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Technical Report

Prairie Creek Property Prefeasibility Update NI 43-101 Technical Report (Amended and Restated)

For 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 715025
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The Prairie Creek Mine Site

1 Executive summary

Revised Technical Report

This Technical Report was revised on 30 September 2016 to correct an error in the life-of-mine economic model presented in Section 22. The error caused an overstatement in gross smelter revenue to $3.7 billion from $3.3 billion over the projected 17-year life of the Prairie Creek Mine. The gross metal value of production (using the same assumptions) remains unchanged. The overestimation resulted from the inclusion in the economic model of smelter revenue for by-product metals in primary concentrates that may not be payable, depending on final concentrate contract terms. Only Sections 1, 2 (first paragraph only), 22 and 25 have been updated, the changes made reflecting the adjustments to the life-of-mine economic model and associated commentary.

All inputs into the economic model and all technical aspects of this report remain unchanged, including all mineral resource and reserve estimates, mining plans and production rates and estimates of capital and operating costs and assumptions on concentrate treatment charges and penalties.

No other changes or additions have been made to the original report which remains effective as of 31 March 2016.

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 Tetra Tech Inc. (Tetra Tech), Vancouver, 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 is an update to an amended and restated report titled *Prairie Creek Property, Northwest Territories, Canada, Technical Report for Canadian Zinc Corporation*, prepared by AMC and with an effective date of 15 June 2012 (2012 AMC Report) and a revised date of 23 July 2014. This report discloses the results of an updated Prefeasibility Study (PFS) based on updated Mineral Resources, and ongoing optimization projects and other engineering studies completed since the date of the 2012 AMC Report. The Mineral Resource estimate incorporates drill assay results that were not available at the time of the 2012 AMC Report.

Highlights

This updated 2016 Prefeasibility Study (2016 PFS), based on optimization work completed over the past three years, including the 2015 underground exploration program at Prairie Creek which increased total Measured and Indicated Resource tonnages by 32%, indicates a Base Case pre-tax Net Present Value (NPV) of $284 million using an 8% discount rate, with an Internal Rate of Return (IRR) of 23%, and a post-tax NPV of $155 million with a post-tax IRR of 18%.

The pre-tax and post-tax net present values, at 5% and 8% discount rates, and internal rates of return, are illustrated in the table below, all at a Canadian/US dollar exchange rate of 1.25:1 (except the Base Case which is also shown at an exchange rate of 1.375:1). 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 ES.1.1 Sensitivities of the Prairie Creek Project

Metal Prices		Pre-Tax				Post-Tax			
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	99	-	(42)	5.0	35	(38)	(70)	2.2
0.90	18.00	405	202	121	15.0	233	100	44	11.1
1.00	**19.00**	**710**	**405**	**284**	**22.5**	**431**	**235**	**155**	**17.9**
1.00 [1]	**19.00 [1]**	**979**	**585**	**429**	**28.5**	**598**	**349**	**249**	**23.1**
1.10	20.00	1,016	608	447	29.1	623	366	262	23.8
1.20	21.00	1,322	811	611	35.2	810	493	366	29.0
1.30	22.00	1,627	1,014	774	40.9	1,002	624	473	34.1

1. Foreign Exchange assumed to be $1.375CAD:$1.00US on this line only

The 2016 PFS indicates average annual production of approximately 60,000 tonnes of zinc concentrate and 55,000 tonnes of lead concentrate, containing approximately 86 million pounds of zinc, 82 million pounds of lead and 1.7 million ounces of silver. The 2016 PFS indicates average annual earnings before interest, taxes, depreciation and amortization (EBITDA) of $64 million per year and cumulative EBITDA earnings of $1.0 billion over an initial mine life of 17 years, using Base Case metal price forecasts of US$1.00 per pound for both zinc and lead and US$19.00 per ounce for silver. Pre-production Capital Costs, including provision for a new all season road, are estimated at $244 million, including contingency, with payback of five years from first revenue.

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 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; this access road is now planned to be all season and will need to 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 Prairie Creek Mine was developed and fully permitted and a processing plant, along with other surface infrastructure, built in the early 1980s, when a decline in metal prices resulted in the closure of the Mine in 1983 prior to commencement of production. San Andreas Resources Corporation became involved with the Property in 1992 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 in 2004 with the acquisition of a 100% interest in the Property and Mine site.

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 within 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, which contains 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-dilatant fractures within a structural 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. It 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 more recent 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 name 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 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 objective of 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 estimate, announced in a press release dated 17 September 2015, was estimated by AMC following completion of the successful 2015 underground exploration program at Prairie Creek, which increased the Measured and Indicated Mineral Resource tonnages by 32%.

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 $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 CAD/USD.
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 AMC Mining Consultants (Canada) Ltd.

Mineral Reserve Estimate

The September 2015 Measured and Indicated Mineral Resource was subsequently converted into a new Mineral Reserve estimate of 7.6 million tonnes of Proven and Probable Reserves at a combined grade of 17% Pb and Zn plus 128 g/t Ag, which represents a 46% increase in Reserve tonnage compared to the 2012 PFS. The estimation of Mineral Reserves by AMC is shown below.

Table ES.1.3 March 2016 Mineral Reserves, Prairie Creek Mine

Mineral Zone	Classification	Tonnes (t)	Silver (g/t)	Lead (%)	Zinc (%)	Zinc Equivalent
Main Quartz Vein (MQV)	Proven	1,199,288	186.00	10.08	12.09	30.70
	Probable	3,966,848	152.62	10.52	8.58	26.79
	Total	**5,166,136**	**160.37**	**10.42**	**9.39**	**27.70**
Stockwork (STK)	Proven	174,656	105.01	4.80	11.48	20.83
	Probable	1,297,665	60.72	3.41	6.64	12.87
	Total	**1,472,322**	**65.97**	**3.57**	**7.22**	**13.81**
Stratabound (SMS)	Proven	-	-	-	-	-
	Probable	965,132	46.09	4.38	9.03	16.12
	Total	**965,132**	**46.09**	**4.38**	**9.03**	**16.12**
TOTAL	Proven	1,373,944	175.70	9.41	12.02	29.45
	Probable	6,229,646	116.97	8.09	8.24	22.24
	Total	**7,603,590**	**127.58**	**8.33**	**8.93**	**23.54**

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.
Average unplanned dilution and mining recovery factors of 14% and 95%, respectively, for LHOS, and 6% and 95%, respectively, for DAF are assumed.
The March 2016 Prairie Creek Mine 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.

Mining

The mine will be an underground operation, based primarily on the MQV and mining an average of 1,350 tonnes per day over a 17-year mine life. During full production, approximately 485,000 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 progresses to depth, ore from the MQV will be supplemented by ore from the deeper SMS deposit, both deposits being accessed by a common ramp development.

Mining 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 good and the existing workings have stood unsupported for thirty years with minimal bolting. CZN commissioned a geotechnical program at the end of 2013, including mapping and examination of drill core. This program demonstrated that the ground is amenable to longhole open stoping and the results of the program were used for rock support design.

The new mine plan envisages slashing out of some of the existing development in order to establish two spiral ramps to access deeper levels. The existing 883 mL adit will be enlarged to 4.5 x 4.5 m; two ramps will be driven downward from this level and one upward to access the MQV and SMS ores. 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 expected to take approximately 18 months, before stope ore becomes available as mill feed. This work will be performed by a contractor. On

completion of the contracted scope of work, CZN will have the options of taking over the work itself or continuing with contract mining.

The mine will be wet and managing groundwater will be a significant aspect of the operation. When in full production, it is estimated that the mine will produce up to 120 L/s of water. The majority of this water will be collected through pre-drainage boreholes and pumped to surface, avoiding contamination. All water discharged from the mine will either be sent to the mill as process water, pumped into the existing impoundment pond, originally planned for tailings storage, which will 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 may occur. Any shortfall will be made up with Dense Media Separation (DMS) float material and development waste rock. Filtered tailings will be stored in a building on surface during times when no stopes are available for backfill. Development waste and DMS float material will be stored in a newly created Waste Rock Pile north of the plant site.

Metallurgy and processing

Metallurgical tests conducted to date have proved positive. Reasonably good metal recoveries have been achieved with both sulphide and oxide material, with a cyanide-free reagent suite.

According to the test results the overall average grade of the blended lead sulphide/oxide concentrate is anticipated to be 67% lead, with an approximately 88% average recovery of total lead in the plant feed. The zinc sulphide concentrate is estimated to be 58% zinc, with an approximately 83% recovery of the total zinc in the plant feed. An average of 87% of the total silver values in the plant feed is anticipated to be recovered within the lead and zinc concentrates. The impurities of antimony, arsenic and mercury are expected to report to both concentrates.

A complete 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, power plant, and water treatment plant.

The current mill crushing facilities have a 1,500 tpd capacity, with an installed jaw crusher, short-head cone crusher, double-decked screen, and conveyor systems feeding a 2,000 t fine ore bin.

A new dense media separation (DMS) plant, with a nominal feed rate of 58 tph, will be installed downstream of the crushing circuit to process 15-mm to + 1.4-mm sized material. Indications from metallurgical testing are that the DMS plant is expected to reject an average of approximately 27% of the feed as waste at minimal metal losses. Milling feed input rates of 1,200 to 1,400 tpd are anticipated; after passing through the DMS plant, the material to be processed in the grinding/flotation circuit of the mill is estimated to be approximately 960 tpd.

Site infrastructure

As the Mine was fully permitted but never achieved production, existing site infrastructure is substantial; these facilities will be utilized and upgraded where possible. This includes upgrading the existing mill, administration building, workshops, sewage treatment plant, diesel storage tank farm, warehouses and accommodation facilities.

The mill will require an electrical upgrade, addition of a thickener, new flotation cells, a reagent storage and mixing facility, a concentrate storage and loadout facility, an on-stream analyzer and control system, and rehabilitation of the building envelope. The new DMS circuit will be added to the north side of the mill and the reagent mixing area and concentrate storage and loadout facility will be added to the south side of the mill. A new paste backfill plant is proposed to be built next to the mine portal.

Five new 1.5 MW diesel powered 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. Maximum power load for the site is estimated at 5.2 MW. The new 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 and mine air. Since the Mine will be accessed by an all season road, dual-fuel generators utilizing diesel/LNG are an option to consider depending on economics at the time of purchase.

Tailings from the mill will be placed permanently underground as paste backfill, produced in a new paste backfill plant, and augmented by DMS reject material in the event of any volume shortfall. 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. 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 runoff 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 new modular camp adjacent to the upgraded administration building complex.

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 that also connects to a new Water Treatment Plant, and an exfiltration pipe in the bed of Prairie Creek to provide discharge of the treated waters. The water management system also includes maintaining the real-time flow measurement gauges presently installed in Prairie Creek upstream of the site.

Access road and transportation plan

The Prairie Creek Mine was originally accessed by a 180 km winter road connecting to the Liard Highway (NWT Highway #7). Canadian Zinc currently holds all land use permits and water licences required for the construction and operation of a revised 184 km winter access road that connects the Prairie Creek Mine to the Liard Highway, and for the construction of two transfer and staging facilities along the road, one near the Liard River crossing and the second inside the Park at about the half-way mark.

The access road, part of which passes over Crown land and part through the expanded Nahanni National Park Reserve, is multi-jurisdictional and the Company has received from both the Water Board and Parks Canada all necessary road related permits and licences related to their respective jurisdictions.

In January 2013, the Mackenzie Valley Land and Water Board (MVLWB) issued LUP MV2012F007 for a period of five years, which permits the construction, maintenance, operation and use of the winter road connecting the Prairie Creek Mine to the Liard Highway. This permit allows the outbound transportation of the zinc and lead concentrates to be produced at the mine and the inbound transportation of fuel and other supplies during the actual operation of the Prairie Creek Mine. The Land Use Permit and Water Licence apply to the portion of the winter road traversing Crown Land, which is under the jurisdiction of the Water Board. There are two sections to this portion of the road, the first being 17 kilometres of road from the mine site to the point where the road enters the NNPR, and the second being 80 kilometres of road from the eastern boundary of the NNPR to the Liard Highway.

In September 2013, the Company received from Parks Canada permits (Parks2012_W001 WL and Parks2012-L001 LUP), both valid for a period of five years valid until August 2018. The permits authorize road access through the NNPR to connect sections of road outside the Park permitted by the MVLWB.

The transportation plan utilized in the 2012 PFS envisaged the use of the access road only in the winter months of each year, both for the outbound transportation of concentrates and for the inbound transportation of equipment and supplies, including diesel fuel. This winter road plan would necessitate a large investment in working capital to finance consumables and supplies and also a large build-up in concentrate inventory awaiting transportation and sale.

In April 2014, the Company submitted an application to the MVLWB and to Parks Canada for Land Use Permits to permit the upgrade of the current winter access road to all season use. The application is now undergoing environmental assessment before the Mackenzie Valley Review Board (MVRB).

Mining operation based on an all season road access, compared to winter road only access, will:

- Decrease working inventory;
- Ensure more timely delivery of product and consistent supply of materials;
- Lower logistical risk of transporting concentrate and supplies;
- Require a smaller trucking fleet throughout the year;
- Allow alternative energy sources such as Liquid Natural Gas (LNG) to be considered; and
- Increase pre-production capital cost.

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 may be producing LNG in the near future, which may provide an opportunity to reduce reliance on diesel fuel. 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.

Transportation over the all season road will utilize an ice bridge in winter and a barge in summer to cross the Liard River.

Canadian Zinc currently holds all the significant regulatory permits for the construction and use of the access road in winter but does not yet hold the permits for the all season road. The Company anticipates that the environmental assessment process for the proposed all season road permit application will take most of the year 2016.

Upon reaching the Liard Highway, concentrates will be trucked to the railhead at Fort Nelson and transported by rail to the port of Vancouver for shipment to smelters overseas. Inbound freight will be trucked as backhaul over the same route. A staging area will be established at the junction of the mine access road and the Liard Highway. A loading area will be constructed at the railhead in Fort Nelson.

Transportation costs included in the 2016 PFS have been estimated at $65 per tonne of ore mined, which includes approximately $33/t for road/truck transportation, $24/t for rail and trans-loading and $8/t for ocean freight.

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. The concentrates will then be transported in enclosed haul trucks via the mine access road and Liard Highway to Fort Nelson, and from there by rail to the Port of Vancouver.

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 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, including deductibles, payable and penalties, vary with smelter location and individual smelter terms and conditions. The Economic Model used in the 2016 PFS has been prepared assuming average blended indicative treatment charges of US$212 per tonne for zinc sulphide concentrates and

US$195 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. The project schedule comprises one year of detailed engineering, including the completion of permitting and design of the all season road, one year to procure long-lead-time items and prepare the site, followed by one year of site completion and mine development. Mobilization will initially be by winter road, concurrent with construction of the all season road. The later 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. 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 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.

The main Licence is the Type “A” Water Licence (MV2008L2-002) which was issued by the MVLWB on 8 July 2013 that 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 application for an all season road was applied for in April 2014 to the MVLWB and is currently in the Environmental Assessment Process with the Mackenzie Valley Review Board. The Environmental Assessment is well advanced and is expected to conclude prior to year-end.

Prior to the main licences being issued in 2013, CZN had been involved, since the passage of the Mackenzie Valley Resource Management Act in 1999, in numerous regulatory processes to obtain various Land Use Permits and Water Licences for exploration and development at the Prairie Creek Mine site. Regulatory processes included both normal-course permitting and numerous Environmental Assessments.

Innovative water management practices are necessary at the Prairie Creek Mine during operations due to the 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 with the construction period being regulated by ‘end-of-pipe’ effluent quality criteria, followed 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 will be via an exfiltration trench below the bed of Prairie Creek that will promote mixing and attenuation of parameter concentrations to meet site specific water quality objectives.

The Prairie Creek Mine site today is surrounded by the Nahanni National Park Reserve. The original NNPR was created in 1972 for the specific purpose of setting aside the South Nahanni River for wilderness recreation. Exploration activity at Prairie Creek had been ongoing for many years prior to 1972, with underground development being well advanced at that time. 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 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 important Impact and Benefits Agreements with each of Nahanni Butte Dene Band and Liidlii Kue First Nation (Fort Simpson), both part of the Dehcho First Nations. 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

Approximately 150 people are expected to be employed on site during initial project construction. A new accommodation block will be constructed at site to accommodate this workforce.

During operations the Mine will employ a total of approximately 316 people on payroll, including truckers, with half of the employees being on-site at any one time, and with an additional 28 off-site in the Fort Liard and Fort Nelson areas. Personnel will work a regular rotation on site, with rest periods off site, to be determined, with transport by charter flights to the existing on-site 1,000 m gravel airstrip. Canadian Zinc'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	Average Tonnes/Year	Average Grade	Payability
Total ore mined (million tonnes)	7.6	Zinc concentrate	60,000	Zinc: 59%	Zinc: 85%
Mining rate (tonnes per day)	1,350			Silver: 136 g/t	Silver: 70%
Milling rate (tonnes per day) after DMS	960	Lead concentrate	55,000	Lead: 65%	Lead: 95%
Life of Mine (years)	17			Silver: 824 g/t	Silver: 95%

Life of Mine Statistics				
	Ore Grade Initial 10 Years	Ore Grade LOM	Mill Recoveries	Average Annual Contained Metal
Zinc	10.0%	8.9%	83%	86M lbs***
Lead	9.8%	8.3%	88%	82M lbs***
Silver	154 g/t	128 g/t	87%	1.7M oz***

Project Assumptions Base Case			
Zinc price	US$1.00/lb	Exchange Rate	$1.25CDN:$1.00US
Lead price	US$1.00/lb	Discount Rate	8%
Silver price	US$19.00/oz		

Operating and Capital Costs			
Operating Costs**	LOM $/t ore mined	Capital Costs	$M
Mining	79	Pre-production capital	216
Processing	41	Contingency	28
Site Services	22	Total Pre-production Capital	244
G&A	23	Sustaining Capital	70
Total On-site Costs	165	Working Capital	36
Transportation*	65		
Total Operating Costs**	230		

* Includes truck, rail, handling and ocean shipping
** Does not include treatment, refining charges or royalty
*** Metal contained in both lead and zinc concentrates

Economic Results	Pre-tax	Post-tax
Cash Flow Undiscounted ($M)	710	431
NPV @ 8% ($M)	284	155
IRR (%)	22.5	17.9
Payback period (years from first revenue)	4	5
Average annual EBITDA ($M)	64	

Note: Rounding of numbers may influence totals.

Capital and operating costs

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 Pre-production capital cost estimate – Prairie Creek Mine

Capital ($M)			
Description	**Project Year 1**	**Project Year 2**	**Total**
Mine development	-	34.5	34.5
Process plant [1]	6.7	12.5	19.2
Support infrastructure [2]	11.5	30.9	42.4
Site completion [3]	14.8	25.6	40.4
All season road [4]	16.3	41.9	58.2
Owner's costs [5]	9.3	12.1	21.4
Total (excluding contingency)	**58.6**	**157.5**	**216.1**
Contingency	14.7	12.8	27.5
Total Pre-Production Capital Cost	**73.3**	**170.3**	**243.6**

1. Includes dense media separator; structural upgrading; instrumentation; flotation circuit upgrade; reagent handling; and piping.
2. Includes power plant; paste plant; water treatment plant; water storage pond; waste rock pile; camp and housing accommodation; warehousing; and facility upgrades.
3. Includes engineering and construction of surface facilities; freight and logistics; initial fills; and spares.
4. Includes Liard River crossing and Fort Nelson load-out facility.
5. Includes reclamation security and insurance.

Pre-Production Capital Cost refers to capital costs incurred until the first processing of mined ore, and have been estimated at a total of $216 million, excluding contingency, and $244 million including a contingency of $28 million, excluding working capital.

Based on proposals or quotations received, a number of capital items were provided on a lease-to-purchase basis. The main items included on a capital lease basis are: diesel generators; water treatment plant; paste plant; and some process equipment. The lease costs of such items incurred in the pre-production period are included in pre-production capital costs; lease costs incurred after production start-up are included in sustaining capital.

Working Capital has been estimated at $30 million, excluding contingency, and $36 million including a contingency of $6 million.

Sustaining Capital over the full life of the mine has been estimated at $70 million, of which approximately 90% is incurred in the first five years, and relates largely to ongoing mine development as the mine is expanded to deeper levels and to the remaining balance of capital lease payments. The financial model indicates that the sustaining capital can be financed from operational cash flows.

The contingency level of accuracy of the capital cost estimates ranges between approximately 5% and 25%, with a blended contingency factor of plus or minus 12.7%. The cost estimate of major equipment packages was determined via a bidding process conducted by Tetra Tech in 2015 and is considered a firm price. Where no bid price was obtained the contingency was determined based upon professional judgement.

The general breakdown of the Operating Cost Estimate for the Prairie Creek Project is indicated in the following table.

Table ES.1.6 Operating cost estimate – Prairie Creek Mine

Operating Cost Summary	
Area	**Unit Operating Cost $/tonne ore mined**
Mining	78.58
Milling/Processing	40.75
G&A costs	22.58
Surface Costs	21.96
Transportation*	65.10
Total Unit Operating Cost	**228.97**

* includes truck/rail/handling/shipping

The following list summarizes key project assumptions used to develop the operating costs, which are in 2016 constant dollars:

- Mine operating costs are estimated according to unit prices tendered by the mine contractor multiplied by annual estimated quantities; after the first two years of operations, it is assumed that mining will revert to the owner for the remainder of the LOM;
- All electrical power will be produced by diesel generators using a delivered price of diesel of $0.72/L yielding an estimated LOM power cost of $0.215/kWhr;
- 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;
- Labour costs are derived from multiple recent sources and include payroll burdens of 46.5%;
- 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$1.00/lb zinc, US$1.00/lb lead, US$19.00/oz silver and an exchange rate of $1.25CDN:$1.00US. These long-term price assumptions are based on consensus price forecasts published by Consensus Economics Inc. as at February 2016, 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.

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 ($M)	43	64	85
Pre-Tax Cash Flow Undiscounted ($M)	379	710	1,041
Pre-Tax NPV @ 8% discount ($M)	107	284	462
Pre-Tax NPV @ 5% discount ($M)	185	405	626
Pre-Tax IRR	14.2%	22.5%	29.7%
Post-Tax Cash Flow Undiscounted ($M)	217	431	639
Post-Tax NPV @ 8% discount ($M)	35	155	272
Post-Tax NPV @ 5% discount ($M)	88	235	377
Post-Tax IRR	10.4%	17.9%	24.3%
Post-Tax Payback Period (years from first revenue)	7	5	3
Exchange Rate Scenario [2]	**$1.125CDN:$1.00US**	**$1.30CDN:$1.00US**	**$1.375CDN:$1.00US**
Average Annual EBITDA ($M)	47	71	81
Pre-Tax Cash Flow Undiscounted ($M)	441	818	979
Pre-Tax NPV @ 8% discount ($M)	140	342	429
Pre-Tax NPV @ 5% discount ($M)	226	477	585
Pre-Tax IRR	15.9%	25.0%	28.5%
Post-Tax Cash Flow Undiscounted ($M)	258	498	598
Post-Tax NPV @ 8% discount ($M)	57	192	249
Post-Tax NPV @ 5% discount ($M)	116	281	349
Post-Tax IRR	12.0%	20.0%	23.1%
Post-Tax Payback Period (years from first revenue)	7	4	3

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 CDN$1.40:US$1.00, would indicate a pre-tax NPV (8%) of $92 million and IRR 13% (post-tax NPV (8%) of $24 million and IRR 10%). Using the average metal prices for the three years ended 31 March 2016 of US$0.90/lb for zinc; US$0.89/lb for lead and US$18.27 for silver, and an exchange rate of CDN$1.33:US$1.00 would indicate a pre-tax NPV (8%) of $199 million and IRR 19% (post-tax NPV (8%) $97 million and IRR 14%).

Recommendations

As part of its assessment in the Technical Report, AMC has recommended that:

- A front-end engineering and design phase to complete detailed engineering and IFC drawings to definitive feasibility study levels to obtain fixed pricing from construction contractors.
- Early completion of site clearance construction, engineering and mine development programs to accelerate start-up times. This would include preliminary earthworks on the water storage pond, waste rock pile, building foundations, portal construction and upgrades of existing infrastructure in tandem with detailed engineering of new structures.
- Complete permitting of the all season access road.
- Additional mill studies to further optimize the mill circuit capacity to increase both ore throughput and metal recoveries.
- Further metallurgical tests to optimize the process flowsheet, particularly reagent regimes, including variability tests on the samples from various mineralization zones and ore types.
- Further study of on-site or off-site processes to reduce deleterious components of concentrates, thereby reducing smelter penalties.

- Studies to optimize the mine operation by automation and adoption of advanced technology.
- Additional underground paste backfill strength studies.
- Additional hydrology studies to better design, size and cost water management facilities.

The above recommendations include both optional and essential items. The cost of essential items is estimated at $9M. The cost of optional items is estimated at $3M. Details are outlined in the body of the report in Section 26.

Conclusions

The Prairie Creek Property contains a high-grade, silver-lead-zinc-copper vein along with other lead-zinc deposits and deposit types. The Base Case economic model indicates a robust project at consensus forecasts for the long-term prices of lead and zinc, generating a pre-tax undiscounted cumulative cash flow of $710 million, a pre-tax NPV (8%) of $284 million with an IRR of 23% and a post-tax NPV of $155 million with a post-tax IRR of 18%. Additional project optimization, as recommended in this report, would further enhance the economics.

The development of the Prairie Creek mine offers significant economic advantages. 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 unique opportunity to enhance the social and economic well-being of the surrounding communities. There will be approximately 220 direct full time jobs. 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 Project is considered to be a viable project, based on the Mineral Reserves, mine plan, and production and economic parameters determined within the 2016 PFS. AMC recommends that Canadian Zinc proceed with the development of the Prairie Creek Project.

Contents

Tables

Figures

LIST OF ABBREVIATIONS

AANDC	Aboriginal Affairs and Northern Development Canada (recently changed to Indigenous and Northern Affairs Canada)
Ag	Silver
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
Cadillac	Cadillac Explorations Ltd.
CBH4	Navigation Canada Designation for the Prairie Creek Airstrip
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
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
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
INAC	Indian and Northern Affairs Canada
IBA	Impact and Benefits Agreement
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	Liquid 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
MEA	Measured Mineral Resource
MMER	Metal Mining Effluent Regulations
MW	Megawatt
MWh	Megawatt-hour
mg/L	Milligrammes per litre
µg/L	Microgrammes per litre
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
NU	Nunavut
PAG	Potentially Acid Generating materials
Pb	Lead
P. Eng.	Professional Engineer
PFS	Pre-Feasibility Study
P. Geo.	Professional Geologist
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
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
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
WRP	Waste Rock Pile
Zn	Zinc
ZnEq	Zinc equivalent

2 Introduction

2.1 Revised Technical Report

This Technical Report was revised on 30 September 2016 to correct an error in the life-of-mine economic model presented in Section 22. The error caused an overstatement in gross smelter revenue to $3.7 billion from $3.3 billion over the projected 17-year life of the Prairie Creek Mine. The gross metal value of production (using the same assumptions) remains unchanged. The overestimation resulted from the inclusion in the economic model of smelter revenue for by-product metals in primary concentrates that may not be payable, depending on final concentrate contract terms. Only Sections 1, 2 (first paragraph only), 22 and 25 have been updated, the changes made reflecting the adjustments to the life-of-mine economic model and associated commentary.

All inputs into the economic model and all technical aspects of this report remain unchanged, including all mineral resource and reserve estimates, mining plans and production rates and estimates of capital and operating costs and assumptions on concentrate treatment charges and penalties.

No other changes or additions have been made to the original report which remains effective as of 31 March 2016.

2.2 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 Tetra Tech Inc. (Tetra Tech), Vancouver, 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 an updated Prefeasibility Study (PFS) based on a re-estimate of the Mineral Resources, and additional engineering information. The Mineral Resource estimate incorporates drill assay results that were not available at the time of the previous estimate in May 2012. This report is an update to a report titled *Prairie Creek Property, Northwest Territories, Canada, Technical Report for Canadian Zinc Corporation*, prepared by AMC and with an effective date of 15 June 2012 (2012 AMC report) and revised in 23 July 2014.

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.

2.3 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.

2.4 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. These are a mix of independent and non-independent QPs as there has not been a material change since the 2012 report and independence as defined in NI 43-101 is not required in this case.

Table 2.1 Persons who prepared or contributed to this Technical Report

Qualified Persons responsible for the preparation of this Technical Report						
Competent 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	Yes	No visit	P.Geo.	Parts of 1-2, 4-12; 14; 23; parts of 25 and 26.
Mr. H.A. Smith	General Manager, Principal Mining Engineer	AMC Mining Consultants (Canada) Ltd	Yes	September 2016	P.Eng.	Parts of 1-2; 15; parts of 16, 25 and 26.
Mr. J. Huang	Senior Metallurgical Consultant	Tetra Tech	Yes	January 2014	P.Eng.	Parts of 1-2; 13; 17; parts of 21, 25 and 26.
Mr. H. Ghaffari	Director, Metallurgy, Mining and Minerals	Tetra Tech	Yes	No visit	P.Eng.	Parts of 1-2, 21
Mr. A. Taylor	Chief Operating Officer	Canadian Zinc Corporation	No	August 2015	P.Geo.	Parts of 1-2; 3; parts of 4-12, 18; 19; 20; parts of 21, 22, 25 and 26; 27
Mr. T. Morrison*	Consultant	Self-employed	Yes	June 2014	P.Eng.	Parts of 1-2, 16, 18, 21, 22; 24; part of 25 and 26.
Other persons who have assisted the Qualified Persons						
Expert	**Position**	**Employer**	**Independent of CZN**	**Visited Site**	**Sections of Report**	
Mr. T.L. Cunningham, CPA, CMA	Chief Financial Officer & Vice President, Finance	Canadian Zinc Corporation	No	August 2012	Sections 19, 21 and 22	
Mr. D. Harpley, P.Geo.	VP Environment & Permitting Affairs	Canadian Zinc Corporation	No	September 2015	Parts of 4 and 24, 20	
Mr. K. Cupit, P.Geo.	Project Geologist	Canadian Zinc Corporation	No	August 2015	Parts of 4-12	
Mr. A. Grice	Principal Backfill Consultant	AMC Mining Consultants (Canada) Ltd	Yes	No	Parts of 16	
Mr. W. Pitman, P.Eng.	Principal Geotechnical Engineer	AMC Mining Consultants (Canada) Ltd	Yes	No	Parts of 16	
Mr. A. Smith	Principal Mining Consultant	AMC Mining Consultants (Canada) Ltd	Yes	No	Parts of 15-16	
Mr. C. McVicar, P.Eng.	Senior Mining Engineer	AMC Mining Consultants (Canada) Ltd	Yes	No	Parts of 15-16	
Mr. Mo Molavi, P.Eng.	Principal Mining Engineer/Mining Services Manager	AMC Mining Consultants (Canada) Ltd	Yes	No	Parts of 16 and parts of 18	
Mr. William Hughes, P.Eng.	Principal Mechanical /Infrastructure Engineer	AMC Mining Consultants (Canada) Ltd	Yes	No	Parts of 16 and parts of 18	

Note: Where QPs accept responsibility for parts of sections, that responsibility is limited to their areas of expertise.
*Is an independent consultant who has been engaged by CZN to assist with this PFS and preparation of this report.

