressed in US dollars, were generally higher in 2016 than in 2015. Metal prices began rising in 2016 and have continued to climb during 2017. Precious metal prices were also generally higher in 2016, reflecting a stronger US$ and low rates of inflation. The price of silver was negatively affected during the current year by a stronger US$ and by expectations of continued low global rates of inflation.

The US$ strengthened significantly against the majority of the world's currencies in the first part of 2016, largely due to the relatively stronger performance by the US economy and expectations of US interest rate increases. GDP growth in the latter part of 2016 and into 2017 has been softer than expected which resulted in a slight weakening of the US$. The US economy, as of late, has picked up pace and economists have projected it to strengthen in the latter part of 2017 and into 2018. A strong US$ results in declining local costs in producing countries, measured in US$.

19.2.2 Price of zinc

Currently more than 80% of the zinc produced is used in Asia, primarily in the production of galvanized steel, which accounts for approximately 50% of the metal's current demand. As a result, demand for zinc remains largely tied to the automotive sector and to infrastructure development, particularly in China, India, and other emerging markets in Asia.

After increasing by 3.1% in 2016, the International Lead and Zinc Study Group ("ILZSG") expected global demand for refined zinc metal to rise by 2.6% to 14.3 Mt in 2017, with this demand to be led by continuing growth in China and India and stable production in Europe and the rest of Asia.

The critical issue for zinc and the zinc price is considered to be not demand, but supply. The ILZSG observed a 5.5% drop in global zinc mine production in 2016 during which time the zinc price went from less than $0.70 per lb. in early 2016 to more than $1.15 per lb. by the year end. For 2017, the ILZSG projected global mine production to increase by 6.7% to 13.70 Mt, as operations that underwent further underground development, such as Hindustan Zinc's Rampura Agucha operation in India, ramp up production. China was also expected to increase zinc mine production in 2017. Preliminary figures for the first eight months of 2017 show global zinc mine production up by only 3.9%.

According to Wood Mackenzie, mine closures, which began in earnest in 2013 with the closure of the Brunswick and Perseverance mines, are having an impact on medium term supply and the situation is becoming more critical as closures related to ore depletion have continued. In 2016, the closure of the Century, Lisheen and Mae Sod mines removed more than 1 Mt of zinc from supply. Other mine closures related either to technical issues, low metal prices or strategic decisions, such as Glencore's decision to suspend production from four of its mines in Australia, Peru and Kazakhstan, have further exacerbated the situation. Glencore's decision alone was expected to lower global production by almost 1 Mt in 2016 and 2017. Elsewhere in China, environmental concerns have led to lower than expected production.

The lower mine production is also having an impact on refined zinc metal. At the start of 2017, the IZLSG anticipated a 2.6% rise in refined zinc metal output to 14.1 Mt in 2017; however, more recent figures suggest that, for the first eight months of 2017, refined metal production was marginally lower by 0.1% and that preliminary data show the global market for refined zinc metal was in deficit by 287 kt. In China the lack of mine production has had a significant effect on Chinese smelters. According to the National Bureau of Statistics, refined production of zinc in July 2017 was 476 kt, which represents a year-on-year decline of 5.9% and a 12.8% drop since June 2017. In an effort to obtain more concentrate, spot treatment charges have fallen and concentrate imports to China have risen further indicating a shortage of concentrates for these smelters.

Past and current production cuts, lower mine production and lower production of refined zinc metal have had an impact on global zinc exchange stocks. Since October 2013, LME zinc warehouse stocks levels are reported to have steadily declined from 1.2 Mt to levels around 250 kt. A similar pattern is seen on the Shanghai Exchange where inventory levels have been around 70 kt. Wood Mackenzie recently stated that Exchange stocks had fallen to just eight days of consumption and estimates that, in order to balance the market, 2.1 Mt of new mine capability must be developed by 2022. This assertion takes into account re-starting Glencore's four mines, where production is currently under suspension, by Q2 2018. And while Wood Mackenzie acknowledges that there are projects around the globe that could supply this much needed capacity these projects have suffered from a lack of capital investment and will not be developed in sufficient time to prevent a significant run-up in zinc prices.

The zinc price has been rallying since early 2016 and recently set a new 10-year high in response to declining levels of zinc concentrates and refined zinc metal. Demand for zinc, which in China had compound annual growth rate between 2010 and 2016 of over 10% has moderated and has been projected to average above 2% between 2016 and 2035. These conditions are expected to lead to a number of years of substantial annual zinc deficits, with higher prices anticipated before the market returns to near balance with longer term prices higher than the current levels being experienced.

19.2.3 Price of lead

The price of lead, which is used mainly for lead-acid batteries for cars and other vehicles, shadows that of zinc as the metals are typically co-produced. Lead has relatively low stocks and, as China barely trades the metal, sell-off shocks are rare. Official data shows that mine output has been falling in China as more stringent environmental controls are applied and many closures have been enforced. Furthermore, China continues to be the leading producer and consumer of lead, accounting for 50% of the world's mine production, and 42% of the refined lead production and usage in 2016.

The demand for lead metal continues to be moderately strong. Demand is currently led by increased need for batteries in Asia. The ILZSG is forecasting increased usage in the automotive and industrial battery sectors in the near-term and also continued decline in the world production of refined lead metal.

In early 2017, global demand in the year for refined lead metal was forecast to increase by 2.3% to 11.4 Mt, primarily due to increased usage in the automotive and industrial battery sectors. In China, usage was expected to increase by 4.3% while demand in the United States and in India was expected to increase by 1.5%.

Mine production was expected to increase by 4.3% to 4.9 Mt due to continued growth from China, Greece, Kazakhstan, Mexico, and India, where higher output from Hindustan Zinc's operations are ramping up following completion of underground development at Rampura Agucha.

The ILZSG has forecast a close balance between global refined lead metal supply and demand in 2017, not anticipating much change in inventory over the course of 2017.

In 2017, the lead price is reported to have outperformed many metals, rising from just over US$0.70 per lb. at the beginning of 2017 to a recent 6-year high of US$1.15 per lb. as LME warehouse stock levels have fallen from 360 kt in November 2012 to current levels near 150 kt.

Provisional data reported by the ILZSG indicate that global refined lead metal demand exceeded supply by 119 kt during the first eight months of 2017. Over the same period total reported stocks decreased by 28 kt.

Global lead mine production also increased rising by 6.8% as a consequence of increases in China, India, and Kazakhstan that more than offset a sharp decline in Australia.

Higher mine output in China and India has given rise to a 4.1% increase in global refined lead metal despite a 10.1% drop in refined lead metal output in the United States.

Imports of lead concentrates to China decreased by 4% to 480 kt, largely resulting from UN sanctions that have halted exports of lead ore and concentrates to China from North Korea.

19.2.4 Price of silver

Over 2016, silver prices averaged $17.14 per ounce, up 9% from 2015 but well below the five-year average of $21.29 per ounce. Silver prices reflect both investor sentiment and industrial demand. As a safe-haven asset class, investors seek precious metals as a store of value in uncertain and turbulent macro-economic environments.

Silver also has a strong underpinning from industrial demand, which represents just under half of the total demand for silver. The use of technology, electronics and renewable sources of power is growing in the modern world, and the silver required for these applications is increasing.

On the supply side, there has been little to no growth in silver because many exploration companies have responded to the precious metals' bear market cycle of the last few years by not investing in exploration and new projects. In fact, total physical demand for silver outstripped total physical supply in 2016 for the 4th year in a row while global silver production declined in 2016 for the first time since 2002.

Over the past decade, significant changes have occurred in the consumption pattern of silver. For many years the major uses were photography and silverware but these have been declining and have been surpassed by industrial demand, mainly for electrical and electronic applications. These industrial uses have developed because of the properties of silver, particularly its high electrical conductivity and its ability to solder itself on to other surfaces. Silver is incorporated into a variety of industrial applications and is generally price-insensitive given the small quantities that are used in some applications and its critical contribution to these applications' functionality.

Silver's use in photovoltaics for solar energy rose 34% in 2016 as compared to 2015 due to the increase in global solar panel installations. Moreover, silver's use in this application is reported to have accounted for approximately 14% of total silver industrial demand in 2016, up from 1% a decade ago.

The sell-off in high yielding assets amid global economic uncertainty has increased demand for silver, as investors search for protection against risk. Silver, like gold, has a price set by members of the London Bullion Market Association which is used as a benchmark for over-the-counter trades.

Silver is also a form of money and, like fiat money such as the U.S. currency, it is a store of value and is used as a medium of exchange. Therefore, it is not simply the intrinsic value of silver that determines its price, but also the state of significant global currencies and economies. Gold is often seen to be silver's primary driver, dominating sentiment in the market for precious metals.

19.2.5 Longer term metal price outlook

Despite the heightened uncertainty caused by the US administration and increasing tensions between the US and North Korea the global economy is indicated as showing stability. Most of the major economies are expanding, and last year's concerns that rising interest rates and the scaling back of quantitative easing could undermine growth have proved unfounded. In the near- to medium-term, the outlook for the global economy remains positive. There is potential that growth in a number of countries could gain further momentum over the balance of 2017 and into 2018.

Solid economic performance in a number of regions is positive news for the mining and metals sector. Global growth has been projected to reach 2.8% this year and is forecast at 3% in 2018. Both the US and Eurozone are reported to be growing steadily at around 2.1% in 2017 and this has been anticipated to continue in the EU in 2018. US economic growth has been projected to possibly be stronger in 2018 at around 2.4% on the back of stimulus plans.

Chinese growth is reported to have surprised on the upside this year at 6.8%, boosting demand for commodities and pushed up prices. Many commodity prices are hitting new highs, such as zinc trading at over US$3,000/t for the first time in a decade and aluminum at over US$2,000/t for the first time since 2014.

Metal demand has increased rapidly for a number of years, but the growth rate tapered off in 2015 as a result of a slow-down in economic growth and a lower industrial production growth rate in China. Metal prices at the end of 2015 were relatively close to cost levels for high-cost mines, which, historically, has often proved to be the low point for prices in a weaker economic climate. Since that time metal prices have steadily risen and base metals in particular has recovered despite a strong US dollar.

Global mine production has, for many years, constituted the limitation on the availability of metals. A large number of smelters have been built, primarily in China, and the smelting industry has now been suffering from over-capacity for several years.

Mines have a limited lifespan and supply will consequently decline if new mines are not opened. An increase in mine capacity requires assumptions that future prices will be sufficiently high to motivate investments in new mines, and when metal prices are low, mining company incentives to develop new mines decline and a number of mines with high cash costs are closed or mothballed.

This leads, in time, to limitations on mine production that halt the fall in the price of metals. Treatment charges fall as a result of the growing scarcity of concentrates, and smelters' profitability weakens. This results, eventually, in production cutbacks or smelter closures, which, in turn, results in an improvement in treatment charges. A fall or stagnation in metal supply results, eventually, in rising metal prices.

Metal prices rise when metal demand increases in an economic upturn. After a period of higher metal prices and growing profitability for mines, new decisions are taken on expanding mine capacity.

The increased growth in the demand for base metals and the decreased supply resulted in price escalation of metals in 2016 that continued into 2017. Treatment charges have decreased dramatically as the supply of concentrates has tightened.

The long-term outlook for both lead and zinc is positive. As indicated above, with the Century mine in Australia, the Lisheen mine in Ireland now closed and Glencore's suspension decision, almost 2 billion lbs. of annual zinc production, representing almost 8% of global production, has been removed from supply. Wood Mackenzie is forecasting an increase in global zinc refined metal demand in 2018, exceeding current estimates for global supply, keeping the refined market in deficit and further reducing global stockpiles of zinc metal.

In the medium to long-term, the key issue is whether the zinc mining industry will be able to develop sufficient new mine capacity to offset scheduled mine closures and the incremental increase in global demand.

On the supply side, the depleting nature of ore reserves, difficulties in finding new orebodies, permitting processes, availability of skilled human resources to develop projects, as well as infrastructure constraints, political risk and significant cost inflation may continue to have a moderating effect on the growth in future production for the industry as a whole. Over the longer term, the industrialization of emerging market economies will continue to be a major positive factor in the future demand for commodities. Therefore, the long-term price environment for the metals remains favourable.

(Sources: Published information extracted from: International Lead and Zinc Study Group; World Bank Group; CRU; Wood Mackenzie; Metals Bulletin Research; Boliden 2016 Annual Report; Teck 2016 Annual Report).

20 Environmental studies, permitting, and social or community impact

20.1 Project overview

The Property contains a base metal deposit containing zinc, lead and silver along with significant infrastructure built on surface. As currently planned, the proposed development involves:

- Rehabilitation and additional development of an underground lead zinc operation that will produce in the order of 1,600 tonnes of ore per day.
- Upgrading or replacing existing mine site facilities.
- Construction of a new water treatment plant, paste backfill plant, dense media separation plant and other facilities at the mine site.
- Construction of a waste rock pile in the Harrison Creek valley.
- Re-design of existing water storage pond.
- Re-clearing of the winter road corridor from the mine site to the Liard Highway and re-aligning portions of the road route which would evolve into an all season road.

The Project is located in the southern Mackenzie Mountains in the south-west corner of the Northwest Territories within the traditional territories of the Nahanni Butte Dene Band (NBDB) and the Liidlii Kue First Nation (LKFN) of the Dehcho First Nations. The nearest community is Nahanni Butte, home of the NBDB, located approximately 90 km to the southeast of the Project site. At this time there is no permanent road access to the Project site other than the existing winter road that was used in the 1980’s to move in all supplies, equipment and infrastructure. Other communities within 200 km of the site include Fort Simpson, Fort Liard, and Wrigley.

20.2 Overview of existing information

There is extensive background information available on the regulatory board’s public registry, which includes environmental baseline studies; regulatory reports / documents; socio-economic data; and summaries of existing agreements with communities and stakeholders. Much of this information has been compiled by CZN or has been collected as part of baseline and environmental assessment activities by expert consultants. Some of the baseline information collected at the site dates back to the 1970s.

The Mackenzie Valley Land and Water Boards’ (the permitting Regulator) public registry of Canadian Zinc Corporation’s associated files can be viewed at: www.mvlwb.com.

The Mackenzie Valley Review Board (the regulatory body responsible for Environmental Assessments) public registry of Canadian Zinc Corporation’s associated files can be viewed at: www.reviewboard.ca.

20.2.1 Environmental setting and potential environmental concerns

Acid rock drainage and metal leaching

As part of ongoing baseline investigations, CZN has evaluated the potential for acid rock drainage and metal leaching (ARD/ML) at the mine site. Mesh Environmental Inc. (Mesh) undertook a broad geochemical study in 2005 and 2006, which analyzed mineralized rock samples, tailings and concentrates. 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 host rock units are non-potentially acid generating (non-PAG), due to generally low amounts of contained sulphur and the substantial effective buffering capacity provided by reactive carbonates;
- MQV and SMS mineralization is classified 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 (DMS) rock is non-PAG and contains relatively low sulphur values;

- Flotation tailings are classified as non-PAG and contain sufficient buffering capacity to maintain neutral conditions under laboratory conditions;
- Sulphide concentrates are classified as potentially acid generating due to slightly elevated pyritic sulphur content and very little neutralization capacity; and
- As a result of substantially higher neutralization potential, lead oxide concentrate is classified as having uncertain acid generation potential.

Mesh also evaluated potential metal leaching as part of their study program. Samples were collected from underground seeps and portal discharge and 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:

- Mineralized material and 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, DMS rock could potentially release elevated concentrations of a number of metals of environmental concern such as arsenic, strontium, cadmium, copper, lead, mercury, selenium, gold, and zinc;
- Humidity cell test results indicate that DMS rock leach rates are lower than those of mineralized vein material;
- 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 similar to those for DMS rock material); and
- Dissolved metals are typical for flotation supernatant.

Given that all mine materials tested by Mesh have the potential to leach metals at neutral pH, CZN incorporated a range of management measures into its operations and closure planning.

Water quality

The Project is located within an environmentally sensitive watershed of the South Nahanni River, which is the highlight of Nahanni National Park Reserve. As a result, concerns were raised by regulators, First Nations, and other stakeholders regarding potential impacts to water quality that may be caused by Project construction and operations. Extensive baseline water sampling has been completed throughout the Project area.

Studies referenced in the Project Developer’s Assessment Report (DAR 2010, submitted as part of environmental assessment (EA) EA08-09) indicate that the historical discharge of untreated mine drainage has had no significant impact on downstream water and stream sediment quality, or aquatic life. While this suggests that the aquatic environment of Prairie Creek is not overly sensitive to discharges from the mine, CZN has committed to a detailed water management strategy as part of its operations.

In 2010, CZN commissioned the Saskatchewan Research Council to complete a study of background metal concentrations in Prairie Creek 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 Prairie Creek that would not exceed the proposed objectives when creek flows are in the normal year-round range. However, the study noted the potential for some water quality exceedances during low flow periods (e.g., winter months). As a result, CZN developed a discharge strategy based on monitoring flows in the creek, and determining the contaminant loads that could safely be discharged without causing exceedance of objectives, with water being stored temporarily in the on-site Water Storage Pond.

Following mine closure, it is expected that there will be no drainage from mine portals as the underground workings and access tunnels will be completely backfilled with a paste tailings mix. Some groundwater seepage from the bedrock surrounding the underground workings may occur, with the water containing some metals, mostly from mineralization considered uneconomic and not mined, and to a lesser extent from the backfilled waste mixture. A small quantity of seepage from the covered Waste Rock Pile is also possible.

Conservative predictions for Prairie Creek water quality after mine closure suggest that all metal concentrations will remain within the water quality objectives when creek flows are in the normal range year-round, although if creek flows are abnormally low in winter, zinc concentrations may be similar to those predicted to have occurred before any mine development. Post-mine predictions also indicate higher cadmium and mercury concentrations in Prairie Creek during the winter if creek flows are unusually low. However, CZN noted that the predictions are conservative since natural attenuation effects will, in all probability, reduce concentrations. Cadmium and mercury are not stable in the natural environment and will be attenuated by various natural reactions.

Terrestrial environment

Terrestrial flora and fauna

Wildlife of concern within the Project area include: Dall's sheep; woodland caribou; wood bison; wolverine; and grizzly bear. While potential impacts to mammalian mega fauna from mine operations are expected to be limited and largely avoidable, there are concerns regarding the potential for road use associated 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. Other mitigation measures include posted and enforced speed limits and the management of flight paths for air traffic.

A variety of vegetation types exist across the Project site. No significant impacts on the types of vegetation communities present are expected due to the relatively small area of disturbance that will result from Project construction and operations.

Terrain and stability

No large-scale landslide features are evident near the mine and access road, and the risk of major slope failure appears to be small. Engineered structures associated with the Project have been designed to be stable during earthquakes.

CZN is proposing to re-align sections of the access road to promote safety, reduce human and environmental risks, and accommodate the wishes of the Nahanni Butte Dene Band (i.e., avoiding wetlands and wildlife habitat) and Parks Canada (i.e., avoiding karst features).

Un-authorized use of the access road has the potential to raise human safety and wildlife concerns, and as a result, CZN will deter unauthorized access, and will closely monitor road activity.

Aquatic environment

The Project is located on the eastern side of, and adjacent to, Prairie Creek, approximately 43 km upstream from the creek's confluence with the South Nahanni River.

Bull trout and mountain whitefish are found in Prairie Creek near the mine. No evidence of spawning has been found downstream of the Project site; however, bull trout were found to spawn in Funeral Creek upstream of the project site. Based on the water quality predictions (including toxicity testing), treated effluent discharge via an exfiltration trench is not expected to impact the aquatic environment. The exfiltration trench will be installed below the bed of Prairie Creek and only part-way across the channel from the mine operation.

Protected areas

The Project is located close to (but outside of) the Nahanni National Park Reserve (NNPR). In 2009 NNPR was expanded to surround, but exclude, the Prairie Creek Mine, and the right of access to the Prairie Creek area was protected in an amendment to the Canada National Parks Act. CZN has an existing MOU with Parks Canada regarding mutual cooperation for the operation and development of the Prairie Creek Mine and the management and protection of the NNPR.

Cumulative effects

Given the remote location of the Project, there is currently no nearby development, and it is expected that there will be very little additional activity in the future which could contribute to cumulative effects. Un-authorized use of the access road to the Project could raise human safety and wildlife concerns; however, CZN will deter unauthorized access, and will closely monitor road activity.

20.3 Environmental management plans

The Project will be developed in a manner that prevents or minimizes potential environmental impacts. Permits for project development include a requirement to submit a number of detailed plans and programs that are expected to prevent or mitigate such impacts. For mine operations, documents that will need to be submitted and activities to be undertaken are:

- Engagement Plan
- Final design, Construction drawings WRP
- Final design, Construction drawings Ore Stockpile
- Final design, Construction drawings WSP
- Sample and test WSP backslope for ARD/ML
- Final design, Construction drawings Exfiltration Trench
- Exfiltration Trench construction as-built report
- Final design, Construction drawings Engineered Structures
- Engineered Structures construction as-built report
- Waste Management Plan
- Waste Rock and Ore Storage Monitoring Plan
- Investigate 930 and 970 pile metal loadings
- Contaminant Loading Management Plan
- Tailings and Backfill Management Plan
- Explosives Management Plan
- Assess options to reduce water inflow
- Construction Phase Water Management Plan
- Operations Phase Water Management Plan
- Report on water treatment effluent quality optimization
- Install a flow gauge on Prairie Creek
- Update the Protocol for Real-Time Estimation of Prairie Creek Flows
- Report on upstream Prairie Creek water quality based on 24 samples
- Terms of Reference for Plume Delineation Study
- Results of Plume Delineation Study
- Variable Load Discharge (VLD) Protocol
- AEMP Design Plan
- Spill Contingency Plan
- Failure Modes and Effects Analysis (FMEA)
- Mine Site Contingency Plan
- Closure and Reclamation Plan
- QA/QC Plan for SNP Water Sampling

For the winter road, the following documents will need to be submitted:

- Spill Contingency Plan
- Cat Camp Remediation Plan
- Spill Risk Analysis Plan
- Engagement Plan
- Sediment and Erosion Control Plan
- Road Operations Plan
- Construction, Operation and Maintenance Plan
- Contaminant Loading Management Plan
- Interim Closure and Reclamation Plan
- Waste Management Plan
- Wildlife Mitigation and Monitoring Plan
- Aggregate Site Plan for Cat Camp Aggregate Pit
- Aggregate Site Plan for Polje-West Aggregate Pit
- Avalanche Assessment
- Construction plans: permafrost & geotextile locations, mitigations
- Any further geotechnical studies re alignments, bridges
- Engineering designs for crossings at km 24, 26.4, 36.8, 28.7, and 43
- Final engineering designs
- Construction drawings showing cut and fill locations and amounts.

Most of the information for the above, including draft plans, was generated during the EA process and permitting phase, and will only require updating and / or reformatting for submission.

Additional plans are likely to be required for an all season road, including an Invasive Species Management Plan.

Key items of environmental management are described below.

20.3.1 Tailings and waste rock management

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. A portion of the DMS reject rock from the mill may also be placed underground in the mix. The remaining DMS reject rock, together with waste rock from mine development, will be placed in an engineered Waste Rock Pile (WRP) 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 Prairie Creek floodplain; and
2 The underground workings will be completely backfilled, removing pathways for mine drainage egress.

During operations, seepage from the WRP 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, either by pipeline to the mill or 883 mL Portal, or by borehole to the underground mine workings where the seepage would be managed with mine water.

20.3.2 ARD / ML management plan

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 and DMS rock underground and covering the WRP.