Frequent visits are made to the Property by CZN personnel. Recent visits are noted in Table 2.1 for those persons who are taking responsibility for sections of this report.

2.5 Source of information

This report relies in part on previous reports on the Property. The preceding Technical Report discloses the results of a PFS in 2012 and is titled *Prairie Creek Property, Northwest Territories, Canada, Technical Report for Canadian Zinc Corporation*, prepared by AMC and with an effective date of 15 June 2012 (2012 AMC Report).

2.6 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 31 March 2016.

3 Reliance on other experts

The QPs have relied upon CZN in regard to legal ownership of, or title to the Property as described in Section 4.2 of this report (fact check by Mr. A. Taylor, Vice President Exploration and Chief Operating Officer of CZN, February 2016).

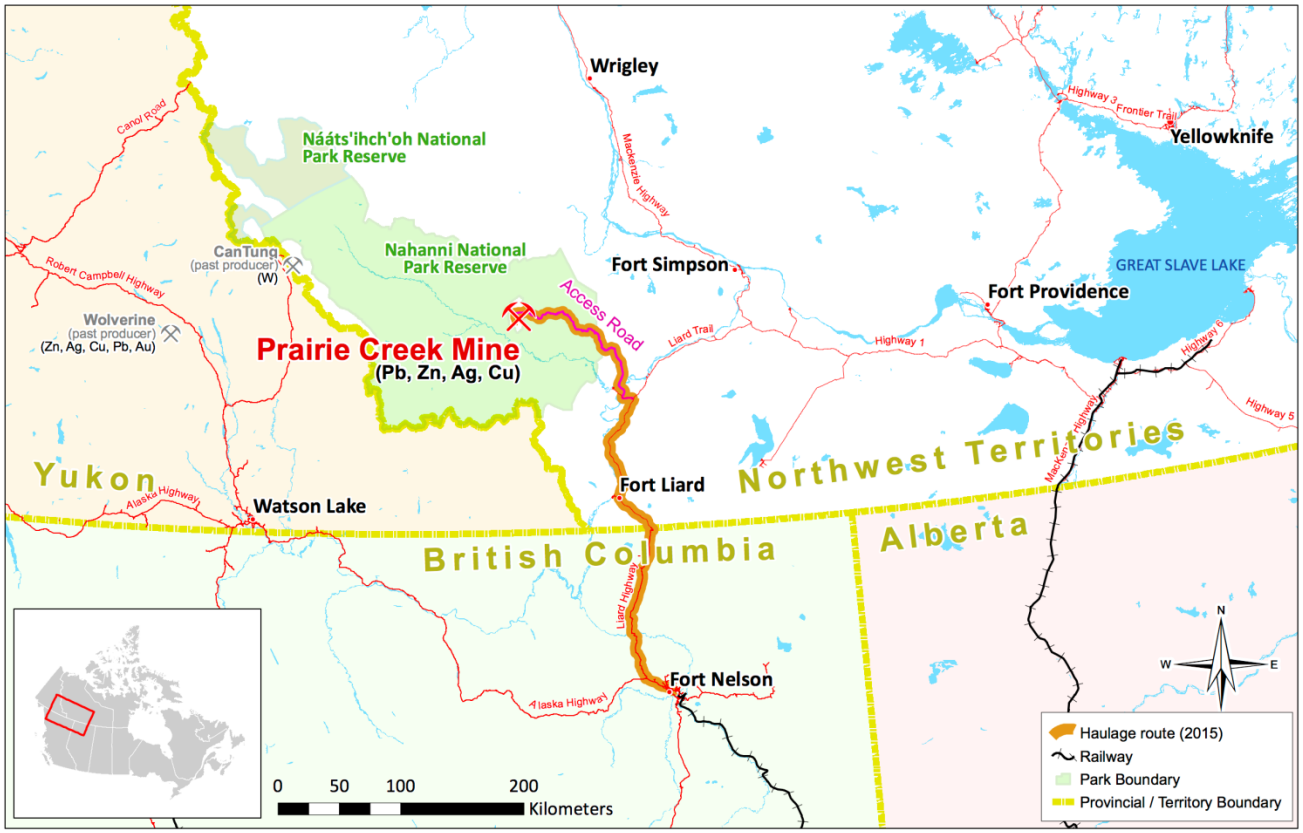
4 Property description and location

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 of access to an active railway system.

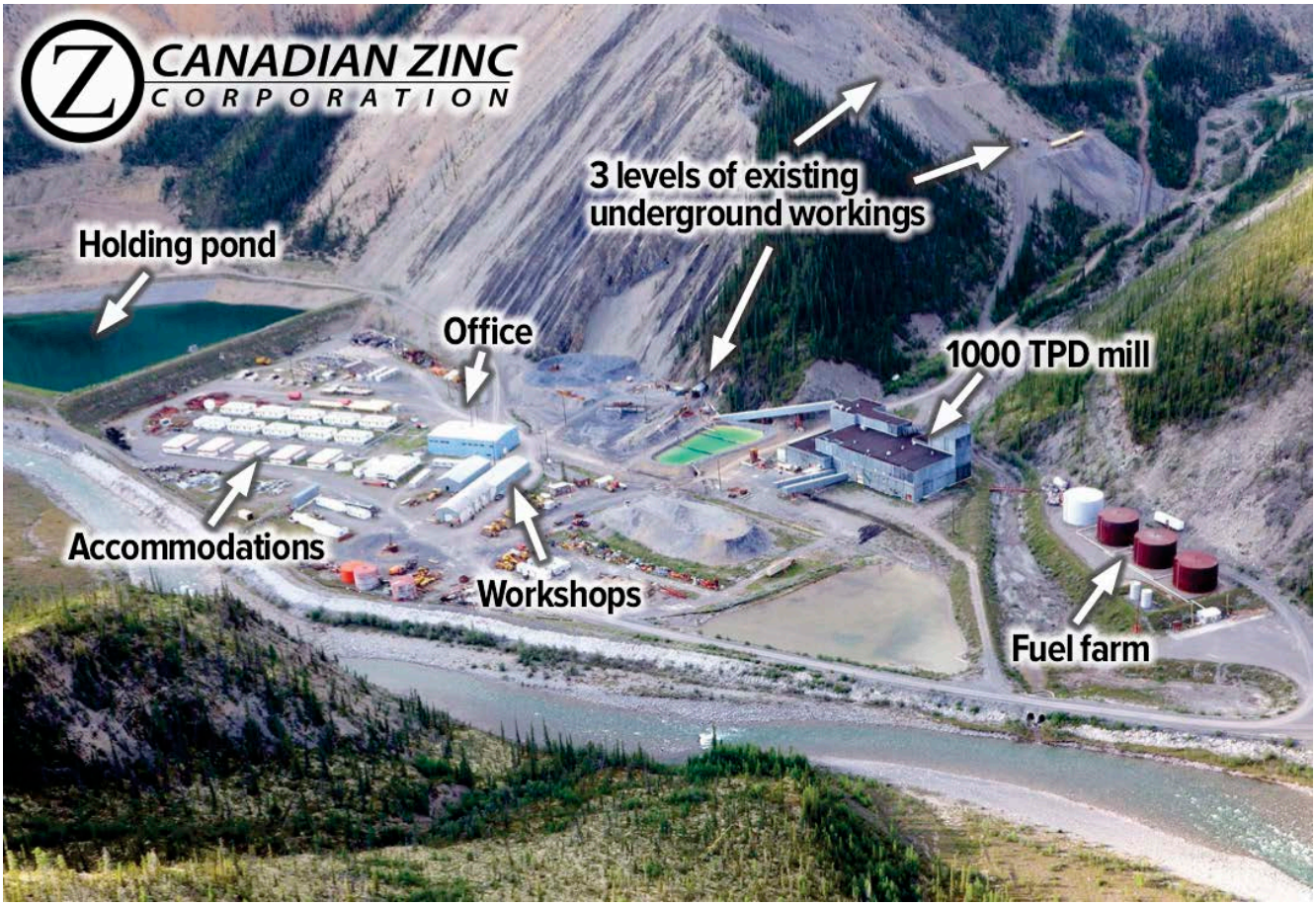
Figure 4.1 shows the location of the Property within the Northwest Territories and relative to various population centres and mining operations.

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 km^2 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.

Figure 4.2 The Prairie Creek Mine site

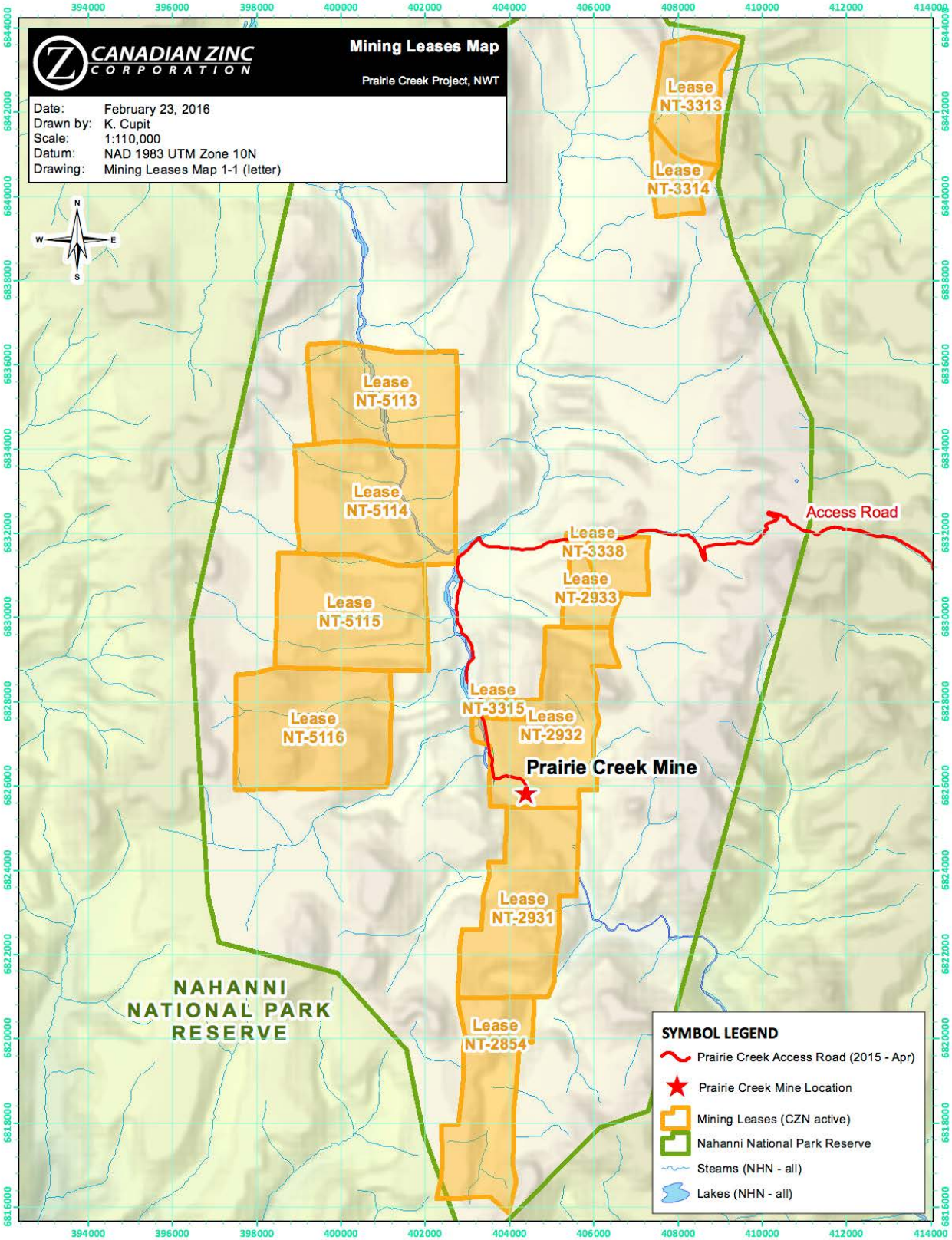


4.2 Property description and ownership

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	Minesite	March 2017	113.6
	95F/10-7-4	Airstrip	March 2017	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 need to 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 is the basis for future security payments tied to construction and operations (refer to Table 20.4).

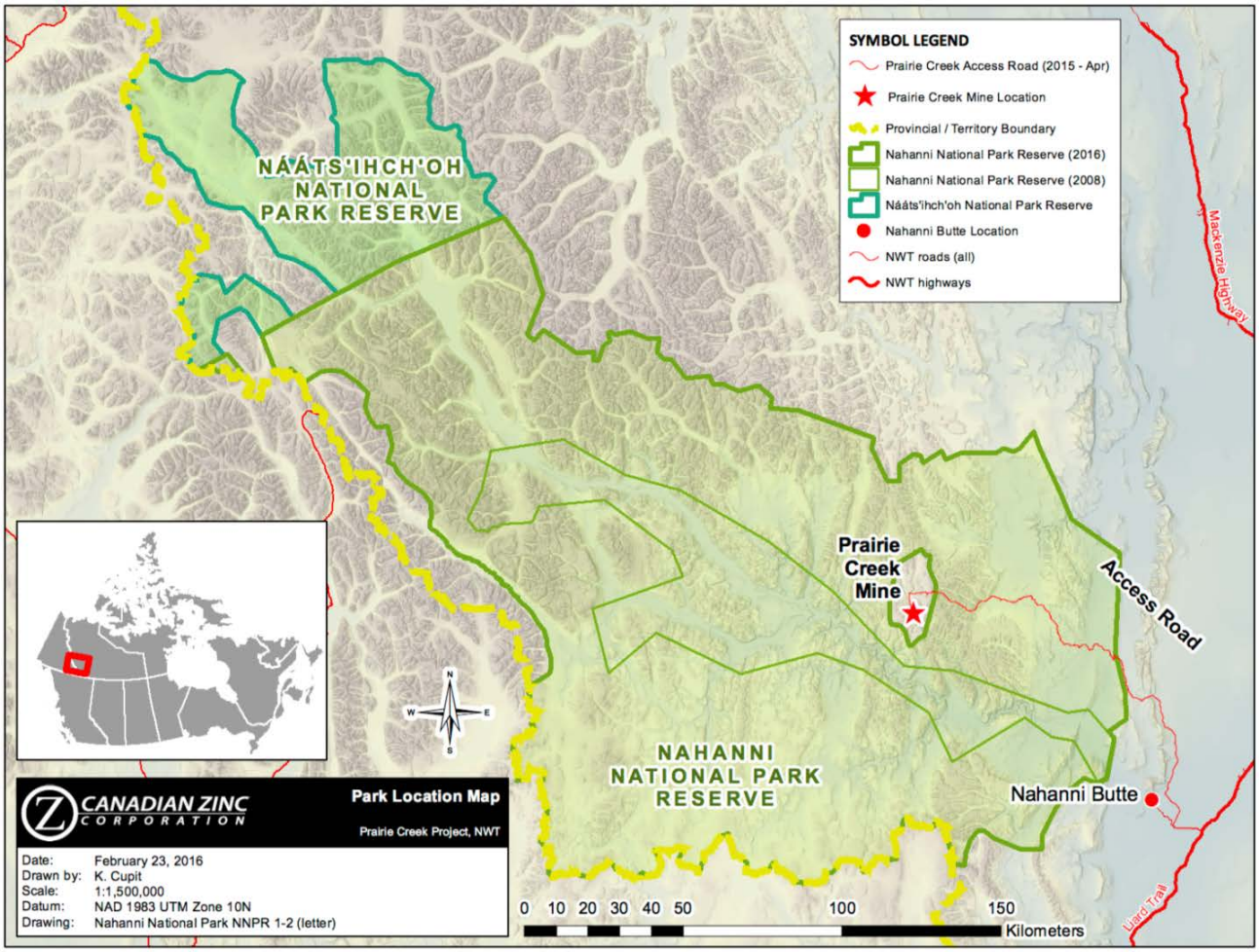
Existing exploration land use permits and water licences issued by the Mackenzie Valley Land and Water Board also have separate security deposits associated with them to ensure reclamation is carried out.

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, B.C. 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, B.C. 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, B.C. (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 diesel 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. 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$64 million (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.

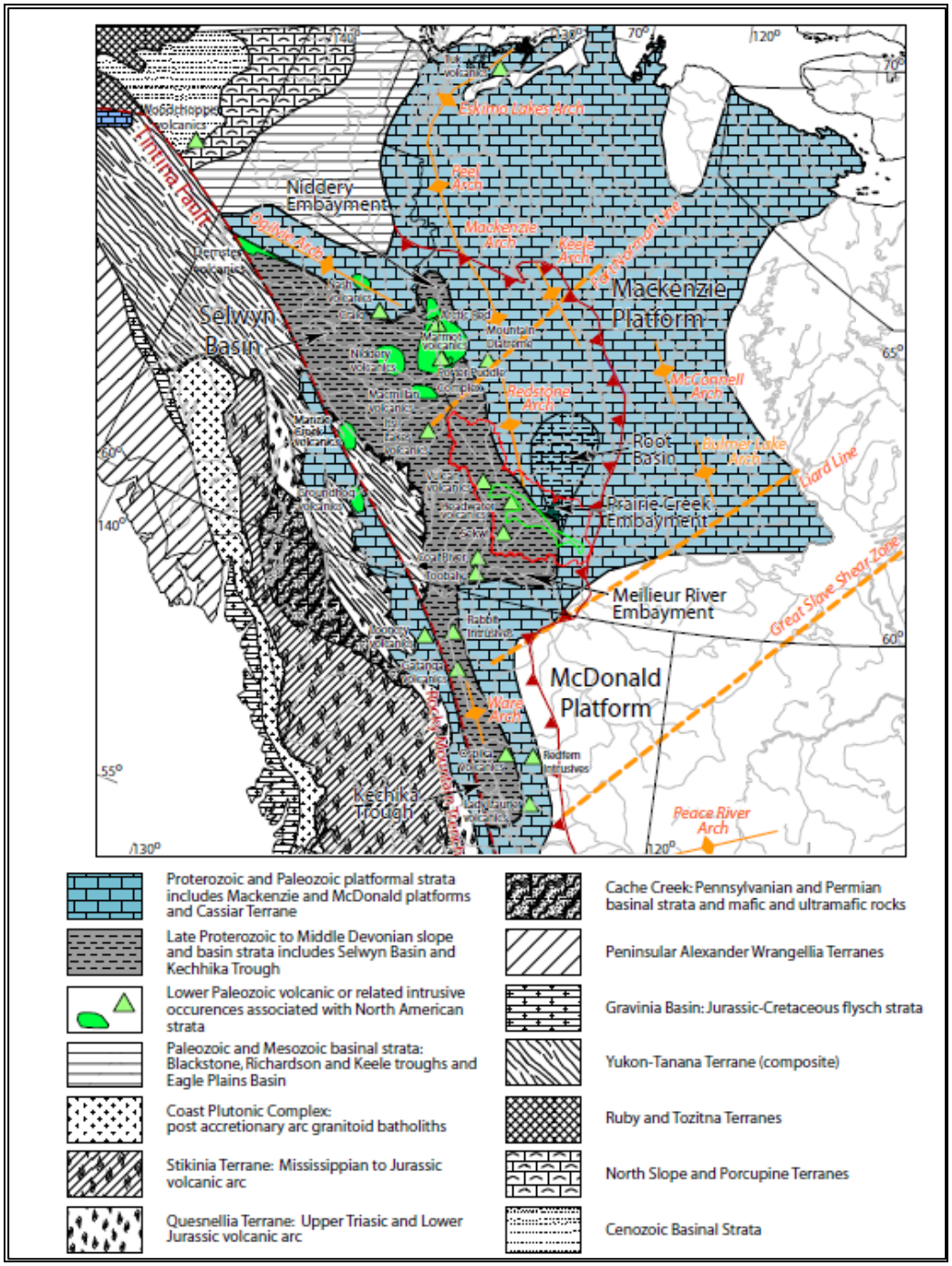
7 Geological setting and mineralization

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

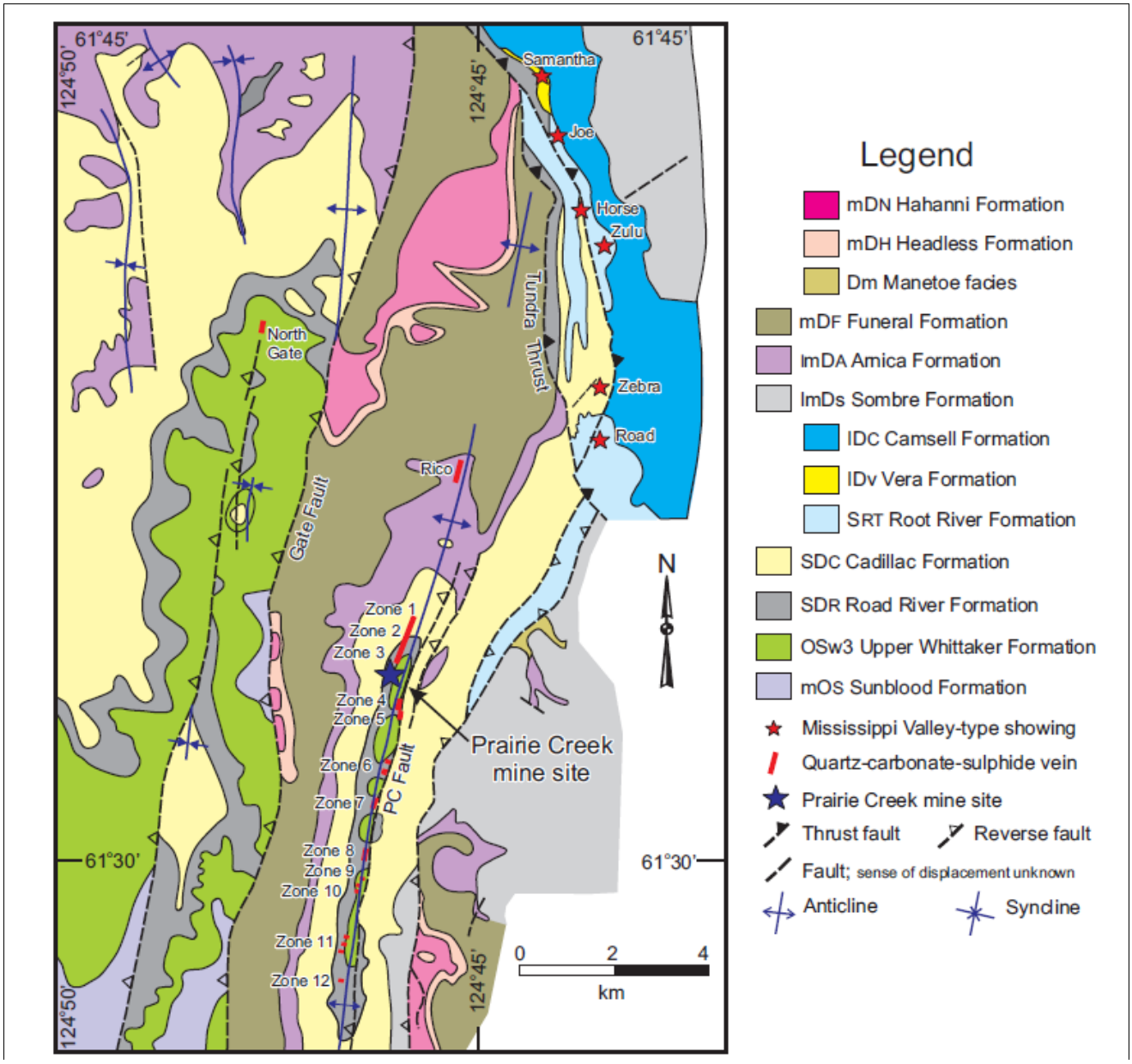
Formation	Code	Thickness (m)	Description
Arnica	ImDAb	200 to 250	Finely crystalline black nodular and banded cherty dolostone and limestone with white quartz-carbonate crackle veining.
Cadillac	SDC	300 to 350	Grey, thinly banded siltstone/shale with minor debris flow.
Road River	SDR	230 to 280	Mid-dark 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.

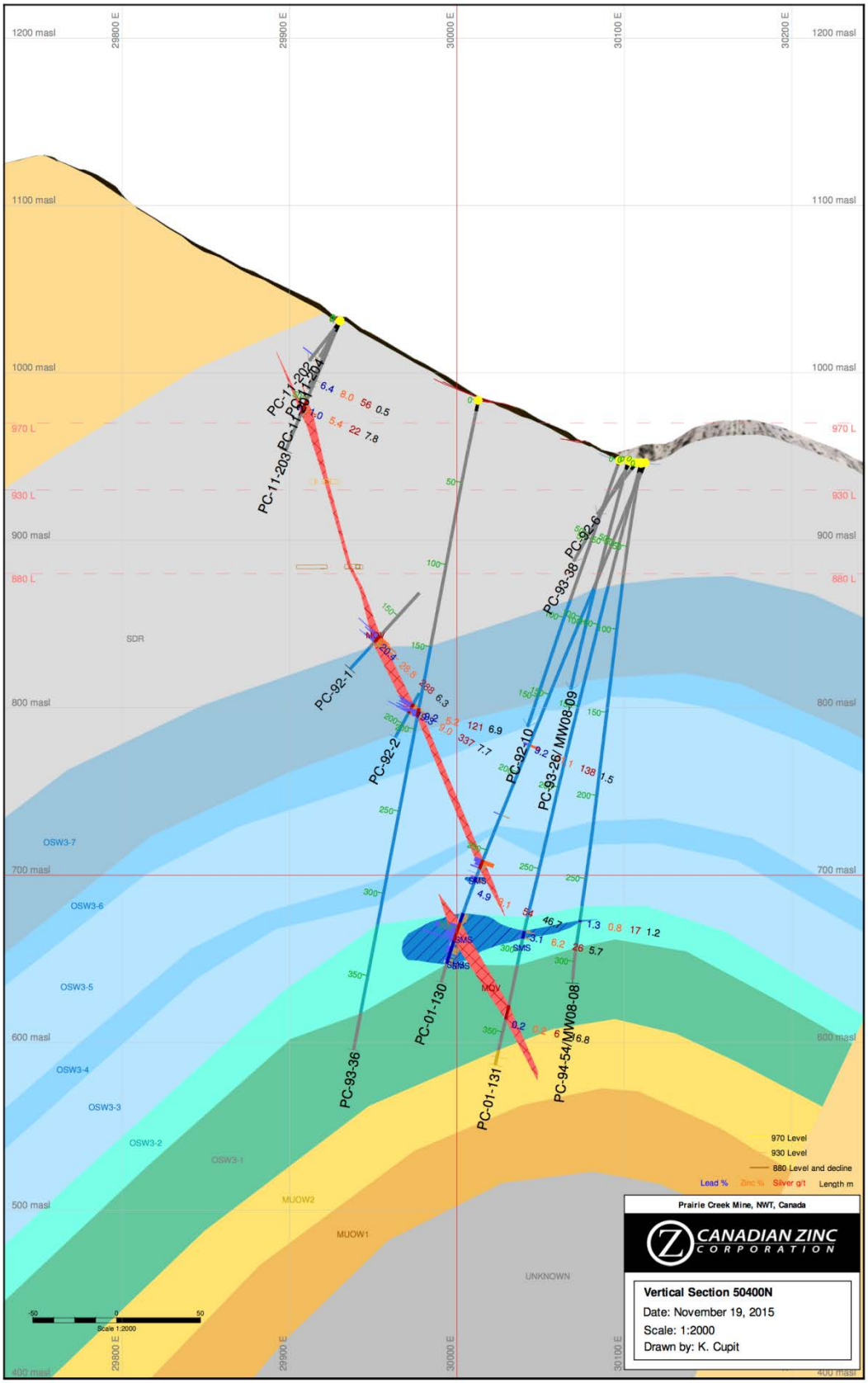
The PCA is bounded to the east by the Prairie Creek Fault and to the west by the Gate Fault. Both are west-dipping thrusts. The Prairie Creek Fault is up to 40 m thick and has a dextral displacement of approximately 1,500 m.