20.3.3 Water management

There are two main sources of water that will need to be managed during mine operations. These are:

- Drainage from the mine
- Process water from the mill

Both water sources will contain metals in varying amounts; although, the process water is expected to contain much higher concentrations of most metals plus residues from flotation chemicals, but to be much lower in volume than the mine drainage.

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 (WSP) for the Project. The WSP will consist of two cells, one for mine water and similar site drainage, and the other for mill process effluent. Up to 50,000 tonnes of flotation tailings may also be placed in the process effluent cell on a temporary basis. The tailings will be reclaimed from the pond at a later date and placed underground.

During operations, the WSP will supply feed water to the mill. Mine water coming into contact with the mine workings will be recycled to stope drills, and the remainder sent to the mill as ‘make-up’ water. The majority of mine water will be drawn from the vein structure up-gradient of the mine workings to avoid metals loadings. This water will be sent to the WSP and treated year-round in a new treatment plant to reduce metal concentrations, as necessary, and then released to Prairie Creek. Excess process water will also be treated in the new plant in a separate circuit; however, process effluent will not be treated and released in January to March (i.e., during low flow periods). As CZN expects to be able to intercept groundwater before it flows into the mine, mine water treatment requirements to meet discharge criteria will be reduced.

Water levels in the WSP will fluctuate seasonally, increasing in the winter as water is accumulated in storage, and decreasing in the summer when water is treated at a higher rate for discharge.

The treatment and release rate of water will vary depending on flows in Prairie Creek at the time of discharge in order to meet in-stream water quality objectives and minimize fluctuations in receiving water metal concentrations. A discharge schedule was developed based on a detailed water balance to guide how water will be stored, managed and treated seasonally. The schedule will be varied based on the magnitude of mine flows that actually occur, as well as the magnitude of flows in Prairie Creek.

Effluent discharge in the north is typically regulated by a Water Licence that specifies end-of-pipe concentrations, and in some cases, volume restrictions. During the EA and permitting, CZN demonstrated that this approach will not work for the Prairie Creek project because of the variable quality and flow rate of the combined effluent stream seasonally, as well as the substantial difference in seasonal creek flows. CZN developed a variable load discharge (VLD) approach whereby the parameter loads in effluent are varied according to the creek flow rate in order to consistently achieve downstream water quality objectives. The site Water Licence includes the downstream objectives as a compliance point, rather than the usual end-of-pipe concentration approach. The VLD approach is further described below.

For the determination of allowable loads for discharge, reference would be made to calculated upstream mean concentrations and to the water quality objectives. These concentrations are fixed for the purpose of regulation. The allowable load for discharge at any time is then the difference between these concentrations multiplied by the creek flow rate at that time. The allowable load would be computed automatically. The effluent load in discharge would also be computed using data on the discharge rate and discharge water quality, and the operator will ensure that the discharged load remains below the allowable load. Safety factors are built into this calculation. For example, concentrations in the effluent will be assumed to be 10% greater than those determined in an on-site laboratory. Not all parameters can be determined on-site due to the very low detection limits required, but a sufficient number of key or sentinel parameters can be determined which also serve as surrogates for others. In the discharge calculation, the computed effluent load should never be more than 95% of the allowable load. The allowable load is also factored to a lower number to account for groundwater discharge that by-passes the effluent discharge location, and to account for incomplete mixing of the discharge with creek water.

CZN will establish a permanent, automated flow monitoring station on Prairie Creek. Flows would be monitored continuously, with data relayed to the Water Treatment Plant (WTP) 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 allowable load calculation will always be based on flows that are known to exist with a high degree of certainty.

Monitoring of discharge flows will be automated, with data relayed continuously. A warning system will be employed to ensure the effluent load is less than the allowable load, and remains in the 90-95% range. This would be done by automatically opening or closing valves controlling the inflow of process effluent and mine water for treatment.

For the construction period prior to mill operation, mine water discharge will be regulated by fixed end-of-pipe concentrations. For operations, the MVLWB Water Licence has included fixed Effluent Quality Criteria as a temporary measure of regulating discharge in the early phases of the project while VLD parameters are being further developed, leading to adopting VLD as the main method for discharge of treated water since it was determined to be a more protective and adaptive way of controlling effluent discharge to Prairie Creek from an operating mine.

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 Prairie Creek. This exfiltration system will promote mixing of the effluent with the receiving waters, thus avoiding higher concentration “hot-spots” which could be temporarily detrimental to fish.

20.3.4 Chemicals, fuel, and hazardous material management

The majority of mine activities, and those associated with chemicals, fuel and hazardous material, will take place within a dyke-protected area, isolated from Prairie Creek. Any spills or contamination can be contained on site, and discharge of site water to the environment can be stopped temporarily. Specific chemicals and fuels will have their own dedicated containments, diesel fuel for example. 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. Control points will also be established upstream of more sensitive locations. Response efforts and spill collection will be focused at these points in the event of a spill.

Concentrates will be transported in sealed containers, and thus dust should be minimal and any spills should be readily recovered. To confirm the absence of impacts, road bed soil samples will be collected along the route annually and compared to a baseline to ensure material is not being lost.

20.4 Permitting

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 Mackenzie Valley Land and Water Board (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 Mackenzie Valley Environmental Impact Review Board (MVRB) for 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, including the preparation of 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 Aboriginal Affairs and Northern Development Canada (AANDC) 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.

CZN made operating permit applications in 2008 prior to the expansion of the NNPR. During scoping of the EA, operation of the winter road access was included in the scope of development, although CZN already held a winter road permit. Since the road crosses through the jurisdictions of both the MVLWB and Parks Canada it was then necessary to apply for separate permits within the different jurisdictions in particular reference to the LUPs and Type B water licences associated with the access road.

20.4.2 Permits and licences

EA decision

One of the key milestones for the Project was the MVRB approval of the Project proposal for operations EA on 8 December 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 CZN) 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.

Table 20.1 Summary of MVRB suggestions

Suggestion	Description
#1	Either option proposed by CZN to increase water storage on site will improve water quality in Prairie Creek; however construction of a second pond may address a broader range of risks and result in better water management on site.
#2	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.
#3	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.

In their final submissions to the MVRB, CZN made an extensive list of commitments regarding environmental protection, and these have become part of the scope of the development. The MVRB noted that it based its decision on the assumption that CZN will fulfil its commitments.

Post-EA permitting process

Following the December 2011 positive EA decision, the project was referred back to the MVLWB for the processing of permits required to operate the mine (mostly related to a Type 'A' Water Licence and a Land Use Permit for the mine site). CZN also applied to both the MVLWB and Parks Canada for Water Licences and Land Use Permits (LUP's) to operate a revised winter road. Parks Canada became a regulator when the NNPR was expanded in 2009 since part of the road crosses a part of the park to access the Mine, which is located on crown land within the expanded park.

To initiate the post-EA process to acquire operating permits, a Consolidated Project Description (CPD) was submitted to MVLWB and Parks Canada on 15 February 2012. The Water Licences for the road are required for proposed permanent bridges and the use of water for seasonal road base construction. The LUP from Parks Canada for a revised winter road includes the mid-point transfer facility located within the expanded NNPR. On 11 May 2012, CZN received a Directive and Work Plan from the MVLWB which defined information requirements and a process schedule leading to operating permits for the Mine.

In October 2012, CZN filed its last response to the Water Board's Directive. In November 2012, a series of technical sessions were held in Yellowknife to review the Company's submissions to the Water Board. The sessions triggered Information Requests which the Company responded to in December 2012. During the period 29-31 January 2013, the Company and Interveners attended Public Hearings held in Fort Simpson, adjudicated by the Water Board.

A Winter Road LUP was issued in January 2013 followed by a draft Water Licence being issued for review on 15 March 2013. CZN provided review comments on 9 April. A final LUP for the mine site was issued in June 2013 and a final Water Licence was issued on 5 July with a term of seven years, with Reasons for Decision issued on 30 July. Ministerial approval was given on 24 September 2013.

Current permits and licences

CZN currently has a number of permits and licences for both exploration and mine operations (refer to Table 20.2 for summary) issued by the MVLWB under the *Mackenzie Valley Resource Management Act*. In addition, CZN also has a LUP and Water Licence from Parks Canada for the portion of an operations winter road that crosses the NNPR.

Table 20.2 Summary of current permits and licences

Permit	Date of issuance (duration)	Description
Water Licence (Class B) MV2001L2-0003	Initially issued 10 September 2003 Renewed in 2008 (for 5 years) Amended 10 May 2012 (extended to 9 September 2019).	Allowed CZN to pursue underground exploration (decline), and subsequently to continue with this exploration while awaiting operational permits -Included application for pilot plant operation (plant was never commissioned).
LUP MV2012C0008	Issued 10 May 2012 – expiring on 9 May 2017. Extended to 9 May 2019.	
LUP MV2012C0002	25 April 2013 (for 5 years)	Allows for surface exploration and diamond drilling activities.
LUP MV2012F0007	10 January 2013 (for 5 years)	Allows CZN to construct and operate a winter access road from the Liard Highway at Nahanni Butte to the Mine, outside of the NNPR.
Water Licence (Class B) MV2012L1-0005	10 January 2013 (valid for 7 years)	Water Licence for the winter road allowing permanent crossings of waterways >5 m wide, and extraction of water from authorized sources for road construction and maintenance.
LUP Parks2012-L001	26 August 2013 (for 5 years)	Allows CZN to construct and operate a winter access road and transfer facility within the NNPR.
Water Licence Parks2012_W001	26 August 2013 (for 5 years)	Water Licence for the winter road allowing permanent crossings of waterways >5 m wide, and extraction of water from authorized sources for road construction and maintenance.
LUP MV2008T0012	17 June 2013 (for 5 years)	Allows CZN to build and operate a transfer facility near the Liard Highway.
LUP MV2008D0014	17 June 2013 (for 5 years)	Allows CZN to construct and operate the Prairie Creek Mine.
Water Licence (Class A) MV2008L2-0002	24 September 2013 (for 7 years)	Allows CZN to use water and deposit waste to operate the Prairie Creek Mine.

Water Licence MV2001L2-0003 and LUP MV2012C0008 allow CZN to continue with underground exploration prior to operations. LUP MV2012C0002 provides for surface exploration and diamond drilling at sites throughout the Prairie Creek property.

The remaining LUP's and Water Licences provide for winter road and mine operations, the main one being the Class A Water Licence MV2008L2-0002, which regulates water use and waste disposal associated with mine and mill operations.

All season road permitting

On 16 April 2014 CZN made applications to the MVLWB and Parks Canada for permits to construct, maintain and operate an all season road from the mine to the Liard Highway. An all season road will enable the haul of concentrates from the mine site to rail year-round, with temporary closures relating to freeze-up and break-up on the Liard River. The all season road would use a similar alignment to the already permitted winter road with some minor re-alignments to take advantage of better ground.

The MVLWB referred the applications to the MVRB on 22 May 2014 for EA, EA1415-001. CZN produced a draft Terms of Reference (ToR) for a Developer's Assessment Report on 4 June 2014. Community meetings to consider the scope of the EA were subsequently held in Nahanni Butte, Fort Liard and Fort Simpson over the period 9-11 June 2014 and a technical scoping meeting was held in Yellowknife on 8 July 2014. The MVRB collated scoping meeting comments, and issued their version of the draft ToR for comment on 31 July 2014. The MVRB produced a final ToR on 12 September 2014.

On 23 April 2015 Canadian Zinc submitted its Developer's Assessment Report (DAR) and the MVRB responded on 22 May 2015 with an Adequacy Review, which required that CZN submit further detail on a number of items in the ToR. CZN submitted an Addendum to the DAR on 2 October 2015, which was followed by more detail in early December 2015. The MVRB issued a Note to File on 20 April 2016 which concluded that CZN had met the requirements outlined in the Reasons for Decision issued by the MVRB in December 2015. CZN subsequently responded to Information Requests, and a Technical Session was held in Yellowknife on 13-15 June 2016 as part of the normal EA process. After replies to a further round of Information Requests, Community Hearings were held in Nahanni Butte and Fort Simpson on 24-25 April 2017 and Public Hearings in Fort Simpson on 26-28 April 2017.

The Mackenzie Valley Environmental Impact Review Board ("Review Board") issued its Report of Environmental Assessment and Reasons for Decision for Canadian Zinc's Prairie Creek All season Road Project for the Prairie Creek Mine (the "EA Report") on 12 September 2017 and submitted the Report to the Honourable Carolyn Bennett, Federal Minister of Crown-Indigenous Relations and Northern Affairs. The Review Board recommends the approval of the Prairie Creek All season Road be made subject to implementation of the measures described in the Report, which it considers are necessary to prevent significant adverse impacts on the environment and local people.

The EA Report has been forwarded to the Federal Minister of Crown-Indigenous Relations and Northern Affairs, with a recommendation that the development be approved, subject to the measures described in the Report, and Canadian Zinc is looking forward to Ministerial approval. The Review Board has concluded that an environmental impact review of this proposed development is not necessary and that the proposed All Season Road project should proceed to the regulatory phase.

Subject to Federal approval, the regulatory phase, conducted by the Mackenzie Valley Land and Water Board with input from territorial and federal agencies, will be the next permitting stage in which the road permit is issued by the Water Board.

20.5 Indigenous groups

The Prairie Creek Mine and access road is located in an area that includes the traditional territory of the Nahanni Butte Dene Band and the Liidlii Kue First Nation, historically both part of the Dehcho First Nations (Dehcho or DCFN). The Nahanni Butte Dene Band is a "band" pursuant to the Indian Act RSC 1985. The members of the Dehcho First Nations are Aboriginal people within the meaning of Section 35 of the Constitution Act, 1982.

The Dehcho Region hosts a distinct group of Aboriginal people, 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 DCFN territory for sustenance, social and ceremonial purposes.

The DCFN is an organization representing Dene and Metis peoples in the Dehcho territory of the Northwest Territories. The DCFN have incorporated a society under the laws of the Northwest Territories in order to provide leadership, governance, administration and program delivery to their member communities. The DCFN is a governing body of the Dehcho peoples' lands, and administers and oversees a number of programs and services for its member communities, including those relating to health, employment, education, and land and resource management.

The outcome of land claims negotiations between the Federal Government, GNWT and the DCFN, 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.

The Company cannot predict the impact, if any, that the Dehcho Final Agreement, if eventually approved and signed, may have on the Prairie Creek Mine.

20.5.1 Nahanni Butte Dene Band

The Prairie Creek Mine is located 90 km from the nearest settled community of Nahanni Butte, located at the confluence of the South Nahanni and Liard Rivers, 146 km downstream of the mine site and home of the Nahanni Butte Dene Band. The population of Nahanni Butte is approximately 90 people and water for domestic purposes is supplied by well. There is no permanent road access into the Prairie Creek Property, other than the existing winter road, which was established in 1981. Regular access is by air only to a private airstrip controlled by the Company. There is no other existing land occupation, nor commercial land or water based activities in the vicinity of the mine. Similarly, no traditional use or trapping activity has been observed in the mine site area in recent history.

In October 2008, Canadian Zinc and the Nahanni Butte Dene Band (NBDB) 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 Prairie Creek. 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 Prairie Creek Project. 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 Prairie Creek, 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 NAHA DEHE DENE PRAIRIE CREEK AGREEMENT (the Nahanni Agreement), which provides for an ongoing working relationship between Canadian Zinc Corporation and the NBDB 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 Prairie Creek Project and confirms their support for the Prairie Creek Mine.

The Nahanni Agreement provides a framework such that training, employment and business contracts are made available to Nahanni to ensure maximization of benefits from opportunities arising from the Prairie Creek Project in a manner that will be to the mutual benefit of both parties.

The Naha Dehe 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 Prairie Creek, which will not have significant adverse environmental effects on the ecological integrity of the South Nahanni River or the NNPR.

CZN has continued engagement with the NBDB during the course of the EA for the all season road. The intent of the parties is to complete a Traditional Land Use Agreement (TLUA). The agreement will cover road activities and will provide additional benefits to the Band. Negotiations are well advanced, however the agreement has not yet been signed.

On 3 May 2017 the NBDB announced that the Band is withdrawing from the DCFN. CZN does not expect that this development will negatively affect relations between the Band and CZN.

20.5.2 Liidlii Kue First Nation

In June 2011, the Company signed an Impact Benefits Agreement (LKFN Agreement) with the Liidlii Kue First Nation (LKFN) of Fort Simpson. The LKFN Agreement is similar in many respects to the above mentioned Nahanni Agreement entered into with the Nahanni Butte Dene Band. The LKFN agreed to support CZN in obtaining all necessary permits and other regulatory approvals required for the Prairie Creek Mine Project. The Agreement is intended to ensure that CZN undertakes operations in an environmentally sound manner. LKFN will appoint a qualified Monitor to monitor environmental compliance and to monitor impacts of the mine on the environment or wildlife and to work with CZN to prevent or mitigate such impacts.

The LKFN Agreement provides a framework such that training, employment and business contracts, and some financial provisions are made available to the LKFN to ensure maximization of benefits from opportunities arising from the Prairie Creek Project in a manner that will be to the mutual benefit of all parties. The LKFN is the largest member of the DCFN.

20.6 Agreements with government agencies

20.6.1 Nahanni National Park Reserve / Parks Canada Memorandum of Understanding

The Nahanni National Park Reserve (NNPR) was created in 1972, following a canoe trip down the river by then Prime Minister Pierre Elliot Trudeau, specifically for the purpose of setting aside the South Nahanni River for wilderness recreational purposes. Exploration activity at Prairie Creek had been ongoing for many years prior to 1972 and underground development was well advanced at that time.

Parliament formally established NNPR of Canada in 1972, legally protecting it as Canada’s 26th National Park under the Canada National Parks Act. It was established as a National Park Reserve in view of the fact that there were outstanding land claims in the area. It will only become a fully-fledged National Park once an agreement has been reached with the DCFN.

NNPR is considered to be of global significance. In 1978, it was the first area added by UNESCO to its list of World Heritage Sites. There are only 13 sites in Canada designated as World Heritage Sites, eight of them being National Parks. Nahanni received this designation because of the geological processes and natural phenomena in the area. In UNESCO’s view, Nahanni is special because it is an unexploited natural area. The presence in this area of three river canyons cutting at right angles to the mountain ranges, with walls of up to 1,000 m high, Virginia Falls which falls over 90 m, hot springs, sink holes and karst topography are considered a special combination.

In considering and approving the nomination of NNPR for World Heritage Status, the World Heritage Committee stated that “it would be desirable to incorporate the entire upstream watershed in the World Heritage Site.” In 1977, the Minister responsible for Parks Canada directed Parks Canada to examine the possibility of expanding NNPR to include more of the head-waters of the South Nahanni and the karst terrain. Several studies were conducted to assess this potential.

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 km^2, making it the third largest National Park in Canada. The enlarged Park covers most of the South Nahanni River watershed and completely encircles the Prairie Creek Mine. However, the Mine itself and a large surrounding area of approximately 300 km^2 are specifically excluded from the Park and are not part of the expanded Park.

The exclusion of the Prairie Creek Mine from the NNPR expansion area has 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 NNPR 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 the Prairie Creek Area, 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 NNPR and specifically for the purpose of providing access to the Prairie Creek Area.

On 29 July 2008, Parks Canada Agency (Parks Canada) and Canadian Zinc entered into a MOU with regard to the expansion of the NNPR and the development of the Prairie Creek Mine, whereby:

- Parks Canada and CZN agreed to work collaboratively, within their respective areas of responsibility, authority and jurisdiction, to achieve their respective goals of an expanded NNPR and an operating Prairie Creek Mine.
- Parks Canada recognized and respects the right of Canadian Zinc to develop the Prairie Creek Mine and was to manage the expansion of NNPR so that the expansion did not in its own right negatively affect development of, or reasonable access to and from, the Prairie Creek Mine.
- Canadian Zinc accepted and supported the proposed expansion of the NNPR and will manage the development of the Prairie Creek Mine so the mine does not, in its own right, negatively affect the expansion of the NNPR.

The 2008 MOU was intended to cover the period up to the development of the Prairie Creek 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 Prairie Creek Mine and the management of NNPR.

The Phase III MOU, signed November 2015 and which is valid for five years, replaces the previous MOU signed between the Parties in 2008. In the renewed MOU:

- 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 Prairie Creek Mine.
- Parks Canada recognizes and respects the right of Canadian Zinc to develop the Prairie Creek Mine and has granted Land Use Permit Parks 2012 – L001 and Water Licence Parks 2012_W001 to provide road access through the Park to the Mine area.
- 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.

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, including convening a Technical Team, including representatives of the DCFN, which will better identify, define and consider issues of common interest, including, among other things, development of the access to and from the Prairie Creek Mine through NNPR and operation of the Prairie Creek Mine.

The Parties also agreed to share with one another and the Technical Team any existing technical and scientific information relevant to a discussion and analysis of issues of common interest to the Parties. The parties have agreed to make reasonable efforts to be timely in regards to permit requests being submitted, with ample time for review and consultation; such review and consultation will occur without unreasonable delay.

To the extent that the Prairie Creek 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 Prairie Creek Mine.

20.6.2 Government of the Northwest Territories Socio-Economic Agreement

In August 2011, the Company signed a Socio-Economic Agreement with the Government of the Northwest Territories related to the planned development of the Prairie Creek Mine. The Socio-Economic Agreement establishes the methods and procedures by which the Company and the GNWT have agreed to work together to maximize the beneficial opportunities and minimize the negative socio–economic impacts arising from an operating Prairie Creek Mine. The Socio-Economic Agreement defines hiring priorities and employment commitments and practices during the construction, operation and closure of the Prairie Creek Mine and across the entire spectrum of project-based employment. The Company has targeted employment levels of at least 60% Northwest Territories residents and 25% Aboriginals. The Company has agreed to implement policies to maximize business and value-added opportunities for businesses in the Northwest Territories. Canadian Zinc will use its best efforts to ensure that purchases of goods and services through or from Northwest Territories businesses will be at least 30% during construction and at least 60% during operations.

On 1 April 2014 *The Northwest Territories Devolution Act,* which provides for the devolution of lands and resource management from the Government of Canada to the Government of the Northwest Territories (GNWT), came into force. Devolution in the Northwest Territories means the transfer of decision-making and administration for land and resource management from the Government of Canada to the Government of the Northwest Territories. The Territorial government is now responsible for the management of onshore lands and the issuance of rights and interests with respect to onshore minerals and oil and gas. The GNWT now has the power to collect and share in resource revenues generated in the territory. The Northwest Territories Devolution Act includes certain amendments to the MVRMA, which impose additional regulations and obligations on mining operations in the Mackenzie Valley.

20.6.3 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 Prairie Creek 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, CZN will provide reports to the Department of Transportation on its anticipated road transportation requirements for the construction and operation of the Prairie Creek 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.

20.6.4 The Northwest Territories Power Corporation

On 14 February 2017 a Memorandum of Understanding (“MOU”) was signed with Northwest Territories Power Corporation (“NTPC”) to examine the supply of electrical power for the development and operation of the Prairie Creek 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. CZN and NTPC have also agreed to evaluate the integration of other energy alternatives, and specifically Liquefied Natural Gas (“LNG”), as part of the energy supply for the mine.

NTPC is a wholly owned subsidiary of NT Hydro, which is 100% owned by the Government of the Northwest Territories. NTPC exists to provide value to its shareholder and customers through the efforts of a highly dedicated, skilled, and productive workforce. Its mission is to generate, transmit and distribute electricity in a safe, reliable, efficient and environmentally sound manner; striving to reduce reliance on fossil fuels.