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 880m L 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

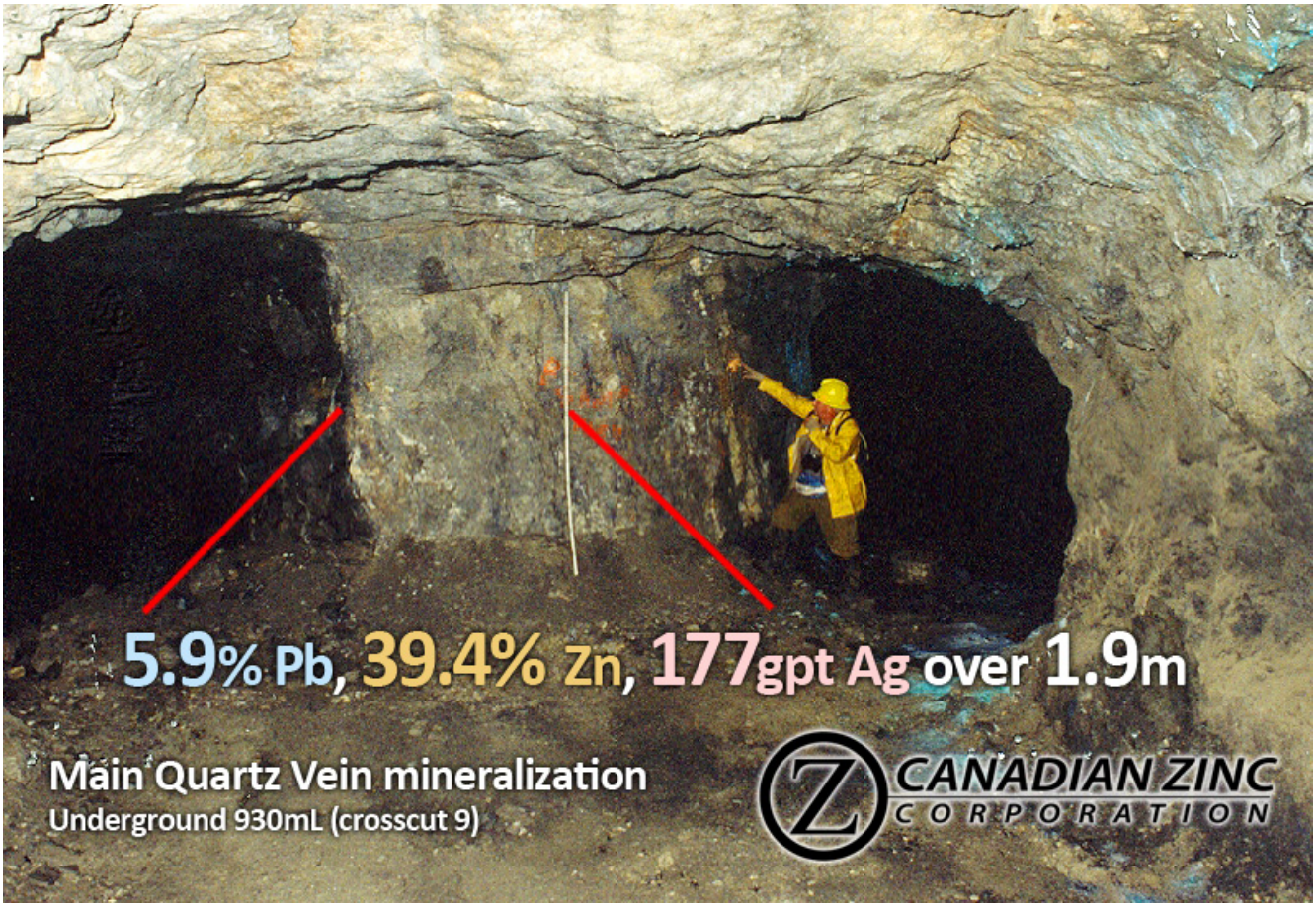
Vein mineralization developed within the cherty dolomites of the Ordovician-Silurian age Upper Whittaker Formation and shaley dolomites of the lower Road River Formation, along the axial plane of the PCA.

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.

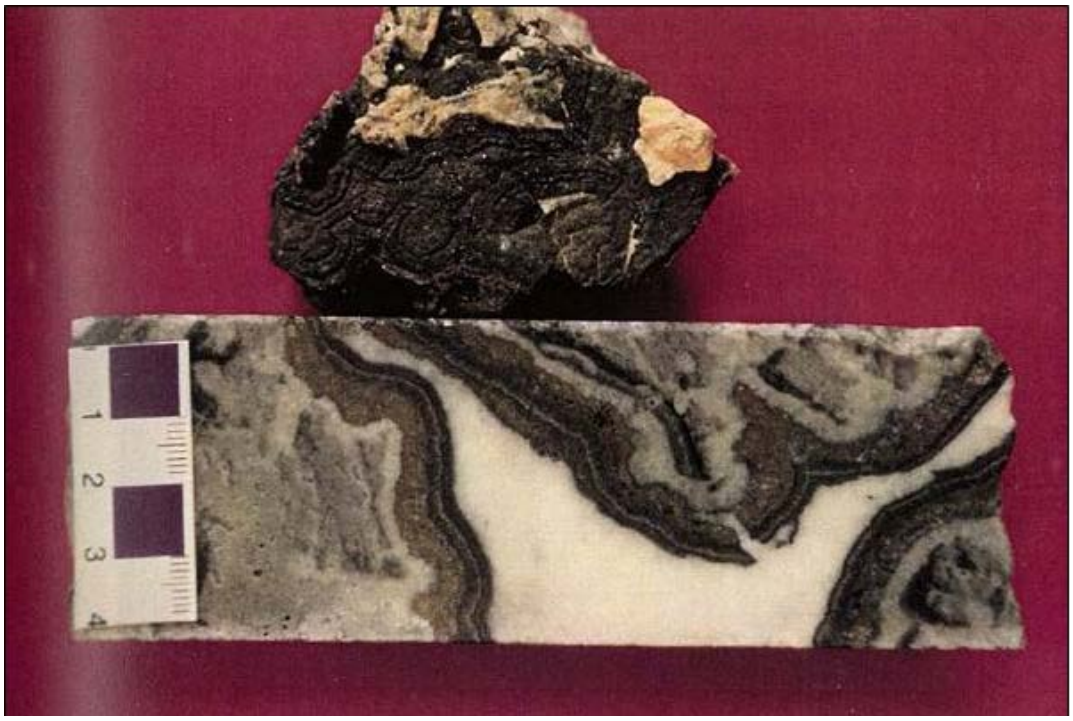
Figure 7.6 Stratabound Mineralization showing massive sphalerite and pyrite in drill core

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 effected 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, 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 B.C., a

subsidiary of Cabo Drilling Corporation of North Vancouver B.C., 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, B.C.

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 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, B.C. Prior to 2011, samples were transported by Greyhound bus to the Acme Labs assay laboratory in Vancouver, B.C. 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, B.C. (Smee). Standards were compiled from a shipment of mineralized samples sent by CZN to CDN Resource Laboratories Limited in Delta, B.C. (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 B.C. (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 2006/07 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

The Prairie Creek deposit comprises three types of mineralization that will be accessed during the current mine plan: Vein (MQV), Stockwork (STK) and Stratabound (SMS). Mississippi Valley-Type (MVT) mineralization does not figure in the proposed mine plan. No metallurgical tests have been conducted on Stockwork and Mississippi Valley-Type mineralization.

Vein mineralization comprises massive to disseminated galena and sphalerite, with lesser pyrite and tennantite-tetrahedrite in a quartz-carbonate-dolomite matrix. Secondary oxidation is locally developed to variable levels, yielding mainly cerussite (oxide lead) and smithsonite (oxide zinc); minor oxidation only of tennantite-tetrahedrite has been found. Oxidation levels of 20 to 30% are found in historic workings. Silver is present in solid solution with tennantite-tetrahedrite, and to a lesser extent with galena.

As reported by CZN, mineralogical characteristics of Stockwork mineralization are similar to those outlined for Vein material.

Stratabound mineralization is much less oxidized, with sulphides generally fine-grained, banded to semi-massive, and comprising massive, fine-grained sphalerite, coarse-grained galena, and disseminated to massive pyrite. Silver mainly occurs in solid solution within galena. There is no tennantite-tetrahedrite, little copper, and only half the galena, but substantially more iron sulphide/pyrite, than typical vein material.

13.2 Test work history

A considerable amount of test work has been done on samples from both Vein and Stratabound mineralization. Samples tested included drill core and underground bulk samples. Several test work reviews have been done, mainly by O'Kane Consultants Inc. (O'Kane) and SNC-Lavalin Inc. (SNC-Lavalin).

As reported by O'Kane, early metallurgical test work was conducted in the late 1960s through to 1974, but few details are available for this early stage work.

From the 1980s through to 2000, various metallurgical tests were done at different laboratories to investigate metallurgical responses and optimize process conditions.

During 2004 and 2009, comprehensive test programs were conducted at SGS Lakefield to optimize metallurgical performance and process conditions, and to test the metallurgical response of various bulk samples to dense media separation (DMS).

More recent work has included flotation and other testing at SGS Lakefield in 2013 and an on-going flotation and mercury removal testing program at Global Mineral Research Limited (GMR), starting from 2014.

The test work programs conducted are summarized in Table 13.1.

Table 13.1 Major metallurgical testing 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	GRM	√	-	-	√

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

13.3 Mineralogy

Process mineralogical determinations were conducted throughout different metallurgical test work phases. Early 1980s work was undertaken at Lakefield (1980), CSMRI (1983) following pilot plant runs in 1982, de Randt (1992), G&T (1994), Hazen (1997), Harris (2000), and Terra (2006).

13.3.1 Vein mineralization

- **Head samples**

In 2006, Terra conducted a mineralogical study on the samples collected from the 930 and 883 adits.

The focus of this work was to assess the nature and characteristics of the main lead and zinc carriers. Results essentially confirmed previous findings, as described below.

The main lead carrier is galena, followed by cerussite (lead carbonate) and anglesite (lead sulphate). Most of the mineral textures with the lead carriers are 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 present locally fine, relatively complex textures, which could require a finer grind size to liberate the target minerals. Lead mineral distribution is illustrated in Figure 13.1.

Sphalerite is the main zinc carrier, occurring mostly as liberated grains, or forming simple coarse-grained intergrowths, predominately with galena and quartz. Smithsonite is also a major zinc carrier, and is commonly intergrown with sphalerite. Smithsonite also commonly forms finer mineral intergrowths with dolomite, which would require a fine grind size (<50 µm) to liberate. The zinc mineral distribution is shown in Figure 13.2.

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 is commonly intergrown with smithsonite and/or cerussite, as well as quartz.

Figure 13.1 Distribution of lead minerals in lead carriers, Terra (2006)

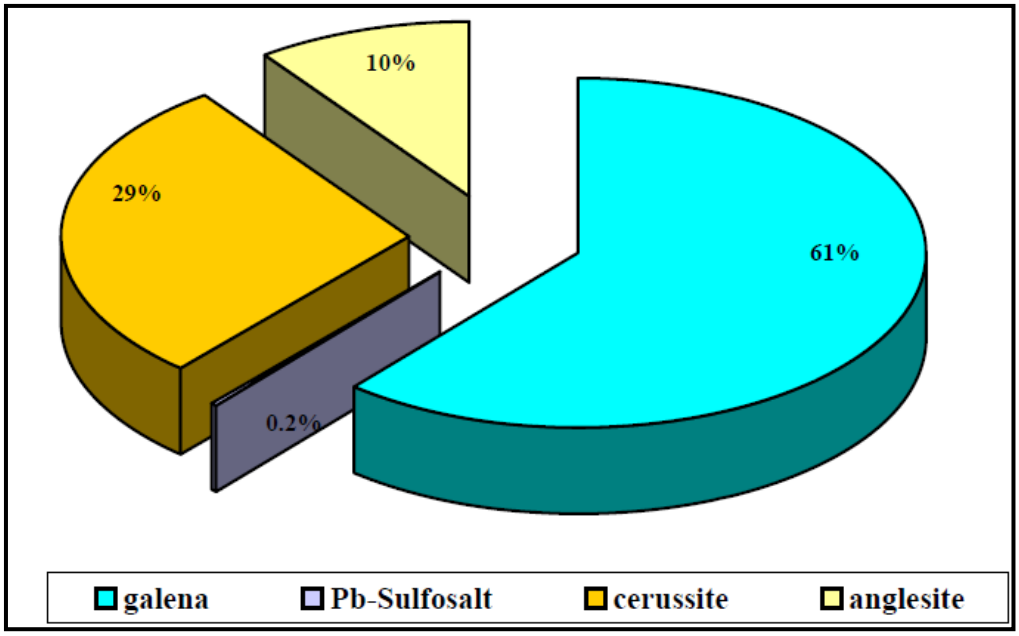
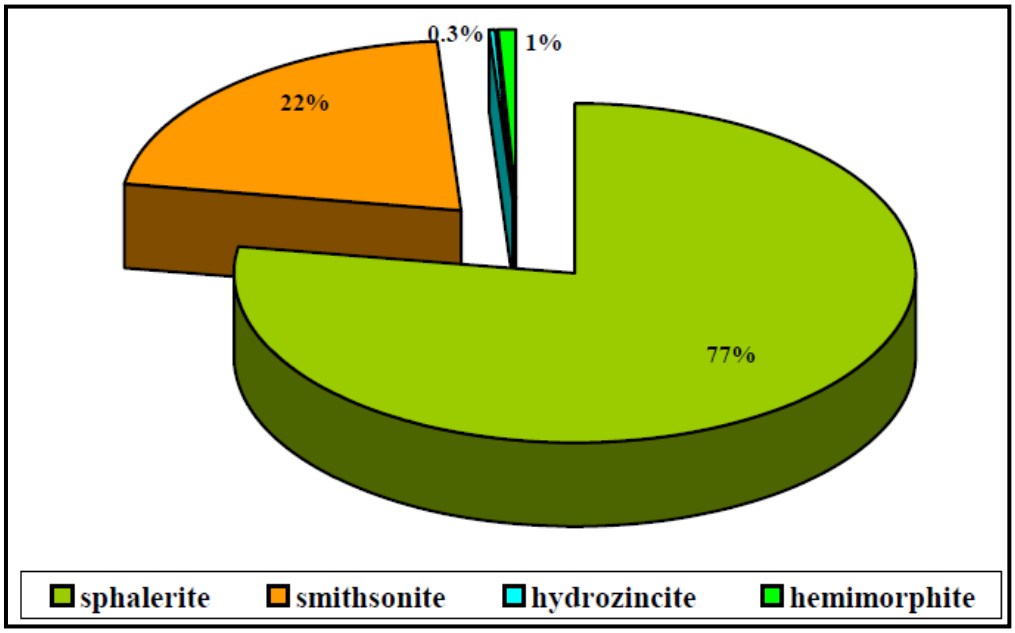


Figure 13.2 Distribution of zinc minerals in zinc carriers, Terra (2006)



In 2014 G&T determined mineral composition and liberation of a vein sample. The data produced are shown in Table 13.2 and Table 13.3.

Table 13.2 Mineral composition, G&T (KM440) (1994)

Mineral Composition – Weight (%)				
Tetrahedrite	**Galena**	**Sphalerite**	**Pyrite**	**Gangue**
1.6	18.0	24.7	1.6	54.1

Table 13.3 Sulphide mineral liberation and distribution, G&T (KM440) (1994)

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

The G&T work indicated that at a primary grind of 80% passing 50 µm, the principal sulphide minerals in the vein sample were well enough liberated from each other to permit a flotation separation; regrinding the rougher concentrate streams might not improve the zinc, and would have a lesser effect on the lead final concentrate grades.

The 1992 de Randt work consisted of mineralogical analysis on fourteen 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. Although most of the metal 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 with a scanning electron microscopy (SEM) energy dispersive x-ray analyzer.

Hazen's 1997 tests were on a Vein bulk head sample and the flotation concentrates from the KM469 test program. Table 13.4 shows the assay results of the head sample, in which approximately two-thirds of the lead and zinc are present as oxide minerals.

Table 13.4 Assay results of head sample, Hazen (1997)

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.

An investigation of mercury in flotation concentrates revealed the following:

- 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, 1.0 to 1.3% Ag).

In 1983, CSMRI mineralogical work demonstrated liberation at approximately 80% for cerussite and smithsonite in the 100 to 150 mesh particle size fraction.

- **Flotation products**

In 2006, Terra conducted a mineralogical investigation on lead and zinc oxide concentrates containing 65.6% lead and 5.76% zinc, and 23.7% zinc and 1% lead respectively. Results are shown in Table 13.5. Main mineral contaminants in the lead concentrate were sphalerite and pyrite.

Table 13.5 Mineral composition and liberation of flotation concentrates, Terra (2006)

Lead Concentrate			Zinc Oxide Concentrate		
Mineral	Weight (%)	Mineral Liberation (%)	Mineral	Weight (%)	Mineral Liberation (%)
Galena	82.1	96	Smithsonite	55.5	93
Sphalerite	8.5	31	Dolomite	20	92
Pyrite	4.6	87	Quartz	18.2	83
Tetrahedrite/Tennantite	4.2	90	Fe-oxides	3.2	79
Chalcocite	0.1	100	Non-opaque	1.3	92
Covellite	0.1	-	Galena	0.9	56
Non-opaque	0.4	92	Zincite	0.7	55
-	-	-	Pyrite	0.1	49
-	-	-	Chalcopyrite	0.1	44
-	-	-	Chalcocite	trace	> 95
-	-	-	Sphalerite	trace	-

A 1994 Cominco study on sulphide flotation tailings concluded that high losses of lead and zinc to tailings were primarily as carbonates, such as cerussite and smithsonite.

A 1993 Cominco study on lead and zinc concentrates produced from the KM370 test work program 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 sphalerite lattice, and arsenic and antimony may be in tennantite.

As a part of 1982 pilot plant test work, CSMRI conducted mineralogical examinations on various products produced from Runs 101 and 103. The results indicated that:

- All samples contained both galena and sphalerite.
- All samples contained detectable amounts of cerussite and smithsonite, except for the lead third cleaner concentrate from Run 103.
- Tetrahedrite was in all samples with silver concentrations greater than 25 oz/ton. The exact tetrahedrite composition in these samples was not known.

Lakefield (1980) and Harris (2000) also examined various flotation products to determine main mineral occurrences and associations among the minerals.

13.3.2 Stratabound mineralization

The sample studied by the 1992 G&T test work contained galena as well-formed crystals and accounting for all of the lead present. Very small 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.

Pyrite was the dominant sulphide, and accounted for most of the iron sulphides present. Some marcasite was also observed, but no pyrrhotite was detected.

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 Table 13.6.

Table 13.6 Mineral composition of ore sample, G&T (KM370) (1992)

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

* Marcasite and pyrite are shown as pyrite

At a grind size of 80% passing 75 µm, liberation of galena and sphalerite was relatively poor. Finer primary grinding to 80% passing 44 µm improved both galena and sphalerite liberation, to the point where a successful flotation separation could be anticipated. Although the galena was relatively well liberated at the finer primary grind level, binary assemblages between galena and sphalerite, pyrite and gangue were observed.

Sphalerite liberation was poor, even at finer primary grind levels. About 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.

It was indicated from the mineral liberation data that a primary grind of about 80% passing 50 µm would be required to give adequate galena and sphalerite liberation.

G&T KM462 work 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.

13.4 Comminution test work

Six composite sample grindability tests were done in 2007 (Table 13.7). Results showed medium hard mineralization with a Bond work index of 8.5 to 11.1 kWh/t, averaging 9.7 kWh/t. The highest Bond work index was for the high oxidation composite.

Table 13.7 Grindability test results, SGS Lakefield (2007)

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

In 1994, G&T (KM440) tested the grindability of a vein mineralization sample at sieve size 106 µm. The Bond ball mill work index of the sample was 9.7 kWh/t.

In 1992, (KM370), a Stratabound sample was tested for grindability. The mineralization was seen to be relatively soft, having a Bond ball mill work index of 8.9 kWh/t. The KM462 test program included a Bond ball mill work index determination using the comparative procedure. The Bond work index was measured to be 9.2 kWh/t.

13.5 Pre-concentration (DMS) test work

Hazen (1997) did a preliminary DMS investigation on Vein samples. As shown in Table 13.4, the sample contained a high proportion of oxide lead and zinc minerals. As shown in Table 13.8, after being crushed to different particle sizes, about 55 - 60% of the material was rejected; lead losses ranged from 3.7 to 6.8%, zinc losses from 6.0 to 11.8% and silver losses from 8.5 to 15%.

Table 13.8 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

Chemical phase analyses of < 10 mesh gravity separation tailings (float product) showed that the majority of the lead and zinc losses occurred as oxides, amounting to 86.5% of the total lead and 70% of the total zinc 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 slimes ranged between 21.2 and 35.5% of these metals.

SGS Lakefield (2005) did DMS tests using heavy liquid separation (HLS). A composite of 50% Vein mineralization and 50% Stratabound, crushed to < 1/2" or 1/4", was used for the tests. The different size materials were screened to remove < 6 mesh material separately. The coarse fractions were tested by three

different heavy liquids with gravity densities of 2.6, 2.8 and 3.0 g/cm^3. As shown in Table 13.9, the test results indicated the following:

- For the coarse particle size (crushed to ½"), at a media gravity density of 3.0 g/cm^3, approximately 42% weight was rejected as waste, with 4.7% of the lead, 3.4% of the zinc, and 4.7% of the silver being lost in the float fraction.
- For the fine particle size (crushed to ¼"), at a media gravity density of 3.0 g/cm^3, the amount of the waste rejected was reduced to approximately 27% compared to 42% at the ½" size.

Table 13.9 Heavy liquid separation test results, SGS Lakefield (2005)

Product	Weight	Assays						Distribution (%)					
	(%)	Pb	Zn	Cu	Ag	Pb Oxide	Zn Oxide	Pb	Zn	Cu	Ag	Pb Oxide	Zn Oxide
		(%)	(%)	(%)	(g/t)	(%)	(%)						
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	0	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	0.9	1	0.9
¼" Sample 2.6 SG Float	0.4	0.71	0.54	-	22.5	0.41	0	0.03	0.03	0	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	0	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	0	1.3	1.9	2.1
½" Sample 2.6 SG Float	0.2	0.52	0.62		59.8	0.32	0	0	0	0	0.1	0	0
Head (calculated)	100	9.44	11.1	0.27	135.7	3.91	2.04	100	100	100	100	100	100

In 2005, Confidential Metallurgical Services (CMS) and SGS Lakefield did HLS testing on 13 samples from the 883 mL and 930 mL adits. The samples were crushed to < ½" particle size and then screened to remove the < 14 mesh (1.4 mm) fraction. The > 14 mesh fraction was tested at heavy liquid specific densities of 2.8 and 3.0 g/cm^3. For 883 mL samples, weight percentages of the HLS rejects (floats) ranged from 14 to 53% at 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 930 mL samples accounted for 8 to 34% of the feed, 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 930 mL samples, and 23% more rejects for 883 mL 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 was rejected as waste. The loss of lead, zinc and silver to the HLS float 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% Ag. High-oxidation samples averaged slightly higher metal losses to the rejects.

In 2009, SGS did large-scale HLS testing on a composite generated from the 883 mL and 930 mL adits. A 530 kg sample was processed through media with a specific density of 2.8. As shown in Table 13.10, approximately 41% of the feed weight was rejected as waste (float product). Loss of economic metals to the HLS float was 2.5% Pb, 4.4% Zn, and 2.6% Ag.

In 2013, SGS Lakefield also did DMS testing on a composite from 883 mL adit samples. The sample was stage-crushed to < 6.35 mm and screened to remove the < 20 mesh fraction. The coarse fraction underwent DMS upgrading at a media specific density of 2.8. As shown in Table 13.11, 97% of the copper, 99% of the lead, 98%

of the zinc and 98% of the silver were recovered to the sink and fine fractions. About 21% of the feed was rejected into the gangue fraction.

In 2014, GRM conducted HLS tests at a media specific density of 2.8 on various samples, including a master composite sample and 17 variability test samples. 20.2% of the feed from the master composite was rejected into the float fraction with losses of the lead, zinc and silver by 1.7%, 1.7% and 1.6% respectively. The floats rejected from the 17 variability samples ranged from 9.0% to 44.6%, averaging 22.7%.

Table 13.10 Heavy liquid separation results at 2.8 SG, SGS Lakefield (2009)

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

Table 13.11 Material balance of overall feed, SGS Lakefield (2013)

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

13.6 Flotation test work

13.6.1 Batch flotation test work before 2001

- **Vein mineralization**

Lead and zinc flotation

As reported by O'Kane, the Galigher Company performed test work on a high-grade sample in 1969, although it was acknowledged that the sample may not have been representative of the mineralization of the vein deposit.

In 1980, Lakefield Research of Canada Ltd. conducted preliminary flotation test work on two head samples: a sulphide composite and an oxide composite. The test results showed 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 high-grade lead and zinc concentrates could be produced. However, metal recoveries decreased significantly at the sulphide flotation stages, although it

appeared that lead oxide minerals were able to recover 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 sulphide to replace sodium cyanide). The work also evaluated effects of primary grind sizes on lead and zinc differential flotation. Optimum results were attained at a primary grind size between 70 and 80% passing 200 mesh. The tests also indicated that 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 were good in grades, but contained significant deleterious elements. Projected metallurgical performance by Kamloops is shown in Table 13.12.

Table 13.12 Metallurgical performance projections, Kamloops (KM019) (1980)

Product	Concentrate Grades (%)		Recovery (%)			
	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 (KM034) using a sample identified as Lower Adit Ore 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.

In later testing (KM077, 1982), potential alternatives to the cyanide-based reagent scheme used previously were examined. Kamloops indicated that complete exclusion of cyanide from the reagent scheme for the mineralization would not be possible.

Optimum primary grind size in both the above tests appeared to be about 75% passing 200 mesh. Regrinding benefits on lead rougher concentrate were of marginal value to lead metallurgical performance. However, zinc rougher concentrate regrinding was seen as key to zinc cleaner circuit stabilization. Other Kamloops testing (KM040) showed regrinding zinc rougher concentrates would not significantly affect the zinc metallurgy.

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 necessary for rejecting zinc from lead concentrate for the Vein mineral sample.

In 1982 CSMRI did pre-pilot plant flotation testing to determine effects of various conditions on metallurgical performances. Results are summarized as follows:

Effect of primary grind size

Primary grind size varied from 49.5% to 93.9% passing 200 mesh. Metal recoveries to lead and zinc rougher concentrates were not significantly affected, although lead rougher recovery was reduced by about 4% at the coarsest grind. A primary grind size of 70 to 75% passing 200 mesh was used for the remainder of the laboratory flotation tests.

Effect of cyanide

Testing of cyanide dosage on lead rougher flotation showed that, with the addition of 2.0 lb/ton (1,000 g/t) of soda ash in the primary grind, 1.0 lb/ton (500 g/t) sodium cyanide produced the highest-grade lead rougher concentrate (49.0% lead) with the lowest zinc content (16.1% zinc), compared to 0.4 and 3.0 lb/ton (200 and 1,500 g/t) sodium cyanide. Lead recovery was 72.7%.

Sodium sulphite to partially replace sodium cyanide

Using a combination of 0.6 lb/t (300 g/t) sodium cyanide and 1.0 lb/ton (500 g/t) or more sodium sulphite significantly decreased lead grade of lead rougher concentrate, although recovery improved by 2 to 3%. Zinc reporting to the lead rougher concentrate increased substantially from approximately 20%, without adding sodium sulphite, to 82% with adding 6 lb/ton (3,000 g/t) of sodium sulphite. This reagent scheme was not satisfactory.

Effect of sodium sulphide

When sodium sulphide dosages were higher than 3.0 lb/ton (1.5 kg/t), lead grade of the lead rougher concentrate improved by 6%, with more efficient rejection of zinc entrainment. The lead recovery to the concentrate also improved by approximately 2%.

Effect of ammoniacal zinc cyanide

CSMRI used ammoniacal zinc cyanide as replacement for sodium cyanide to suppress zinc in the lead rougher flotation circuit. The testing showed that at tested dosage levels, the mixture could not effectively reject zinc minerals or improve lead recovery.

Lime for pH control

To study pH modification effect on lead flotation, lime was used in place of soda ash to condition pulp pH. At similar pH level as soda ash, lime produced inferior results; lead metallurgical performance also deteriorated at pH 12.0 compared to pH 9.5.

Lead cleaner flotation - cyanide addition and regrinding

Addition of sodium cyanide in lead cleaner flotation stages did not improve final lead concentrate grade. The effect of regrinding of 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.

Zinc cleaner flotation - frother and copper sulphate

Higher frother dosage of 0.14 lb/ton (0.07 kg/t) compared to 0.04 lb/ton (0.02 kg/t) reduced zinc grade of the first zinc cleaner concentrate from 53.6% to 36.5%. It appeared that addition of copper sulphate improved zinc metallurgical performance in zinc cleaner flotation circuits.

Soda ash in zinc flotation for pH control

Soda ash instead of lime for zinc rougher pulp conditioning did not affect zinc metallurgical performance in the zinc rougher flotation. However, large amounts (10.5 lb/t (5.25 kg/t)) of soda ash were required to maintain a pH of 10.5, compared to 3.0 lb/ton (1.5 kg/t) of lime.

Effect of zinc scavenger flotation

Additional flotation retention time in the zinc rougher flotation did not appreciably increase the recovery of lead or zinc. An increase in flotation retention time of five minutes produced a zinc scavenger concentrate of 6.64% lead and 4.56% zinc, but which only accounted for 2.1% of the lead and 0.9% of the zinc in the feed sample. In 1993, G&T did preliminary testing (KM424) on a few Vein samples. Lime and sodium cyanide were used in lead circuits to suppress zinc minerals at pH 8 to 9.5. Lime and copper sulphate were used in zinc circuits to suppress pyrite and activate zinc minerals. It appears that zinc minerals were not effectively suppressed in the lead circuits. The zinc contents of the lead third cleaner concentrates were high, ranging from about 14 to 20%.

In 1994, G&T (KM440) 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 almost constant, despite relatively large changes in reagents and treatment conditions. Compared to 500 g/t each of calcium oxide and sodium cyanide as zinc mineral suppressants, sodium metabisulphite (SMBS) addition of 2,000 g/t in the primary grind resulted in very similar concentrate grades and recoveries. Lead recovery in the lead circuit was < 80% despite large collector additions and extended flotation times.

SMBS dosage test results showed that increasing SMBS from 2,000 to 5,000 g/t or decreasing SMBS from 2,000 to 0 g/t increased zinc contamination in the lead concentrates. Figures Figure 13.3 and Figure 13.4 show the G&T test results in terms of recovery and grade relationships for lead and zinc cleaner open circuits.