20.7 Local employment training programs

Canadian Zinc, the Mine Training Society, Government of the Northwest Territories and the Prairie Creek Mine’s neighbouring aboriginal communities successfully completed a three year, federally funded training program entitled “*More Than a Silver Lining*” (MTSL) under the Skills Partnership Fund with the Government of Canada. The expected training program’s total cost was $4.3M, with the majority of the funding being provided by the federal department of Human Resources and Skills Development Canada. The program was solely focused on the workforce needs of the Prairie Creek Mine.

The MTSL program delivered 19 training projects in the Dehcho Region over the three year period ending in 2013. Of the 19 training projects, six were facilitated by Canadian Zinc at the Prairie Creek Mine. Over the course of three years approximately 300 local individuals were assessed for participation in the training programs with 250 people actually participating, of which approximately 70 are reported to have returned to employment and others have moved on to higher education.

20.8 Mine closure

An interim Closure and Reclamation Plan (CRP) for the Prairie Creek Mine was prepared during the permitting process. The plan followed the “Mine Site Reclamation Guidelines for the Northwest Territories”, issued by Indian and Northern Affairs Canada, Yellowknife, NWT, in January 2007.

A CRP 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 and provides estimated closure costs.

20.8.1 Temporary closure activities

Definition of temporary closure:

> *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.*

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.

Process plant

On temporary closure, 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, bagged and stored in the Concentrate Shed. For extended temporary closures, the ball mill will be jacked up off its bearings to prevent wear.

Water storage pond

The WSP will continue to receive water mainly from underground and the Waste Rock Pile.

The Process Plant will not be producing process water.

The WSP will be kept in balance by sending water to the WTP.

Water treatment plant

The WTP will remain at full operational status and will be operated as required to maintain an annual balance in the Water Storage Pond.

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 infrastructures will need to be maintained.

Off-site infrastructure

Off-site infrastructure includes the all season road, winter road and any transfer facilities.

The transfer / staging area at the Liard River is a “concentrate storage/operating supplies staging” facility designed to facilitate the simultaneous flow of lead / zinc / silver concentrates from the mine, and the flow of the annual operating supplies into the mine. This facility will also be closed during temporary closure of the mine after all concentrates have been shipped out. All supplies and materials will have been removed from the site and any waste will also have been removed.

No temporary closure activities are contemplated for the all season or winter roads over and above the normal seasonal closure requirements each year.

20.8.2 Permanent closure activities

Definition of permanent closure

Permanent closure is deemed to occur when a mine exhausts ore reserves that can be economically extracted and ceases operations without the intent to resume mining activities in the future.

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 WRP.

Waste rock pile

The WRP will store up to 3.2 Mt of waste rock and an additional 100,000 m^3 of inert solid waste to provide landfill disposal. The following solid waste components will probably be landfilled following removal of all contaminants:

- 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 WRP will be covered with a minimum one-metre thick layer of waste rock. The final cover for the WRP will be designed to promote runoff and minimize infiltration and the generation of leachate. An initial study recommended a 1 to 2 m thick 'till' (clayey soil) layer be applied.

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 WRP may need to be temporarily directed underground so that it can be managed with mine water.

Water storage pond

At the point that the Water Storage Pond (WSP) is no longer required for mine operations or reclamation activities, it will be reclaimed as follows:

- Pond water will be processed through the WTP and discharged;
- Sediment and any remaining tailings in the WSP will be removed and sent to the Paste Backfill Plant to be subsequently deposited underground;
- When the WSP is free of contaminated solids and water, the liner will be removed and placed in the WRP; and,
- The WSP embankment will be breached in two places to prevent the structure from impounding water, and the outlets to Prairie Creek will be stabilized as necessary.

Underground

The intent is to completely backfill the underground workings and all access tunnels with a backfill mix. The workings will be completely 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 quality of surface water. These predictions indicate that surface water quality objectives will be met without any need for further actions. While these predictions will be refined during operations and at mine closure, as a contingency, a groundwater pumping and water treatment scheme has been devised and provided for.

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 WTP, followed by discharge to the environment. Over time, the quality of groundwater underground is expected to improve as leachable metals diminish. Therefore, the contingency pumping scheme, if required, will likely not need to operate for an extended period. Initial indications are that the pumping might be required for four to eight years.

Mine equipment

All contaminated mine equipment will be removed from underground before mine closure.

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 WRP.

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 wastes that do not qualify for disposal in the WRP;
- Dismantle all equipment and building structures, reduce the material to manageable pieces, and place them in the WRP.

For the post-closure monitoring phase, a scaled-down mine water treatment circuit may remain, along with reduced accommodations, fuel storage and warehouse facilities.

Off-site infrastructure

The off-site infrastructure at the road construction/maintenance camps will be removed and either salvaged, taken to the mine for disposal, or taken to a suitable off-site disposal location with the approval of local authorities. Equipment for disposal will include trailers and generator sets. The sites will be reclaimed by scarifying the surfaces and placing a growth medium to promote revegetation by the natural invasion of native species.

20.8.3 Post-closure monitoring, maintenance, and reporting program

Post-closure monitoring will include inspection of mine access barricades, the WRP cover and runoff controls, observation of reclaimed surfaces for erosion, and the collection of water samples. 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.

20.8.4 Closure schedule and cost estimate

Upon cessation of operations, closure activities at the Prairie Creek 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, if required:

- Backfill underground workings;
- Mine and mill equipment removal to WRP;
- Building and on-site infrastructure demolition;
- WSP – Tailings and Sediment removal and disposal;
- WSP – Liner removal and dyke breaching;
- Off-site infrastructure demolition and salvage/disposal, site reclamation; and
- Substantial reclamation of the WRP, 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 WRP;
- Final reclamation of the WRP; and,
- Reclamation of the access road.

The total amount of security deposits are as described in the issued permits associated with future operations and summarized in Table 21.4 of this report. The all season road permit application is currently in Environmental Assessment and the security deposit that will be required for this is yet to be determined; therefore an estimated amount has been used in the cash flow model.

21 Capital and operating costs

21.1 Sources of estimated costs

The 2017 Feasibility Study capital and operating cost estimates for developing and commissioning the Prairie Creek project were compiled from information provided by AMC, Ausenco and other specialist consultants, and include vendor quotes and leasing proposals.

The capital cost estimate for the mine, process plant and site infrastructure has been developed using engineering quantities and budget pricing obtained from vendors and contractors. Where such information has not been available, estimates have been provided by AMC and Ausenco, derived using in-house estimating procedures and database information, with specialty input from:

- Golder Associates: Waste Rock Pile and Water Storage Pond;
- Outotec: Paste Plant
- Canadian National Railway and Kinder Morgan: rail and port costs.

Table 21.1 summarizes the sources of the capital and operating cost estimates.

Table 21.1 Sources of inputs to capital and operating cost estimates

Component of input to the capital and operating cost estimates	Source
Mine development capital cost estimate	AMC
Mill capital cost estimate	Ausenco
Surface facilities capital cost estimate	Ausenco
Mine workforce cost estimate	AMC
Mill workforce cost estimate	Ausenco
Surface & G&A workforce cost estimate	Ausenco / CZN
Mine operating cost estimate	AMC / CZN
Mill operating cost estimate	Ausenco
Surface facilities & G&A operating cost estimate	Ausenco / CZN
Power requirements	Ausenco / AMC
Power cost estimate	Ausenco
All season road installation & maintenance cost estimate	CZN / Allnorth
Owner's costs	CZN
Reclamation security estimate	CZN

AMC = AMC Mining Consultants, CZN = Canadian Zinc Corporation, Ausenco = Ausenco Engineering.

21.2 Capital Cost estimate

The Capital Cost estimate is broken down into pre-production capital and sustaining capital, and is presented as a summary in Table 21.2 below. The sustaining capital is carried over operating years 1 through 16.

Pre-Production Capital Cost refers to capital costs incurred until the first processing of mined ore, and has been estimated to be a total of $252.9 million excluding contingency. This total includes $16.7 million in capitalized operating costs and lease costs during construction. Total pre-production capital is $278.9 million including a contingency of $26.0 million for the mine, process plant and site infrastructure. Total sustaining capital is $117.0 million and includes an off-set of $5.5 million in salvage.

The overall contingency was calculated at 10.3% and the accuracy of the estimate is projected at -10% to +15%.

Table 21.2 Summary of capital costs

Description	Total ($M)
Pre-production capital (incl. contingency of $26.0M)	278.9
Sustaining capital	117.0
Total capital cost (life of mine)	**395.9**

21.2.1 Pre-production capital

The Pre-Production Capital cost estimate is sub-divided by Work Breakdown Structure (WBS) cost areas as indicated below in Table 21.3.

Table 21.3 Pre-production capital cost estimate

Description	Project Y-02 ($M)	Project Y-01 ($M)	Project Y01 ($M)	Total cost ($M)
Mine development	2.6	13.6	21.5	**37.7**
Site preparation	4.3	12.5	2.6	**19.4**
Mill process plant [1]	9.0	18.9	3.2	**31.1**
Paste tailings plant and process	2.9	16.6	3.4	**22.9**
Construction indirects including EPCM [2]	10.9	7.8	5.1	**23.8**
Other site infrastructure	6.7	7.7	1.5	**15.9**
All season road	13.0	41.6	13.9	**68.5**
Owner's costs	6.8	15.3	11.5	**33.6**
Total (excluding contingency)	**56.2**	**134.0**	**62.7**	**252.9**
Contingency	5.5	12.3	8.2	**26.0**
Total pre-production capital	**61.7**	**146.3**	**70.9**	**278.9**

1. Includes dense media separator, structural upgrading, instrumentation, flotation circuit upgrade, reagent handling and piping.
2. Includes engineering and construction of surface facilities, freight and logistics, initial fills and spares.

Based on the proposals received, several capital items will be supplied on a lease-to-purchase basis including the accommodation camp, paste plant, flotation cells and thickeners. The lease costs of such items incurred during the pre-production period are included in Pre-Production Capital costs, and lease costs incurred after production start-up are included in Sustaining Capital costs.

21.2.2 Salvage value

Some of the capital equipment is expected to have minor salvage value at the end of mine life. The salvage value for the general site equipment includes reselling construction equipment not required for operations at the completion of the construction works, as shown in Table 21.4.

Table 21.4 Capital salvage value

Description	Capital cost ($M)	Residual value (%)	Salvage value ($M)
Underground mining equipment	20.0	12.5	2.5
General site equipment[1]	10.8	27.8	3.0
Total	**30.8**	**17.9**	**5.5**

1. Includes crushing and grinding equipment, camp accommodation equipment, assay lab and barge.

21.2.3 Capital cost exclusions

The following cost items have not been included in the Pre-Production Capital cost estimate:

- Project sunk costs and any additional studies;
- Project financing costs;
- Any bonding costs (performance bonds or completion bonds);
- Inflation or escalation during construction;
- Foreign exchange variations;
- Operating costs (except where noted as capitalized opex).

21.2.4 Direct costs

Mine pre-development

In 2014, CZN invited expressions of interest from five major Canadian mining contractors, resulting in two tenders, for a 2½ year scope of work to rehabilitate the existing mine, drive appropriately-sized new development and start stope production by means of longhole mining. The cost estimate in this 2017 FS study is derived from the unit prices indicated by the successful tenderer, Procon. Procon updated their 2014 cost estimate in 2017, and the 2017 unit prices were multiplied by the quantities estimated by AMC. The tendered work items were sub-divided into development capital and operating costs. The capital cost of mine development was therefore the sum of those items identified as capital works. Longer-term, unit prices have been factored for increasing mining depth as appropriate, and the estimation for the take-over of mining operations by CZN has included building costs from first principles.

Procon unit prices are all-inclusive, except for CZN-supplied diesel fuel and camp accommodation.

During 2015, the Mineral Resource was significantly increased and better definition was obtained through diamond drilling. As a result, CZN requested AMC to revise the previous mine plan. The new mine plan accelerated the ramp-up to full production and biased initial production towards sulphide, rather than oxide, mineralization with beneficial effects on mine economics.

The mine development capital works comprise:

- 883 mL portal reconstruction and enlargement of drift to 5.0 m x 5.0 m, 538 m;
- 4.5 m x 4.5 m ore drive slashing, 367 m;
- 4.6 m x 4.6 m ramp, 1,680 m;
- 4.5 m x 4.5 m vein-access crosscuts, 587 m;
- 4.5 m x 4.5 m remucks, paste fill line access drifts, ventilation drifts, sumps, miscellaneous development, 954 m;
- 4.5 m x 4.5 m electrical cut-outs, miscellaneous development, 112 m;
- 4.0 m x 4.0 m waste development, 16 m;
- 4.5 m x 4.5 m vein drifts, 448 m;
- 4.0 m x 4.0 m vein drifts, 724 m;
- 3.0 x 3.0 m vertical development, 244 m;
- General mine rehabilitation and installation of new mine infrastructure.

Overall, mine pre-development and equipment capital costs, including contingency and capitalized operating costs, are estimated at $42.6M.

Individual contingencies were applied to each capital cost item and the amount of contingency varied depending on the cost uncertainty associated with each item. The UG mine overall contingency cost of $4.9M is 13.0% of the UG mine estimated capital cost.

Process plant, onsite, and offsite infrastructure

Ausenco completed the initial design, developed budgetary quotation packages and completed technical and cost evaluations for the following capital items within the process plant and site infrastructure:

Process Plant:

- Flotation Cells
- DMS Plant
- Thickeners
- Mine Water Treatment Plant
- Fire Water Pumps
- Slurry Pumps
- Water and Solution Pumps
- Reagent Pumps
- Cyclones
- Lime Silo
- Air compressors
- Blowers
- Agitators
- Flocculent System
- Vibratory Feeders
- Bag Breaker
- Conveyors and Feeders.

Surface Infrastructure:

- Accommodation Camp
- Power Generators
- Potable Water Treatment Plant
- Pre-Engineered Buildings
- Mine Water Treatment Plant
- Paste Backfill Plant
- Plant Control System.

Approximately 89% of the mechanical equipment supply cost estimates were received from vendors via non-binding budgetary quotes for the process plant and site infrastructure. Civil, concrete, structural steel, piping, electrical and instrumentation quantities were developed by Ausenco's engineering team and the unit supply and installation rates are based on the company's in-house database.

Refurbishment costs for the major process equipment, including the ball mill, crushers and filters, are based on site inspection reports and were developed with input from specialist contractors. All materials, plant and equipment items within the direct costs are based on delivered-to-store on-site. Freight costs include inland transportation, export packing, all forwarder costs, ocean freight and air freight where required, insurance, receiving port custom agent fees and local inland freight to the project site. A growth allowance has then been allocated to each cost element (direct and indirect costs) to reflect the level of definition of design (quantity maturity) and pricing strategy (cost maturity) and accounts for approximately 7.7% of the initial capital direct costs.

21.2.5 Reclamation costs

Reclamation and Closure costs have been established by the Mackenzie Valley Land and Water Board and are associated with issued Land Use Permits and Water Licences, as shown in Table 21.5.

Table 21.5 Reclamation and closure costs for Prairie Creek Mine

Mine site "A" WL	1,550,000	Security posted per 22 May 2015 amendment on existing leases
(MV2008L2-0002)	2,450,000	Prior to extracting waste rock from underground
	2,000,000	Within 12 months of extracting waste rock from underground
	2,000,000	Within 24 months of extracting waste rock from underground
	5,070,000	Prior to commencing milling
	13,070,000	
Mine site LUP	250,000	Previously posted security on existing leases
(MV2008D0014)	1,750,000	Prior to commencement of construction upgrades
	1,000,000	Within 12 months of commencement of construction upgrades
	1,000,000	Within 24 months of commencement of construction upgrades
	4,000,000	
Access road (crown) LUP	220,000	Prior to commencement of the operation
(MV2012F0007)	**220,000**	
Access road (crown) "B" WL	220,000	Post and maintain security
(MV2012L1-0005)	**220,000**	
Access road (NNPR) LUP	674,376	Prior to commencement of operation
	674,376	
Access road (NNPR) "B" WL	330	Year 2
(Parks2012_W001)	330	Year 3
	330	Year 4
	330	Year 5
	683,021	Prior to bridges at km 23 & km 53
	813,896	Prior to temp bridges at km 24, 26.4, 26.8, 28.7, 43
	1,498,237	
Grand total	**19,682,613**	
Previously remitted	**1,800,000**	
Ongoing requirements	**17,882,613**	

Reclamation costs of $12.9M incurred prior to mine start-up are included in pre-production capital costs with the balance of $5.0M included within sustaining capital. The all season road permit application is currently in Environmental Assessment and the reclamation security deposit related to this permit is yet to be determined. A preliminary estimate has been included in the cash flow model within sustaining capital.

21.2.6 Indirect costs

Engineering, Procurement, and Construction Management (EPCM)

The cost of EPCM services includes all related work and activities required for the complete engineering package necessary to construct the intended facilities, procurement, contract administration, office services and construction management activities. This EPCM item may be subdivided approximately as 50% for engineering and procurement and 50% for site construction management.

Construction indirects

Project facilities and services required for support during the construction period, including accommodation camp for use during construction (and subsequently operations), EPCM offices, maintenance services, provision of temporary roads, scaffold, power, water and effluent disposal, are included as construction indirects in the estimate.

Flight costs for the construction workforce are included and are based on commercial flights within Alberta and British Columbia to Yellowknife and charter flights between Yellowknife and site.

Catering costs are estimated at $75 per person per day and are based on CZN’s actual camp costs for prior site work and the manpower curves for direct and indirect labour.

Marshalling yard, spare parts, vendor representatives

Access to the site during construction will be via an existing access road from Fort Nelson, and the estimate allows for a marshalling yard at Fort Nelson inclusive of yard rental, security, on-site management / dispatch office, mobile equipment and manpower.

Ausenco estimated the cost of spare parts, first fill lubricants, first fill reagents, and vendor assistance.

Pre-commissioning, commissioning

Commissioning assistance from mechanical completion to hand-over has been included in the estimate.

21.2.7 Contingency

Individual contingencies were estimated by AMC for each capital cost item in the underground mine and Ausenco used a Monte Carlo simulation model (to an 80% confidence level) to determine contingencies for the plant and surface infrastructure. The overall contingency is 10.3%.

21.2.8 Sustaining capital

Sustaining capital costs have been stated in 2017 Canadian dollars without any allowance for escalation or inflation. Sustaining capital costs (Table 21.6) have been developed as follows:

- Sustaining capital for mining, including purchase and rebuild of mobile equipment, is estimated at $93.4M over the remainder of the mine life, including buy-out of mine contractor equipment, fleet additions and renewals and extensions to underground infrastructure, continuation of capitalized lateral and vertical mine development, but excluding salvage.
- Costs for mine equipment used in this estimate were sourced from a variety of equipment suppliers, including Atlas Copco, Sandvik, McDowell Brothers Industries, Orica Mining Services, Miller Technology, Normet, and from AMC’s cost database.
- Leasing costs of capital items of $11.0M are included in the first five (5) years of sustaining capital costs.

Table 21.6 Total sustaining capital costs

Sustaining capital	LOM $M
Mining	73.4
Mobile equipment	20.0
Capital leases	11.0
All season road	8.6
Infrastructure	9.5
Salvage	(5.5)
Total	**117.0**

21.2.9 Working capital

Working capital of $36 million is estimated to be required over the first six months subsequent to commercial production, based on the assumptions as set out in Section 22.1.

21.3 Operating cost estimate

21.3.1 Total operating cost

The operating cost estimate is expressed in 2017 Canadian dollars and has been developed to a feasibility study estimate level of -10%/+15% overall intended accuracy, over the life of the mine.

Operating costs (per tonne of ore feed) including transportation to the smelter are $222.59 as shown in Table 21.7.

The mining contractor quotes for the first two years of operation, based on a detailed scope of work and schedule, provide a particularly high level of confidence in the estimated mining costs. The indicative proposal from the Northwest Territories Power Corporation to supply turnkey type power generation provides further support in the key area of power costs

Table 21.7 Total operating cost summary

Total operating cost	**($/t)**
LOM (8,071,463 t)	
Mining	58.23
Milling and processing	46.76
G&A	30.32
Site services	18.55
Sub-total	**153.86**
Transportation[1]	68.73
Total	**222.59**

Totals do not necessarily equal the sum of the components due to rounding.
1. Includes truck / rail / handling / shipping.

The life-of-mine operating plan indicates that mine operating costs will vary from year to year, both absolutely and per tonne mined.

The annual tonnes and grade of mined ore are based on a detailed mine plan prepared by AMC, reconciled with the acceptance rate by the mill. The tonnage accepted by the mill beyond the grinding stage is limited by the saturation point of the flotation plant and is, therefore, a function of mill head grade.

Figure 21.1 shows the cash cost, by area, over the mine concentrate production years.

Figure 21.1 Annual cash operating cost by area

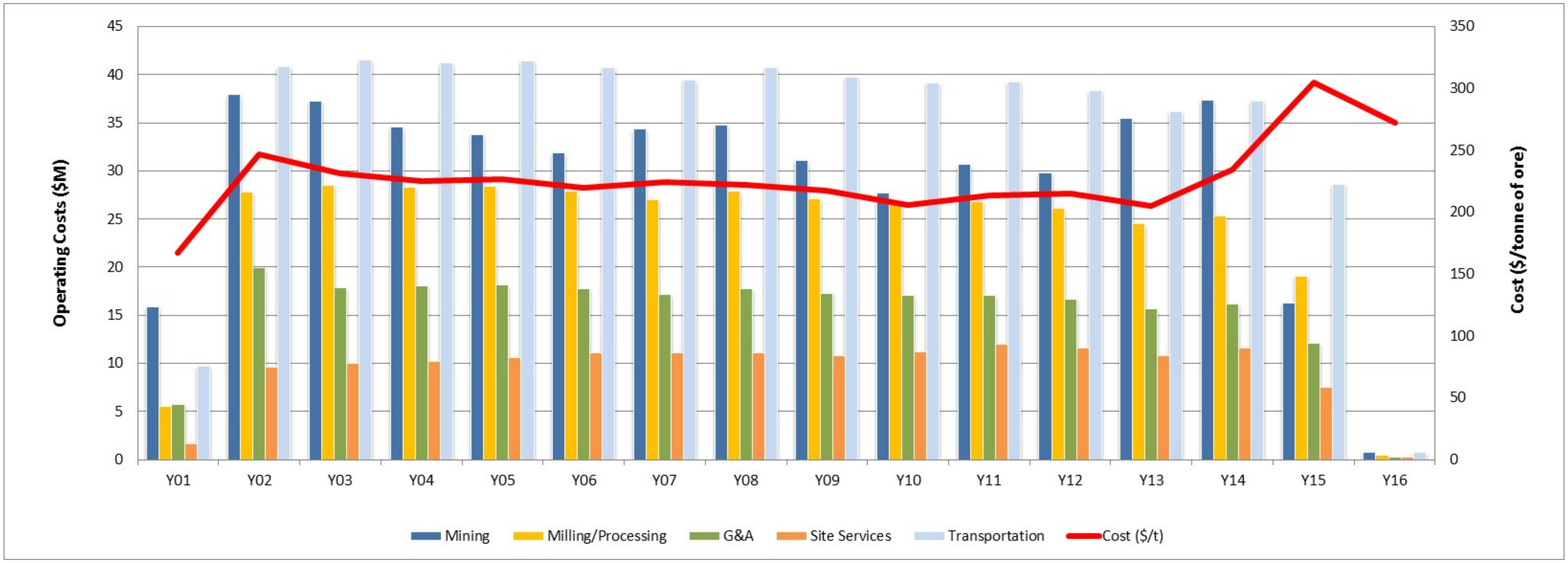
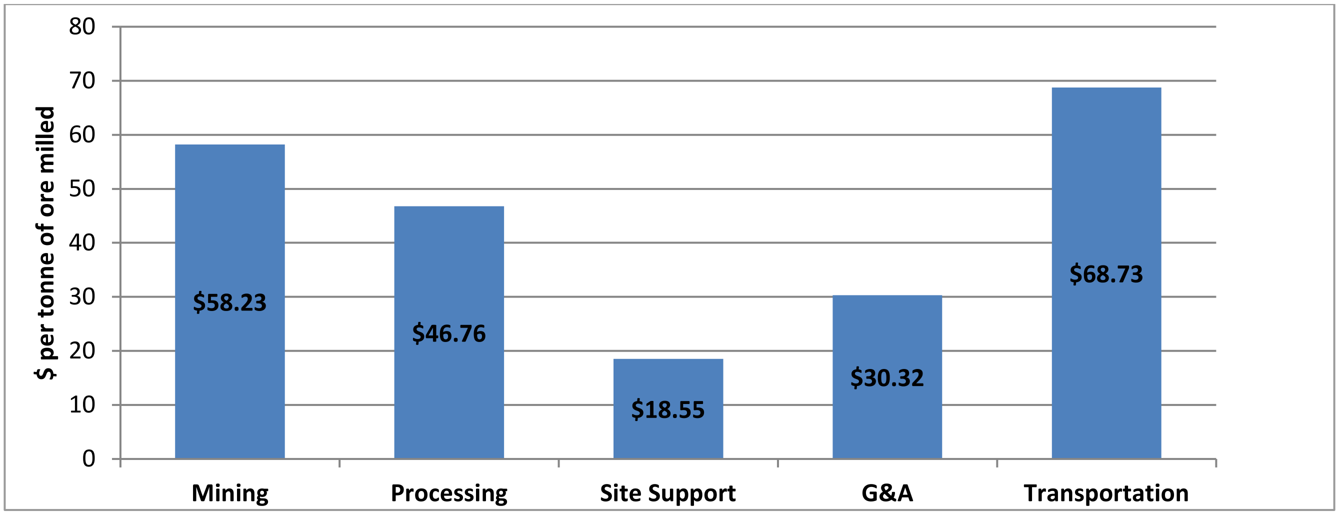


Figure 21.2 shows the cash operating cost distribution.