Figure 13.3 Lead grade recovery curves for cleaner open circuits, G&T (KM440) (1994)

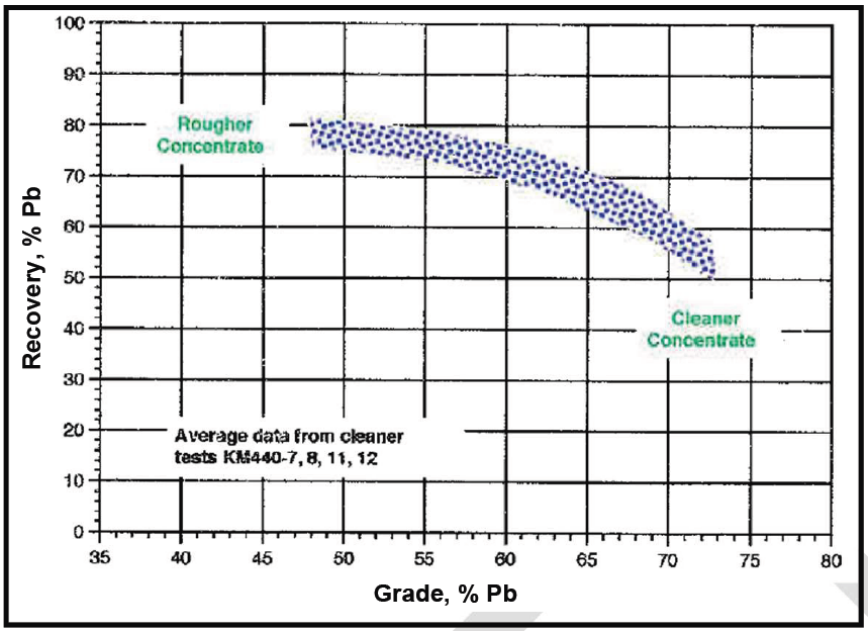
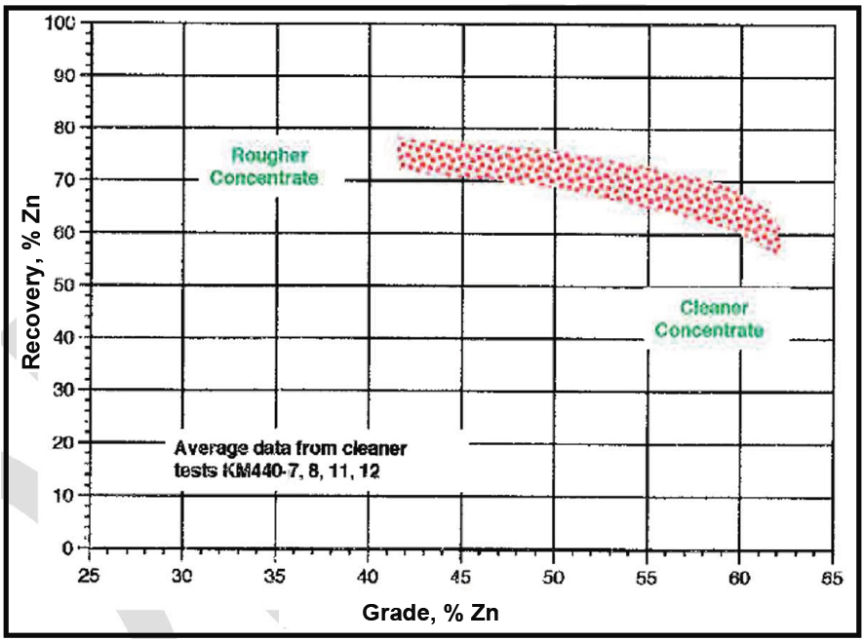


Figure 13.4 Zinc grade recovery curves for cleaner open circuits, G&T (KM440) (1994)



Regrinding of cleaner feed streams did not appear to enhance lead circuit performance, but regrinding of zinc cleaner feed stream was considered important.

KM454 testing used a copper-lead-zinc differential flotation technique, in an effort to produce separate copper, lead and zinc concentrates. 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 might not be practicable in industrial operations.

G&T (KM469) tested a batch differential flotation protocol to prepare flotation concentrates of copper, lead and zinc for smelting studies. A high-grade head sample, containing approximately 0.77% copper, 20.8% lead and 26.6% zinc was used. 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 high grades deemed the sample to not represent overall Vein mineralization.

In 1994, G&T did test work (KM488) on four different Vein samples. No detailed test conditions were reported. G&T conducted further tests (KM497) using increased reagent levels in the flotation. Removing the < 45 µm fraction prior to grinding and flotation resulted in higher zinc first rougher concentrate (61.1% zinc), compared to the concentrate grade of 45.9% zinc by using 700 g/t butyl xanthate and without fines removal.

Copper/Lead flotation separation

Several test programs investigated copper metallurgical response: copper and lead sequential flotation and 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. 66% of galena 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 copper concentrate assayed 8.5% copper and 57.3% lead.

In 1980 Kamloops (KM019) investigated potential methods to produce a silver-rich copper concentrate from the composite sample tested by the L.R.2252 program at 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 one of two copper and lead separation techniques:

- Potassium dichromate with sodium silicate or sulphur dioxide to depress galena
- Conditioned by sulphur dioxide, then pH raised with lime and used a weak collector

Both procedures apparently gave similar results. The separation procedure was deemed complex and seen as unlikely to be able to be controlled to the degree required.

In 1980, Kamloops (KM034) performed copper and lead separation tests on the 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 with those obtained by Lakefield. Kamloops concluded that producing a silver-rich copper concentrate from both the Upper and Lower 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 a further testing (KM040) 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, copper concentrates 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 (KM048) 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 could be 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 was seen in metallurgical results; however, the starch and sulphur dioxide technology is more complicated.

In 1994 G&T sequential flotation was done on Vein sample KM454 in an effort to produce a separate copper concentrate. Three collectors were tested, with Minerec 2030 giving 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 KM454 program, a large-scale, open-circuit flotation test (KM469-1) was carried out to produce copper, lead, and zinc concentrates for smelting testing from a Vein 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.

The subsequent study (KM474) 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 MBS. 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, Vein sample tests 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 previous results.

Effect of recycling process water

In 1981, Kamloops tested (KM040) the Upper Adit Ore sample for recycle water effect on metallurgical response, using the soda ash/cyanide and SMBS reagent scheme:

- Case 1: fresh water throughout
- Case 2: rod mill dilution water made of 50% recycle water
- Case 3: all flotation feed water made of recycle water, all other fresh water
- Case 4: all phases of the test carried out in recycle water

Recycle water use reduced copper grade in copper concentrate from 28.6 to about 20% in Cases 2 and 3, and 15% in Case 4. Lead grade in copper concentrate increased from 15 to about 30% in Cases 2 and 3, and 39% in Case 4. Case 3 had the best copper recovery.

Lead grade and recovery were similar whether fresh or 50% recycle water was used in the primary grind, but decreased significantly in Cases 3 and 4.

50% recycle water use in the primary grind improved zinc metallurgy significantly. A zinc third cleaner concentrate of 56% zinc and 4.3% lead at 59% zinc recovery was produced.

Cerussite and smithsonite flotation

In 1980, Lakefield (L.R.2252) did preliminary oxide mineral flotation tests on a sulphide composite and an oxide composite. About 20% of lead and zinc minerals in the sulphide composite and about 50% of lead and zinc minerals in the oxide composite were in oxide forms. The oxide mineral flotation was conducted separately on the two samples.

- Sulphide composite: The sulphide flotation tailings was filtered, re-pulped, and then conditioned with sodium sulphide. Cerussite was floated with potassium amyl xanthate collector (Z6). 9.4% of the lead reported to the oxide concentrate with 18.5% lead and 4.2% zinc. The oxide lead flotation tailings were 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.
- Oxide composite: The sulphide flotation tailing was similarly conditioned with sodium sulphide. Cerussite was floated with Z6 collector. Oxide lead concentrate averaging 11.5% lead and 7.8% zinc was produced. The additional lead flotation recovered about 22% of the lead. No oxide zinc flotation was conducted.

CSMRI (1982) 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 excess sodium cyanide depressed 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 (NP831003) did a series of open-circuit tests to determine if cerussite and smithsonite could be recovered by 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 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 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.

Test 15 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.

The test work showed that napthenic acid as a collector for smithsonite was not effective and silica flotation prior to smithsonite flotation was not successful.

The process patented by 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%.

The 1994 KM488 test program targeted lead and zinc non-sulphides recovery potential. However, the sample lead and zinc non-sulphide contents were much lower than expected and few conclusions were drawn from the test program.

Stratabound mineralization

Stratabound mineralization generally has little copper, low silver and about 50% pyrite/marcasite. Lead and zinc minerals mainly occur as sulphides. Testing was done to determine optimum grind size and investigate suppression reagent response.

Kamloops (1992) did test work to establish a processing flowsheet and to select an appropriate reagent regime. Two primary grind sizes of 80% passing 80 and 50 µm were tested. Lead circuit performance was seen to be insensitive to change in grind size. However, the zinc circuit was very significantly influenced by grind size variations.

Tests of two sphalerite/pyrite/marcasite suppression reagent schemes, lime-cyanide, and lime alone, showed similar lead and zinc flotation differential. 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% lead grade and 80.6% recovery, and a zinc concentrate with 52.0% zinc 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 the KM424 tests, 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.

KM462 testing used a simple two-product process at a primary grind of 80% passing 50 µm. However, it was noted that a slightly coarser primary grind between 80% passing 50 and 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 Vein mineralization did not produce an acceptable metallurgical response for the Stratabound 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.

2000 UBC testing showed the Stratabound sample very sensitive to overgrinding. For both lead and zinc, there were elevated losses in the finer than 325 mesh fraction in the flotation tailings. It was recommended to evaluate staged grinding with flash flotation.

13.6.2 Flotation test work after 2001

- **Metallurgical samples**

Testing by SGS Lakefield after 2001 was done on composites generated from samples collected from several underground adit crosscuts. In the Phase I & II programs, three samples were generated from material from the Vein zone (Upper (930 mL) and Lower (883 mL) adits) and the Stratabound zone. Composites were labelled 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.

Main element assays for the zone composites are shown in Table 13.13.

Table 13.13 Head assay – phases I AND II, SGS Lakefield (2005)

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 III, flotation test samples were diluted Lower Zone (870 m) and Upper Zone (930 m) composites from Phases I and II. The samples were upgraded by the HLS procedure. Flotation head assay results are shown in Table 13.14.

Table 13.14 Head assay – phase III, SGS Lakefield (2005)

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

Three composite samples were tested in Phase IV, 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 13.15.

Table 13.15 Head assay – phase IV, SGS Lakefield (2007)

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 V 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 13.16.

Table 13.16 Head assay – phase V, SGS Lakefield (2009)

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 to generate flotation concentrates, flotation tailings, and supernatants. Sample Head assay results (883 mL adit composite) are shown in Table 13.17.

Table 13.17 Head assay – SGS Lakefield (2013)

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

In 2014 and 2015, GRM conducted a test program to generate concentrate samples for a bio-leaching program and simplify flotation reagent regime. One master composite sample and 17 variability test samples from the mine adits were collected for the testing. The total head grade ranged from 8.0 to 43.5% for total lead and 4.3 to 28.9% for total zinc respectively. However, the averaged oxidation rate is high at approximately 47% for lead and 24% for zinc. This implies that the samples may not be representative to the Vein mineralization.

- **Flotation testing**

Between 2004 and 2013 SGS Lakefield did flotation test work that focused on:

- Process flowsheet optimization, including reagent scheme optimization.
- Utilizing DMS pre-concentration ahead of flotation to reject gangues.
- Determining recoveries of oxide lead and zinc minerals.

Open batch results are summarized below; locked cycle test (LCT) results are summarized in Section 13.6.3.

Phase I and II – SGS Lakefield (2005)

Objectives of the flotation test work were:

- To develop a treatment process for beneficiation of both the Vein and Stratabound mineralization using a cyanide-free reagent scheme.
- To confirm if both mineralization types could be co-mingled in the milling process.

- To determine treatment processes for recovering oxide lead and zinc minerals.

The overall results showed that the sulphide and oxide minerals can be recovered using a sequential flotation process, including 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:

Sulphide and oxide lead flotation circuits

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.

Sulphide and oxide zinc flotation circuits

Optimum pH levels 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 13.18.

Table 13.18 Reagent scheme for Prairie Creek mineralization – SGS Lakefield (2005)

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)

Optimum grind size for the Lower Zone composite was 80% passing about 80 µm. However, testing appeared to show better lead and zinc performance with finer primary grinding. Also, all mineralization responded reasonably well to the process flowsheet developed, and it was seen that the different types could be co-mingled in processing.

Phase III – SGS Lakefield (2006)

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 are described below:

- 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.

Table 13.19 shows the modified reagent scheme and test program results.

Table 13.19 Prairie Creek mineralization modified reagent scheme – SGS Lakefield (2006)

Reagent	Reagent Dosage (g/t)			
	Sulphide Lead Flotation	Oxide Lead Flotation	Sulphide Zinc Flotation	Oxide Zinc Flotation
Modifier and Depressant				
Na_2CO_3	1500 – 1800	-	1500 – 2500	-
$Na_2S \cdot 9H_2O$	500	800	-	1700
MQ3	600	-	-	-
MKF	200	-	-	-
Sodium Silicate 'N'	-	700	-	1500
DV177	-	450	-	1100
Calgon/Acumer 9000	-	-	100	-
$CuSO_4 \cdot 5H_20$	-	-	1500	-
Collector and Frother				
DF067	18	6	-	-
SIBX	28	70	70	-
Cytec 3894	-	-	16	-
PAX/Armeen C (50:50)	-	-	-	125
MIBC	4	-	-	-

MQ3: 70% ZnSO4, 20% Na2S2O3 and 10% Na2S2O5
DV177: short chain polyacrylamide; commercial code
DF067: collector, commercial code

For Upper Zone testing, results were better with pre-concentration than without. Main metal recoveries were 91% lead, 87% zinc, and 98% silver in lead and zinc concentrates.

For the Lower Zone, lead metallurgical performance was significantly better 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%, lower than that obtained from the as-received sample.

Phase IV – 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 III 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 on sulphide lead flotation of lime 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 showed better metallurgical performance than MKF. Therefore, 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. 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 grade obtained was 34.7% zinc.

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.

Table 13.20 shows the modified reagent scheme test results.

Table 13.20 Modified reagent scheme for Prairie Creek – SGS Lakefield (2007)

Reagent	Reagent Dosage (g/t)			
	Lead Sulphide Flotation	Oxide Lead Flotation	Zinc Sulphide Flotation	Oxide Zinc Flotation
Modifier and Depressant				
Na_2CO_3	2500-3600	-	1000-1200	-
$Na_2S \cdot 9H_2O$	500	800-1000	-	1000-1500
AQ4	200-300	150-250	200-300	-
P82	1000-1200	-	-	-
Sodium Silicate 'N'	-	500-1000	-	800-1200
$CuSO_4 \cdot 5H_2O$	-	-	1200-1500	-
DV177	-	-	-	250-400
Calgon/Dispersogen (1:1)	-	-	-	200-400
Collector and Frother				
DF067	20-25	Oct-15	-	-
SIBX	20-24	40-60	40-60	40-60
3894	-	-	15	-
Normac S	-	-	-	150-200
MIBC	5-Oct	As required	-	-

P82: 50% $ZnSO_4$, 25% $Na_2S_2O_3$ and 25% $Na_2S_2O_5$
AQ4: 33% Accumer 9000, 34% Na_2SiO_3 and 33% Na_3PO_4

Batch testing of process recycle water effect on flotation selectivity showed deteriorated selectivity between lead and zinc minerals when LCT test flotation water was used.

Phase V – 2009, SGS Lakefield

SGS tested a bulk composite sample to generate flotation products for environmental and concentrate marketing review in 2009. It seemed that secondary copper minerals in the ROM sample resulted in zinc minerals activation, which caused a sulphide lead flotation selectivity problem. Using the Phase IV 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, 15% EDTA (ethylene diamine tetra acetic acid)).

Mine water did not affect sulphide lead and zinc flotation, but oxide lead flotation deteriorated. By adjusting reagent dosages and modifying depressant AQ4, oxide lead floatability was restored. Reagents used for the LCT tests are shown in Table 13.21.

Table 13.21 Modified reagent scheme for Prairie Creek – SGS Lakefield (2009)

Reagent	Reagent Dosage (g/t)		
	Lead Sulphide Flotation	Oxide Lead Flotation	Zinc Sulphide Flotation
Modifier and Depressant			
Na_2CO_3	4,800	-	1,900
$Na_2S \cdot 9H_2O$	500	1,000	-
SQ4	550	-	400
P82	1,200	-	-
Sodium Silicate 'N'	-	900	-
$CuSO_4 \cdot 5H_2O$	-	-	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)

2011 Test work, SGS Lakefield

No detailed data is available for 2011 SGS flotation test work for environmental issues.

2013 Test work, SGS Lakefield

In 2013, SGS did flotation test work on a DMS pre-concentrated sample, including two batch flotation tests and two locked cycle tests. The first batch flotation test used Vancouver tap water; the remaining tests used water from the 883 mL 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 V testing. Concentrate grades and metal recoveries of the sulphide lead and zinc concentrates produced were inferior to these generated in previous testing.

2014-2015 Test work, GRM

In 2014 and 2015, GRM conducted a flotation test program to generate concentrate samples for a bio-leaching program and simplify 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. A full test work report is not available at this time of review.

13.6.3 Flotation locked cycle test work

- **Vein mineralization**

Several test programs used LCT tests to investigate the effect of recycling middling streams on metallurgical performance. LCT tests before 2001 focused on sulphide flotation. Some tests included copper and lead separation. During 2004 and 2013 SGS did extensive LCT testing, including sulphide, and also oxide lead and zinc flotation.

Tables 13.22 and 13.23 show Vein mineralization test results before and after 2001.

Table 13.22 LCT test results before 2001 – Vein mineralization

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

Table 13.23 LCT test results after 2001 – Vein mineralization by 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	-

Product	Weight	Grade				Distribution			
	(%)	Pb (%)	Zn (%)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)	Ag (%)	Cu (%)
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	-
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	-

Product	Weight	Grade				Distribution			
	(%)	Pb (%)	Zn (%)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)	Ag (%)	Cu (%)
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	-
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

* Vein and Stratabound blended sample

The pre-2001 processing flowsheet did not include recovery of oxide lead and zinc minerals. LCT tests were focused on lead and zinc sulphide recovery. Some testing also investigated copper recovery. Although different reagent schemes and flowsheets were used, the test results indicated significant amounts of the target metals lost into the flotation tailings or as impurities into the concentrate products.

The lead concentrate recovered approximately 66 to 70% of the lead and 59 to 69% of the silver. The lead grades of the concentrates were approximately 50 to 69%.

Between 60% and 75% of zinc reported to zinc concentrates, with grades ranging from 50% to 59% Zn. Approximately 16 to 18% of silver was recovered into the concentrates.

LCT tests appeared to show that a copper concentrate containing about 20% Cu could be produced. Approximately 50% of the copper reported to the copper concentrates. The KM048 test program indicated silver content of the copper concentrate > 5,000 g/t.

Testing after 2001 included recovery of oxide lead and zinc minerals. Also most LCT tests used samples treated by HLS/DMS pre-concentration. A flowsheet of sulphide lead and zinc flotation followed by oxide lead and zinc flotation was adopted. As discussed above, reagent schemes were much more complex than those used before 2001. In general, 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. The tests showed better slime suppression control needed for oxide flotation circuits, especially for oxide zinc. The oxide lead flotation was relatively stable.

The sulphide lead concentrate recovered about 50 to 70% of the lead and 54 to 73% 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 6 to 15% (average 12%), excluding the 2013 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 were high, ranging from 55 to 62%. Oxide zinc flotation after oxide lead flotation further recovered about 5 to 20% of the zinc (average 12%). Average oxide zinc concentrate grade was 30% zinc (range 23 to 38%). The oxide concentrate further recovered about 4% of the silver.

LCT Test 3 (2013) had 27 cycle tests. Generally, 2013 testing gave inferior metallurgical performance in sulphide lead and zinc flotation compared to previous test results. The reasons for the lower metal recoveries are unknown, but possibly this was due to a high oxidation rate in the head sample or requiring further optimization in the test conditions.

In 2015 GRM 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. A full test work report is not available at this time for review.

2005 testing (Phase I & II) also included a LCT test using a sample of 50% Vein and 50% Stratabound mineralization. Compared to Vein composite sample (LR10916-001/Test 59/Master Composite 1), it appeared that co-processing of the two types of mineralization would not cause a significant impact on metallurgical performance.

- **Stratabound mineralization**

Several test efforts (including 2005) had LCT tests to investigate the effect of recycling middling streams on the metallurgical performance of Stratabound samples. Because of the low degree of oxidation of Stratabound material, the developed flowsheet does not include oxide mineral flotation. Early test work used finer primary grinding than in 2005. Table 13.24 summarizes the results.

Table 13.24 LCT test results – Stratabound mineralization

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

Generally, the Stratabound material gave better metallurgical performance than Vein material. The lead concentrate recovered about 73 to 89% of the lead (grade about 53 to 60%) and 37 to 63% of the silver. The best result (2005) was LR10916-001/Test 60.

The zinc concentrate recovered 77 to 92% of the zinc. Flotation concentrate grades were high at 53 to 61%. About 17 to 41% of the silver was recovered into the concentrates.

As discussed above, it appeared that co-processing of the Stratabound and Vein mineralization would not have a significant impact on metallurgical performance.

13.6.4 Pilot plant tests

CSMRI (1982) did flotation pilot plant testing to simulate the proposed mill process. Test objectives were to provide operating and metallurgical data from the continuous operation of three flow schemes and produce lead and zinc concentrates representative of the full-scale mill operation. The flowsheets tested were similar, excluding Pilot Plant Runs 102 and 103, which employed a regrinding step in the lead cleaner circuit:

- Run 101 used lead and zinc differential flotation – see flowsheet in Figure 13.5.
- Run 102 had 1st lead cleaner tailings and rougher scavengers concentrate regrinding.
- Run 103 reground all rougher and scavenger concentrates and lead cleaner flotation with three stages of flotation instead of the two used in the above test runs.

The pilot plant feed rate was approximately 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 200 mesh. Soda ash/ sodium cyanide was used in the lead circuit and lime/copper sulphate in the zinc circuit.

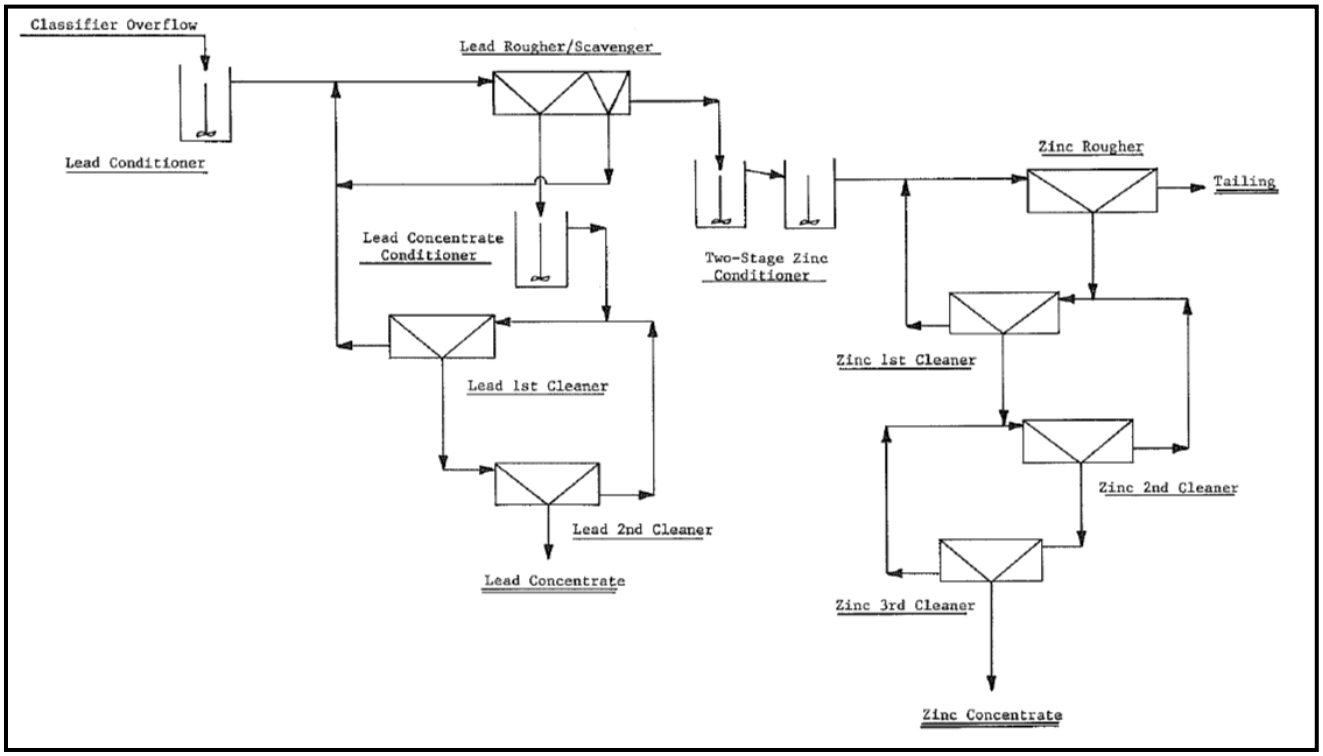
Selected results of the Pilot Plant Runs are shown in Table 13.25.

Pilot plant runs produced lead concentrates containing 64 to 69% lead and 8 to 11% zinc, and zinc concentrates with 57 to 61% zinc and 3 to 6% lead. The results showed that the lead content of the final lead concentrate was only slightly increased through a lead regrind step. A lead third cleaner step employed in Pilot Plant Run 103 appeared to improve the final lead concentrate grade.

Table 13.25 Pilot plant test results – CSMRI (1982)

Pilot Plant Run	Process Stream	Process Rate	Weight	Grade				Distribution			
		(lb/h)	(%)	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

Figure 13.5 Pilot plant flowsheet - base case (run 101) - CSMRI (1982)



13.7 Other test work

13.7.1 Dewatering testing

Settling tests - flotation tailings

Outotec (2011) did a thickening test on two lead/zinc flotation tailings samples, one wet and one dry – see Table 13.26. 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/m^2/h respectively.

Table 13.26 Settling test results obtained on flotation tailings – Outotec (2011)

Flocculant		Solid Loading Rate	Rise Rate	Underflow	Overflow	Vane Yield Stress
Type	Dosage (g/t)	(t/m^2/h)	(m/h)	(w/w%)	(ppm)	(Pa)
MF10	30	0.54	2.4	71	36	67
		0.84	3.8	69	56	47

SGS (2006) dewatering tests on tailings and concentrates from Phase III testing included assessment of flotation tailings settling characteristics, and settling and filtration characteristics of the four concentrates. Table 13.27 shows settling test results.

Tailings from both Upper Zone and Lower Zone 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 had to be reduced to about 7.0 and a fairly large amount of flocculant 351 was added.

Table 13.27 Settling test results obtained on flotation tailings – SGS Lakefield (2006)

Test No.	pH	Flocculant		Initial Settling Rate	Thickener U/F Unit Area	Thickener Hydraulic Unit Area *
		Type	g/t	$m^3/m^2/d$	$m^2/mt/d$	$m^2/mt/d$
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

* Overflow capacity

CSMRI (1982) settling tests on Pilot Plant Run 103 flotation tailings showed a clear supernatant produced from all samples. Flotation tailings settled rapidly and settling rate improved with lime or flocculant (Superfloc 1202, non-ionic flocculant) added – see Table 13.28. In the test program, percolation tests were also conducted on the tailings.

Table 13.28 Settling test results obtained on flotation tailings – CSMRI (1982)

Flocculant		Critical Time*	Unit Rate
Type	Dosage (lb/ton)	(min)	$(ft^2/st/d)$
-	-	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

* Time to reach a solid density of 55% w/w

Settling tests - flotation concentrates

SGS (2006) did settling tests on oxide and sulphide concentrates from Phase I & II tests. 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 13.29 shows results. Good settling rates were achieved for all concentrates. Adding small quantities of flocculant, Magnafloc 351, improved settling rate and supernatant clarity.

Table 13.29 Settling test results on flotation concentrates – SGS Lakefield (2006)

Test No.	pH	Flocculant		Initial Settling Rate	Thickener U/F Unit Area	Thickener Hydraulic Unit Area *
		Type	g/t	$m^3/m^2/d$	$m^2/mt/d$	$m^2/mt/d$
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

* Overflow capacity

The CSMRI 1982 work included settlings tests by the modified Kynch method on flotation concentrates produced from Pilot Plant Run 103. Table 13.30 shows results. 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 0.84 ft^2/st/d or 0.086 m^2/mt/d for the lead concentrate and 1.25 ft^2/st/d or 0.128 m^2/mt/d for the zinc concentrate, indicating that both the concentrates settled rapidly.

Table 13.30 Settling test results obtained on flotation concentrates – CSMRI (1982)

Test No.	Flocculant		Critical Time*	Unit Rate
	Type	Dosage (lb/st)	(min)	(ft^2/st/d)
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

Filtration tests - flotation concentrates

SGS (2006) filtration tests on Phase I & II oxide and sulphide concentrates used a vacuum pour-on method. Results are shown in Table 13.31. Good cake production rates were achieved on each concentrate. Filter cake moisture ranged from 9.5% to 10.9%.

Table 13.31 Filtration test results - SGS Lakefield (2006)

Sample	Slurry			Total Filtration Cycle Time	Filter Cake		
	Percentage	pH	Mag 351	(min)	Thickness	Moisture	Filtration Rate
	Solids (%)		(g/t)		(mm)	(%)	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

13.7.2 Oxide mineral leaching at ambient temperature

1983 CSMRI sulphide flotation tailings leach tests with galena removed used ammonium hydroxide, sodium hydroxide, and sulphuric acid as lixiviants 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. 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 and due to the method complexity leaching was not investigated further.

13.7.3 Gravity concentration

CSMRI (1983) did gravity concentration tests on the head sample and sulphide flotation tailings. Using shaking tables, 75.8% of lead and 63.8% of zinc was recovered to a gravity concentrate with 24.4% lead and 20.1% zinc 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.

As per O'Kane, hydrocyclone concentration to recover non-sulphide values from sulphide zinc flotation tailings was attempted. Available data showed this route as not successful.

13.7.4 Pyrite flotation

Due to the high pyrite content in Stratabound mineralization, pyrite flotation following zinc flotation was performed on several samples. The zinc flotation tailings were obtained by adding lime and cyanide in the lead flotation circuit and copper sulphate in the zinc flotation circuit. The sulphide zinc flotation tailings were conditioned with sulphur dioxide and floated with butyl xanthate collector. Pyrite rougher concentrates from three different Stratabound samples graded 37.5 to 40.4 % iron.

A similar Vein zone test produced a pyrite concentrate of 9.3% iron and 26.1% zinc.