Figure 21.2 Mining, process plant, site infrastructure and transportation operating cost distribution



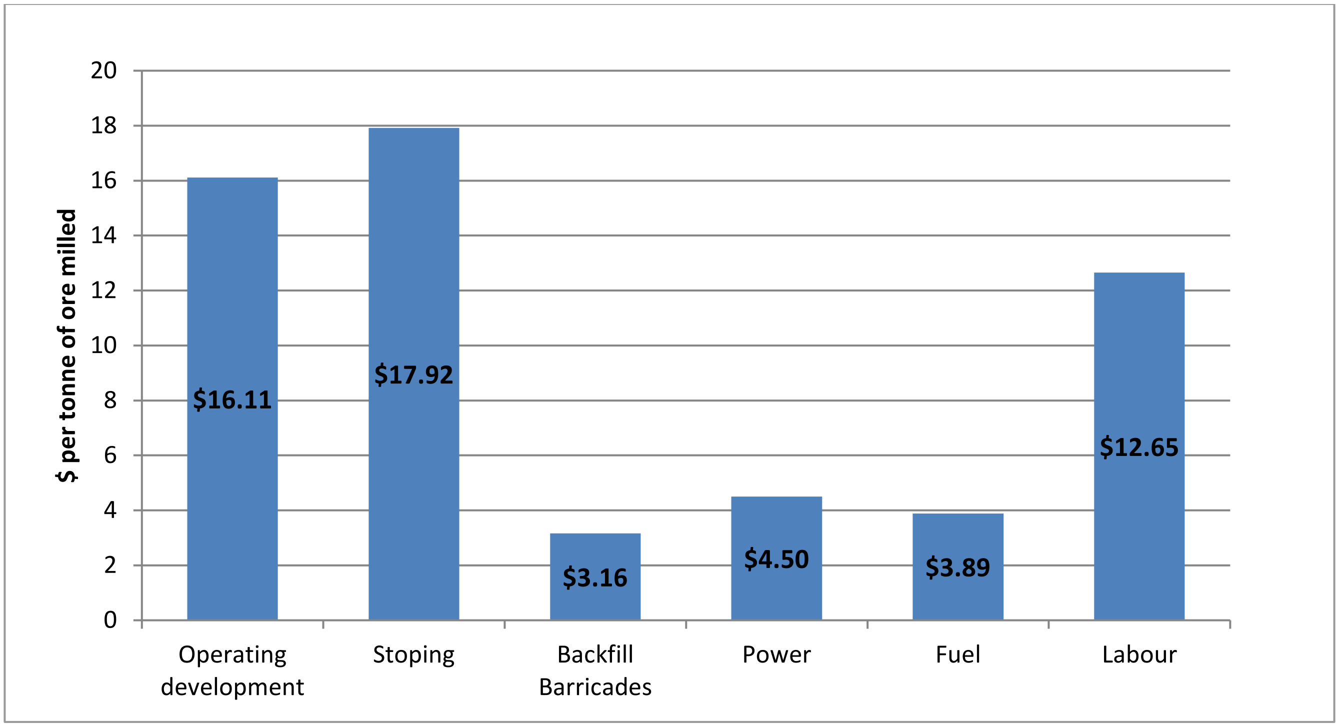
Mine operating cost

Mine operating costs are estimated according to unit prices tendered by the selected mining contractor, Procon, and on development rates and quantities developed by AMC based on first principles; these have been multiplied by annual quantities estimated by AMC and factored for increasing mining depth as appropriate. During the period where CZN has taken over mining operations, operating labour costs have been generated from personnel numbers and work-ups of individual personnel labour and benefits cost to the company. Mine infrastructure costs were estimated by AMC, comprising:

- Ventilation power and hardware;
- Dewatering power and hardware;
- Mobile equipment power supply;
- Mobile equipment diesel fuel;
- Mine air heat, LNG fuel.

The distribution of mine cash operating costs is shown in Figure 21.3.

Figure 21.3 Mine cash operating cost distribution



Processing, site surface, and general and administrative operating cost

Processing, site support service and G&A operating costs have been estimated based on staffing levels and labour rates agreed between CZN and Ausenco, and materials costs provided by Ausenco. The actual cost, year by year, will vary in accordance with the tonnage feeding the processing plant.

The operational cost includes the following:

- Process plant operating costs – mill staff, plant operators, process plant maintenance, laboratory staff, process plant consumables, grinding media, reagents, power generation (process plant and site infrastructure).
- G&A operational costs – administrative staff site office expenses, consultants, personnel transport, camp accommodation, insurance, contract services, mobile equipment fuel and maintenance.
- Site support operating costs – maintenance labour and LNG.
- Mine water treatment operating costs – power, consumables, reagents and labour (estimated by specialist vendor).

Figure 21.4, Figure 21.5, and Figure 21.6 show the respective split of costs in the processing, site surface support and G&A areas.

Figure 21.4 Process plant operating cost distribution

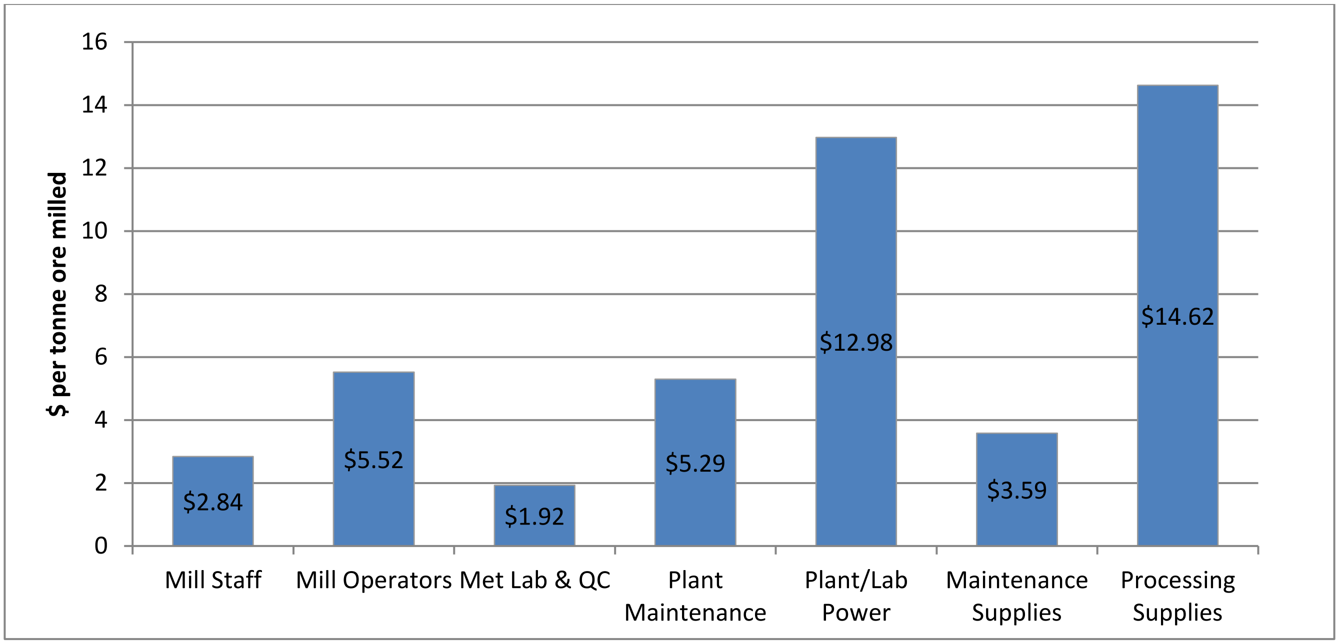


Figure 21.5 Site support operating cost distribution

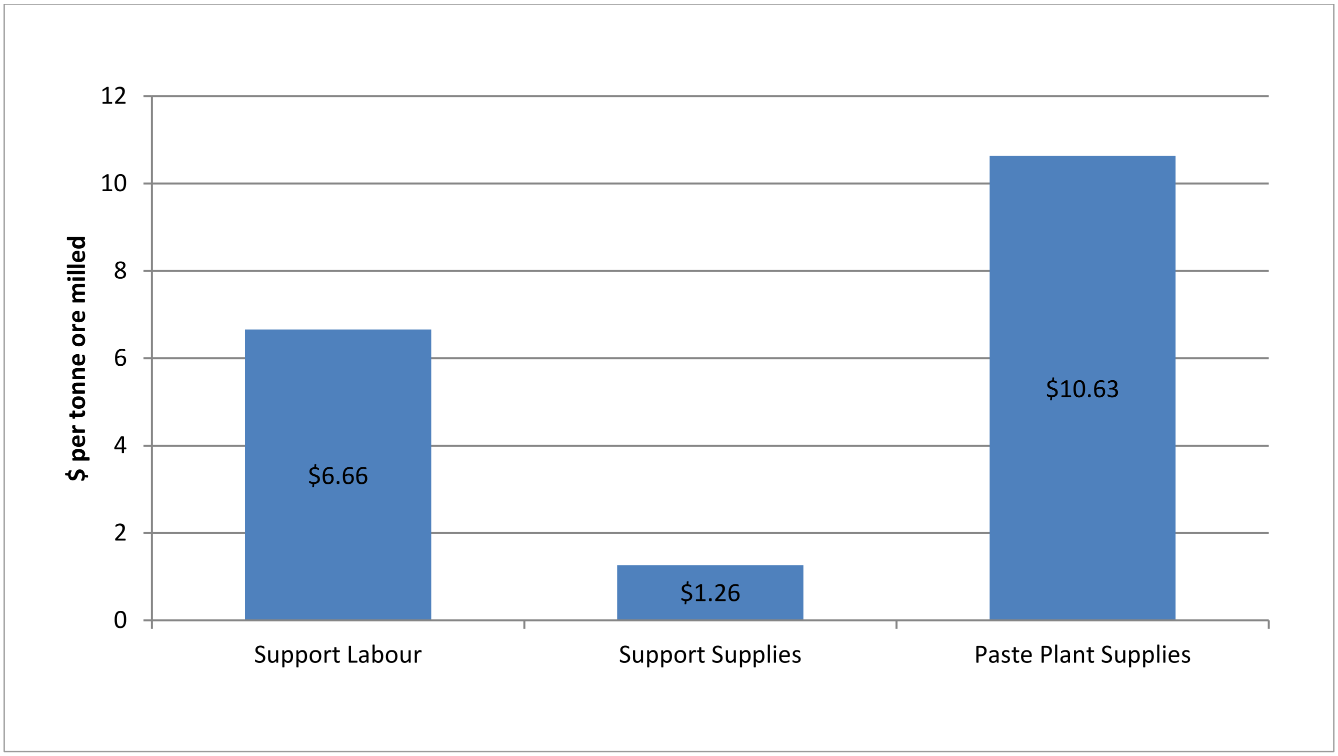
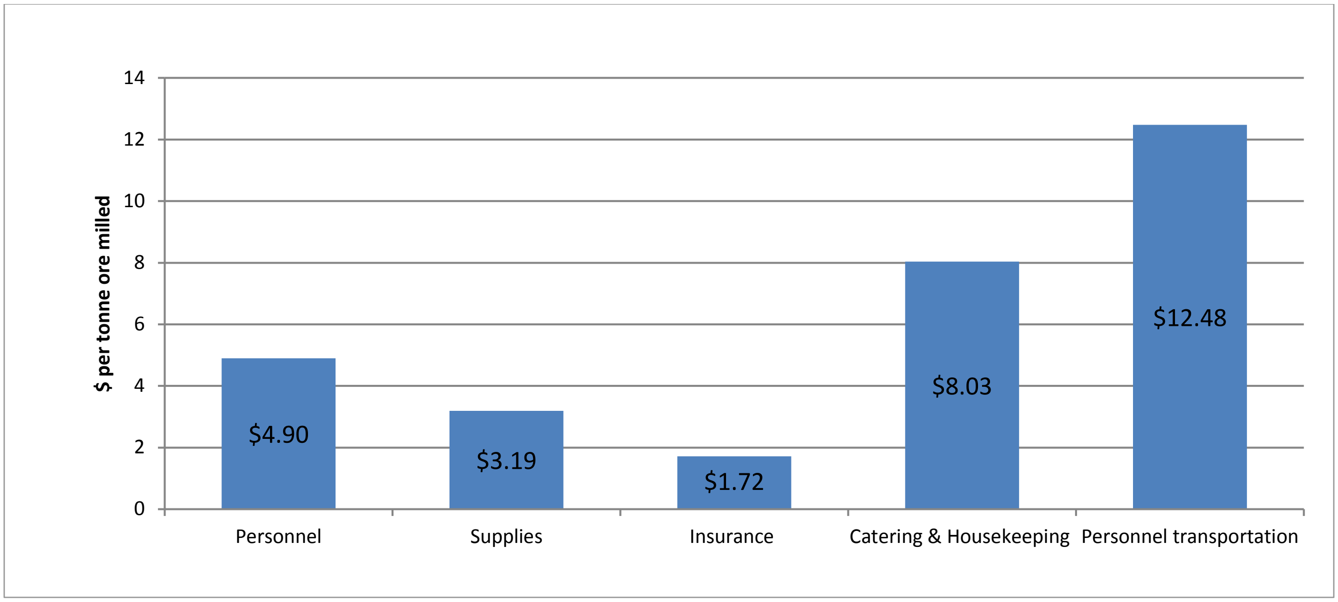


Figure 21.6 G&A cash operating cost distribution



21.3.2 Annual operating manpower costs

The process plant manpower includes plant staff, plant operators, plant maintenance crew, and laboratory technicians.

The manpower for site services includes all site maintenance labour including maintenance superintendent, maintenance planner, foreman, electricians, mechanics, pipefitters, carpenters, welders, labourers and mobile equipment operators.

The manpower for G&A includes all positions required for general and administrative support at the site including general manager, accounting, purchasing / warehouse personnel, human resources, and health and safety. Catering manpower costs are included in the camp-person day rate.

Labour costs for the underground mine include mine administrative staff and technical services personnel for all years of mine operation, including the years when the mining contractor is on-site, and, as indicated above, the costs also include development and production miners, and underground maintenance personnel, for the years of operation following the mining contractor's departure.

The concentrate transport manpower costs are included in the transportation cost.

Labour costs were compiled by CZN and accepted by AMC and Ausenco as appropriate and are based on Canadian Mine Salaries, Wages, and Benefits Survey, published by InfoMine USA Inc.

The total manpower required (on payroll) to operate the Prairie Creek site (mine, process plant and site infrastructure) is shown in Section 24 (Table 24.1).

21.3.3 Annual supplies costs

Processing supplies costs are based on all consumables and supplies for the process plant, including equipment wearing parts, grinding balls, reagent chemicals and power.

The cost of mining is based on contractor unit pricing and on development rates and quantities developed by AMC based on first principles; material costs for the contractor period are included in the unit rates and are not separated out. Major categories of G&A costs have been estimated by CZN and have been accepted by AMC and include costs of transporting all site personnel; catering and housekeeping for all personnel on site; and administrative personnel and associated supplies.

Site support costs include the day-to-day operation and maintenance of the surface facilities and LNG used for heating of site infrastructure buildings such as the camp and the warehouse.

Power cost and diesel fuel consumption

The steady-state, site-wide average power demand is estimated at 6.5 MW. The 2017 Feasibility Study incorporates a non-binding indicative proposal from Northwest Territories Power Corporation to supply turn-key type power generation utilizing four new 2.77 MW dual-fuel LNG/diesel-powered generator units that will provide power and heat for the site. The cost for LNG generated power is projected at $0.25/kWh.

Table 21.8 shows the estimated average power consumption by main area of use.

Table 21.8 Power consumption

Area	Average GWh/year	%
Mine	9.4	21.2
Process plant	30.0	67.7
Site infrastructure (camp and offices)	4.9	11.1
Total	**44.3**	**100**

The estimated annual diesel fuel consumption is indicated in Table 21.9 (LOM average).

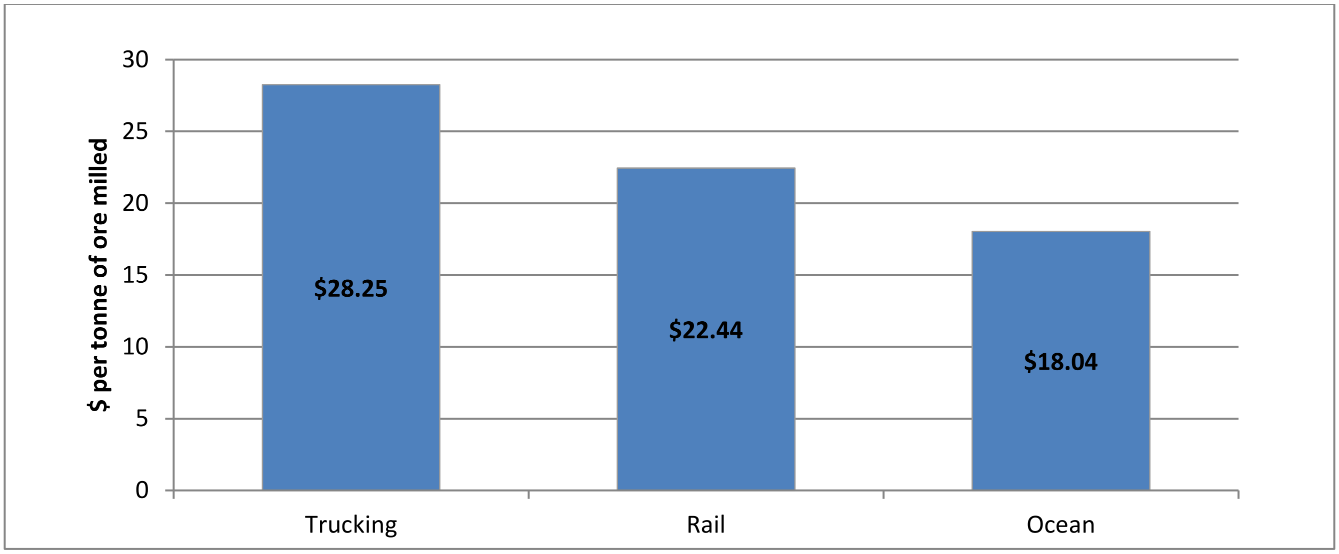
Table 21.9 Annual diesel fuel consumption

Area	Million litres	%
Mine	1.3	65
Surface mobile equipment	0.7	35
Total	**2.0**	**100.0**

21.3.4 Transportation cost

The total transportation cost is estimated as $68.73 per tonne of process plant feed (ore), which includes $28.25 per tonne (ore) for road / truck transportation, $22.44 per tonne (ore) for rail transport, $18.04 per tonne (ore) for terminal handling, ship loading and ocean freight, as shown in Figure 21.7.

Figure 21.7 Transportation operating cost distribution



Concentrate will be hauled from the Prairie Creek Mine site to Fort Nelson by truck; the operating cost includes:

- Trucks and trailers
- Labour including drivers, mechanics, supervision and logistics
- Maintenance of equipment
- Fuel
- Accommodation of drivers at site

Road maintenance was estimated by Allnorth and is contained in sustaining capital costs. Cost estimates for rail transport were provided by CN Rail, and terminal handing and loading charge estimates were provided by Kinder Morgan. Cliveden provided the estimated for ocean freight and insurance to Asia.

22 Economic analysis

The key parameters and assumptions used in the base case economic model and the key production and economic results are summarized in Table 22.1.

Table 22.1 Economic summary of the 2017 FS

Mine and mill parameters		Concentrates			
		Type	10 yr W. avg. tonnes	Average grade	Payability
Total ore mined (million tonnes)	8.07	Zinc concentrate	64,800	Zinc: 59%	Zinc: 85%
Mining rate (tonnes / day)	1,600			Silver: 136 g/t[3]	Silver: 70%
Milling rate (tonnes / day) post-DMS	1,200	Lead concentrate	71,600	Lead: 62%	Lead: 95%
LOM (years)	15			Silver: 800 g/t	Silver: 95%
Mine and mill statistics					
Metal		10 yr ore grade (weighted average)	Ore grade LOM (weighted average)	Mill recoveries LOM (weighted average)	10 yr average annual contained metal
Zinc		8.50%	8.70%	83%	95M lbs[4]
Lead		9.30%	8.10%	88%	105M lbs[4]
Silver		139 g/t	124 g/t	87%	2.1M oz[4]
Project assumptions base case					
Zinc price	**US$1.10/lb**	**Treatment charges**		Exchange rate	C$1.25:US$1.00
Lead price	**US$1.00/lb**	US$172/tonne Zn Con		Discount rate	8%
Silver price	**US$19.00/oz**	US$130/tonne Pb Con			
Operating and capital costs					
Operating costs[2]	LOM $/t ore mined	Capital costs			$M
Mining	58	Pre-production capital			253
Processing	47	Contingency			26
Site services	19	Total pre-production capital			279
G&A	30	Sustaining capital			117
Total on-site costs	154	Working capital			36
Transportation[1]	69				
Total operating costs[2]	223				
[1] Includes truck, rail, handling and ocean shipping		[3] Subject to a deduction of 3 oz. per tonne of concentrate			
[2] Does not include treatment, refining charges, royalty		[4] Total metal contained in both lead and zinc concentrates			
Economic results (LOM)				Pre-tax	Post-tax
Cash flow undiscounted ($M)				899	562
NPV @ 8% ($M)				344	188
NPV @ 5% ($M)				497	291
IRR (%)				23.8	18.4
Payback period (years from first revenue)				4.4	4.6
Average annual EBITDA ($M)				81	

22.1 Sources of estimated costs and assumptions used

This Technical Report contains forward-looking information regarding projected mine production rates, construction schedules and forecast of resulting cash flows. Factors such as the ability to obtain major equipment or skilled labour on a timely basis, to achieve the assumed mine production rates at the assumed grades, or to achieve the projected capital and operating costs 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 21.0 of this report and are presented in 2017 dollars.