13.7.5 Mercury removal

O'Kane summarized mercury removal test work using leach methods. Test reports were not available for this review. The following is from the O'Kane report:

> *Polysulphide leach tests were conducted on lead and zinc concentrates in an effort to reduce mercury level. Using a 4% sodium sulphide/2% sodium hydroxide (NaOH) solution reduced mercury levels in the concentrate by 16%.*
>
> *Sodium cyanide leach tests were also conducted on lead and zinc concentrates. At a concentration of 5 g/l, mercury levels were reduced by 23% in the lead concentrate. Residue from the zinc concentrate showed no reduction in mercury levels.*
>
> *Leaching with hypochlorite and roasting were two methods recommended by CSMRI, following the poor results of the above leach tests, but no documentation of this test work, if any, was found.*

Results from 1997 PRA zinc concentrate tests showed that mercury levels can be reduced significantly from 1,988 to 155 ppm for Vein zone concentrate, and from 477 to 157 ppm for Stratabound concentrate, after the concentrates were heated to 750°C.

13.8 Flotation concentrate quality

13.8.1 Lead concentrate

In general, the lead grade of sulphide lead concentrates from both Vein and Stratabound material is high, especially for Vein concentrates. Vein concentrates also contained, on average, about 1,000 g/t silver. Oxide lead concentrates from Vein material contain about 38 to 57% lead, much lower than for sulphide lead concentrates. Some Vein zone concentrate impurity levels are high, especially for arsenic and antimony, which will likely be penalized by smelters. Mineralogical studies showed that arsenic and antimony in Vein material are intimately associated with the copper minerals, which are expected to be concentrated into the lead concentrate.

The grades of the lead concentrate from the Stratabound zone were lower than those from Vein zone concentrates. However, the impurity concentrations in the concentrates from the Stratabound zone are much lower than the concentrates from the Vein zone. The main impurity contents in the lead concentrates are shown in Table 13.32 for the Vein concentrates and Table 13.33 for the Stratabound concentrates.

Table 13.32 Impurity contents – lead concentrates – Vein mineralization

Concentrate		**Sulphide Lead Concentrate**								**Oxide Lead Concentrate**		
Test Program	**LR10916-001**	**LR11098-001**		**LR11098-002**	**KM440**	**CSMR**	**LR2252**		**LR10916-001**	**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

Table 13.33 Impurity contents – lead concentrates – Stratabound & blended mineralization

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% Vein and 50% Stratabound)

13.8.2 Zinc concentrate

Zinc sulphide concentrates from both Vein and Stratabound material were low in iron, copper and some impurities, such as arsenic and antimony. However, mercury content in Vein concentrates was high, averaging over 2,000 g/t, which will likely be penalized by smelters and may limit marketability. Mineralogical investigation showed the mercury to be closely associated with zinc minerals. Concentrate from Stratabound samples showed lower mercury content. As estimated by Turnbull Engineers Inc., a Stratabound concentrate mercury content around 900 ppm is anticipated. Cadmium in Vein mineralization was concentrated into the zinc sulphide concentrates. Average cadmium concentration in the range of 0.2 to 0.5% is anticipated.

The oxide zinc concentrates produced from the Vein mineral samples show low zinc contents of approximately 30% zinc. The mercury contents of the oxide concentrate are much lower than those for the sulphide concentrates.

The main impurity contents in the zinc concentrates are shown in Table 13.34 for the Vein concentrates and Table 13.35 for the Stratabound concentrates.

Table 13.34 Impurity contents – zinc concentrates – Vein mineralization

Concentrate		**Sulphide Zinc Concentrate**							**Oxide Zinc Concentrate**			
Test Program	**LR10916-001**	**LR11098-001**		**LR11098 - 002**	**KM440**	**CSMR**	**LR2252**		**LR10916 -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 13.35 Impurity contents – zinc concentrates – Stratabound & blended mineralization

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
Aluminum ($A_{12}0_3$) %	-	<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)

13.9 Metallurgical performance projection

From the test results, preliminary metallurgical performance was projected, as shown in Table 13.36. As the proposed process flowsheet and reagent system are very complex, further test work should be conducted to update the projections.

Table 13.36 Preliminary metallurgical performance projection

Process Circuit	Value	Note
Vein Mineralization		
DMS Pre-concentration		
- Mass Recovery, %	= 0.8484 x (Head Grade, % - Lead +Zinc) + 55.013 = 98	< 50% Pb+Zn > 50% Pb+Zn
- Lead Recovery, % - Sulphide	= 1.293 x ln (Head Grade, % - Sulphide Lead) + 95.794 = 0.0628 x (Head Grade, % - Sulphide Lead) + 97.97 = 99.2	0 - 7.88% Pb-Sulphide 7.88 - 16% Pb-Sulphide > 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 = 99.7	< 50% Zn-Sulphide > 50% Zn-Sulphide
- Oxide	= 85 = 4.211 x ln (Head Grade, % - Oxide Zinc) + 87.579 = 97.5	< 0.5% Zn-Oxide 0.5 – 10.5 Zn-Oxide > 10.5% Zn-Oxide
- Silver Recovery, %	= 0.0436 x (Mass Recovery, %) + 94.508 = 90	> 50% Mass Recovery < 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) $^{0.1253}$) = (Head Grade, % - Sulphide Lead)/(Head Grade, % - Total Lead) x 98	0 - 17.4% Pb-Sulphide > 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 = (Head Grade, % - Sulphide Zinc)/(Head Grade, % - Total Zinc) x (0.2217 x (Head Grade, % - Sulphide Zinc) + 90.135) = (Head Grade, % - Sulphide Zinc)/(Head Grade, % - Total Zinc) x 98	< 1% Zn-Sulphide 1 - 30% Zn-Sulphide > 30% Zn-Sulphide
- Silver	= 0 = 3.0962 x (Lead Recovery to Sulphide Zinc Concentrate, %) - 3.7592 = 20	< 1.5% Lead Recovery to Sulphide Zinc Concentrate 1.5 – 7.5% Lead Recovery to Sulphide Zinc Concentrate > 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 = 1.0	> 2% Pb - Oxide < 2% Pb - Oxide
Stratabound Mineralization		
DMS Pre-concentration		
- Mass Recovery, %	= 88	
- Lead Recovery, % -	= 98	

Process Circuit	Value	Note
Sulphide		
- 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 = 10	> 55% Lead Recovery to Sulphide Lead Concentrate < 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%

13.10 Recommendations

The test results show that both the Vein and Stratabound mineralization responded reasonably well to the DMS pre-concentration followed by sulphide lead and zinc flotation and oxide lead and zinc flotation. Due to high oxidation rates, the Vein mineralization showed inferior metallurgical performance in comparison with the Stratabound mineralization. The preliminary test results showed that it is possible to generate a separate copper concentrate from the copper-lead bulk flotation concentrate when the copper grade is high. However, the grade of the copper concentrate produced is expected be low and to contain high percentages of arsenic and antimony because the separation will also concentrate these elements into the copper concentrate. According to the test results, Tetra Tech (author of this section) would suggest further test work to optimize the process flowsheet, especially the reagent regimes. The further tests should also include variability tests to investigate metallurgical responses of various mineralizations from different lithological zones and spatial locations, including Stockwork mineralization.

In general, the main impurities are closely associated with their bearing minerals, which are also the target minerals. For example, arsenic and antimony are closely associated with copper minerals, while mercury and cadmium are associated with zinc minerals. Separation using flotation is unlikely to reject these impurities from concentrates. However, the optimum process conditions would reduce mingling of target minerals and reduce impurity contents in concentrates. Hydrometallurgical and pyrometallurgical processes may need to be studied to develop process flowsheets that can more effectively recover the target metals from the mineralization.

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 metres) and 22 channel samples from the 930 Level (50 samples with an aggregate length of 48 metres) that have been acquired since the previous estimate.

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. (of AMC as of the effective date of this report, now with Global Mineral Resource Services) 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 samples are indicated on Figure 14.1.

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

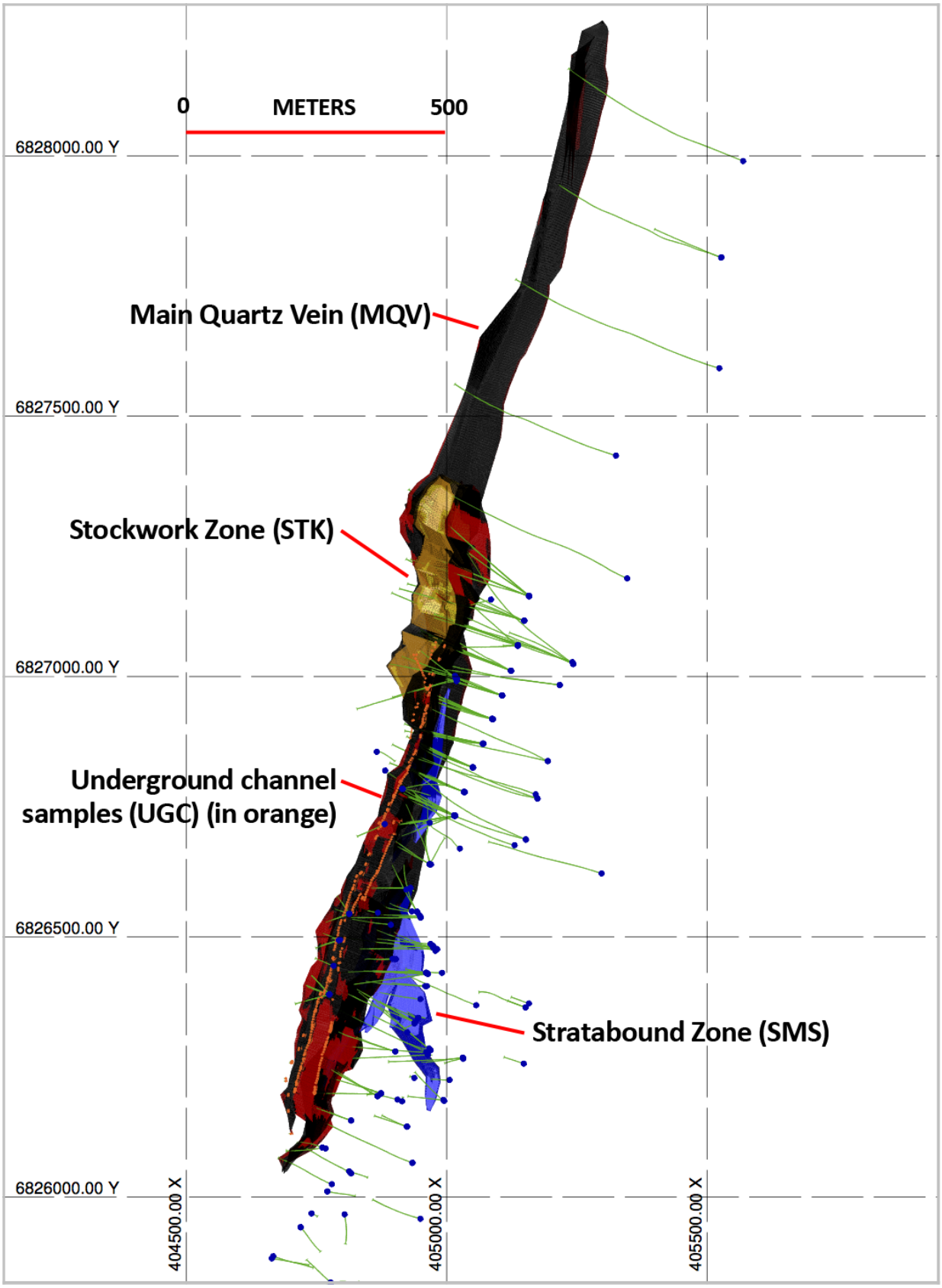
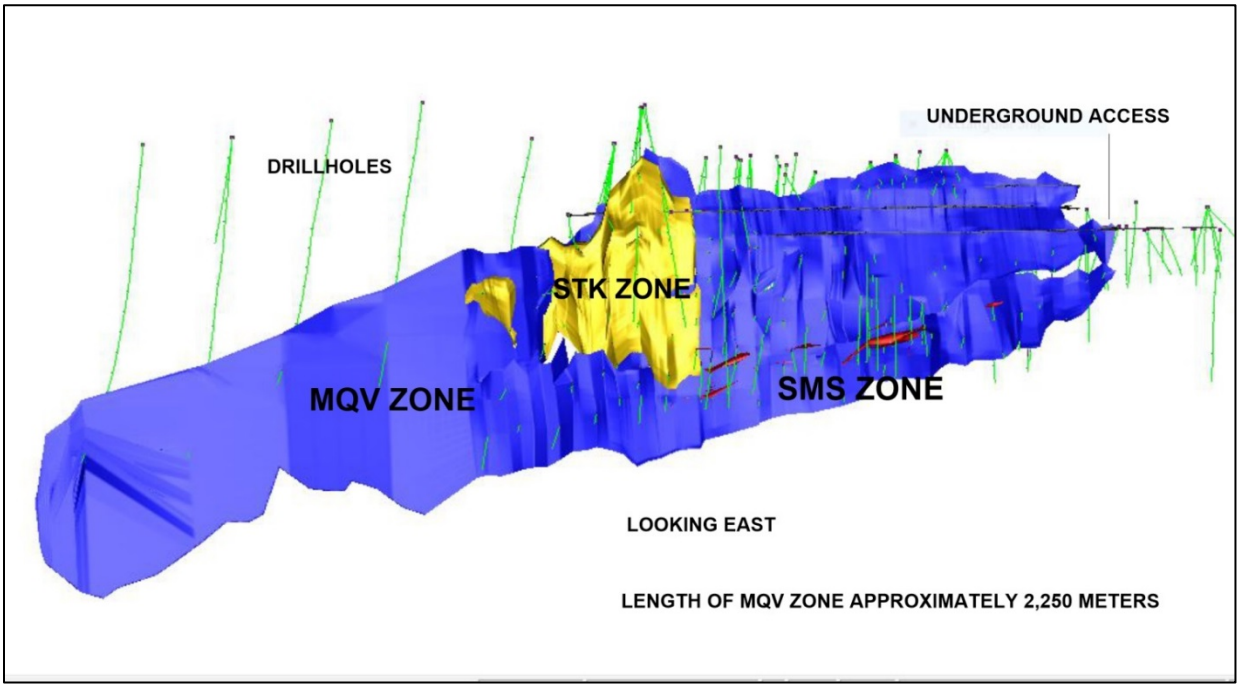


Figure 14.2 Illustrative longitudinal view showing Mineral domains, drillholes and underground channel samples



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 14.2.

Table 14.3 shows the correlation coefficients of silver, lead and zinc relative to a number of other elements for each of the three Mineral domains MQV, STK and SMS, as well as the underground channel samples. 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

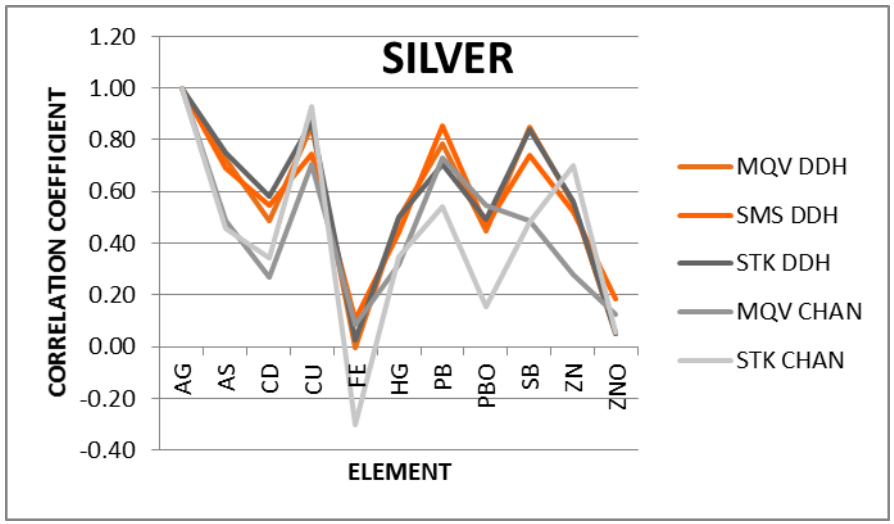
STK DDH COMPOSITES	Ag (g/t)	Pb (%)	Zn (%)
Mean	50	3.31	3.86
Standard Error	6	0.41	0.34
Median	19	1.08	2.37
Mode	4	0.27	
Standard Deviation	85	5.66	4.58
Kurtosis	20	11.96	7.77
Skewness	4	3.19	2.47
Range	618	35.91	29.19
Minimum	1	0.01	0.03
Maximum	619	35.92	29.22
Count	183	187	186

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

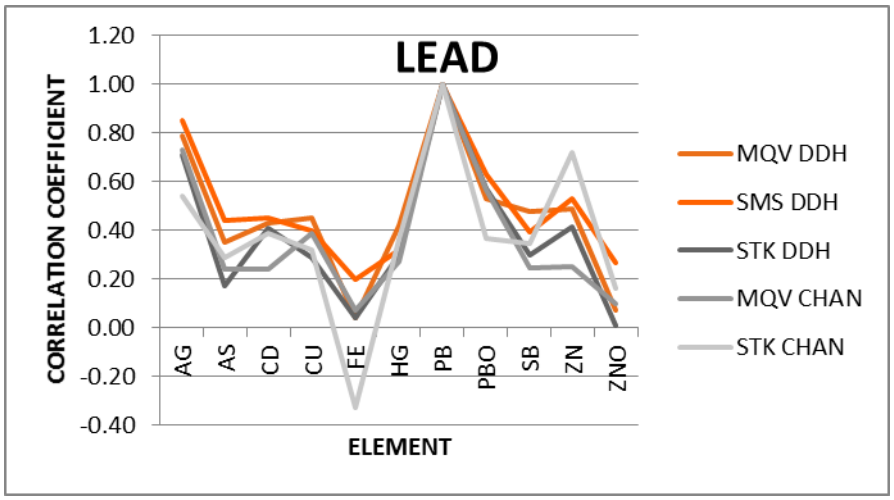
STK CHANNEL COMPOSITES	Ag (g/t)	Pb (%)	Zn (%)
Mean	126	6.16	13.30
Standard Error	14	0.55	1.00
Median	112	5.40	12.40
Mode	123	7.41	21.50
Standard Deviation	104	3.99	7.31
Kurtosis	13	-0.05	-0.64
Skewness	3	0.61	0.16
Range	654	16.04	30.24
Minimum	3	0.14	0.24
Maximum	658	16.18	30.48
Count	52	53	53

Table 14.3 Correlation coefficients for silver, lead, and zinc

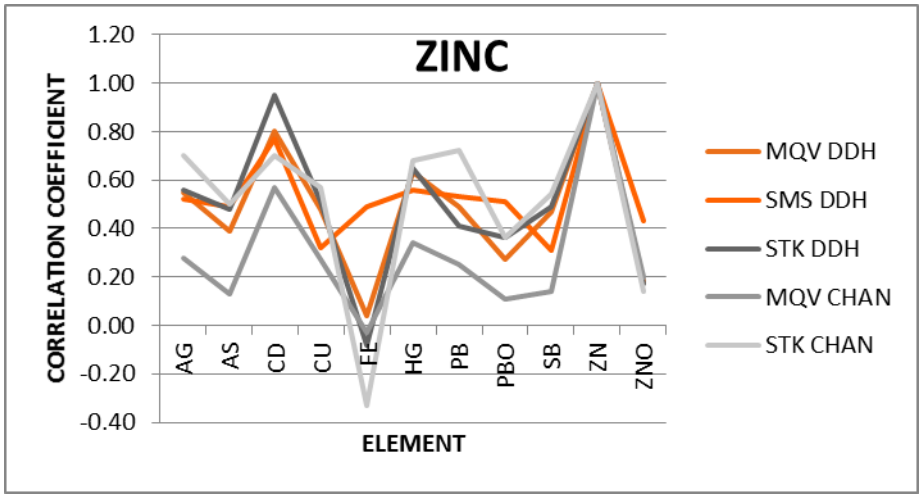
Ag	MQV DDH	SMS DDH	STK DDH	MQV CHAN	STK CHAN
Ag	1.00	1.00	1.00	1.00	1.00
As	0.73	0.69	0.75	0.49	0.46
Cd	0.49	0.55	0.58	0.27	0.34
Cu	0.85	0.74	0.87	0.70	0.93
Fe	0.00	0.11	0.03	0.08	-0.30
Hg	0.50	0.44	0.50	0.32	0.35
Pb	0.79	0.85	0.71	0.73	0.54
PbO	0.45	0.47	0.49	0.55	0.16
Sb	0.85	0.74	0.84	0.49	0.48
Zn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

14.2.5 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.

14.3 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 14.4 contains the variography parameters and Table 14.5 the search ellipse parameters that were used in the estimate. The search ellipse dimensions were standardized to provide similar coverage for all elements.

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

14.4 Mineral Resource block model

Block model parameters are summarized in Table 14.6. 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

14.5 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.

14.6 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 14.7.

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

14.7 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; Figures 14.4, and 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

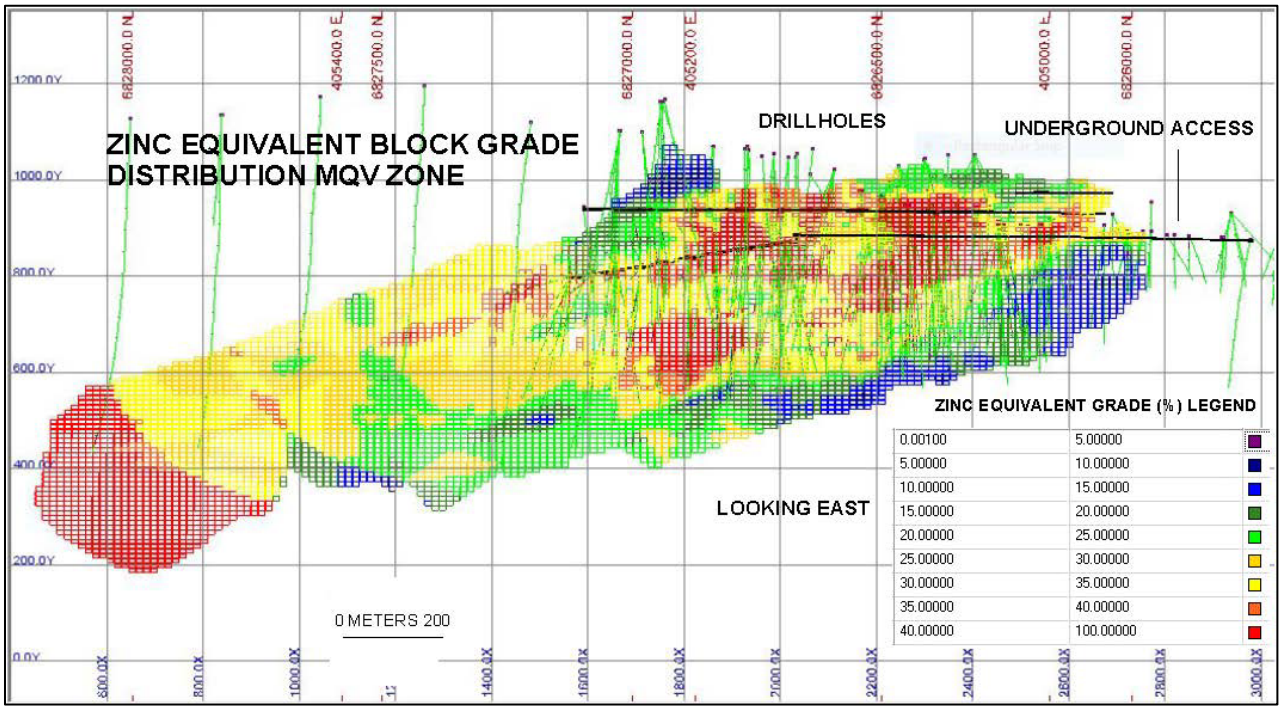


Figure 14.4 Zinc-Equivalent block grade distribution SMS zone

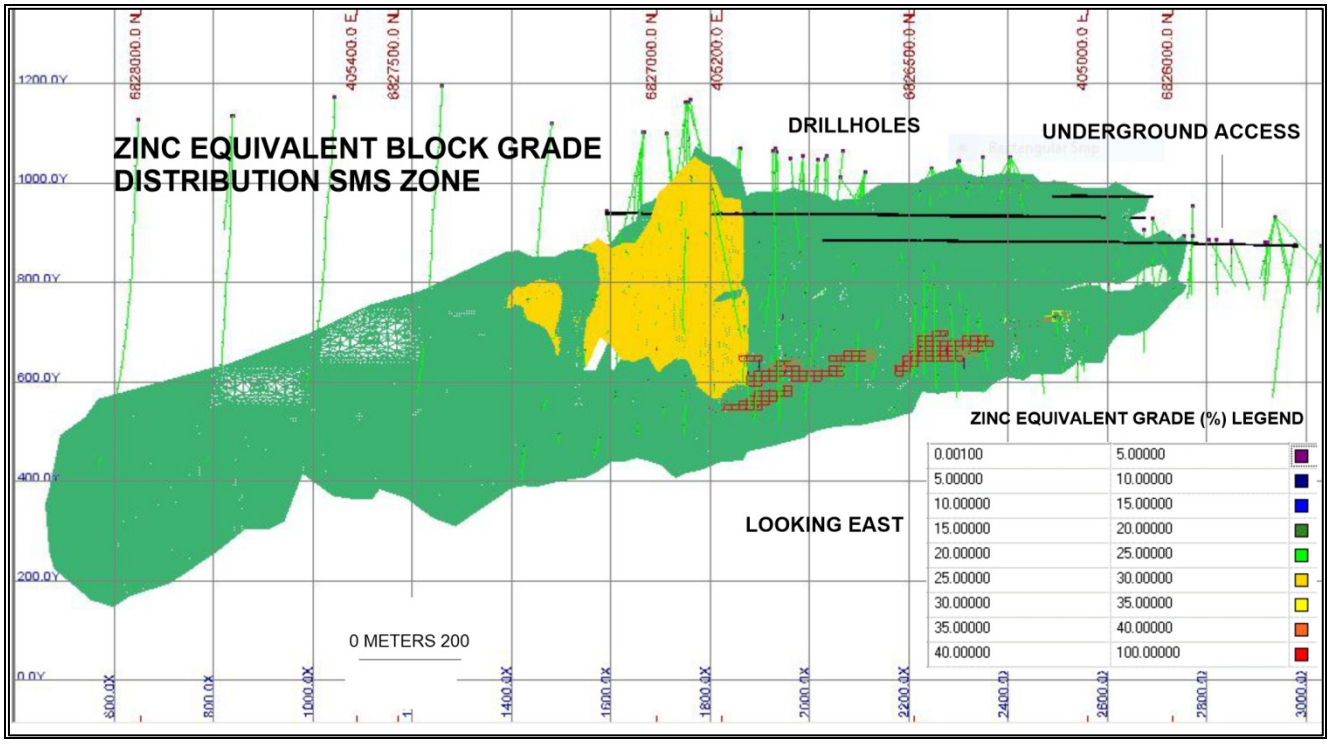


Figure 14.5 Zinc-Equivalent block grade distribution STK zone

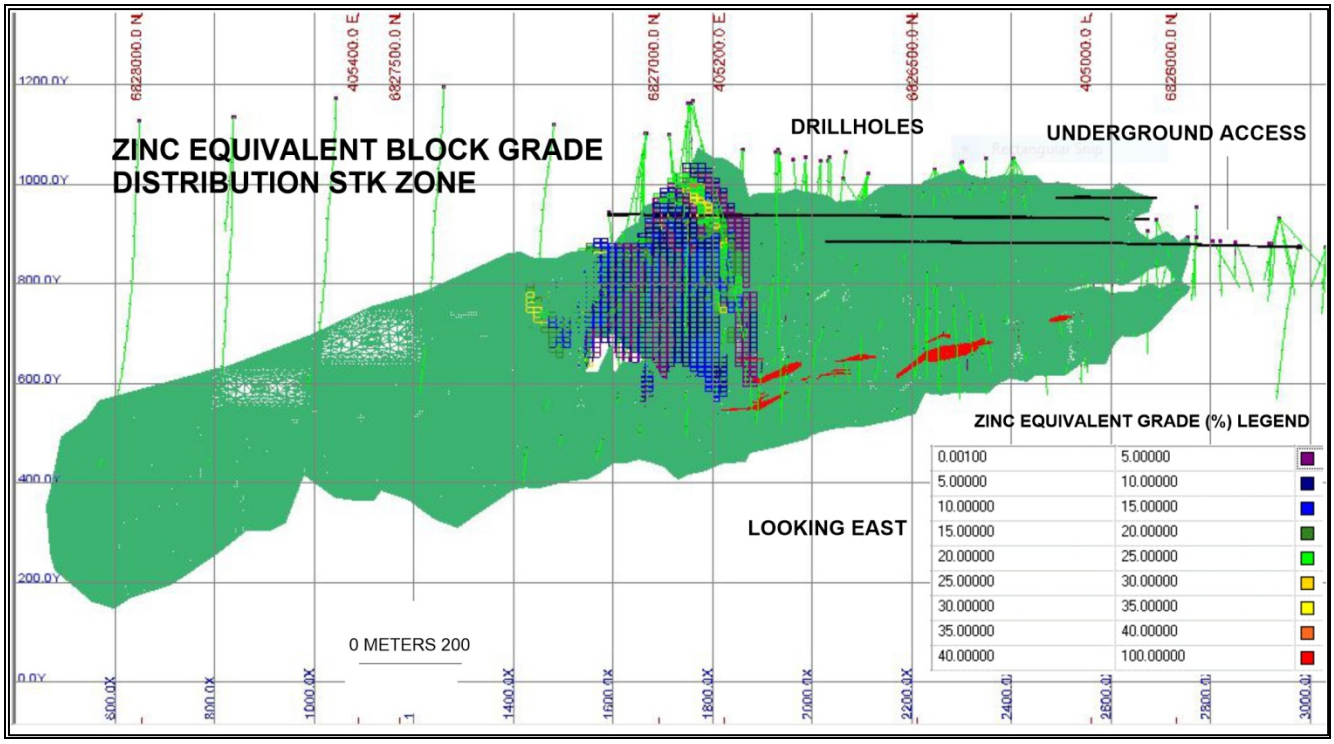
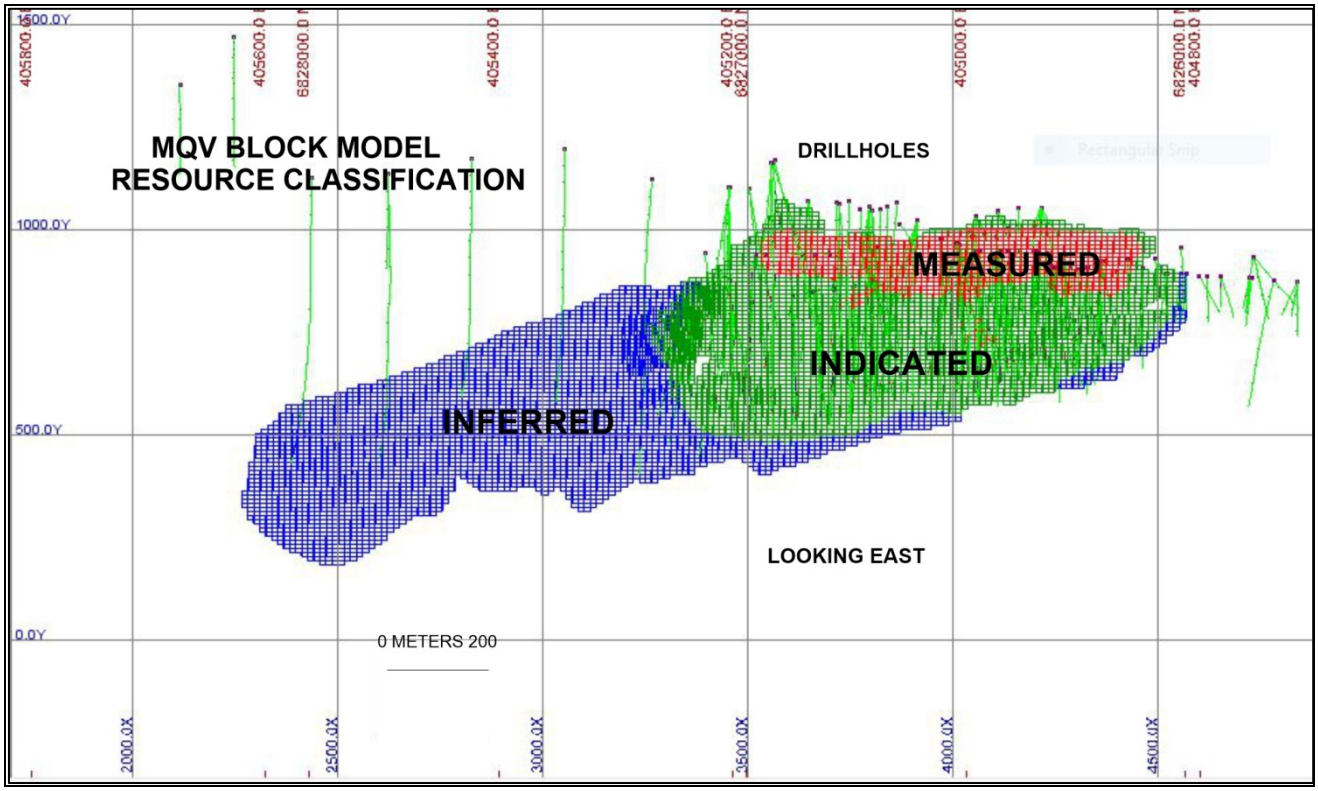


Figure 14.6 Mineral Resource classification MQV domain



14.8 Mineral Resource tabulation

Table 14.8 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 CAD/USD.
Numbers may not compute exactly due to rounding.

Tables 14.9, 14.10 and 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.

14.9 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 & block model

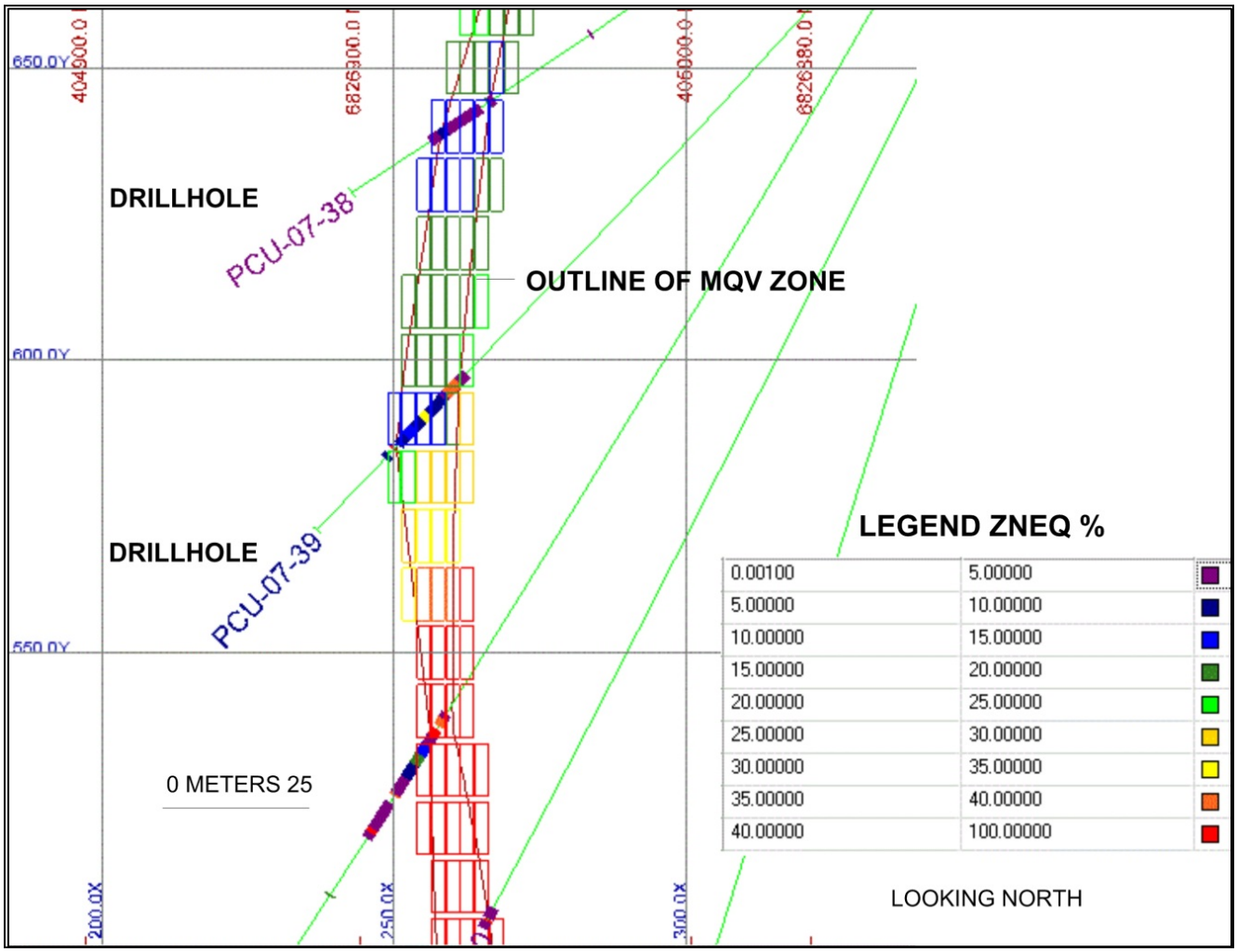


Figure 14.7 shows intercepts within the MQV Zone; mineralized intercepts outside the MQV Zone are located within the STK Zone which is not shown in this figure.

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

14.10 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

As defined by the Canadian Institute of Mining, Metallurgy and Petroleum within the CIM Definition Standards on Mineral Resources and Mineral Reserves as adopted by CIM Council on May 10 2014, the definition of a Mineral Reserve is 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

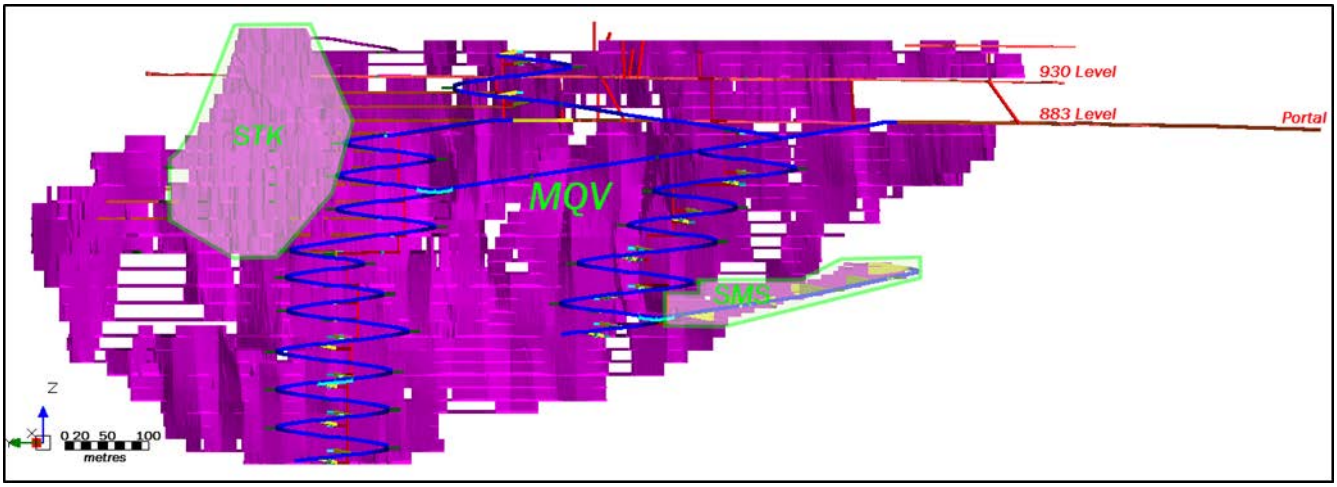
All design and scheduling has been completed to a level of detail appropriate for a prefeasibility study using a Mineral Resource model generated by AMC on 10 September 2015. All mining at Prairie Creek will be by underground methods.

15.2.1 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



15.2.2 Cut-off grade

Zinc equivalent (ZnEq) cut-off grades (COG) were calculated using the parameters shown in Table 15.1. Primary ZnEq cut-offs were 12% for longhole open stoping (LHOS) and 11% for drift and fill (DAF). 9.7% ZnEq was applied as an incremental stoping cut-off and 7.1% was applied as a cut-off for development ore. These design

cut-offs respectively equate to C$331/t, C$303/t, C$267/t and C$196/t 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 15.1 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	17.00
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 %)*

15.2.3 Mining shapes

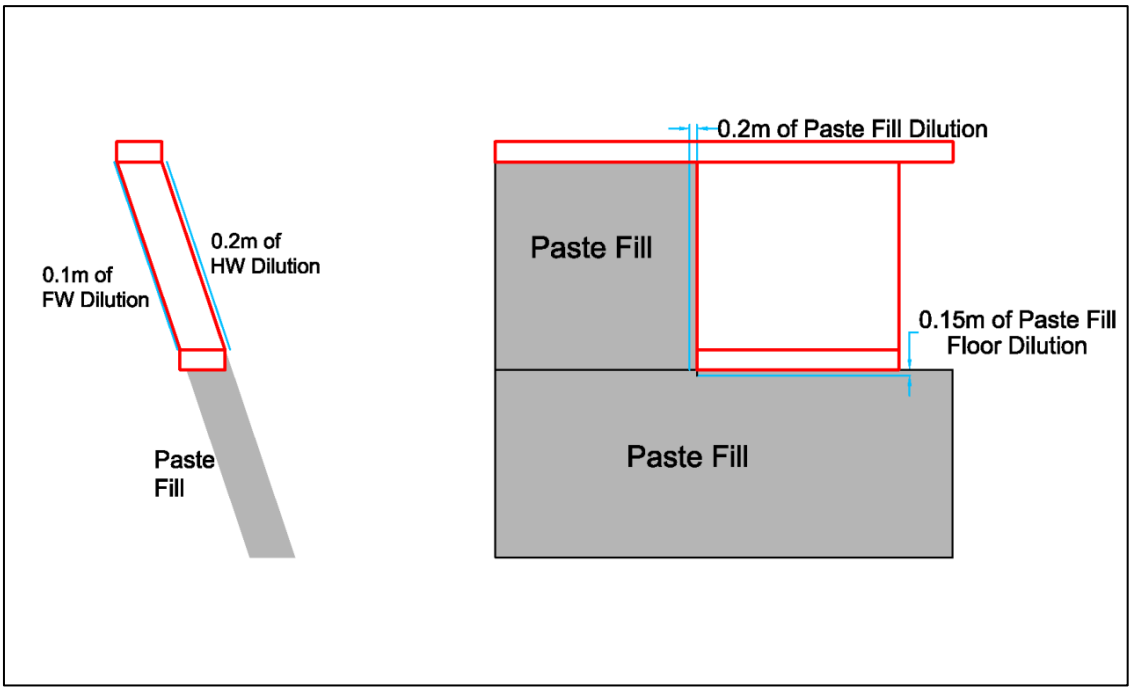
For longhole stope generation in the MQV, AMC used the software Mineable Shape Optimizer (MSO) to produce stope shapes based on the 12% COG and other key design criteria. Shapes were manually combined into full-length stopes of about 30 m along strike. In the STK, MSO was also used to generate LHOS shapes but these were subsequently divided manually into stopes of 15 m maximum width. The SMS DAF stopes were generated manually based on 11% COG grade-shells on a 5 m cut interval.

On the outliers 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.

15.2.4 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 0.15 m and average endwall dilution from an adjacent filled stope was estimated to be 0.2 m. The resultant overall average unplanned dilution estimate for LHOS is 14%. Figure 15.2 represents a single LHOS panel with the unplanned dilution applied. Mining recovery for all LHOS was estimated to be 95%.

Figure 15.2 Typical LHOS with the unplanned dilution applied



In the DAF shapes, an unplanned dilution factor of 6% and a mining recovery of 95% were estimated. Unplanned dilution in development ore was estimated to be 9% and mining recovery was estimated to be 100%. Further details on dilution estimates are provided in Section16.

15.2.5 Mineral Reserves

The Prairie Creek Mineral Reserves, in total and broken down by zone, are summarized in the Table 15.2.

Table 15.2 March 2016 Mineral Reserves, Prairie Creek Mine

Mineral Zone	Classification	Tonnes (t)	Silver (g/t)	Lead (%)	Zinc (%)	Zinc Equivalent
Main Quartz Vein (MQV)	Proven	1,199,288	186.00	10.08	12.09	30.70
	Probable	3,966,848	152.62	10.52	8.58	26.79
	Total	**5,166,136**	**160.37**	**10.42**	**9.39**	**27.70**
Stockwork (STK)	Proven	174,656	105.01	4.80	11.48	20.83
	Probable	1,297,665	60.72	3.41	6.64	12.87
	Total	**1,472,322**	**65.97**	**3.57**	**7.22**	**13.81**
Stratabound (SMS)	Proven	-	-	-	-	-
	Probable	965,132	46.09	4.38	9.03	16.12
	Total	**965,132**	**46.09**	**4.38**	**9.03**	**16.12**
TOTAL	Proven	1,373,944	175.70	9.41	12.02	29.45
	Probable	6,229,646	116.97	8.09	8.24	22.24
	Total	**7,603,590**	**127.58**	**8.33**	**8.93**	**23.54**

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.
Average unplanned dilution and mining recovery factors of 14% and 95%, respectively, for LHOS, and 6% and 95%, respectively, for DAF are assumed.

15.2.6 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 15.3.

Table 15.3 Conversion of Mineral Resources to Mineral Reserves

		MINERAL RESOURCES			MINERAL RESERVES			CONVERSION FACTOR*		
		ALL	MEA	IND	ALL	PRV	PRB	ALL	MEA/PRV	IND/PRB
TOTAL										
Tonnes	kt	8,704	1,482	7,222	7,604	1,374	6,230	87%	93%	86%
Silver	g/t	136	200	123	128	176	117	82%	81%	82%
Lead	%	8.9	10.8	8.5	8.3	9.4	8.1	82%	81%	82%
Zinc	%	9.5	13.2	8.7	8.9	12.0	8.2	82%	85%	81%
MQV										
Tonnes	kt	5,540	1,313	4,227	5,166	1,199	3,967	93%	91%	94%
Silver	g/t	178	211	168	160	186	153	84%	81%	85%
Lead	%	11.6	11.5	11.6	10.4	10.1	10.5	84%	80%	85%
Zinc	%	10.2	13.2	9.2	9.4	12.1	8.6	86%	83%	87%
STK										
Tonnes	kt	2,122	169	1,953	1,472	175	1,298	69%	103%	66%
Silver	g/t	66	116	61	66	105	61	70%	94%	66%
Lead	%	3.6	5.3	3.5	3.6	4.8	3.4	68%	94%	65%
Zinc	%	7.1	12.6	6.6	7.2	11.5	6.6	70%	94%	66%
SMS										
Tonnes	kt	1,042	-	1,042	965	-	965	93%	-	93%
Silver	g/t	54	-	54	46	-	46	78%	-	78%
Lead	%	5.2	-	5.2	4.4	-	4.4	78%	-	78%
Zinc	%	10.8	-	10.8	9.0	-	9.0	78%	-	78%

*Metal conversion factors reflect total metal content.

15.2.7 Comparison of 2016 and 2012 Mineral Reserves

Table 15.4 compares 2016 Mineral Reserves with those for the previous Mineral Reserve estimation in 2012.

Table 15.4 Comparison of 2016 and 2012 Mineral Reserves

MINERAL RESERVES COMPARISON		MINERAL RESERVE 2016			MINERAL RESERVE 2012			TONNES & CONTAINED METAL DIFFERENCE FACTOR*		
		ALL	PRV	PRB	ALL	PRV	PRB	ALL	PRV/PRV	PRB/PRB
TOTAL										
Tonnes	t	**7,603,590**	1,373,944	6,229,646	**5,221,000**	1,278,000	3,943,000	146%	108%	158%
Silver	g/t	**127.58**	175.7	116.97	**150.87**	172	144.02	123%	110%	128%
Lead	%	**8.33**	9.41	8.09	**9.49**	9.4	9.52	128%	108%	134%
Zinc	%	**8.93**	12.02	8.24	**9.34**	10.8	8.86	139%	120%	147%
MQV										
Tonnes	t	**5,166,136**	1,199,288	3,966,848	**4,418,000**	1,278,000	3,140,000	117%	94%	126%
Silver	g/t	**160.37**	186	152.62	**167.02**	172	165	112%	101%	117%
Lead	%	**10.42**	10.08	10.52	**10.18**	9.4	10.5	120%	101%	127%
Zinc	%	**9.39**	12.09	8.58	**9.31**	10.8	8.7	118%	105%	125%
STK										
Tonnes	t	**1,472,322**	174,656	1,297,665	-	-	-	-	-	-
Silver	g/t	**65.97**	105.01	60.72	-	-	-	-	-	-
Lead	%	**3.57**	4.8	3.41	-	-	-	-	-	-
Zinc	%	**7.22**	11.48	6.64	-	-	-	-	-	-
SMS										
Tonnes	t	**965,132**	-	965,132	**803,000**	-	803,000	120%	-	120%
Silver	g/t	**46.09**	-	46.09	**62**	-	62	89%	-	89%
Lead	%	**4.38**	-	4.38	**5.7**	-	5.7	92%	-	92%
Zinc	%	**9.03**	-	9.03	**9.5**	-	9.5	114%	-	114%

For 2016 Mineral Reserves see footnotes to Table 15.2.
For 2012 Mineral Reserves:
(1) Stated as of 31 May 2012.
(2) Mining cut-off grade of $162/t including mining, processing and transportation.
(3) Metal prices assumed are zinc = $1.10/lb, lead = $1.10/lb and silver = $28/oz.
(4) Average processing recovery factors of 75% for zinc, 88% for lead and 92% for silver.
(5) Average payables of 85% for zinc, 95% for lead and 81% for silver.
(6) Exchange rate: C$1= US$1
* % comparison 2016 vs 2012

16 Mining methods

16.1 Introduction

The Prairie Creek Mine will be a 100% underground mining operation extracting the majority of ore from a steeply-dipping, narrow, base metal-bearing vein-fault structure referred to as the Main Quartz Vein (MQV). Three levels of underground adits (970L, 930L, 883L) were established previously at the Mine. Several shrinkage stopes were partly mined above the 930 and 883 levels, generating a stockpile of approximately 10,000 tonnes of mixed ore and waste that is stored near the mill.

The MQV deposit covers a known strike distance of about 1,000 m and a vertical distance of about 400 m. Below 883L, additional mining levels will be established at generally 60-m intervals with 15- or 20-m sublevels. As mining on the MQV structure progresses to depth, ore mined will be supplemented by the STK and SMS deposits – see Figure 16.1. Initial stoping will be focused in the upper parts of the Mine. Lower levels will be developed to depth through trackless ramp access over the first seven years of operation.

Mining will be by the longhole open stoping (LHOS) method on the MQV vein and in the STK area. The drift and fill (DAF) method is projected for the SMS material. An average mining rate of 1,350 tonnes per day of ore is targeted (including dilution). At steady-state, approximately 485,000 tonnes of ore per year will be mined. Mine life is projected to be 17 years for the total Mineral Reserve of 7.6 million tonnes.

MQV material will be the main focus of mine production and will be extracted throughout the life of the mine. The vein structure is exposed in over 800 m of backs within the existing three levels of underground development.

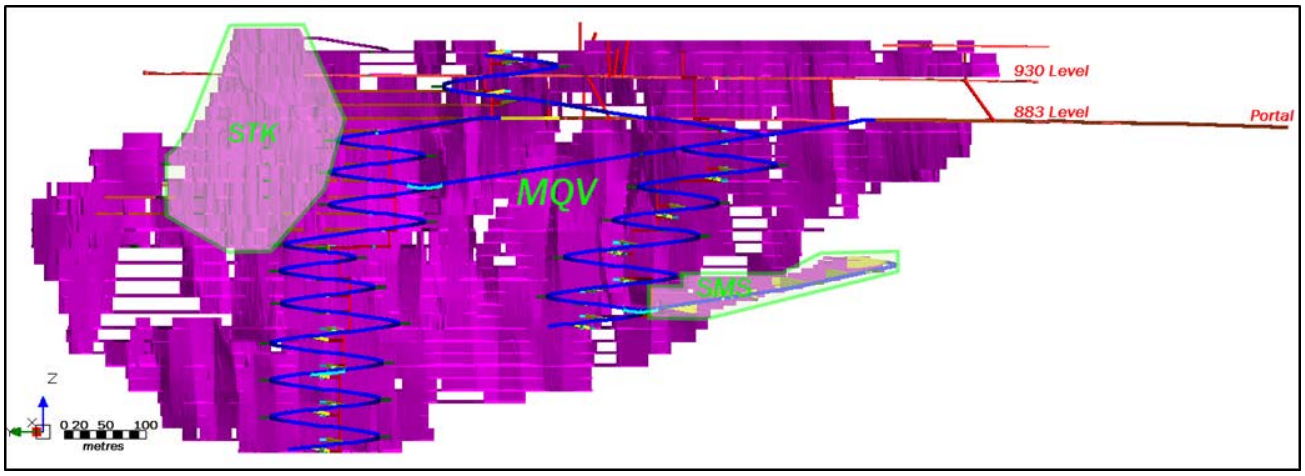
The SMS mineralization occurs approximately 200 m below the 883L portal elevation and requires significant underground development for access. SMS mining is scheduled to start during the eighth year of the mine life.

LHOS will use electric-hydraulic drill jumbos and battery-powered scoops for development and diesel powered scoops and trucks for production, along with electric-hydraulic and/or compressed-air longhole drills. The SMS zone will use electric-hydraulic drill jumbos, diesel powered scoops and trucks, and mechanized bolters.

Main access to the mine will be through the existing 870 portal. It should be noted that previously the 883L has been referred to variously as the 870L and 880L. The 883L, 870L and 880L names thus refer to the same level development. This access will be slashed out to 4.5 m high by 4.5 m wide, after removal of the existing track, to accommodate larger trackless equipment. The existing underground development will help to minimize the amount of development needed for mine operation. Stoping on the existing 883 to 930 levels of the mine, along with the 823 to 883 levels, will account for the majority of mill feed in the initial years of operations. As development progresses, ore will be accessed from the lower part of the mine to maintain the total feed required to keep the mill running at full capacity.

Ground conditions in existing development underground are good. The current workings have stood unsupported for close to 30 years with minimal bolting. Figure 16.1 is a longitudinal view of the underground workings at completion of the current 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

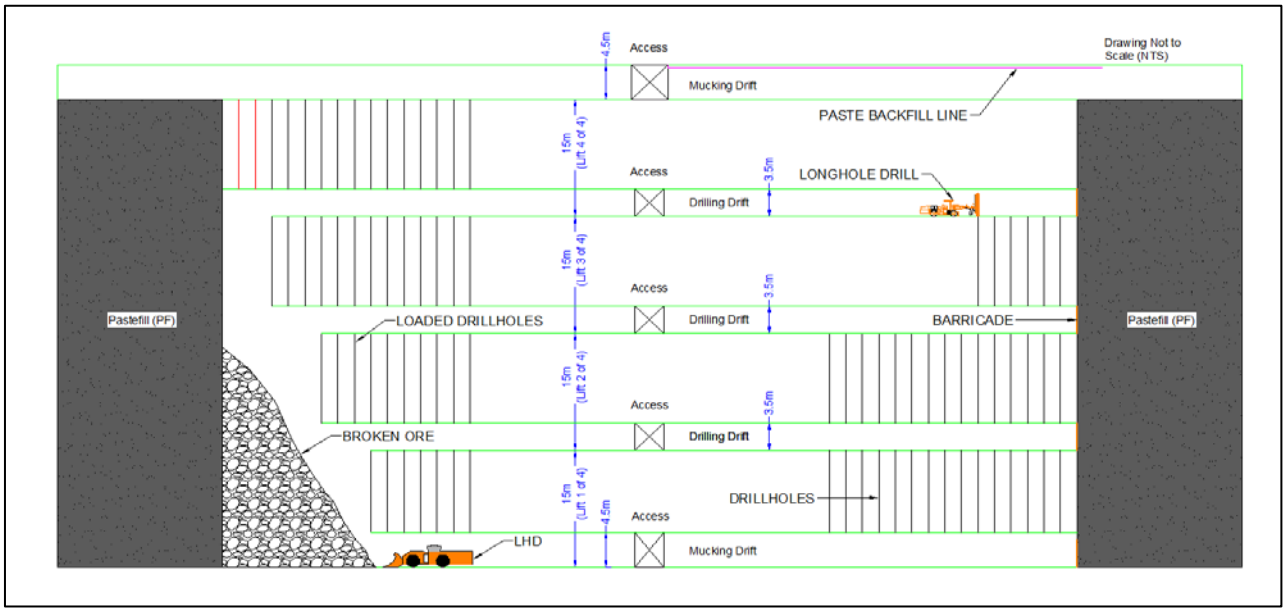


16.2 Mining methods

16.2.1 Longhole Open Stoping (LHOS)

Mining of the MQV will be by LHOS. The mining levels at the 883 m elevation and above have previously been established but need to be slashed to 4.5 m W x 4.5 m H for the equipment proposed. 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 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 sublevel driven every 15 m or 20 m at 3.5 m x 3.5 m dimensions. 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 300 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 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.

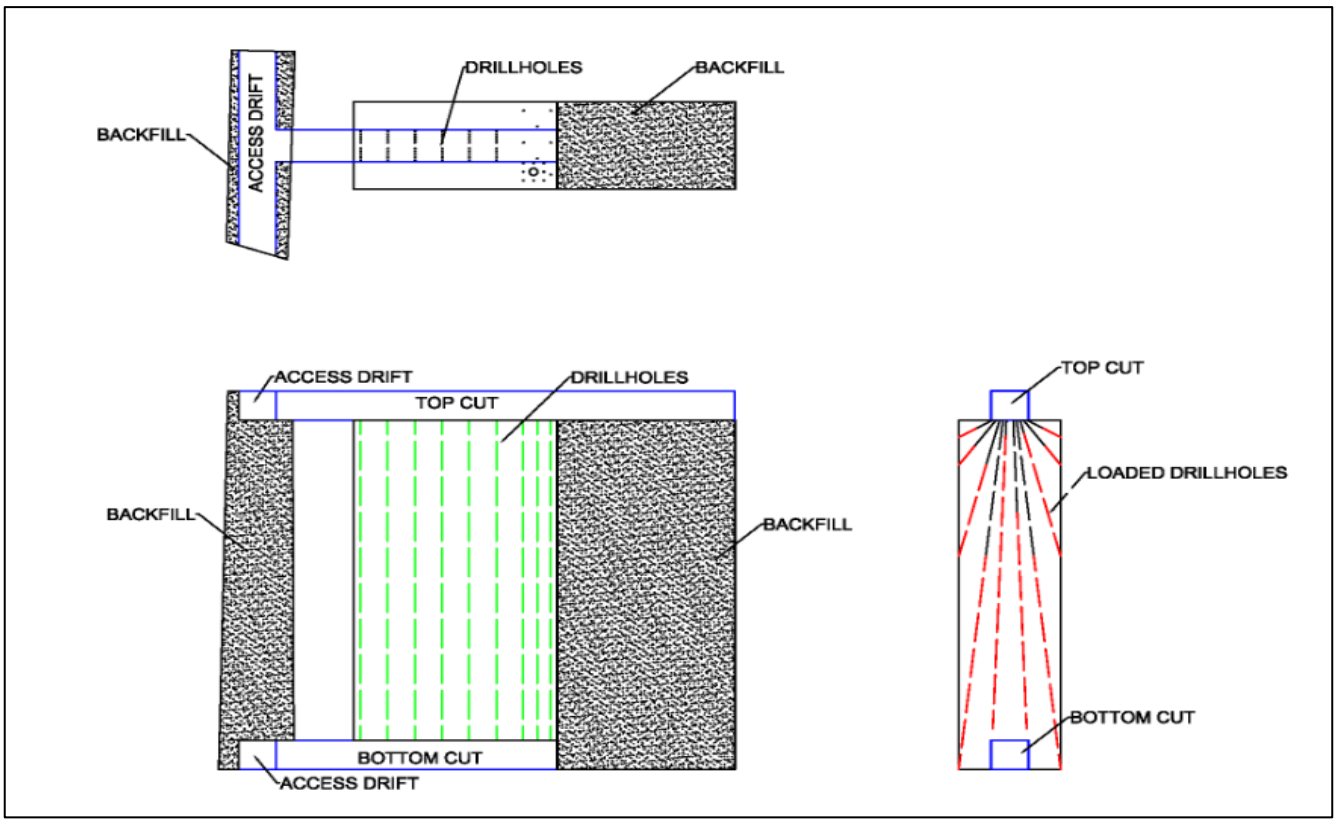
Figure 16.2 Proposed longhole open stoping method (longitudinal section view)



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, access development through fill is projected for the later stages in the mine life. Figure 16.3 illustrates design and extraction for a single STK panel. The feasibility of drifting through cured paste fill has been demonstrated in other mines.