Inputs into the financial model were the responsibility of Ausenco and AMC for the post-tax economic model and associated sensitivity charts, owner's cost, commodity prices, associated road costs, power cost, net smelter return / smelter terms and reclamation estimates. Ausenco was responsible for the surface capital and operating costs, and mill and surface workforce. Ausenco and Frank Wright were responsible for metallurgical balance, and mill capital and operating cost estimates. AMC completed the mine production schedule and the mine capital and operating cost estimates.

Assumptions

The project is assumed to be financed at 100% equity.

All costs included in this report and in the financial model are expressed in 2017 Q3 Canadian dollars (C$) without allowance for escalation or inflation, unless specified otherwise.

Metal prices used in the Base Case economic analysis are discussed in Section 19 and presented in Table 22.2 reported in US dollars (US$).

Table 22.2 Forecast Base Case Long-term metal prices for zinc, lead, and silver

	Life of mine
Zinc ($/lb)	$1.10
Lead ($/lb)	$1.00
Silver ($/oz)	$19.00

Other economic factors used include the following:

- Discount rate of 8% (sensitivities using other discount rates have been calculated).
- Nominal 2017 dollars.
- Exchange rate equal to C$1.25 to US$1.00.
- Life of mine equal to 16 years (15 years of concentrate production).
- Inflation not included.
- Provincial Sales Tax not included.
- Numbers are presented on 100% ownership and do not include management fees or financing costs.
- Exclusion of all pre-development and sunk costs (i.e. exploration and resource definition costs, engineering fieldwork and studies costs, environmental baseline studies costs, etc.).

Timing of revenues and working capital

Mine revenue is derived from the sale of concentrates into the international marketplace subsequent to processing at a smelter. Revenue is assumed to be received three months after concentrate production at the Prairie Creek mine site.

Operating costs are assumed to be paid immediately in the case of labour and on net 30 day accounts payable terms in the case of supplies and contractors.

Working capital of $36 million is estimated to be required over the first six months subsequent to commercial production based on the above assumptions.

22.2 Capital costs

The initial capital required to bring the Prairie Creek Project to production is estimated to be $278.9 million, including a contingency of $26.0 million and excluding working capital. The pre-production capital includes mine development, refurbishment of the processing plant and site infrastructure and services, construction of the all season road and owner's costs. The pre-production capital costs sub-divided by Work Breakdown Structure (WBS) cost areas are summarized in Table 22.3 below.

Table 22.3 Pre-Production capital cost summary

Description	Project Y-02 ($M)	Project Y-01 ($M)	Project Y01 ($M)	Total cost ($M)
Mine development	2.6	13.6	21.5	**37.7**
Site preparation	4.3	12.5	2.6	**19.4**
Mill process plant [1]	9.0	18.9	3.2	**31.1**
Paste tailings plant and process	2.9	16.6	3.4	**22.9**
Construction indirects including EPCM [2]	10.9	7.8	5.1	**23.8**
Other site infrastructure	6.7	7.7	1.5	**15.9**
All season road	13.0	41.6	13.9	**68.5**
Owner's costs	6.8	15.3	11.5	**33.6**
Total (excluding contingency)	**56.2**	**134.0**	**62.7**	**252.9**
Contingency	5.5	12.3	8.2	**26.0**
Total Pre-Production Capital	**61.7**	**146.3**	**70.9**	**278.9**

1. Includes dense media separator, structural upgrading, instrumentation, flotation circuit upgrade, reagent handling, and piping.
2. Includes engineering and construction of surface facilities, freight and logistics, initial fills, and spares.

22.3 Sustaining capital

Sustaining capital costs (Table 22.4) over the remainder of the mine life are estimated at $117 million, including mine development of $73.4 million, the purchase and rebuild of mobile mining equipment, estimated at $20.0 million, leasing costs of capital items of $11.0 million, and all season road capital in the amount of $8.6 million. Process and infrastructure sustaining capital was estimated at a total of $9.5 million over the life of the mine.

Table 22.4 Sustaining capital summary

Sustaining capital	LOM $M
Mining	73.4
Mobile equipment	20.0
Capital leases	11.0
All season road	8.6
Infrastructure	9.5
Salvage	(5.5)
Total	**117.0**

22.4 Operating costs

Operating costs per tonne milled and delivered to the smelter are shown in Table 22.5 for the life of the mine.

Table 22.5 Total operating cost summary

Total operating cost	($/t)
LOM (8,071,463 t)	
Mining	58.23
Milling and processing	46.76
G&A	30.32
Site services	18.55
Sub-total	**153.86**
Transportation[1]	68.73
Total	**222.59**

Totals do not necessarily equal the sum of the components due to rounding.
1. Includes truck / rail / handling / shipping.

22.5 Concentrate smelter terms

Canadian Zinc has signed MOUs with Korea Zinc and Boliden for the sale of zinc and lead concentrates. The MOUs with each of Korea Zinc and Boliden set out the intentions of Canadian Zinc and each of Korea Zinc and Boliden to enter into concentrate sales agreements for the concentrates to be produced from the Prairie Creek mine on the general terms set out in the MOUs, including commercial terms which are to be kept confidential.

The sale agreements will account for all of the planned production of zinc concentrate and about half of the planned production of lead concentrate for the first five years of operation at the Prairie Creek mine. The sales agreements will provide that treatment charges will be set annually at the annual benchmark treatment charges and scales, as agreed between major smelters and major miners.

Payables, penalties and quotational periods will be negotiated in good faith annually during the fourth quarter of the preceding year, including industry standard penalties based on indicative terms and agreed limits specified in each MOU.

Treatment and refining charges, payables, including deductibles, and penalties, vary with smelter location, and individual smelter terms and conditions. The economic model in the 2017 FS includes mercury penalties ranging from $500,000 to $3M per year, with an average of approximately $1.5M per year.

The following assumptions regarding concentrate marketing have been used in generating the economic model inputs:

- Concentrates will be transported to the Port of Vancouver and from there shipped to smelters in Asia.
- Concentrates will be shipped in 10,000 t lots.

Based on these assumptions, estimates for indicative “world smelter terms” with respect to treatment charges, penalties, and other terms have been estimated – see Table 22.6 and Table 22.7. These estimates were reviewed in light of the current market, as well as historic and future expected trends. The Economic Model used in the 2017 FS has been prepared assuming average blended indicative treatment charges of US$172 per tonne for zinc sulphide concentrates and US$130 per tonne for lead concentrates, both referenced as being substantially higher than current spot treatment charges. Industry standard penalties have also been modelled, including mercury penalties of US$1.75 for each 100 ppm above 100 ppm Hg per tonne of concentrate.

Table 22.6 Zinc concentrate indicative terms used in Economic Model

Zinc sulphide concentrate indicative terms used in economic model	
Payables	
Zinc Silver	Pay for 85% of the final zinc content, subject to a minimum deduction of 8%. Deduct 3 ounces per dry metric tonne and pay for 70% of the balance.
Deductions	
Treatment charge	US$172 per dry metric tonne of zinc concentrates
Penalties	Per dry metric tonne
Lead Arsenic Antimony Silica Cadmium Mercury	US$1.50 for each 1% above 3% US$1.50 for each 0.1% above 0.3% US$1.50 for each 0.1% above 0.3% US$1.50 for each 1% above 3% US$2.00 for each 0.1% above 0.25% US$1.75 for each 100 ppm above 100 ppm

Source: Cliveden Trading AG.

Table 22.7 Lead concentrate indicative terms used in Economic Model

Lead oxide and sulphide concentrates indicative terms used in economic model	
Payables	
Lead Silver	Pay for 95% of the final lead content, subject to a minimum deduction of 3%. Pay for 95% of the final silver content subject to a minimum deduction of 50 grams.
Deductions	
Treatment charge Refining charge	US$130 per dry metric tonne of lead concentrates Silver: US$1.50 per troy ounce payable.
Penalties	Per dry metric tonne
Zinc Arsenic Antimony Mercury	US$0.00 for each 1% above 4% US$1.50 for each 0.1% above 0.3% US$1.50 for each 0.1% above 0.3% US$1.75 for each 100 ppm above 100 ppm

Source: Cliveden Trading AG.

22.6 Taxes and royalties

The Prairie Creek Mine is subject to all applicable Canadian federal and Northwest Territories territorial taxes and royalties. This is essentially a three-tiered system including federal income tax, territorial income tax and territorial tax royalty. Federal income taxes are estimated to be 15% of taxable income. The mine will be subject to an 11.5% territorial income tax rate on taxable income.

It should be noted that Canadian Zinc is incorporated in the province of British Columbia and the Company will therefore be subject to a different taxation scheme than that of the mine.

Federal and provincial corporate income tax

Federal tax rate of 15.0% and a NWT (11.5%) rate were used to determine a blended 26.5% rate, which was used to calculate income taxes.

Mineral property tax pools

Canadian Exploration Expense (CEE) and Canadian Development Expense (CDE) tax pools were used with applicable opening balances to calculate income taxes.

Federal Investment Tax Credits

Applicable opening balances were used to calculate the Federal Investment Tax Credits for the project with respect to the pre-production capital costs.

Capital Cost Allowance (CCA)

Capital cost specific CCA rates were applied to and used to calculate the applicable amount of CCA the Company can claim during the life of the Project.

NWT Mineral Tax Royalty

The Northwest Territories Mining Regulations impose a mining royalty on an operator or owner of a mine located in the Northwest Territories. The royalty is a percentage of the mine's annual profit. The profit is calculated as the mine's total revenue less the cost of mining and processing and other deductions and allowances. The royalty rate applied to the annual mine profit is the lesser of 13% of the total profit and the sum of escalating tiered marginal royalty rates ranging from zero percent to the maximum of 13%.

Northwest Territories royalties as estimated in the Economic Model are approximately 11% of the value of the life of mine output, although this may vary depending upon the profit of the mine during a given year. Calculations of what is considered taxable income and the value of the output of the mine depend upon a number of factors and variables. The value of the output of a mine for a fiscal year is the amount by which the fair market value of minerals produced exceeds the permitted deductions and allowances.

Royalty

The Company has agreed to a net smelter return royalty (NSR) arrangement with Sandstorm Gold of 1.2%. The cost of this royalty is reflected in the economic model.

Aboriginal participation

Impact Benefits Agreements (IBA) have been signed with the Nahanni Butte Dene Band (Nahanni Butte) and the Liidlii Kue First Nation (Fort Simpson), two local Bands of the Dehcho First Nations, in proximity to the Prairie Creek Mine. While the terms of the IBAs are confidential, the associated or related costs have been included in the Economic Model.

22.7 Summary of outputs

A summary of the financial model output values is shown in Table 22.8.

Table 22.8 Output values of financial model

Financial analysis	Pre-tax	Post-tax
Average annual EBITDA	$81 M	-
NPV (8% discount rate)	$344 M	$188 M
IRR	23.8%	18.4%
Payback period	**4.4 years**	**4.6 years**

Over the first ten years of production, the 2017 FS indicates average annual production of approximately 64,800 tonnes of zinc concentrate and 71,600 tonnes of lead concentrate with a combined metal content of approximately 95 million pounds of zinc, 105 million pounds of lead and 2.1 million ounces of silver. The 2017 FS indicates average annual earnings before interest, taxes, depreciation and amortization of $81M per year and cumulative EBITDA earnings of $1.294 billion over an initial mine life of 15 years, using Base Case metal price forecasts of US$1.10 per pound for zinc, US$1.00 per pound for lead and US$19.00 per ounce for silver, with an exchange rate of C$1.25: US$1.00.

22.8 Financial analysis

22.8.1 Introduction

The Base Case economic cash flow model has been developed using long-term metal price assumptions of US$1.10/lb zinc, US$1.00/lb lead, and US$19.00/oz silver, and an exchange rate of C$1.25:US$1.00. Determination of metal prices for use in the 2017 FS has included consideration of consensus price forecasts published by Consensus Economics Inc. as at September 2017, and a review of market commentary published by various services, including the International Lead and Zinc Study Group, CRU, Metals Bulletin Research, Wood Mackenzie, and other industry sources as discussed in Section 19. Current metal prices, rolling three-year averages, and prices used in recent similar mining project studies were also considered for the Prairie Creek economic evaluation.

Based on the evaluation of consensus forecasts and review of published information from various market commentators and other sources as referenced above, the long-term metal prices selected for the base case economics of the Prairie Creek Project are considered to be reasonable and in line with market projections.

Project cash flows

Table 22.9 is a summary table from the economic cash flow model developed by CZN.

The table shows the year-by-year mill feed grades delivered to the mill from the associated Prairie Creek Mine production and, after processing, the total metal content and gross metal value of the concentrates.

The table also shows the year-by-year gross smelter revenue (after calculating payablility) and resultant net smelter revenue, after deducting treatment and refining charges and penalties.

Associated annual operating costs (including site costs and transportation costs) are deducted from the annual net smelter revenues to generate income before taxes and royalties, yielding net income from mining operations.

Total Capital Expenditures are accounted for in the year incurred to generate undiscounted cash flow on a year-by-year basis.

Finally, discounted Pre and Post-tax Cash Flows are shown using a discount rate of 8%.

Table 22.9 Prairie Creek Mine Project cash flows

Period			-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16
PHYSICAL																				
Mill Feed	kt	8,071.5	-	-	**135.5**	**596.9**	**607.3**	**602.6**	**606.3**	**594.7**	**575.5**	**594.6**	**578.9**	**570.5**	**571.9**	**557.1**	**523.4**	**541.0**	**406.1**	**9.1**
Mill Feed Grades (from Mine)																				
Lead	%	8.1%	-	-	8.4%	9.7%	8.9%	9.2%	9.7%	10.1%	9.3%	8.3%	9.9%	8.5%	5.7%	7.7%	6.0%	4.4%	4.0%	3.8%
Silver	g/t	124	-	-	139	161	141	137	152	143	130	129	143	119	97	127	85	74	73	80
Zinc	%	8.7%	-	-	8.8%	11.4%	9.3%	7.8%	8.8%	7.0%	7.2%	8.6%	8.0%	8.0%	9.3%	10.0%	9.9%	7.6%	8.0%	7.9%
Concentrate Tonnages																				
Lead Concentrate - Oxide	kt	129.5	-	-	4.4	18.5	10.1	7.1	7.4	8.4	7.3	8.1	8.0	6.5	8.8	16.3	9.1	6.5	2.9	0.1
Lead Concentrate - Sulphide	kt	794.6	-	-	10.9	61.8	65.5	71.4	76.0	77.4	69.0	60.5	73.8	64.0	36.7	45.9	38.6	24.9	17.9	0.4
Zinc Concentrate - Sulphide	kt	975.4	-	-	13.5	82.9	76.6	67.4	78.2	58.7	60.8	75.1	66.5	68.0	74.0	72.0	72.8	58.9	48.8	1.1
Total Concentrate Produced	**kt**	**1,899.5**	**-**	**-**	**28.8**	**163.3**	**152.1**	**145.9**	**161.7**	**144.6**	**137.0**	**143.7**	**148.3**	**138.5**	**119.5**	**134.2**	**120.5**	**90.3**	**69.7**	**1.5**
Metal Contained in Concentrates[1]																				
Lead	kt	606.0	-	-	9.8	52.4	50.4	52.5	56.2	56.8	50.8	46.0	54.5	46.3	29.2	38.1	28.4	20.2	14.3	0.3
Silver	t	871.3	-	-	15.6	84.5	75.7	71.4	81.5	71.7	64.4	67.8	71.4	58.9	48.0	59.7	37.9	35.4	26.8	0.7
Zinc	kt	639.0	-	-	9.0	54.7	50.6	45.3	52.1	40.7	41.3	49.2	45.0	44.8	46.6	46.5	45.6	36.7	30.3	0.7
COMMODITY PRICES & FX																				
Zinc	USD/lb	1.10	1.10	1.10	1.10	1.10	1.10	1.10	1.10	1.10	1.10	1.10	1.10	1.10	1.10	1.10	1.10	1.10	1.10	1.10
Lead	USD/lb	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00
Silver	USD/oz	19.00	19.0	19.0	19.0	19.0	19.0	19.0	19.0	19.0	19.0	19.0	19.0	19.0	19.0	19.0	19.0	19.0	19.0	19.0
Exchange Rate	USD:CAD	1.25	1.25	1.25	1.25	1.25	1.25	1.25	1.25	1.25	1.25	1.25	1.25	1.25	1.25	1.25	1.25	1.25	1.25	1.25
OPERATIONS, CASHFLOWS & VALUATION[2]																				
Revenue from:[2]																				
Lead		1,576.8			25.5	135.2	130.6	137.3	146.0	149.8	133.2	119.0	142.7	121.4	75.0	99.3	74.1	51.4	35.6	0.8
Silver		665.3			11.9	64.5	57.8	54.5	62.2	54.7	49.2	51.8	54.5	45.0	36.6	45.6	28.9	27.0	20.5	0.5
Zinc		1,735.8			24.1	148.3	137.0	120.5	139.9	105.0	108.7	134.3	118.8	120.7	131.0	126.8	127.0	104.5	87.2	1.9
Gross Revenue		**3,977.9**			**61.5**	**348.1**	**325.4**	**312.4**	**348.2**	**309.6**	**291.0**	**305.1**	**316.0**	**287.1**	**242.6**	**271.7**	**230.0**	**182.9**	**143.3**	**3.1**
Smelter - Deductions[3]		(454.0)			(6.6)	(39.4)	(36.5)	(33.5)	(38.1)	(31.2)	(30.7)	(34.9)	(33.5)	(32.1)	(30.8)	(32.4)	(30.3)	(24.2)	(19.5)	(0.4)
Gross Smelter Revenue		**3,523.9**			**54.9**	**308.7**	**288.9**	**278.9**	**310.0**	**278.4**	**260.3**	**270.2**	**282.6**	**254.9**	**211.8**	**239.3**	**199.6**	**158.7**	**123.8**	**2.7**
Smelter - TCRC		(401.6)			(6.17)	(34.91)	(32.38)	(30.75)	(34.32)	(30.18)	(28.63)	(30.53)	(31.11)	(28.91)	(25.53)	(28.41)	(25.05)	(19.35)	(15.09)	(0.33)
Smelter - Penalties		(31.2)			(0.29)	(2.98)	(2.65)	(3.89)	(3.44)	(3.14)	(2.75)	(3.14)	(1.69)	(1.98)	(1.27)	(1.10)	(1.26)	(0.79)	(0.82)	(0.02)
Net Smelter Revenue		**3,091.0**			**48.4**	**270.8**	**253.9**	**244.3**	**272.3**	**245.1**	**228.9**	**236.6**	**249.8**	**224.0**	**185.0**	**209.8**	**173.3**	**138.6**	**107.9**	**2.4**
Operating Costs																				
Mine		(469.5)			(15.9)	(38.0)	(37.2)	(34.6)	(33.8)	(31.9)	(34.4)	(34.7)	(31.1)	(27.7)	(30.6)	(29.8)	(35.5)	(37.3)	(16.2)	(0.8)
Processing		(377.4)			(5.6)	(27.8)	(28.5)	(28.2)	(28.4)	(27.9)	(27.0)	(27.9)	(27.1)	(26.7)	(26.8)	(26.1)	(24.5)	(25.4)	(19.0)	(0.4)
Surface Support		(151.2)			(1.7)	(9.6)	(10.0)	(10.2)	(10.6)	(11.1)	(11.1)	(11.1)	(10.8)	(11.2)	(12.0)	(11.6)	(10.8)	(11.6)	(7.5)	(0.3)
Transport		(554.7)			(9.7)	(40.8)	(41.5)	(41.2)	(41.4)	(40.7)	(39.5)	(40.7)	(39.7)	(39.2)	(39.2)	(38.3)	(36.1)	(37.3)	(28.6)	(0.8)
G&A		(244.7)			(5.8)	(19.9)	(17.9)	(18.0)	(18.1)	(17.8)	(17.2)	(17.8)	(17.3)	(17.0)	(17.1)	(16.6)	(15.6)	(16.2)	(12.1)	(0.3)
Total		**(1797.5)**			**(38.6)**	**(136.3)**	**(135.0)**	**(132.3)**	**(132.4)**	**(129.3)**	**(129.1)**	**(132.1)**	**(126.0)**	**(121.8)**	**(125.8)**	**(122.4)**	**(122.6)**	**(127.7)**	**(83.5)**	**(2.5)**
EBITDA		**1,293.5**			**9.8**	**134.5**	**118.9**	**112.0**	**139.9**	**115.8**	**99.8**	**104.4**	**123.8**	**102.2**	**59.2**	**87.4**	**50.7**	**10.9**	**24.4**	**(0.1)**
Taxes and Royalties		**(335.8)**			(0.8)	(5.7)	(6.0)	(15.2)	(39.7)	(37.3)	(32.8)	(35.8)	(44.1)	(36.2)	(20.3)	(31.9)	(17.9)	(3.4)	(8.4)	(0.3)
Income From Mining Operations		957.7			9.0	128.8	112.9	96.8	100.2	78.5	67.0	68.6	79.7	66.0	38.9	55.5	32.8	7.5	16.0	(0.4)
Total Capital Expenditures		**(395.9)**	**(58.4)**	**(151.6)**	**(76.8)**	**(35.5)**	**(35.1)**	**(12.1)**	**(8.0)**	**(4.3)**	**(3.5)**	**(2.8)**	**(4.1)**	**(3.6)**	**(2.0)**	**(1.4)**	**(1.4)**	**(0.7)**	**(0.1)**	**5.5**
Project Cash Flows																				
Undiscounted Pre-tax Cash Flows		897.6	(58.4)	(151.6)	(94.1)	69.1	92.2	93.1	129.3	122.4	98.9	96.2	119.3	110.6	54.1	90.1	55.0	17.4	38.7	15.2
Undiscounted Post-tax Cash Flows		561.8	(58.4)	(151.6)	(94.2)	66.4	86.6	87.3	104.3	68.8	66.4	65.1	81.2	61.6	22.6	78.4	16.9	7.5	43.9	4.4
Discounted Pre-tax Cash Flows		344.5	(56.2)	(135.1)	(77.6)	52.8	65.2	60.9	78.4	68.7	51.4	46.3	53.2	45.6	20.7	31.9	18.0	5.3	10.9	3.9
Discounted Post-tax Cash Flows		188.3	(56.2)	(135.1)	(77.7)	50.7	61.2	57.2	63.3	38.6	34.5	31.4	36.2	25.4	8.6	27.8	5.5	2.3	12.3	1.1
Cumulative Discounted Post-tax Cash Flows		**188.3**	**(56.2)**	**(191.2)**	**(268.9)**	**(218.2)**	**(157.0)**	**(99.8)**	**(36.6)**	**2.0**	**36.6**	**67.9**	**104.1**	**129.5**	**138.1**	**165.9**	**171.4**	**173.7**	**186.0**	**187.2**

[1] Total metal contained in both lead and zinc concentrates
[2] Figures in nominal CAD millions, unless stated otherwise
[3] See Section 22.5 for concentrate smelter terms and payability factors

Figure 22.1 shows the cash flows projected for the Prairie Creek Project using the base case assumptions. The Base Case demonstrates a pre-tax Net Present Value (NPV) of $344 million using an 8% discount rate, with an Internal Rate of Return (IRR) of 23.8%, and a post-tax NPV of $188 million, with a post-tax IRR of 18.3%, and yielding average annual earnings before interest, taxes, depreciation, and amortization (EBITDA) of $81 million per year and cumulative EBITDA earnings of $1.294 billion over an initial mine life of 16 years (15 years of concentrate production) with a post-tax payback of 4.6 years from first revenue.