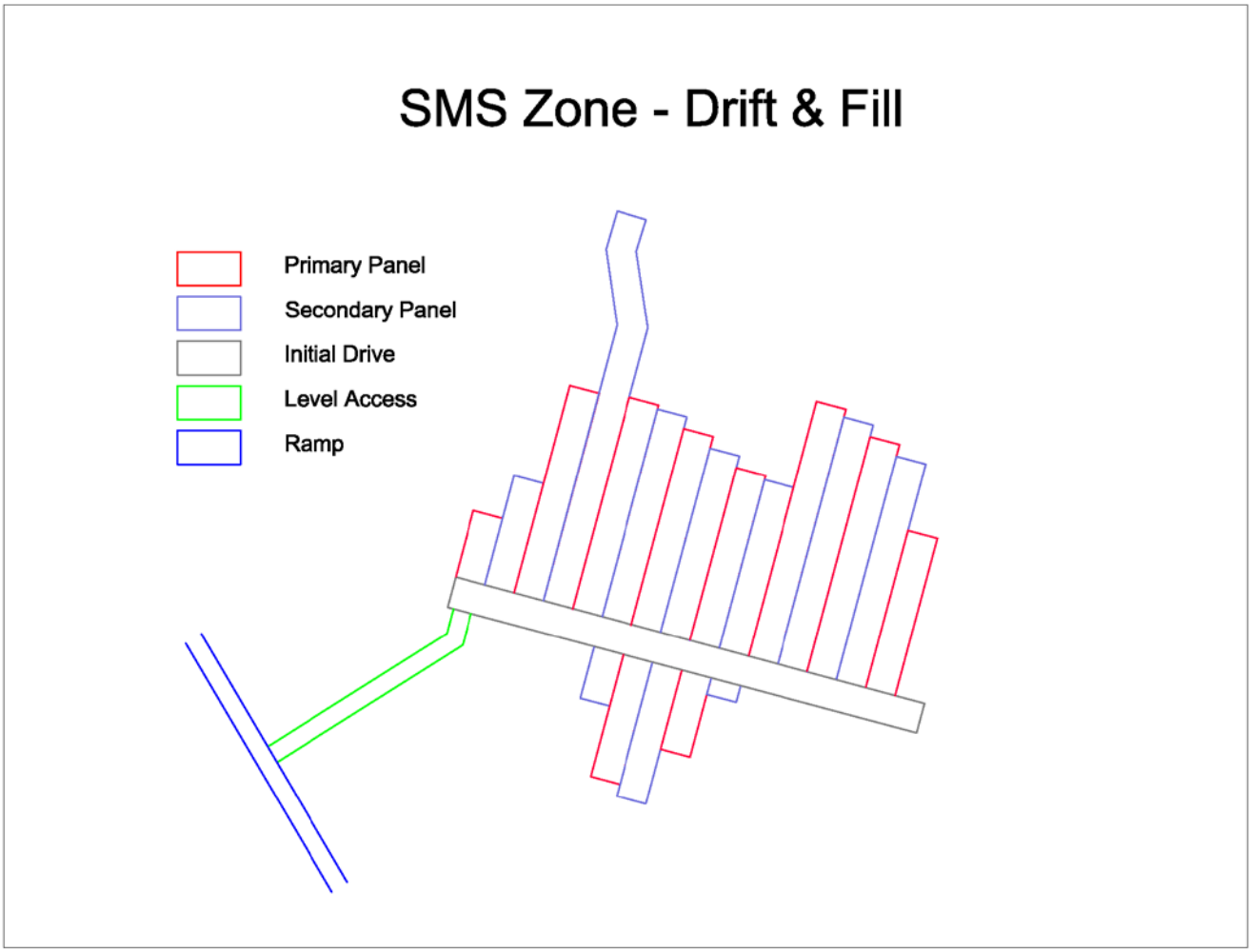
Figure 16.3 STK zone longhole open stoping, plan and section views showing development access through fill



16.2.2 Drift and Fill (DAF)

The SMS zone is more tabular in nature, gently-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. Primaries will thus generally have ore on either side, while secondaries 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 four cuts by taking down backs in the access after each cut is complete.

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

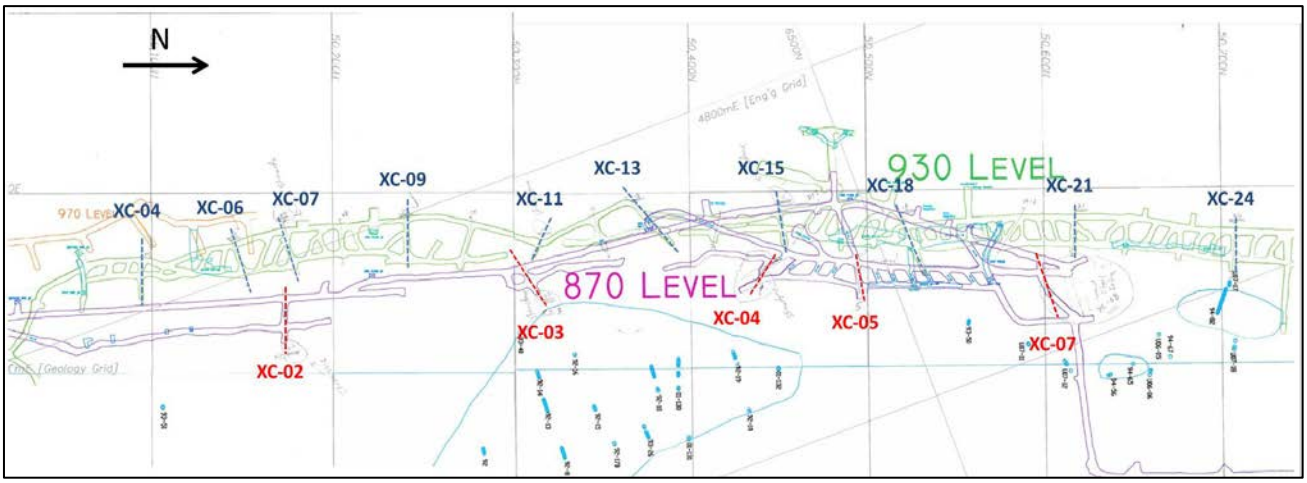


16.3 Geotechnical considerations

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).

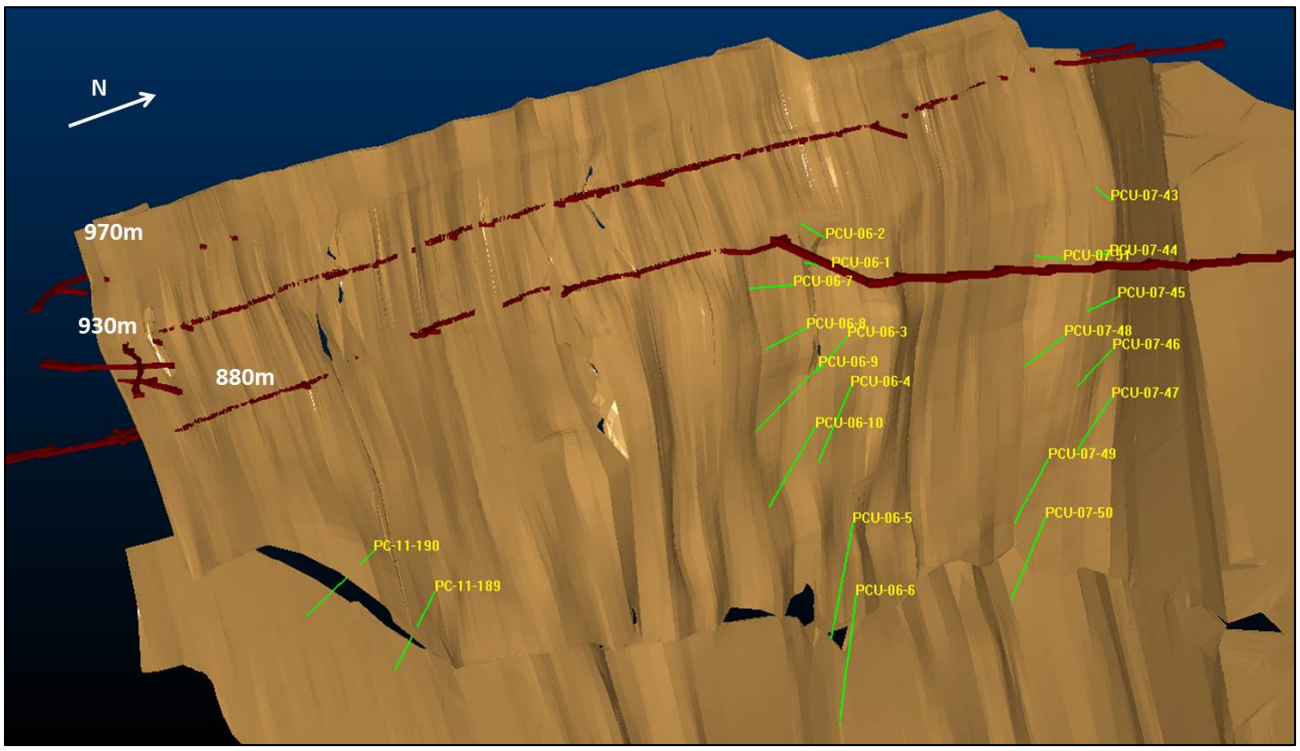
A summary of mapping locations and the levels accessed is presented in Figure 16.5. Geotechnical mapping data and geological information for each mapping section are provided in an AMC geotechnical report (2014).

Figure 16.5 Geotechnical mapping sections



Due to the uneven distribution of geotechnical information (significant data for the upper part of the mine and much less at depth), and to assist with the required analysis, core photographs from exploration holes drilled in 2006, 2007 and 2008 were 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 zone contact, used for geotechnical mapping



In general, the core photograph assessment indicates 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%.

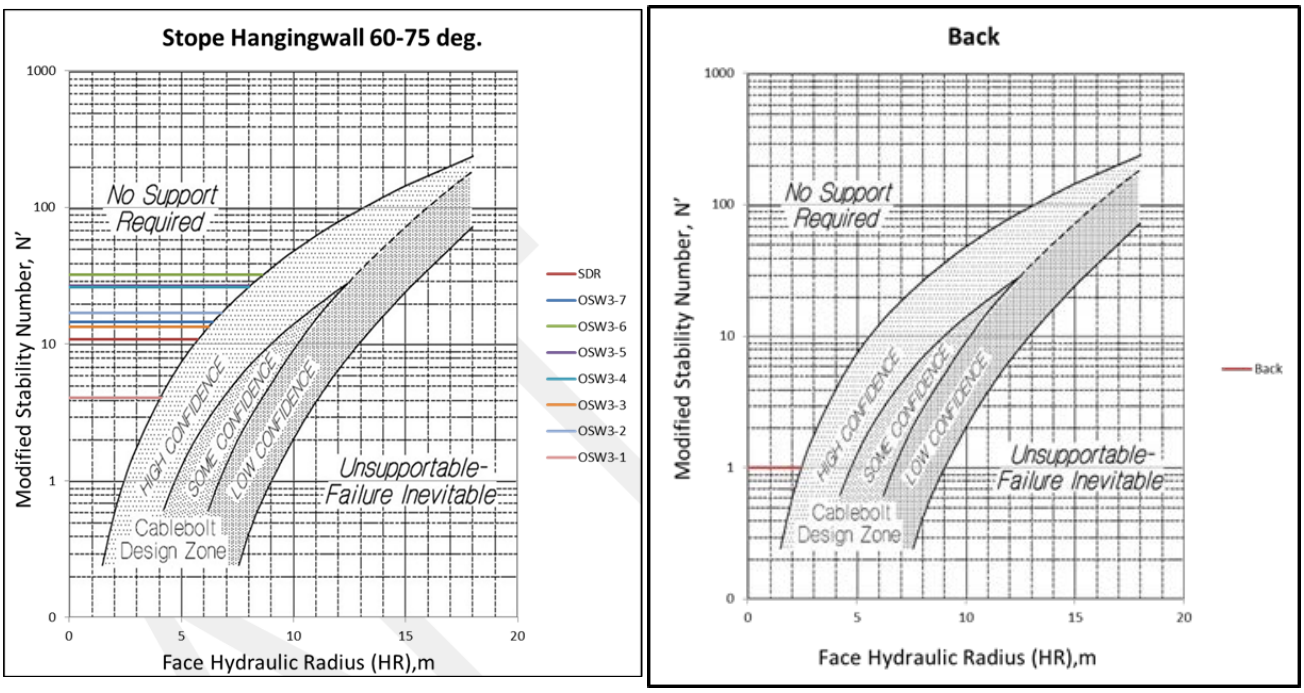
Rock mass quality in drift areas mapped is generally seen to be variable, with transition zones between the main lithological units and the veined ore zone. In some areas a poor-quality shear zone has been identified in the hangingwall. The HW rocks were noted to be usually of poorer quality than the 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. 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 30m 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.

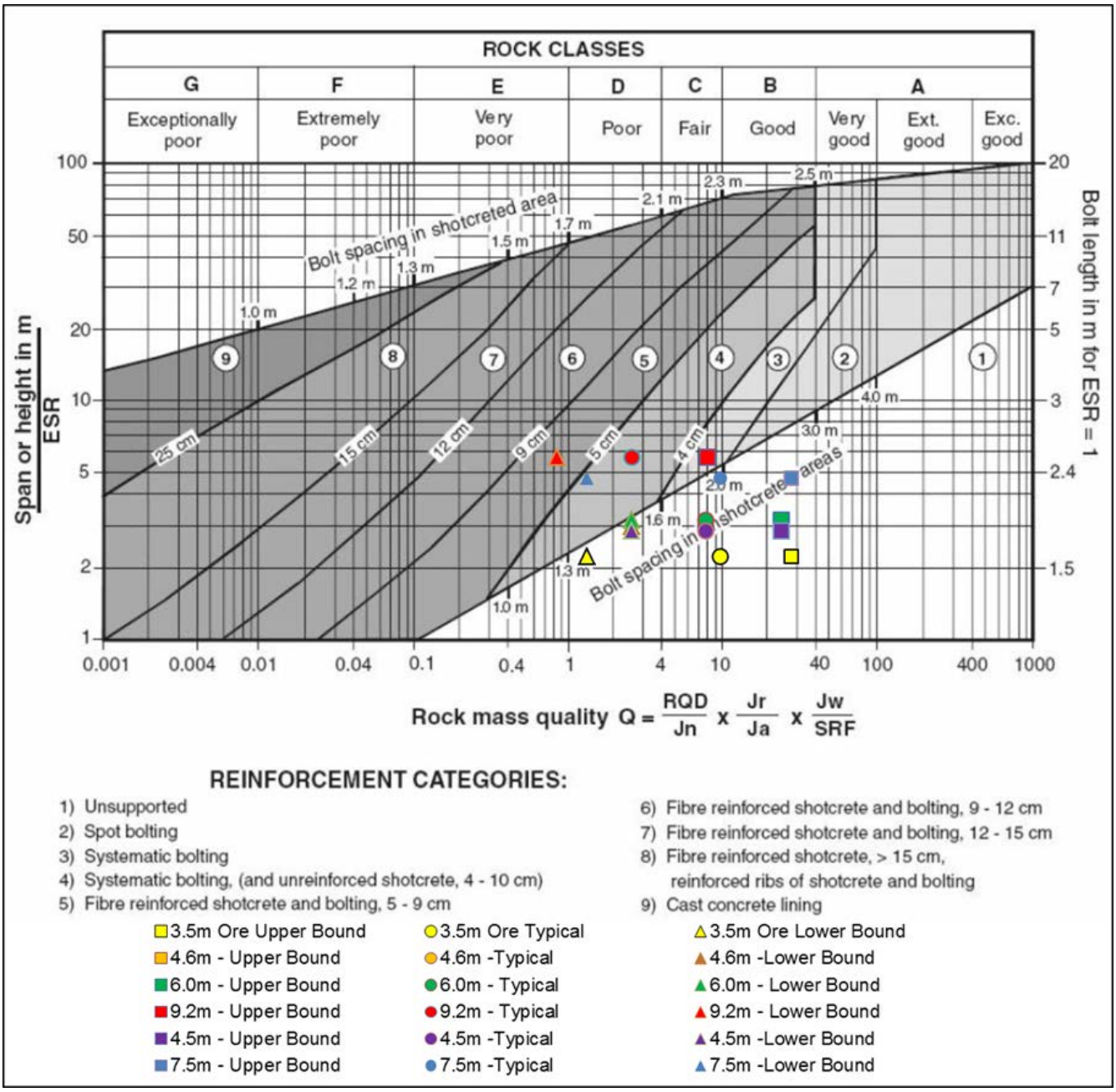
The stability number (N') for rock mass conditions of each lithological sub-unit is plotted on the stability graphs shown in Figure 16.7. These graphs are representative of the stope walls and stope back, respectively. They illustrate the correlation between N' and the excavation surface hydraulic radius (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. Timely placement of paste fill in stopes will impact ground stability.

Figure 16.7 Stability graph results for the stope walls (typical average dip 60–75 deg.) and stope backs



Preliminary empirical support requirements are presented in Figure 16.8. 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. 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. Preliminary ground support standards are outlined in Table 16.1.

Figure 16.8 Support chart from the Q System



It should be noted that Figure 16.8 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 (3.5 m × 3.5 m)	3.5 m Ore Upper Bound	1.8	1.8	Swellex or Split set	-	-	-
	3.5 m Ore Typical	1.8	1.5	Swellex or Split set	-	-	-
	3.5 m Ore Lower Bound	1.8	1.5	Swellex or Split set	50	-	-
Main development (Waste) (4.6 m × 4.6 m)	4.6 m - Upper Bound	1.8	1.8	Resin Rebar	-	-	-
	4.6 m -Typical	1.8	1.5	Resin Rebar	-	-	-
	4.6 m - Lower Bound	2.4	1.2	Resin Rebar	50	-	-
Main development (Waste) (6.0 m × 4.6 m)	6.0 m - Upper Bound	2.4	1.8	Resin Rebar	-	-	-
	6.0 m - Typical	2.4	1.5	Resin Rebar	-	-	-
	6.0 m - Lower Bound	2.4	1.2	Resin Rebar	50	-	-
Main development Intersections (9.2 m span)	9.2 m - Upper Bound	2.4	1.8	Resin Rebar	-	6	2
	9.2 m -Typical	2.4	1.5	Resin Rebar	50	6	2
	9.2 m -Lower Bound	2.4	1.2	Resin Rebar	70	6	2
Mucking Drive (Waste) (4.5 m × 4.5 m)	4.5 m - Upper Bound	1.8	1.8	Resin Rebar	-	-	-
	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 (Ore) (7.5 m × 5 m)	7.5 m - Upper Bound	1.8	1.8	Swellex or Split set	-	-	-
	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 4.5 m (H) x 4.5 m (W) to provide adequate main access from surface for personnel, equipment, fresh air and materials handling. The main access will connect to two internal ramps; one at the north end to service the MQV and STK zones, driven between elevations 965 and 478, and a south ramp, between elevations 883 and 628, to service the narrower areas of the MQV and the SMS zones. The ramps have been designed at a maximum +/-15% gradient with a minimum 20 m turning radius. Remucks and truck loading areas will be placed at every level access.

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 a minimum of 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; raises up to 30 m will be by drop raising. One existing raise to surface will be rehabilitated to exhaust return air from the mine. The 930L will be an additional exhaust airway.

Safety considerations during excavation limit the cross-section of Alimak raises to 3 m x 3 m. Where ventilation needs entail greater cross-section areas, raises will be duplicated or drop-raised.

Ore drives in the MQV zone will be for a distance of up to 300 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 7.1% zinc equivalent (ZnEq) to cover the cost of processing and selling the material. Any development material below the marginal cut-off grade will be considered waste and will be placed in stopes or on the surface stockpile.

Development heading design standards consider equipment needs, placement of mine services and regulatory requirements. Ore drives at the base of a mining block will be used for mucking access and will be larger than sub-level ore drives. Figures 16.9 to 16.12 show typical drift cross-sections, considering the dimensions of mobile equipment and ventilation ducting. Figure 16.13 shows a representative level development in plan view.

Figure 16.9 3.5 m by 3.5 m drift for MT2000 truck

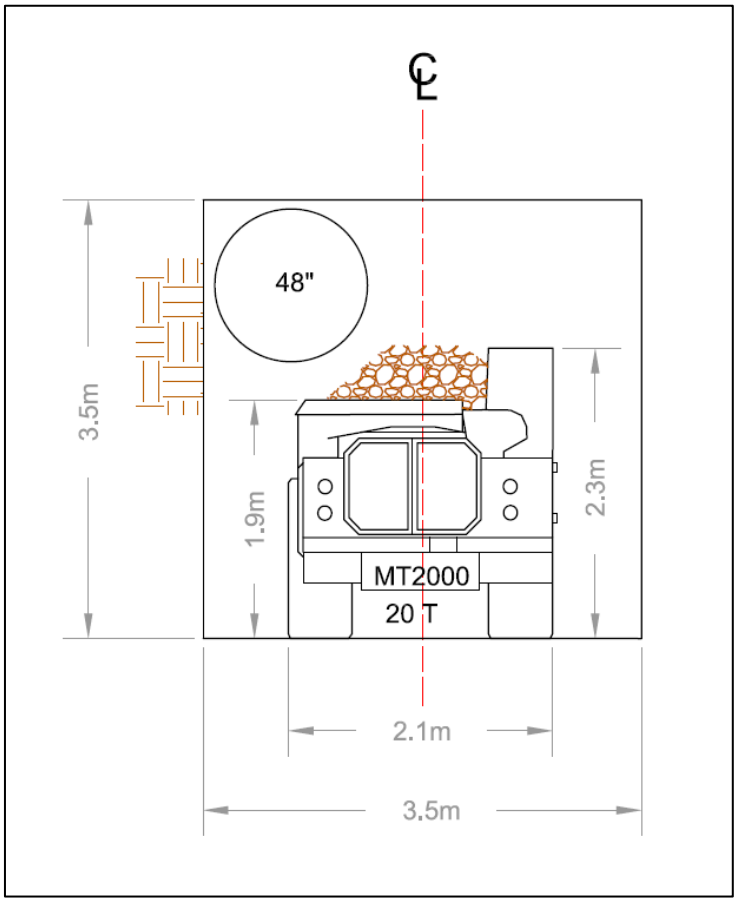


Figure 16.10 4.5 m by 4.5 m drift for D40 truck

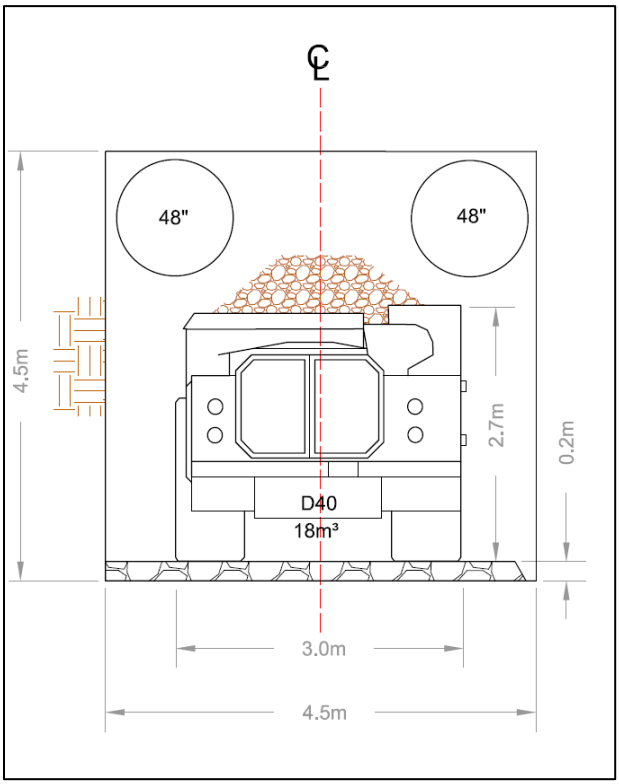


Figure 16.11 3.5 m by 3.5 m drift for 300EB scoop

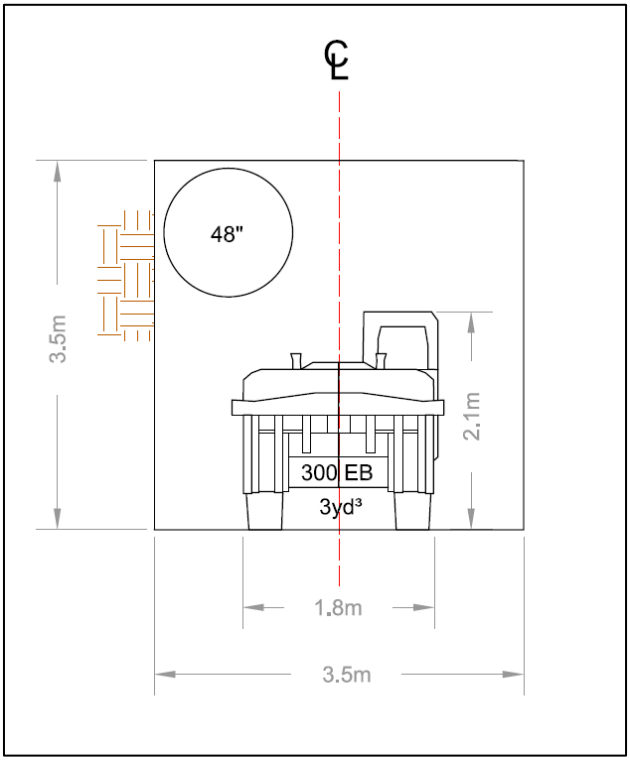


Figure 16.12 4.5 m by 4.5 m drift for R1700 scoop

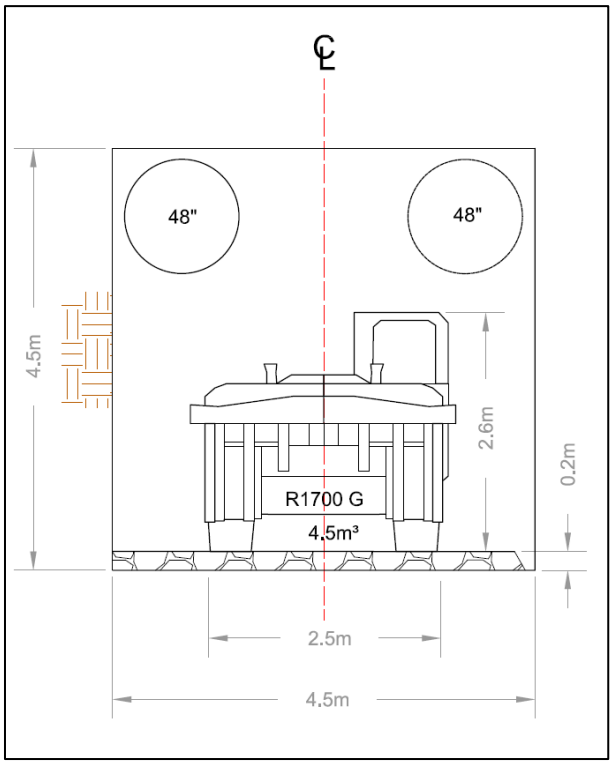
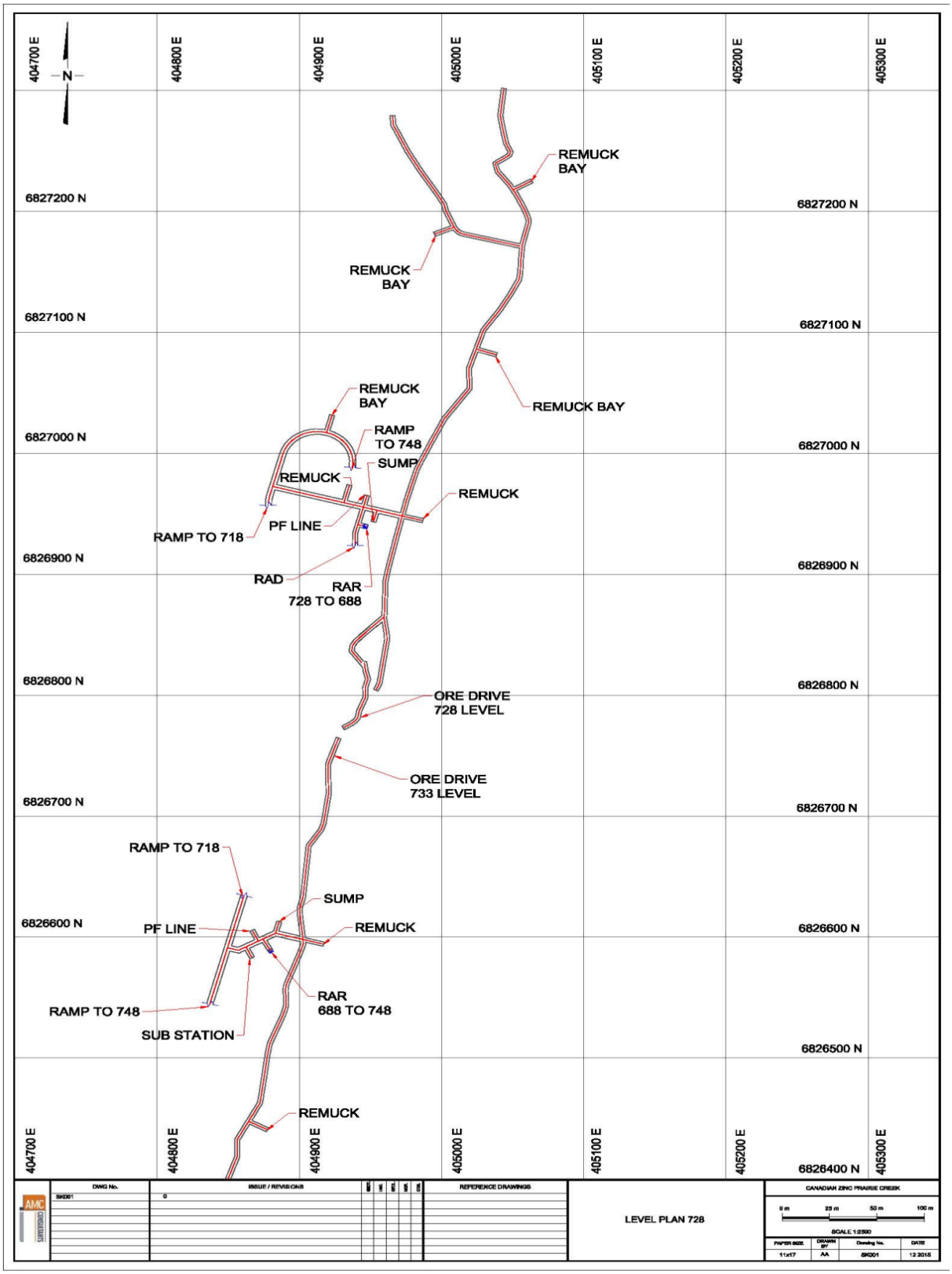


Figure 16.13 Representative sublevel arrangement in plan



*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 nominal longitudinal LHOS length and mine sequencing requirements. In addition, the stope shapes were economically assessed relative to development requirements.