Figure 22.1 Project cash flow

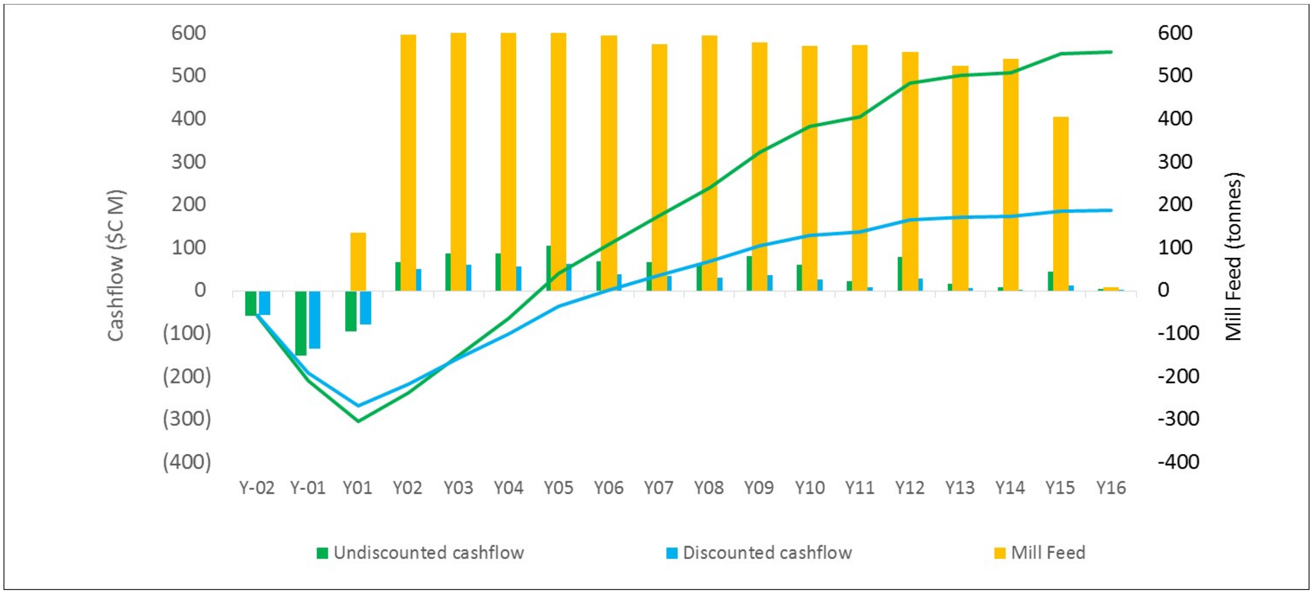
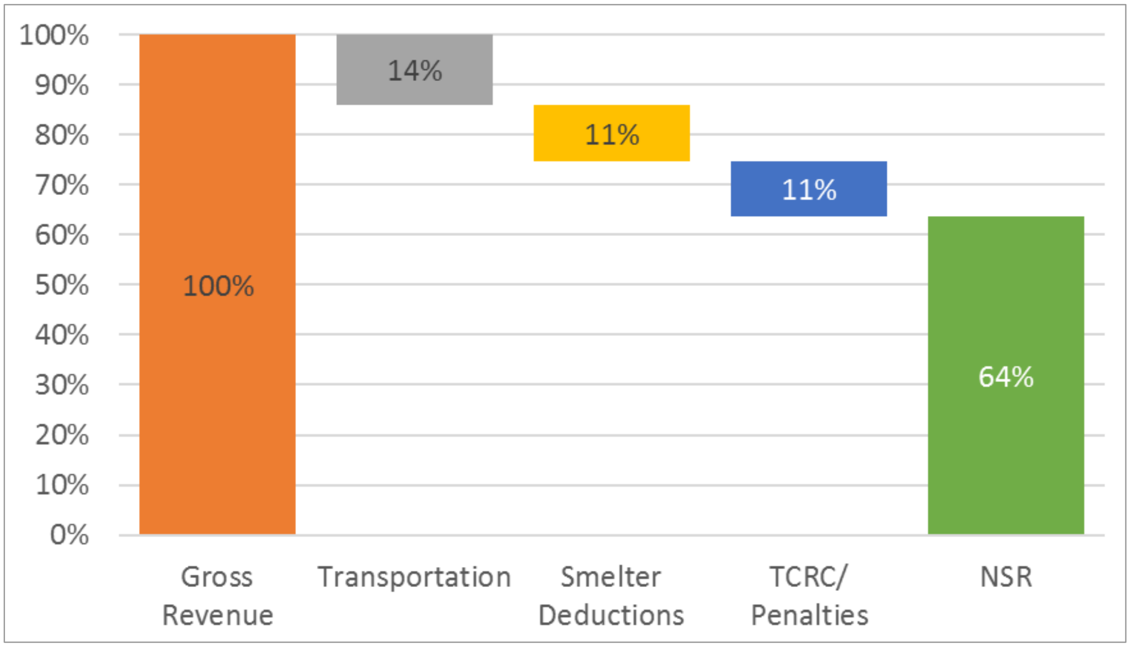


Figure 22.2 shows the net smelter return for the life of mine, which is projected to be 64% of LOM gross metal value after all off-site costs (smelter deductions; smelter treatment charges and penalties; and transportation costs) are incurred.

Figure 22.2 Life of mine net smelter return



22.9 Sensitivity analysis

A sensitivity analysis was conducted on the Base Case model to evaluate robustness against variations in financial parameters, specifically Base Case metal prices +/- 10% and the Base Case foreign exchange rate +/- 10% and +4%. The financial analysis centering on the Base Case, showing average annual EBITDA, NPV (at 8% and 5% discount rates), IRR and payback periods, on a pre-tax and post-tax basis is presented in Table 22.10 below. The Prairie Creek Mine is sensitive to movements in lead and zinc prices and in exchange rates. For example, a 10% improvement in the base case metal prices, or a 10% improvement in the assumed exchange rate would yield a post-tax NPV (8%) of $301M and IRR of 23.7%, or a post-tax NPV of $287M and IRR of 23.1%, respectively. Conversely, a 10% decline in the base case metal prices, or a 10% decline in the assumed exchange rate would yield a post-tax NPV (8%) of $74M and IRR of 12.4%, or a post-tax NPV of $88M and IRR of 13.2%, respectively.

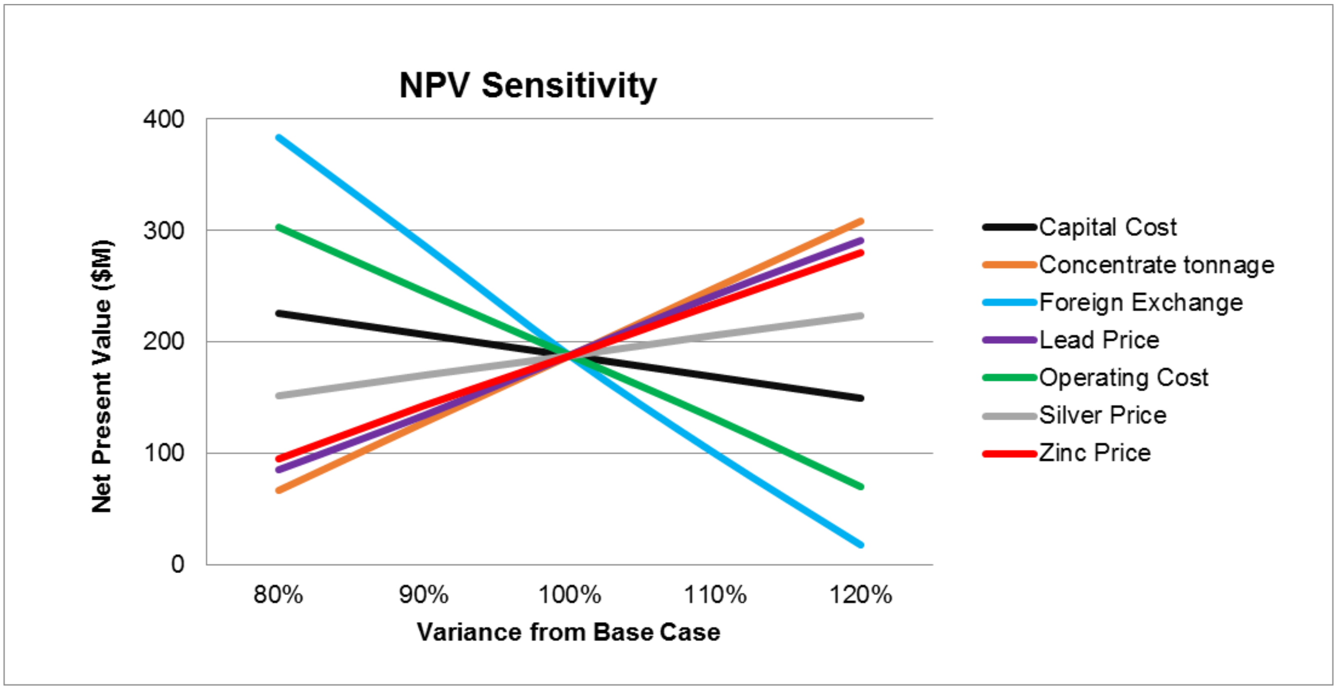
Table 22.10 Sensitivity analysis – Prairie Creek Mine

Metal price scenario[1]	**90%**	**100%**	**110%**
Average Annual EBITDA ($M)	59	81	103
Pre-Tax Cash Flow Undiscounted ($M)	546	899	1,251
Pre-Tax NPV @ 8% discount ($M)	166	344	523
Pre-Tax NPV @ 5% discount ($M)	270	497	724
Pre-Tax IRR	16.5%	23.8%	30.2%
Post-Tax Cash Flow Undiscounted ($M)	345	562	779
Post-Tax NPV @ 8% discount ($M)	74	188	301
Post-Tax NPV @ 5% discount ($M)	148	291	433
Post-Tax IRR	12.4%	18.4%	23.7%
Post-Tax Payback Period (years from first revenue)	5.7	4.6	4.1
Exchange rate scenario[2]	**C$1.125:US$1.00**	**C$1.30:US$1.00**	**C$1.375:US$1.00**
Average Annual EBITDA ($M)	62	89	100
Pre-Tax Cash Flow Undiscounted ($M)	589	1,022	1,208
Pre-Tax NPV @ 8% discount ($M)	188	407	501
Pre-Tax NPV @ 5% discount ($M)	298	577	696
Pre-Tax IRR	17.4%	26.2%	29.5%
Post-Tax Cash Flow Undiscounted ($M)	372	638	752
Post-Tax NPV @ 8% discount ($M)	88	228	287
Post-Tax NPV @ 5% discount ($M)	166	341	416
Post-Tax IRR	13.2%	20.3%	23.1%
Post-Tax Payback Period (years from first revenue)	5.6	4.5	4.2

1. Metal prices varied plus/minus 10% and exchange rate unchanged.
2. Exchange rate varied plus/minus 10% and plus 4%, and metal prices unchanged.

Figure 22.3 is a spider diagram showing the post-tax sensitivities to lead, zinc, and silver prices, operating costs, capital costs, and foreign exchange. Operating costs include smelter charges and taxation, as distinct from the cash operating costs itemized in Section 21.

Figure 22.3 Post-tax sensitivity chart



The NPV (and IRR) of the Prairie Creek project is most sensitive to foreign exchange rates, as most costs, except smelter treatment charges and penalties, are denominated in Canadian dollars, while revenue is paid in United States dollars. The project is moderately sensitive to operating costs and to zinc and lead prices, and least sensitive to silver prices and capital costs. A pre-tax sensitivity chart would show a similar shape to the post-tax scenario.

The Project pre-tax and post-tax net present values at 5% and 8% discount rates, and internal rates of return, are illustrated in Table 22.11, all at a Canadian / US dollar exchange rate of 1.25:1 (except for the Base Case also being shown at exchange rates of 1.375:1 and 1.125:1). The table also demonstrates the sensitivities of the Prairie Creek Project to zinc, lead, and silver prices and to the Canadian / US dollar exchange rate.

Table 22.11 Metal price and exchange rate sensitivity table

Metal prices		Pre-tax				Post-tax [1]			
Zinc / lead US$/lb	**Silver US$/oz**	**Undiscounted $M**	**NPV (5%) $M**	**NPV (8%) $M**	**IRR %**	**Undiscounted $M**	**NPV (5%) $M**	**NPV (8%) $M**	**IRR %**
0.80	17.00	139	10	(39)	5.5	75	(29)	(68)	3.3
0.90	18.00	452	211	120	14.4	282	109	43	10.6
1.10/1.00	**19.00**	**899**	**497**	**344**	**23.8**	**562**	**291**	**188**	**18.4**
1.20/1.00	19.00	1,033	582	410	26.2	644	344	230	20.4
1.10	20.00	1,077	614	437	27.3	671	364	247	21.3
1.20	21.00	1,390	815	596	32.7	863	489	346	25.7
1.30	22.00	1,703	1,017	755	37.7	1,053	612	444	29.8
1.10/1.00 [2]	**19.00 [2]**	**1,208**	**696**	**501**	**29.5**	**752**	**416**	**287**	**23.1**
1.20/1.00 [2]	**19.00 [2]**	**1,355**	**789**	**574**	**31.9**	**842**	**473**	**332**	**25.0**
1.10/1.00 [3]	**19.00 [3]**	**589**	**298**	**188**	**17.4**	**371**	**166**	**88**	**13.2**

1. Post-tax results include all taxes, royalties, aboriginal participation costs and the Sandstorm 1.2% NSR.
2. Foreign exchange assumed to be C$1.375:US$1.00 on these lines.
3. Foreign exchange assumed to be C$1.125:US$1.00 on this row.

A 'stressed case' sensitivity analysis using assumed metal prices of US$0.80/lb for zinc and lead and US$17/oz for silver, and an exchange rate of C$1.375:US$1.00, would indicate a pre-tax $NPV_{(8\%)}$ of $80M and IRR 12% (post-tax $NPV_{(8\%)}$ $16M and IRR 9%).

22.10 Summary

The 2017 Feasibility Study is based on optimization work completed over the past four years, including the 2015 underground exploration program at Prairie Creek, which resulted in increased total Measured and Indicated Resource tonnages of 32%. The FS indicates a Base Case Pre-tax Net Present Value of $344M using an 8% discount rate, with an Internal Rate of Return of 23.8%, and a post-tax NPV of $188M with a post-tax IRR of 18.4%.

An economic analysis was completed to assess the performance of the Project under the key production, cost and revenue assumptions adopted in the FS. Sensitivity analyses were performed for variation in metal prices, operating costs, capital costs, discount rates and exchange rates to evaluate their relative importance as project value drivers.

The Base Case economic model has been developed using long-term metal price assumptions of US$1.10/lb zinc, US$1.00/lb lead, and US$19.00/oz silver. Determination of metal prices for use in the 2017 FS has included consideration of consensus price forecasts published by Consensus Economics Inc. as at September 2017, and a review of market commentary published by various services, including the International Lead and Zinc Study Group, CRU, Metals Bulletin Research, Wood Mackenzie, and other industry sources as discussed in Section 19. Current metal prices, rolling three-year averages, and prices used in recent similar mining project studies were also considered for the Prairie Creek economic evaluation.

The 2017 FS indicates over the first ten years of production an average annual production of approximately 64,800 tonnes of zinc concentrate and 71,600 tonnes of lead concentrate, containing approximately 95 million pounds of zinc, 105 million pounds of lead and 2.1 million ounces of silver. The 2017 FS indicates average annual earnings before interest, taxes, depreciation and amortization of $81 million per year, and cumulative EBITDA earnings of $1.294 billion over an initial mine life of 15 years.

The Project economic analysis is based on 100% ownership of the Prairie Creek Project and is presented on a pre-tax and after-tax basis. It should be noted, however, that tax estimates involve many complex variables that can only be accurately calculated during operations and, as such, the projected after-tax results are only estimates. The Project net present value and internal rate of return values have been calculated using a discount rate of 8%.

The net smelter revenues were derived using processing schedules, process recovery and metal prices, net of concentrate payment terms, treatment charges and penalties. All revenues and costs have been accumulated annually. The sources of the key technical assumptions for mining and processing schedules and capital and operating costs are discussed in the corresponding sections of this report.

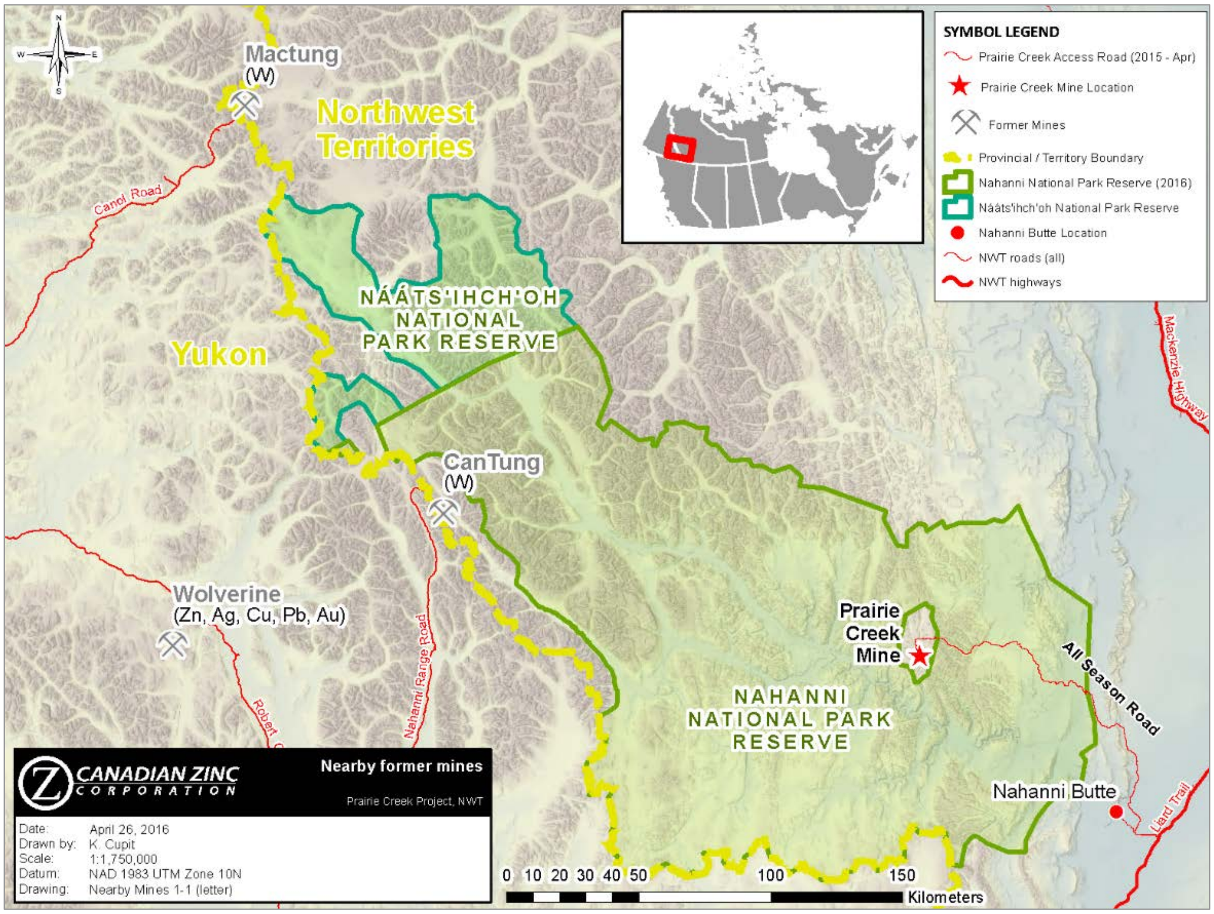
Summary highlights of the 2017 FS

- Post-tax Net Present Value, using an 8% discount rate, of $188 million, with a post-tax internal rate of return of 18.4%, based on base case metal price forecasts of US$1.10/lb for zinc, US$1.00/lb for lead and US$19.00/oz silver for the Life of Mine production at an exchange rate of C$1.25:US$1.00.
- Average EBITDA of $81 million per year and cumulative EBITDA of $1,294 million over the LOM.
- 15 year mine life based exclusively on a Mineral Reserve of 8.1 million tonnes, grading 8.6% zinc and 8.1% lead, with 124 g/t silver, including a Mineral Reserve in the Main Quartz Vein of 5.7 million tonnes, grading 8.7% zinc, 9.7 % lead and 149 g/t silver.
- Over the first ten years of production an average annual production of 64,800 dmt of zinc concentrate and 71,600 dmt of lead concentrate containing 95M lbs of zinc, 105M lbs of lead and 2.1M ounces of silver.
- Pre-production capital cost is estimated to be $279 million, of which $56 million will be incurred in Year 1, $134 million in Year 2 and $63 million in Year 3, with an additional contingency of $26 million.
- Average LOM cash operating costs per tonne of ore mined (before transportation costs) are estimated at $154/t.

23 Adjacent properties

There are no mineral properties immediately adjacent to the Prairie Creek Project since the site is somewhat uniquely located in relation to the Nahanni National Park Reserve. The NNPR was expanded in 2009 and, as part of the expansion agreement, the Prairie Creek Mine itself, and a large surrounding area of approximately 300 square kilometres, was specifically excluded from the Park. This excluded area remains under the jurisdiction of the GNWT and is surrounded by the NNPR as shown in Figure 23.1. In addition, road access into the Prairie Creek Mine area through the expanded Park area was also provided through an amendment to the *Canada National Parks Act,* solely for the NNPR.

Figure 23.1 Mineral deposits in the vicinity of Prairie Creek Mine



The closest mineral property of significance is the Cantung Mining Property, located 185 km to the west of Prairie Creek. 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, while located in the Northwest Territories, is distant from and had little influence on the Prairie Creek Mine, since the Cantung Mine was accessed by a road from Watson Lake in the Yukon Territory. On 19 November 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 (refer to Figure 23.1), 100 km west of Cantung Mine, was the closest base metal underground mine to Prairie Creek Mine but also closed in 2015. The Howard's Pass project contains a significant amount of drill-defined base metal Mineral Resources but remains undeveloped at this time.

24 Other relevant data and information

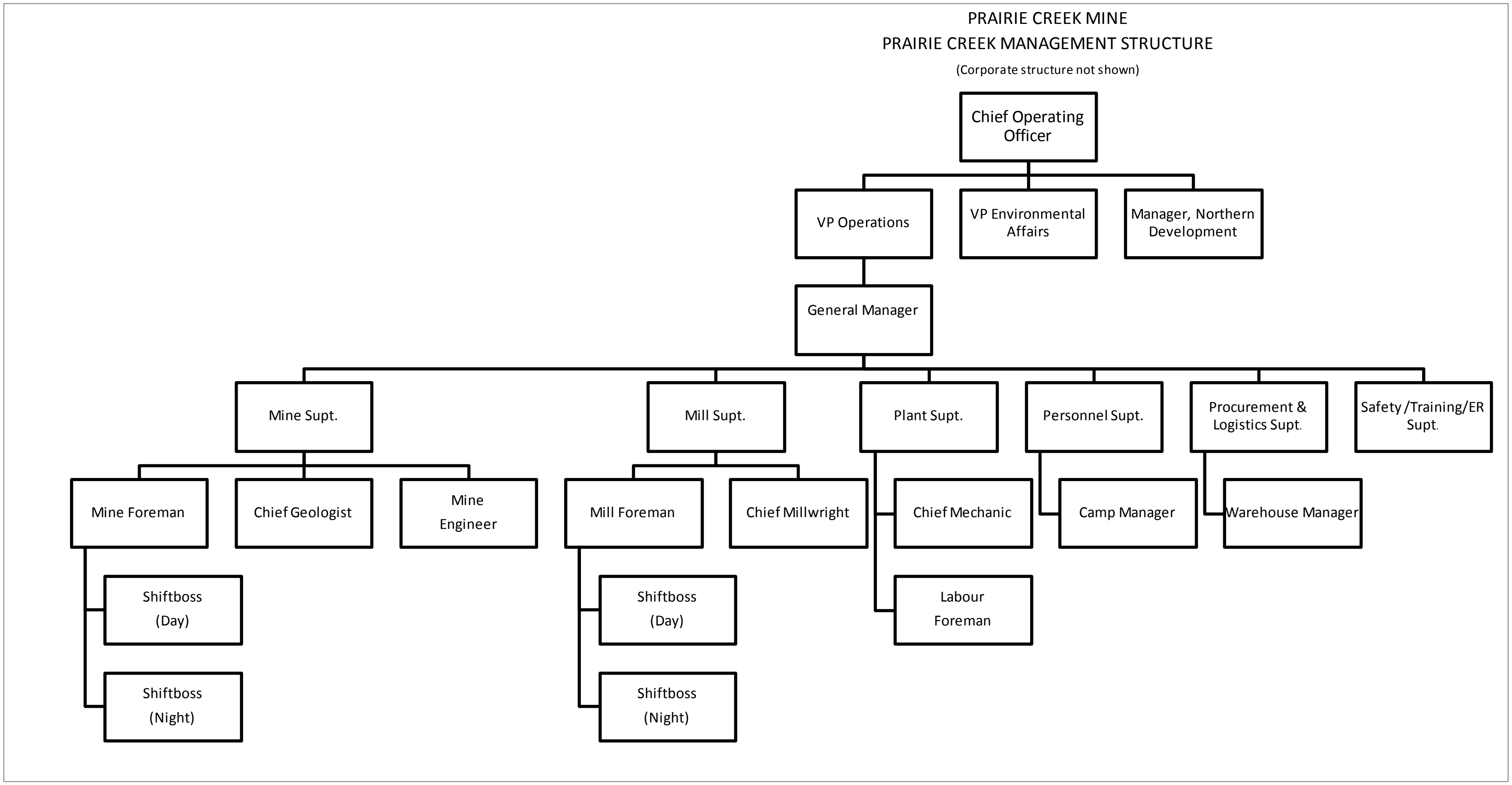
24.1 Organization and staffing

The site organization will be conventional for a mine of this size and type in a remote, isolated location. Figure 24.1 shows the projected management structure to operate the mine. The General Manager and subordinate positions will be site-based. Positions senior to the General Manager level will be head office-based.

Site employees will work a regular fly-in-fly-out rotation based on a two weeks on, two weeks off roster with paid travel by CZN.

The organization structure consists of supervisory and technical positions, shared between senior incumbents (A Team), and assistants (B Team), and skilled / manual positions, which are occupied continuously.

Figure 24.1 Management structure



24.1.1 Staffing levels

Currently envisaged average steady state mine site operation staffing levels are shown in Table 24.1. There will also be a trucking workforce, largely based in Fort Nelson, as shown in Table 24.2.

Table 24.1 Mine site operation staffing levels

Position	Total
Mill Superintendent	1
Mill General Foreman	1
Mill Foremen	4
Mill Operators	28
Metallurgical Engineer	1
Metallurgist	2
Mill Clerk	2
Assayer	6
Sample & Reagent Preparation Labourer	6
Sub-total, mill operations	**51**
Mill Maintenance Foreman	1
Maintenance Planner	1
Millwrights	8
Electricians	4
Welders	2
Crane / Equipment Operators	4
Apprentices / Labourers	4
Sub-total, mill maintenance*	**24**
Labour Foreman	2
Mechanics	4
Welder / Fabricators	2
Carpenters	2
Freight / Concentrate Handlers	4
Equipment Operators	14
Labourers / Apprentices	4
Water Treatment Plant	4
Sub-total, surface facilities	**36**
Chief Engineer	1
Mine Planning / Ventilation / Ground Control	4
Mine Technologist / Surveyor	2
Chief Geologist	1
Senior Geologist	1
Grade Control / Beat	2
Mine Superintendent	1
General Foreman	1
UG Supervisors	3
Safety / Training Co-Ordinator	2
Sub-total, mining supervision and technical	**18**
Miners	32
Longhole drillers	5
Longhole blasters	8
Services crew	8

Position	Total
LHD - production	8
Haul truck	12
Backfill - surface paste plant (costed to surface crew)	8
Construction / Fill Barricades / Pumping	8
Labour	4
Sub-total, mining	**93**
Maintenance Superintendent	1
Chief Mechanic	1
Chief Electrician	2
Maintenance Planner	2
Welder	4
Electricians	6
Diesel Mechanics	8
Plumber / Pipefitter	4
Sub-total, underground maintenance*	**28**
General Manager	1
Payroll Supervisor	1
Procurement & Logistics Superintendent	1
Purchaser	1
Warehouse / First Aid	4
Receptionist / Dispatcher	2
Human Resources Superintendent	1
Human Resources Coordinator	1
IT Technician	2
Safety / Training / Emergency Response Superintendent	1
Safety / Training / Emergency Response Officer	1
Nurse / Paramedic	2
Environmental Technician	2
Sub-total, G&A	**20**
Camp Operations	32
Trucking Supervisors	3
Concentrate Haul Truckers	18
Truck Mechanics	7
Sub-total, other	**60**
Total	**330**

*Some maintenance resources will be shared between underground and surface.

Table 24.2 Fort Nelson staffing levels

Position	Total
Manager	1
Supervisors	2
Drivers	22
Mechanics	7
Administrator	1
Total	**33**

24.1.2 Recruitment

Preference for site employment will be given to First Nations band members and northern residents; numbers and skill levels, particularly in the early part of the mine life, may be insufficient to staff more than a minority of positions.

24.1.3 Wage and salary levels, bonus

CZN will engage a specialist consultant to estimate the requisite wage and salary levels to recruit and retain a satisfactory workforce. For the FS, CZN has obtained basic information from authoritative sources for wage, salary and bonus levels appropriate to a mine of this size, type and location.

24.2 Project execution: path to production

24.2.1 Basic engineering

Basic engineering is projected to commence in early 2018 and will include:

- Process design
- Piping and instrumentation diagrams (P&IDs)
- Paste Plant optimization
- Civil design
- Fire protection preliminary design
- Preliminary mechanical and electrical equipment lists
- Preliminary process control philosophy
- Power and grounding study
- Preliminary single line diagrams
- Site plan frozen
- Initial engineering for portal structure and installation

Procurement services to support Basic Engineering will also be provided. The award of vendor engineering (with deferred purchase / delivery of components as late as possible) has been identified as a key opportunity to minimize capital spend during the Project execution phase.

24.2.2 Detailed engineering

Detailed design will continue immediately after basic engineering and will include:

- Concrete and structural steel
- Piping
- Electrical
- Instrumentation and controls
- Fire protection
- Process control philosophy
- Final site plan and drawings
- Single line diagrams
- Finalization of design for portal structure and installation
- Design for underground fans, magazines, and first dewatering
- Underground electrical system design
- Rock support design

Procurement and contracts services will be provided to purchase materials, equipment, and develop site contracts including:

- Procurement of long lead items.

- Procurement of equipment and bulk materials.
- Development of site contracts.
- Expediting vendor data and deliveries.
- Logistics coordination and material management.

24.2.3 Path to production - projected 2018 (Y-02) site works

The focus of the projected initial 2018 site work is to assess the condition of the major process plant equipment and to complete site investigations that will benefit the detailed design. Any rehabilitation of the site infrastructure required to support the site activities will also be completed. The current site facilities are capable of supporting a work crew up to 50 people on a short term basis in the existing ‘orange stripe’ trailers. Access to the site during early 2018 is limited to light aircraft and the main activities planned for this period minimize the import of materials due to the high cost of transport. These include:

- Mobilization of construction management personnel in early 2018.
- Refurbishment of the existing accommodation trailers (approx. 50 beds) including plumbing and connection to utilities.
- Refurbishment of sewage treatment plant (to support early works).
- Borrow pit and aggregate testing.
- Site stock-take of bulk materials and site clean-up.
- Testing of existing conveyor belting and bulk components, e.g. rubber seals, valves, fittings, cables, etc.
- Redline / as-built of existing facility including major commodities such as piping, electrical bulks and structural steel to feedback into the detailed design.
- Refurbishment and recertification of overhead cranes within the Mill Building. This is required before refurbishment of the grinding and crusher area major equipment can commence.
- Inspection and assessment of the HVAC system / air handling units within the mill building to determine suitability for reuse / refurbishment during construction.
- Temporary heating of facilities to support any cold weather work.
- Assessment and refurbishment of major crushing and grinding process plant components. The main focus will include determining the extent of repairs and to determine the need to dismantle and refurbish off-site in advance of winter road campaign #1.
- Set up Fort Nelson Marshalling Yard in advance of winter road campaign #1 (March 2018).

A key aspect of the envisaged 2018 site work is to execute critical activities to de-risk the project ahead of securing senior financing. The ‘orange stripe’ trailer facilities will be used to support the construction labour during this period. A winter road is projected to be available to mobilize materials and construction consumables (including diesel fuel) during March 2018 to support the construction and portal set-up activities. The main activities associated with this are:

- Geotechnical assessment and testing for the all-season road (performed during the availability of the winter road).
- Demolition of redundant infrastructure / equipment at site.
- Site clean-up.
- Temporary repairs to the administration and mill building roof.
- Refurbish ‘yellow stripe’ trailers for contractor’s trailers.
- Refurbish workshop to support refurbishment of process equipment.
- Continue refurbishment of process plant components (small crew).
- Perform compliance inspection and refurbish diesel tanks and pumps in advance of receiving incoming diesel on winter road campaign #2 to support projected 2019 (Y-01) site works.
- Refurbish and recertify propane tanks to provide back-up heating source.
- Drain WSP in advance of freezing conditions.

- Set up Fort Nelson marshalling yard in advance of winter road campaign #2 (projected for March 2019).
- Dress slope, remove track, and slash 50 m from 883 L adit entrance and set up new portal structure.

24.2.4 Path to production – projected 2019 (Y-01) site works

During 2019, a projected increase in the site construction activities is anticipated to include bulk earthworks, concrete works, erection of buildings, ramping up of construction of the process plant, and start of major underground development. The installation of the permanent camp, incinerator and potable water plant is planned to support the increased manning levels. The majority of equipment, permanent bulk materials, construction equipment and construction consumables (including diesel fuel) will be delivered over the winter road campaign #2 (projected for February to March 2019) to support the surface construction and underground activities. The main activities projected in 2019 are:

- Permanent camp – used camp purchased on lease-to-own commercial agreement. Set-up to commence as soon as mobilized to site, with progressive construction to support ramp-up of site manning levels, starting with the kitchen.
- Aggregate crushing and screening plant mobilization and set-up including production of aggregate for civil works.
- Site civil works – ROM pad, container storage area, catchment pond, site drainage, roads, water storage pond, waste rock facility, and haul road.
- Potable water treatment plant to support the camp development.
- Incinerator to support the camp development.
- Refurbishment of sewage treatment plant (to support peak manning levels).
- Connection of utilities to camp.
- Concrete batch plant mobilization and set-up.
- Detailed earthworks and concrete with a focus on completing the concrete foundations during window of most favourable weather conditions (June to September).
- Erection of the pre-engineered buildings and overhead cranes including paste plant building, lead oxide building, DMS plant and concentrate storage building. These items would be completed after completion of the concrete foundations and also during the favourable weather conditions. This would allow the construction of the inside building components to be completed using the overhead crane in an enclosed and heated building during the less favourable winter conditions.
- Refurbish HVAC system / air handling units for mill building for use during construction (pending results of condition assessment).
- Continue refurbishment of process plant components.
- DMS plant structural, mechanical, piping and electrical works.
- Set up Fort Nelson marshalling yard in advance of winter road campaign #3 (March 2020).
- Demobilise heavy earthworks equipment.
- Remove remainder of 883 L track, remove old chutes, set up powder magazines, strip old underground services and install new electrical system, rehab refuge station, set up preliminary underground services, including for initial dewatering.
- Installation of main fans and heaters.

24.2.5 Path to production – projected 2020 (Y01) site works

During 2020, the balance of the plant and infrastructure is envisaged to be completed. The mill commissioning is planned for completion during Q3 2020 and the completion of the all season road will be used to support the mobilization of first fills and consumables required to support the plant start-up.

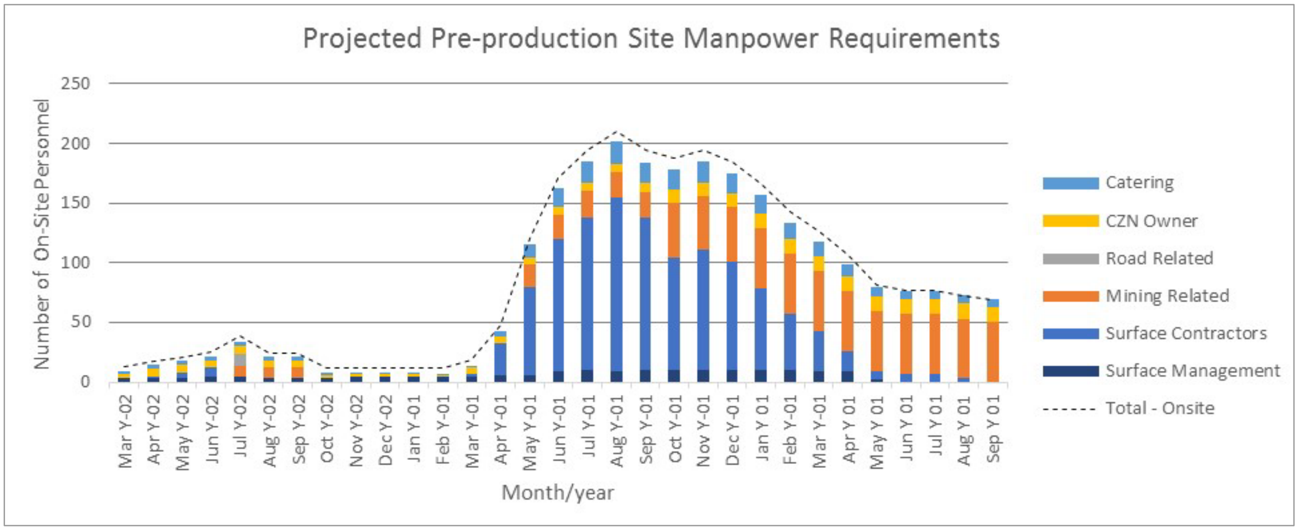
Any remaining equipment, permanent bulk materials, construction equipment, and construction consumables (including diesel fuel) are planned to be delivered over the winter road (the base course of the permanent all season road) campaign #3 during February to March 2020 to support the construction activities. The main activities planned in 2020 are:

- Completion of refurbishment of the process plant and installation of new equipment within the mill building.
- Installation of the power generation plant and associated heat recovery equipment.
- Paste plant and tailing stockpile buildings and associated conveyors.
- Installation of the mine water treatment plant in advance of the underground mine dewatering program.
- Lead oxide plant.
- Commissioning of plant and handover to operations.
- Advancement of underground development and services.
- Installation of first main dewatering station.
- Ore development and first stoping in preparation for mill start-up.

24.2.6 Path to production workforce

Figure 24.2 shows the manpower profile envisaged for the path to production site works.

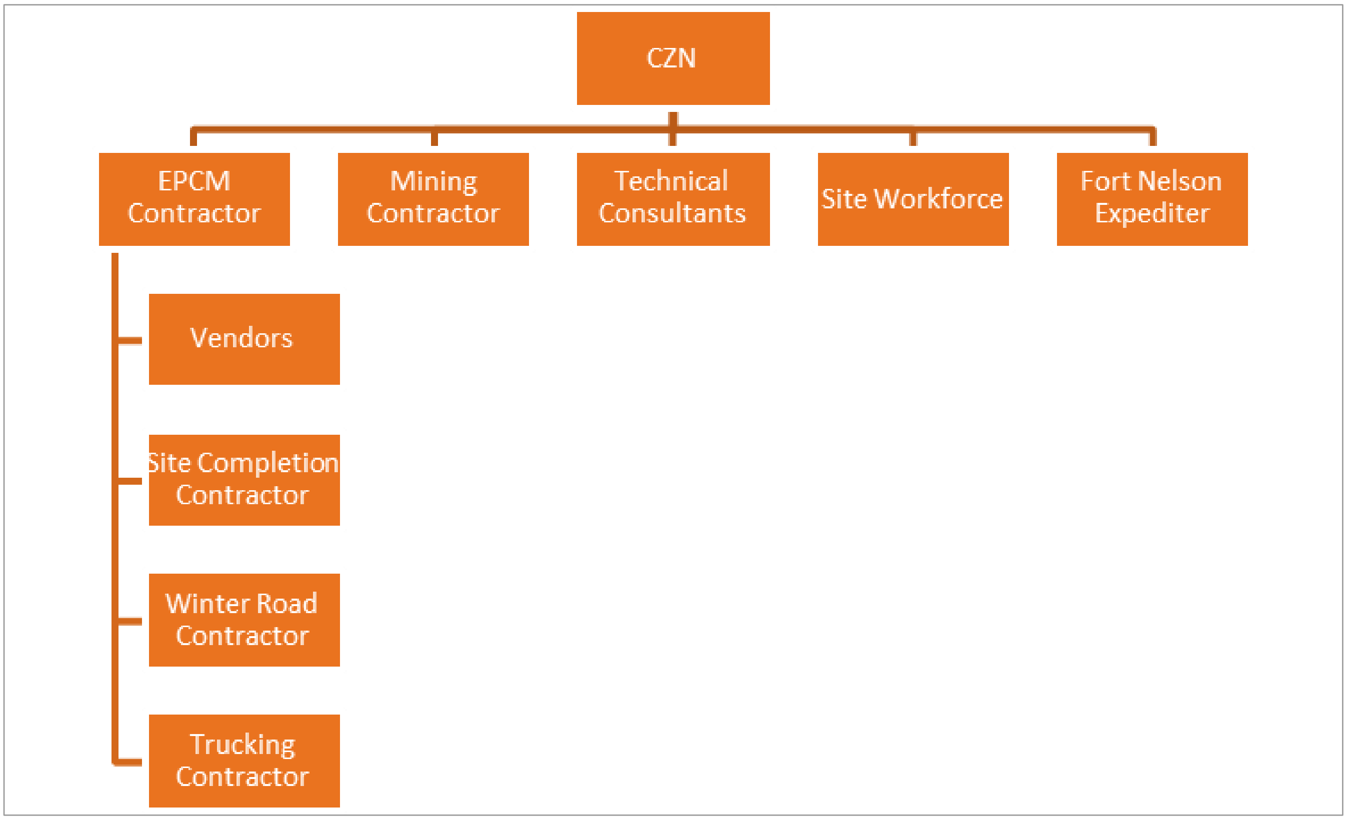
Figure 24.2 On site manpower requirements for path to production



24.2.7 Project execution organization

Figure 24.3 shows the organization and relationship of the different entities involved in executing the project.

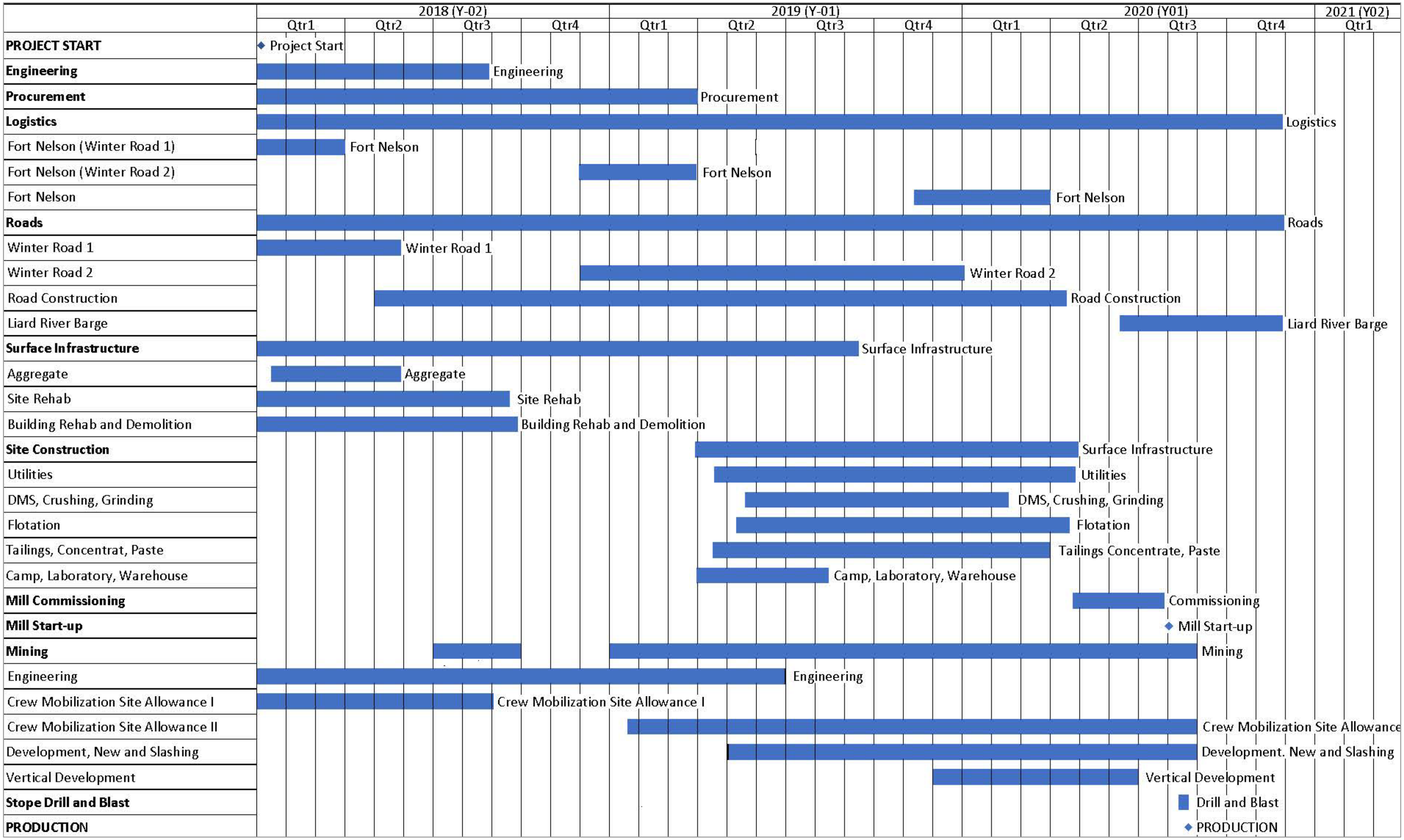
Figure 24.3 Project execution organization chart



24.2.8 Project schedule

The summary project execution schedule is shown in Figure 24.3:

Figure 24.3 Project schedule



25 Interpretation and conclusions

The Prairie Creek Property contains a high-grade, silver-lead-zinc-copper vein along with other lead-zinc deposits and deposit types.

The 2017 FS indicates a Mineral Reserve of 8.1 Mt and an LOM from mill start-up of 15 years at a steady-state production rate of 584,000 tpa.

Mill start-up is projected for August 2020, with a pre-production period during which detailed engineering, mill and camp refurbishment, underground development from existing workings, and construction of key surface infrastructure items, including a paste plant and all season road, will take place.

The economic assessment indicates a base case Pre-Tax Net Present Value ("NPV") of $344M using an 8% discount rate, with an Internal Rate of Return ("IRR") of 23.8% and a post-tax NPV of $188M with a post-tax IRR of 18.4%. Corresponding pre-tax and post-tax payback periods from mill start-up are 4.4 and 4.6 years respectively. The Base Case metal price assumptions used in the model are: Zn US$1.10/lb., Pb US$1.00/lb., Ag US$19.00/oz., with a foreign exchange rate of C$1.25 = US$1.00.

The development of the Prairie Creek mine is projected to offer significant economic advantages on a wider scale. CZN has indicated that there is broad support among aboriginal organisations and communities in the Dehcho region for the direct benefit and economic stimulus that the mine would bring to this region of the Northwest Territories. Its envisaged operation presents a particular opportunity to enhance the social and economic well-being of the surrounding communities. There will be approximately 330 direct full-time jobs on site and a further 33 off-site, mostly trucking associated. In addition, there will be many indirect business and employment opportunities, related to transport, supply of the mine site, and environmental monitoring and management.

The Prairie Creek Mine is shown to be a viable project, based on the Mineral Reserves, mine plan, and production and economic parameters determined within the 2017 FS. AMC recommends that Canadian Zinc advance this project to the next stage, which will include: detailed design of the required services, construction of the all season road, refurbishment of the mill, ordering the long lead equipment for power generation, portal refurbishment, access widening, and development of ramp declines and underground infrastructure in preparation for ore production and processing.

Dense media separation (DMS) of mine production is envisaged prior to grinding and flotation. According to the mineralogical test results and interpretation, the anticipated overall average grade of the lead sulphide concentrate is 65% lead, with an approximately 90% average lead recovery. The zinc sulphide concentrate is estimated to be 59% zinc, with an approximately 90% average zinc recovery. An average of 86% of the total silver in the plant feed is estimated to be recovered within the lead and zinc concentrates.

Tailings from the mill will be placed permanently underground as paste backfill, produced in a new paste backfill plant, augmented by DMS reject material. The remainder of the DMS reject and mine development waste will be trucked to a newly created Waste Rock Pile, located approximately 700 m north of the mill. Although the waste rock is considered non-acid generating due to its high content of carbonate material, appropriate precautions will be taken to prevent and mitigate any leaching that may occur from surface run-off through the Waste Rock Pile.

Existing infrastructure at the site is substantial. It includes a partially complete processing plant, a pond originally built as a 1.5 million tonne capacity tailings impoundment but repurposed as a water storage pond, a power plant, a sewage treatment plant, an administration building, a tank farm for diesel fuel storage, accommodations and workshops. CZN plans to rehabilitate and upgrade these facilities.

Four new 2.77 MW dual fuel LNG / diesel powered generator units, provided by the Northwest Territories Power Corporation are planned to provide power and heat for the site during production operations. These generators will be outfitted with heat recovery systems in order to maximize energy efficiency. The waste heat from the generators will be used to heat the surface facilities.

A 150-person camp and cookhouse exists on the site, but most of the buildings are old and beyond economic repair and will be replaced by a new modular camp.

Concentrates will be trucked over the 184-km road between the mine site and Liard Highway, which will be upgraded for all season use and is currently being permitted. The Liard Highway 7 will be used for transport to the CN railhead at Fort Nelson, whereby shipping will be by rail to the Port of Vancouver for shipment to smelters overseas. CZN has signed offtake MOUs with Boliden and Korea Zinc. Inbound freight will be trucked over the same route and will take advantage of backhaul opportunities.

The site workforce will follow a regular fly-in-fly-out rotation on an anticipated two-weeks-in, two-weeks-out basis, travelling from and to a designated marshalling area. Fort Nelson and Yellowknife are the nearest communities served by scheduled air services with large aircraft. Additional movements to and from site are anticipated for visitors and senior management.

CZN will charter flights from one or both of Fort Nelson and Yellowknife region, depending on the availability and reliability of scheduled services. The Prairie Creek airstrip is usable by DHC-7 aircraft, which will serve for all foreseeable passenger movement needs. Flights will be restricted to day VFR conditions; some weather delays may be anticipated.

The project capital cost is broken down into pre-production capital and sustaining capital as summarized below in Table 25.1. Pre-production capital will be expended during the first three years of the project.

Table 25.1 Capital cost summary

Description	Total cost ($M)
Pre-production capital	
Mine development	37.7
Site preparation	19.4
Mill process plant	31.1
Paste tailings plant and process	22.9
Indirects including EPCM	23.8
Other site infrastructure	15.9
All season road	68.5
Owner's costs	33.6
Total (excluding contingency)	**252.9**
Contingency	26.0
Total pre-production capital	**278.9**
Sustaining capital	
Mining	73.4
Mobile equipment	20.0
Capital leases	11.0
All season road	8.6
Infrastructure	9.5
Salvage	(5.5)
Total sustaining capital	**117.0**
Total project capital cost	**395.9**

Average operating costs over the LOM for the process plant and site infrastructure are estimated at $223/t ore mined, as shown in Table 25.2.

Table 25.2 Total operating cost summary

Total operating cost	($/t)
LOM (8,071,463 t)	
Mining	58.23
Milling and processing	46.76
G&A	30.32
Site services	18.55
Sub-total	**153.86**
Transportation[1]	68.73
Total	**222.59**

Totals do not necessarily equal the sum of the components due to rounding.
1. Includes truck / rail / handling / shipping.

Table 25.3 and Table 25.4 respectively indicate perceived risks and opportunities for the Project.

Table 25.3 Project risks

Preliminary project risks		
Risk	**Explanation / potential impact**	**Risk mitigation**
Low commodity prices and adverse foreign exchange rate.	The project is sensitive to low commodity prices and a high Canadian dollar, negatively affecting economics.	A focus on efficiency throughout the operation will minimize the economic impact of lower commodity prices and adverse exchange rate.
High materials and labour prices.	The Project is sensitive to materials and labour prices over which it has limited control. The Project economic model is based on a combination of pricing for metals, materials and labour. This combination may change to the Project's advantage or disadvantage.	Further optimization of all operation processes to minimize cost of production would assist in reducing the economic impact of high materials and labour prices.
Site completion	Some unknown factors as to the details of the present equipment/buildings still exist, including materials quality and specifications. As-built drawings of existing buildings are not available.	The buildings have stood without distortion or significant weather damage since they were built. As-built inspections, material stock takes, building surveys and materials testing will be made.
Condition of existing process plant equipment.	The major process plant components (mill, crushers, filters) could require more extensive and, possibly, offsite refurbishment.	Preliminary site investigations have been completed with specialist vendor input. Additional work to further define the condition of the components is recommended. Further risk analysis and assessment should be carried out.
Road closures	Road closures due to poor road or weather conditions will interfere with a consistent feed of concentrate to market and impede replenishment of mine supplies.	Work with local authorities to improve the road conditions and install better visual aids along the route for safer passage.
Airstrip closure	Poor weather conditions can limit the aircraft able to service the mine site and cause scheduling problems, especially in relation to shift rotations.	Install further approach beacons and ground control; consider extension of runway.
Schedule	Seasonal restrictions for access are variable and could affect Project schedule and alter project economics.	Experienced management and sound operating plans will minimize the effect of this potential problem
Ability to attract a qualified workforce	High turnover rates and availability of appropriate experienced technical and management staff could result in difficulties meeting project goals.	Careful recruitment of experienced senior management will be essential. Continuation with plans for comprehensive training programs for local people and northern residents will be important. Effective management, incentive bonus systems and an understanding of the importance of morale will minimize the effects of this problem.
Excessive dilution	Higher than projected wall dilution could reduce projected mill feed grade.	Stope mining plan maintains broken ore as wall support for much of stope extraction period. Stope design and blasting plan can also be adjusted.
Higher than projected water in-flows	Excessive ground water entering the mine could slow underground production.	Advance dewatering is planned. Location, number and positioning of drain holes, and capacity of pumps, can be adjusted.

Table 25.4 Project opportunities

Project opportunities		
Opportunity	**Explanation**	**Potential benefit**
Increase in daily mining / milling rate.	The resource could support a significant increase in mining/milling rates and an expanded or new mill.	Improved economics with more metal being mined and milled per year.
Increasing road haulage days.	The all season road will have times of closure / restrictions and any increased availability could lower the cost of $/t transport by reducing fleet size and increasing capacity.	Upgrading of the route would lower the restrictions and extend the operations of the ice bridge (faster construction and longer life) and keep the barge operating for longer periods. Hoverbarge would facilitate all haulage operations.
Alternate energy sources	With an all season road accessing the mine, alternate energy sources such as LNG have been included in the design. Further green energy sources may be considered.	Lower power costs, lower emissions, less environmental risk.
Used equipment	Obtain used equipment for surface. Numerous sites are downsizing or closing and have available equipment.	Lower capital costs.
Supplemental uses for the road.	The all season road could provide access to areas that could be utilized by locals and visitors.	Nahanni Butte is hoping to build a wellness centre at Grainger Gap and the road could also provide additional tourist opportunities into NNPR.
Reduced cement use.	Opportunities exist for reduced paste fill cement in certain circumstances.	Lower operating costs.
Supplement for underground heating	Excess heat from the generators and LNG fuel to supplement mine air heating system.	Lower operating costs.
Additional Mineral Resources	Exploration and drilling in MQV during mine operations may bring additional Mineral Resources into mine plan.	Longer mine life and greater economic benefit.

The Prairie Creek Mine is shown to be a viable project, based on the Mineral Reserves, mine plan, and production and economic parameters determined within the 2017 FS. AMC recommends that Canadian Zinc advance the Project to the next stage, which will include: detailed design and planning of the required services, construction of the all season road, refurbishment of the mill, ordering the long-lead equipment for power generation, portal refurbishment, access widening, and development of ramp declines and underground infrastructure in preparation for ore production and processing.

26 Recommendations

As a result of the feasibility study assessment, AMC recommends the following:

- Early completion of engineering and mine development programs to facilitate achievement of scheduled access development, initial dewatering and first ore.
- Completion of permitting of the all season access road.
- Study of opportunities for enhanced mine operation through use of automation and advanced technology.
- Further underground paste backfill strength and flow property studies.
- Additional study of paste binder requirements and backfill methodology.
- Further hydrology study to enhance understanding of water ingress to the mine and optimize dewatering strategy and water treatment.
- Selection of appropriately qualified and experienced mining contractor for the pre-production phase and the first phase of production operations.
- Completion of a detailed trade-off study further examining the economic merits of recovering lead oxide and oxide recovery at low grades.
- Further detailed mine design and schedule to optimize grades, and balance ore and tailings stockpile requirements.
- Detailed design of key underground infrastructure including magazines, dewatering and sump set-ups, service bays, main fan and heating set-ups, electrical infrastructure.
- Finalization of arrangements for active and passive tailings stockpiles.
- Ordering of key long-lead items for underground operation including main fans and heaters.

Ausenco recommends the following in the execution phase of the project:

- Further investigation of the use of LNG (liquefied natural gas) for heating of buildings and underground mine.
- Further investigation of the use of excess heat from generators to supplement underground heating.
- Investigation of the use of used construction equipment and mobile equipment for operations.
- Optimization of the paste plant design, to include best use of test results in progress at the time of writing this report.
- Complete detailed engineering and IFC drawings to support the procurement and construction of the process plant and site infrastructure.
- Completion of early works site activities including removal of existing generators from the power house, repair of the mill roof, initial work on the Water Storage Pond and Waste Rock Pile, site clearance of derelict buildings, equipment and scrap material.
- Selection of appropriately qualified and experienced contractor(s) to construct the surface works.
- Selection and engagement of appropriately qualified and experienced contractor(s) to carry out the construction of the process plant and site infrastructure.

Total associated preliminary cost estimate for the above recommendations from AMC and Ausenco is $18.2M, of which the majority is included in the FS Capital Cost estimate.

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Mackenzie Valley Review Board public registry lists all pertinent documents associated with past and present Environmental Assessments of Land Use Permits and Water Licences for Canadian Zinc Corporation. Go to www.reviewboard.ca and enter into Public Registry.

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28 QP Certificates

CERTIFICATE OF HERBERT A. SMITH, P.ENG.

I, Herbert A. Smith, P.Eng., of Vancouver, British Columbia, do hereby certify that:

1. I am currently employed as a Senior Principal Mining Engineer with AMC Mining Consultants (Canada) Ltd., with an office at Suite 202, 200 Granville Street, Vancouver, British Columbia V6C 1S4;
2. This certificate applies to the technical report titled "Prairie Creek Property Feasibility Study NI 43-101 Technical Report", in Northwest Territories, Canada, with an effective date of 28 September 2017, (the "Technical Report") prepared for Canadian Zinc Corporation ("the Issuer");
3. I graduated with a degree of B.Sc in Mining Engineering in 1972 and a degree of M.Sc in Rock Mechanics and Excavation Engineering in 1983, both from the University of Newcastle upon Tyne, England. I have worked as a Mining Engineer for a total of 40 years since my graduation and have relevant experience in underground mining, feasibility studies and technical report preparation for mining projects;

I have read the definition of "Qualified Person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101;

4. I have visited the Prairie Creek Property in the capacity of a Qualified Person;
5. I am responsible for Sections 2,15, 16, 19, 20, 22, 24 and parts of 1, 3, 18, 21, 25, 26 and 27 of the Technical Report;
6. I am independent of the issuer as described in Section 1.5 of NI 43-101;
7. I have had prior involvement with the property that is the subject of the Technical Report, having co-authored multiple studies, including the Prairie Creek Property Prefeasibility Update NI 43-101 Technical Report dated 31 March 2016;
8. I have read the NI 43-101 guidelines, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1;
9. As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: 28 September, 2017
Signing Date: 31 October 2017

(original signed by) Herbert A. Smith, P.Eng.

Herbert A. Smith, P.Eng.
Senior Principal Mining Engineer
AMC Mining Consultants (Canada) Ltd.

CERTIFICATE OF L. P. STAPLES, P.ENG.

I, L. P. Staples, P.Eng., of Penticton, British Columbia, do hereby certify that:

1. I am currently employed as VP & Global Practice Lead with Ausenco Engineering Canada Inc. with an office at 855 Homer Street, Vancouver, BC V6B 2W2;
2. This certificate applies to the technical report titled “Prairie Creek Property Feasibility Study NI 43-101 Report, North West Territories, Canada, Technical Report”, with an effective date of 28 September 2017 (the “Technical Report”) prepared for Canadian Zinc Corporation (“the Issuer”);
3. I am a graduate of Queen’s University in Kingston, Ontario, Canada (Bachelors of Science, Materials and Metallurgical Engineering, 1993). I am a member in good standing of the Association of Professional Engineers and Geoscientists of New Brunswick (License #4832), and a member of the CIM and AusIMM. I have relevant experience in the delivery of feasibility studies over the past 15 years and previous to this, relavant operating experience with this style of lead-zinc deposit;

I have read the definition of "Qualified Person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101;

4. I have visited the Prairie Creek Property in the capacity of a Qualified Person;
5. I am responsible for Section 17 and parts of 1, 18, 21, 25, 26 and 27 of the Technical Report;
6. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;
7. I have had no prior involvement with the property that is the subject of the Technical Report;
8. I have read the NI 43-101 guidelines, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1;
9. As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: 28 September, 2017
Signing Date: 31 October 2017

(original signed by) L. P. Staples, P.Eng.

L. P. Staples, P.Eng.
VP & Global Practice Lead
Ausenco Engineering Canada Inc

CERTIFICATE OF SCOTT C. ELFEN, P.E.

I Scott. Cameron Elfen, of Vancouver, British Columbia, do hereby certify that:

1. I am currently employed as a Global Lead – Geotechnical Services with Ausenco Engineering Canada Inc. with an office at 855 Homer Street Vancouver BC Canada V6B 2W2;
2. This certificate applies to the technical report titled “Prairie Creek Property Feasibility Study NI 43-101 Report, North West Territories, Canada, Technical Report”, with an effective date of 28 September 2017, (the “Technical Report”) prepared for Canadian Zinc Corporation (“the Issuer”);
3. I am a graduate of University of California, Davis, United States (Bachelors Civil Engineering in 1991). I am a member in good standing of the Board for Professional Engineers, Land Surveyors, and Geologists, California (License # C056527), and a member of the American Society of Civil Engineers, Society for Mining, Metallurgy & Exploration, The Canadian Geotechnical Society and Canadian Dam Association. I have 23 years of experience in the mining sector. I have been directly involved in the mining and evaluation of mineral properties internationally for both previous and base metals;

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 Prairie Creek Property in the capacity of a Qualified Person;
5. I am responsible for parts of Sections 1 and 18 of the Technical Report;
6. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;
7. I have not had prior involvement with the property that is the subject of the Technical Report;
8. I have read the NI 43-101 guidelines, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1;
9. As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: 28 September, 2017
Signing Date: 30 October 2017

(original signed by) Scott Elfen, P.E.

Scott Elfen, P.E.

Global Lead – Geotechnical Services

Ausenco Engineering Canada Inc

CERTIFICATE OF GREGORY Z. MOSHER, P.GEO.

I, Gregory Z. Mosher, P.Geo., of Vancouver, British Columbia, do hereby certify that:

1. I am currently employed as a Principal Geologist with Global Mineral Resource Services, with an office at 603 – 131 East Third Street, North Vancouver, British Columbia V7L 1E5.
2. This certificate applies to the technical report titled "Prairie Creek Property Feasibility Study NI 43-101 Report, Northwest Territories, Canada, Technical Report", with an effective date of 28 September 2017, (the "Technical Report") prepared for Canadian Zinc Corporation ("the Issuer").
3. I am a graduate of Dalhousie University (B.Sc. Hons., 1970) and McGill University (M.Sc.

   Applied, 1973). I am a registered member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, Licence #19267. My relevant experience with respect to lead-zinc Mineral deposits extends over 40 years and includes exploration, Mine geology and Mineral Resource estimations.

   I have read the definition of "Qualified Person" set out in National Instrument 43-101 (NI

   43101) 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 Prairie Creek Property in the capacity of a Qualified Person.
5. I am responsible for Sections 4-12, 14 and 23 and parts of Sections 1, 3, 25, 26 and 27 of the Technical Report.
6. I am independent of the issuer as described in Section 1.5 of NI 43-101.
7. I have had prior involvement with the property that is the subject of the Technical Report; I have been coauthor of several technical reports and have previously inspected the Property on two occasions but prior to the implementation of National Instrument 43-101.
8. I have read the NI 43-101 guidelines, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
9. As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: 28 September 2017
Signing Date: 31 October 2017

(original signed by) Gregory Z. Mosher, P.Geo.

Gregory Z. Mosher, P.Geo.

Principal Geologist

Global Mineral Resource Services

CERTIFICATE OF F. WRIGHT, P.ENG.

I, Frank R. Wright, P.Eng., of Delta, BC, do hereby certify that:

1. I am currently employed as a Metallurgical Engineer with F. Wright Consulting Inc. with an office at #45-10605 Delsom Cr. Delta BC, Canada V4C 0A4;
2. This certificate applies to the technical report titled “Prairie Creek Property Feasibility Study NI 43-101 Report, North West Territories, Canada, Technical Report”, with an effective date of 28 September 2017, (the “Technical Report”) prepared for Canadian Zinc Corporation (“the Issuer”);
3. I am a graduate of University of Alberta, in Edmonton, AB Canada with a Bachelor of Science in Metallurgical Engineering in 1979, and from Simon Fraser University in Burnaby, BC Canada with a Bachelor of Business Administration in 1983. I am a member in good standing with the Engineers and Geoscientists British Columbia (License #15747), and a member of the Canadian Institute of Mining and Metallurgy. I have continuously practiced my profession in the areas of hydrometallurgy, environmental, and mineral process engineering since 1979, as an employee of various resource companies and consulting firms. Since 1998, I have been the principal and self-employed consultant with F. Wright Consulting Inc., primarily providing services, including the co-authoring of technical reports for junior and mid-tier mineral exploration and mining firms.

I have read the definition of "Qualified Person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101;

4. I have visited the Prairie Creek Property in the capacity of a Qualified Person;
5. I am responsible for Section 13 and part of Section 1 of the Technical Report;
6. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;
7. I have not had prior involvement with the property that is the subject of the Technical Report;
8. I have read the NI 43-101 guidelines, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1;
9. As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: 28 September 2017
Signing Date: 28 October 2017

(original signed by) F. Wright, P.Eng.

F. Wright, P.Eng.
Metallurgical Engineer
F. Wright Consulting Inc.

CERTIFICATE OF DON WILLIAMS, P.ENG.

I, Don Williams, P.Eng., of Prince George, British Columbia, do hereby certify that:

1. I am currently employed as a Divisional Manager with Allnorth Consultants Limited with an office at 2011 PG Pulpmill Road, PO Box 968, Prince George, British Columbia, V2L 4V1;
2. This certificate applies to the technical report titled “Prairie Creek Property Feasibility Study NI 43-101 Technical Report, North West Territories, Canada, Technical Report”, with an effective date of 28 September 2017, (the “Technical Report”) prepared for Canadian Zinc Corporation (“the Issuer”);
3. I am a graduate of University of British Columbia in Vancouver, Canada (Bachelors of Applied Sciences in 1995). I am a member in good standing of the Northwest Territories Association of Professional Engineers and Geoscientists(License #L2046), and a member of the EGBC (BC), APEGY (Yukon), APEGA (Alberta), APEGS (Saskatewan), and APEGM (Manitoba) . I have experience in Resource road and bridge design and construction.
4. I have read the definition of "Qualified Person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101;
5. I have not visited the Prairie Creek Property in the capacity of a Qualified Person;
6. I am responsible for Sections 18.27.2 of the Technical Report, portions of the All Season Road Capital and Operating Costs included within the Technical Report, and part of Section 1 of the Technical Report;
7. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;
8. I have not had prior involvement with the property that is the subject of the Technical Report;
9. I have read the NI 43-101 guidelines, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1;
10. As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: 28 September, 2017
Signing Date: 27 October 2017

(original signed by) Don Williams, P.Eng.

Don Williams, P.Eng.

Divisional Manager, Prince George, B.C.

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OUR VISION

ADVISER OF CHOICE TO THE WORLD'S MINERALS INDUSTRY

OUR PURPOSE

To optimize the value of the world's mineral resources

OUR VALUES

- We regard safety as fundamental
- We are client-focused
- We act with integrity
- We are always professional
- We collaborate
- We share our knowledge & expertise