A listing of key design parameters for each mining zone and method is shown in Table 16.3.

The MQV Mineralization widths generally vary between less than 0.1 m and up to 5 m. Most of the MQV width considered for mine design is in the range of 2.5 m to 4 m. In the STK zone, the stopes have been divided up into 6.5 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 807L 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	%
Cut-off Grade	NSR	137.1	%
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		0.2	m
Footwall Dilution		0.1	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*

Table 16.3 Key design parameters by zone and mining method

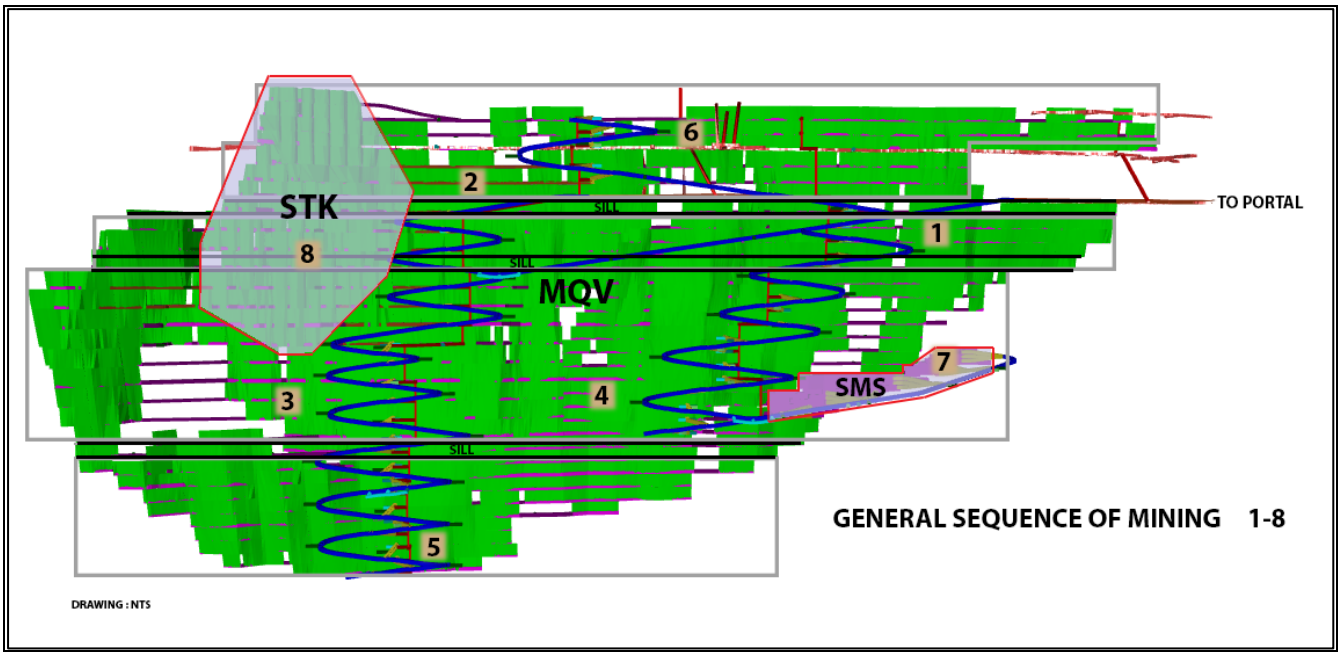
	UNITS	MQV LHOS 15	MQV LHOS 20	STK LHOS 15	STK LHOS 20	SMS DAF
Cut-off Zinc Equivalent	%	12	12	12	12	11
Minimum Mining Width	m	2	3	2	3	7.5
Mining Height	m	15	20	15	20	5
Mining Length	m	30	30	30	30	4
Minimum HW Angle	º	55	55	55	55	90
Minimum FW Angle	º	55	55	55	55	90
HW Unplanned Dilution	m	0.2	0.2	0.2	0.2	0.1
FW Unplanned Dilution	m	0.1	0.1	0.1	0.1	0.1
Floor Unplanned Dilution	m	0.15	0.15	0.15	0.15	0.1
LHOS Sidewall Unplanned Dilution	m	0.2	0.2	0.2	0.2	
Unplanned Dilution*	%	14	14	14	14	6
Mining Recovery	%	95	95	95	95	95

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

16.4.3 Zone and mining block definition

Figure 16.14 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, including sill locations. The general sequence of mining is also shown in Figure 16.14, although mining of the different zones will overlap significantly.

Figure 16.14 Longitudinal view of zones and mining blocks looking east



16.4.4 Stope cycle and sequence

For longhole stoping, LHOS shapes were created on both 15 and 20 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 will have LHOS blocks 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 vein. From each access, ore drives for drilling and mucking will be developed along the strike extents of the ore block. Forced ventilation through ducting will provide fresh air to the ore drive face and exhaust out of the ore drives into the return air system connected at each 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 16.2 above. For the MQV and STK LHOS mining sequence, an entire mining block will be exhausted before moving to the one above it unless it is separated by a designed sill.

After a block of stopes is exhausted of ore and ancillary activities 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 various mining activities and the total aggregate stoping rate per 60 m high set of longhole stopes.

Table 16.4 LHOS cycle and production rate for 60 m stoping block

Activities	Units	Number of Lifts
		4
DBM Rate (drill, blast, muck)	t/d	750
CMS*, Fill fence	days	7
Fill	m^3/d	1,368
Cure	days	21

*Cavity measuring system

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 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 Paste description

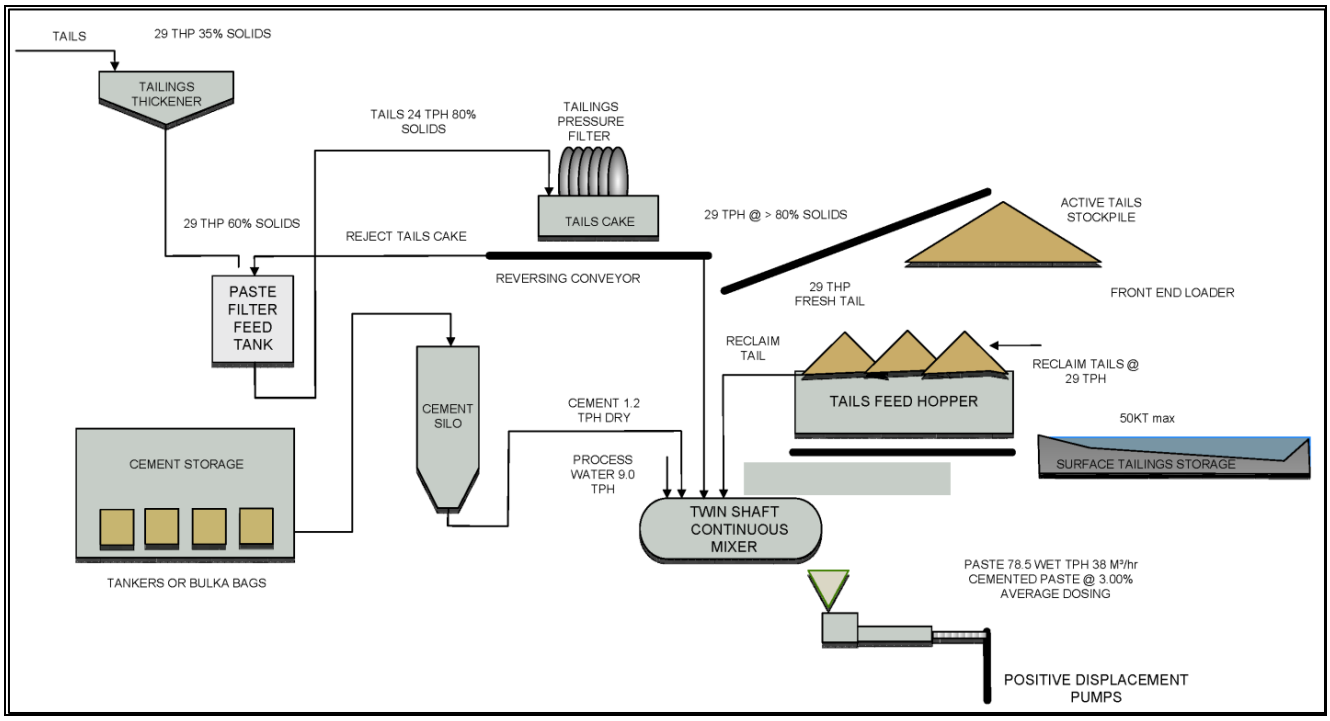
Paste fill will generally be the material used as backfill within the mine. The existing mill plant will be upgraded with a newly installed paste backfill plant that will return 100% of the flotation tailings to underground. The planned DMS plant, at the front end of the mill, will remove a significant amount of waste rock from the mill feed, resulting in the production of less 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 northwest of and directly adjacent to the mill. Paste will be produced from dewatered tailings mixed with either cement or cement and fly ash binders 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 be typically 3%, varying between 1% (where only low-strength fill is required) and 10% (sills only), dependent upon the required strength and slump. (The density of paste 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 will be carried out to determine the actual paste recipes required.

16.5.2 Paste system

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 1,350 tpd will produce approximately 365 tpd of heavy media reject aggregate, approximately 325 tpd of mineral concentrates and approximately 665 tpd of flotation tailings. These flotation tailings, when combined with cement, will produce on average 410 m^3/day of paste backfill, matching an average production void of 435 m^3/day. The small shortfall will allow for the placement of some development waste rock in stopes, the remainder being transported to surface for disposal. The continual availability of DMS reject material will be an additional source of backfill. A schematic of the paste fill plant is shown in Figure 16.15.

Figure 16.15 Prairie Creek paste system schematic



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 29 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 and place extra tailings cake in nearby piles as required. Any excess material will be hauled to the water storage pond area for temporary storage.

When backfill is required, 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 58 tph. This will result in a feed rate of 87 tph of tailings solids to the mixer.

At the completion of production at each longhole stope, a structural shotcrete barricade will be built at the lowest draw point elevation in the stope 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.

On request from underground, the plant operator will select the required fill recipe, specifying density, cement and delivery rate and will start up the paste mixing plant. The tailings, cement and process water will be mixed in the continuous mixer to produce a cemented paste fill. At an average dosing rate of 3%, cement will be added at 4.75 tph with make-up water to produce 57 m^3/hr of cemented paste fill for delivery underground by one of two positive displacement pumps.

The paste fill will be pumped 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 flow by gravity (other than for stopes above 883L) 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 the point of discharge. Where convenient in low-pressure zones, 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 vertical borehole in the event of a blockage.

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.

16.5.3 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 1,350 tpd there is a net shortfall of 25 m^3/day of available paste backfill. This small volume will be made up from development waste rock and DMS reject, including that required for top dressing of paste fill in the longhole stopes. The heavy media rejects and remaining development waste rock will be stored permanently on surface in an engineered stockpile, located in a ravine west of Harrison Creek.

16.5.4 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 live and active 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 heated shed. The volume of the tailings stored here will be approximately 6,500 m^3 and will contain approximately 9,500 t of tailings solids. When the backfill plant is running, this stockpile will be drawn down to residual levels as the tailings are placed back underground. The stockpile will be fed from an overhead conveyor and managed with a front-end loader.

If backfill is not placed for about 13 days at full production rate, excess tailings will be stacked in rows adjacent to the backfill plant. When backfilling, the live and active storage areas will be managed to minimize external storage volumes.

In the event of extended downtime of the backfill system, tailings will 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 over 70 days of additional storage capacity at peak production rates.

One of the design objectives of the backfill system 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, which, for the longhole stopes, will vary from one to four panels in volume.

16.5.5 Development waste rock

The projected development waste, ore and backfill schedules are detailed in Table 16.5. On average, 20,000 t of waste rock will be used each year to supplement the paste backfill. Excess waste rock will be hauled to surface and disposed of in the surface waste stockpile. The waste stockpile will grow to about 2.1 Mt over the life of the mine, but there may be opportunities to return some rock fill underground in later years as the development rate reduces.

16.5.6 Heavy media rejects

Table 16.5 also quantifies the heavy media reject material. This material is not an integral part of the backfill system but is included here for completeness of description of the site waste management strategy. At an average 27% reject rate of heavy media reject to ROM ore, the stockpile will grow to 2.05 Mt by the end of the LOM (Year 17).

Table 16.5 Ore, waste and backfill material movements

Year	Total Ore Tonnes	Total Waste Rock Tonnes produced	Total Pastefill Volume	Waste rock tonnes used UG for fill	Waste stockpile on surface tonnes	Total DMS tonnes	Total DMS Stockpile tonnes
1	108,722		99,920	20,000		29,355	29,355
2	376,144	76,765	179,032	20,000	56,765	101,559	130,914
3	485,517	157,170	187,137	20,000	193,934	131,090	262,004
4	484,950	138,631	115,921	20,000	312,565	130,937	392,940
5	487,229	160,473	22,324	20,000	453,038	131,552	524,492
6	485,858	161,124	112,554	20,000	594,163	131,182	655,674
7	488,103	146,851	77,041	20,000	721,013	131,788	787,462
8	487,940	160,317	24,527	20,000	861,331	131,744	919,205
9	485,655	156,918	12,865	20,000	998,249	131,127	1,050,332
10	485,619	146,679	10,830	20,000	1,124,928	131,117	1,181,449
11	489,451	146,681	15,477	20,000	1,251,608	132,152	1,313,601
12	485,551	167,965	10,325	20,000	1,399,573	131,099	1,444,700
13	487,568	160,066	74,568	20,000	1,539,639	131,643	1,576,343
14	487,020	164,131	84,769	20,000	1,683,770	131,495	1,707,839
15	467,475	162,482	56,196	20,000	1,826,252	126,218	1,834,057
16	449,738	146,242	10,595	20,000	1,952,494	121,429	1,955,486
17	361,049	152,661	7,344	20,000	2,085,155	97,483	2,052,969
Totals	7,603,590	2,405,155	1,101,422	340,000		2,052,969	

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 necessary to ensure 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 Act.”

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 (883L), and that contaminants are removed to the exhaust air system and ultimately to the surface via the 930L portal and an exhaust raise.

A maximum flow of 143 m^3/s is planned for ventilation of the underground operation to account for the proposed underground diesel equipment fleet and 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 16.16.

Fresh air circuit

Level distribution is designed so that fresh air will be sourced from the main decline at 883L, splitting between the two ramps in the north and south of the orebody. Fresh air will be delivered along each ore drive by auxiliary fan and duct installations. The ramps will thus be in fresh air in the event that mine rescue operations become necessary.

Return air circuit

Contaminated air from development and production activities will exhaust to the return air raises, which break through the level accesses at twenty or fifteen metre intervals and ultimately exhaust to the surface. The 930L portal as well as the existing 2.4 m x 2.4 m raise to the surface will serve as the primary exhaust routes. These will connect to two 4 m x 4 m raises (or, possibly, four smaller raises of equivalent total resistance to air flow) extending from the 883L to the projected bottom of the deposits, also connected to levels above the 883L.

Second egress

The exhaust air raises will be fitted with ladderways to serve as the second means of egress. The return air raise from 883L to surface will be for ventilation only. Further details can be found in the emergency preparedness section 16.10.

Ventilation power

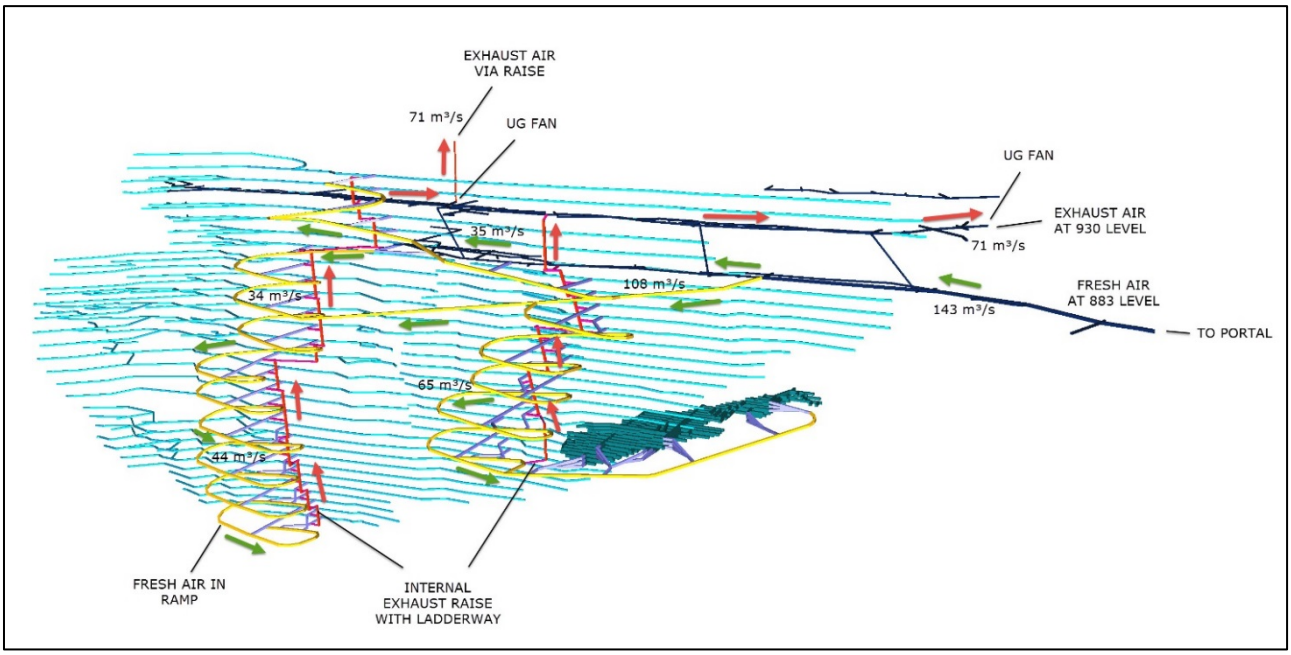
A natural ventilation pressure exists, which is upcast in winter and weak or downcast in summer. The power requirement for the primary fans to enable a maximum airflow of 143 m^3/s in summer is 202 kW in total. This is achieved by the two fans having a 150 hp (112 kW) motor each, pulling the exhaust air through the return air raise and 930L exhaust to the surface. During the winter season, Natural Ventilation Pressure (NVP) will assist the air movement such that the power consumed will be 185 kW.

During the development of the 4.5 m x 4.5 m ramps, 150 hp (112 kW) auxiliary fans will provide fresh airflow through twin 48” (1219 mm) ducts. Similarly, 150 hp (112 kW) fans with twin 48” (1219 mm) ducts will be required for development and mucking from the haulage drifts. For the non-haulage ore drifts, development and subsequent activities will require 75 hp (56 kW) auxiliary fans with 48” (1219 mm) duct to provide fresh airflow, noting that smaller battery-operated LHDs are projected to be utilized in these smaller drifts.

Mine air heating

A glycol heating system will be sited at the existing 883L 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. Waste heat from the main generators will be used in a glycol loop to supply the majority of the heat for the mine ventilation air. A direct-fired heating system will be used to provide additional “top-up” heat during extremely cold weather. Different phases of development and production will require different volumes of ventilation air.

Figure 16.16 Ventilation layout



16.7 Drill and blast

16.7.1 Explosives and initiation systems

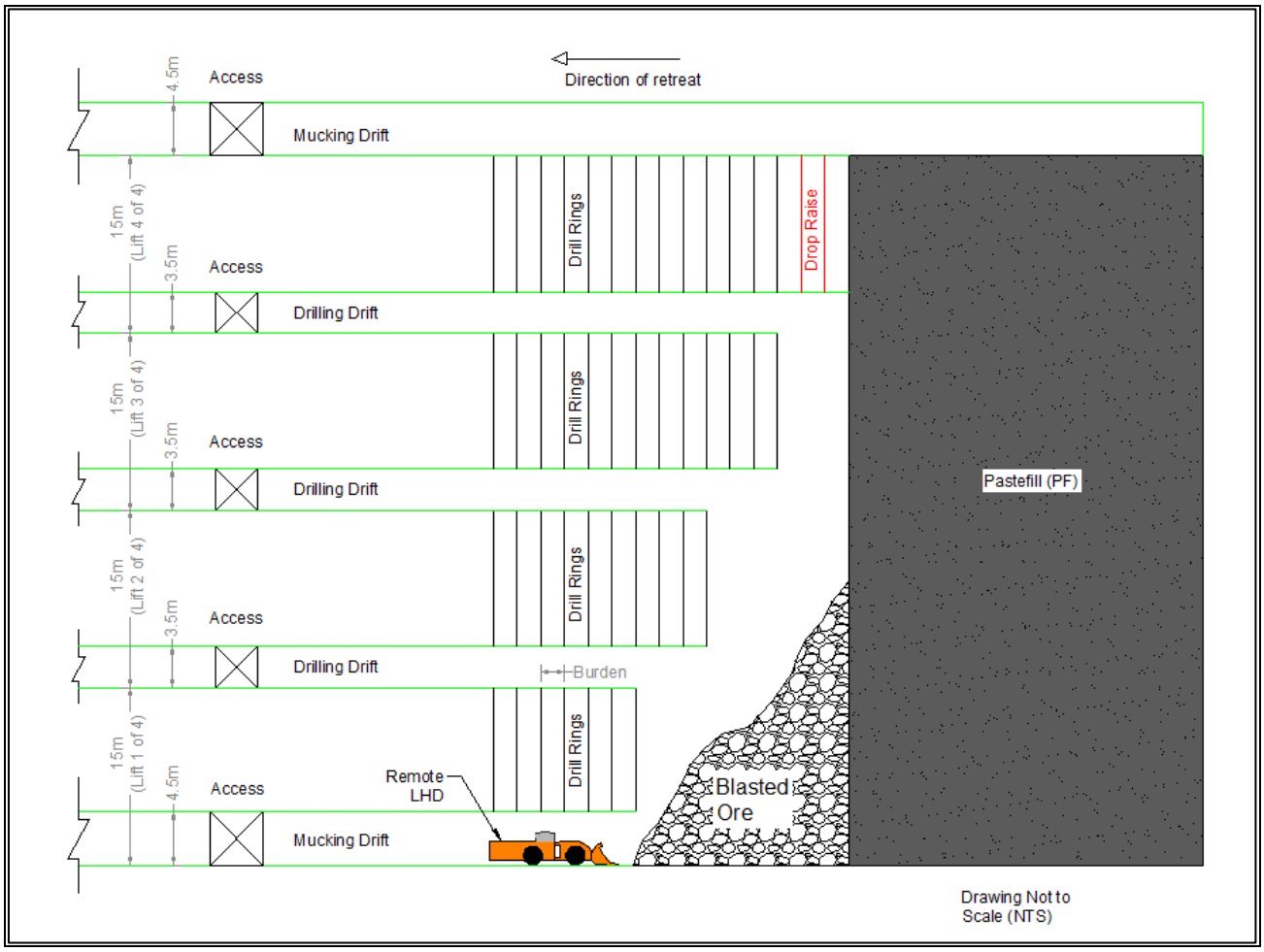
Canadian Zinc has undertaken not to use Ammonium Nitrate Fuel Oil (ANFO) blasting agent because of the high rate of release of nitrates into the drainage water. Packaged emulsion will be used for all stoping and lateral development. Emulsion is a commonly utilized bulk explosive product that is water resistant, produces low levels of ammonium nitrate in leachate, can be handled in bulk deliveries to reduce costs, has reliable and consistent performance and, depending on the product, can be left in the ground for up to four weeks after loading and before firing.

Boosters are planned for development and stoping, initiated by high-strength electric, electronic and non-electric detonators. Non-electric detonators are suggested for lateral development and stoping.

16.7.2 Longitudinal downhole stope design

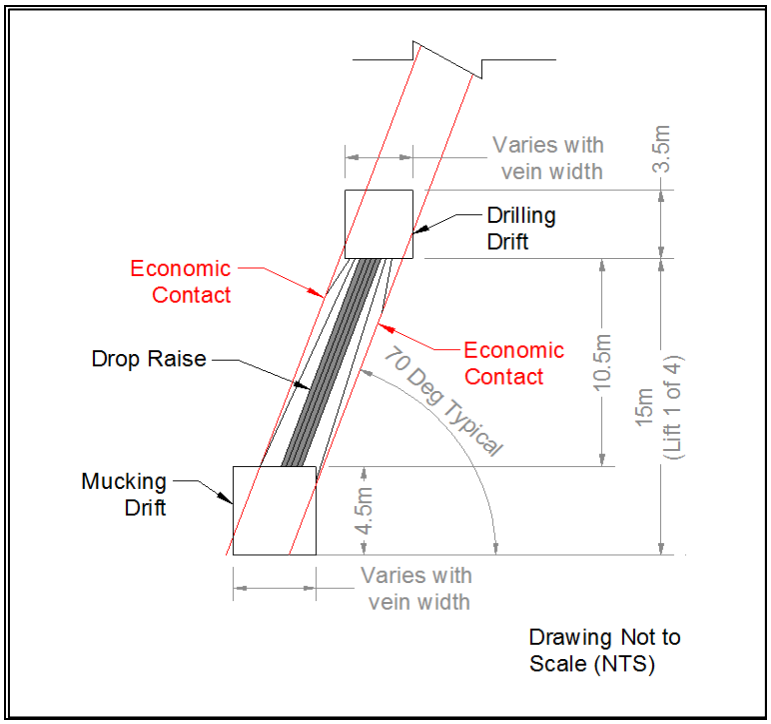
Figure 16.17 and Figure 16.18 show a typical longitudinal section and cross section of a downhole stope of 15 m panel height (floor to floor), typical width of 4 m, and length of 30 m. Longhole stopes account for approximately 75% of the planned stope tonnage. Approximately 10% is envisaged from transverse stoping and longhole uppers. The remaining 15% of planned stope tonnage will be from drift and fill (see Section 16.7.3).

Figure 16.17 Longitudinal section through longhole open stope



The overall powder factor for the downhole drilling stope design shown in Figure 16.18 is estimated at 0.49 kg/t. Depending on the stope shape and drill pattern, the design powder factor ranges from 0.35 to 0.50 kg/t for all types of longhole open stopes.

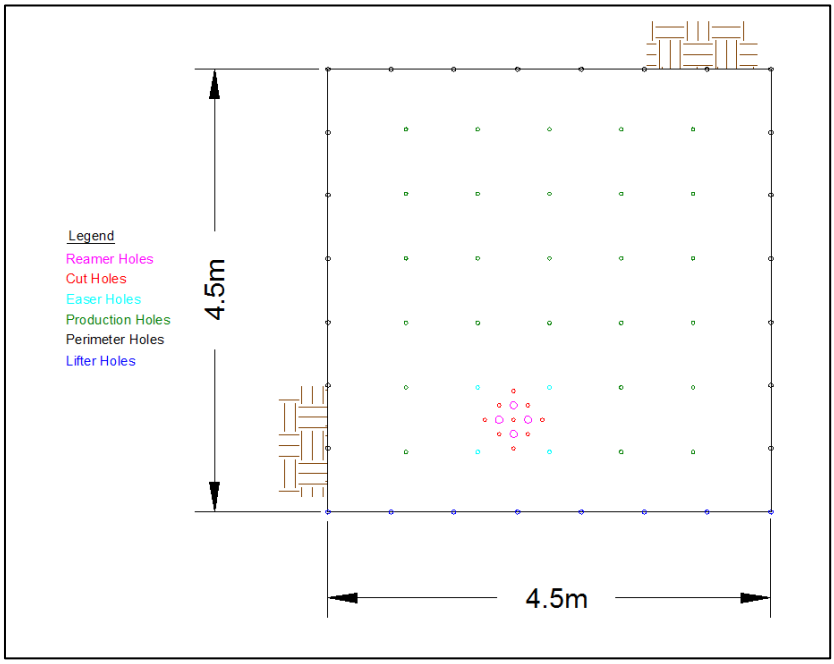
Figure 16.18 Long stope cross section showing downhole drilling.



16.7.3 Lateral development blast design

Figure 16.19 shows the drill layout for a typical lateral development round measuring 4.5 m (H) x 4.5 m (W) x 3.9 m deep.

Figure 16.19 Lateral development blast design



The overall powder factor for the lateral development design shown is 1.05 kg/t. The powder factor range for all lateral development is from 0.95 to 1.25 kg/t, including DAF stopes and the sill ore drives for the longhole open stopes.

16.7.4 Explosives consumption

At steady state production of 1,350 tpd and peak development advance of 8 mpd, the monthly mine explosives consumption will be approximately 34 tonnes of packaged emulsion, 5,000 detonators and 5,000 boosters.

Table 16.6 shows the design explosives consumption and powder factors for typical downhole stopes and lateral development. The stope tonnage shown is for a 60 m high stope at 4.0 m width after removal of the tonnage associated with ore development within the 60 m height.

Table 16.6 Powder factors for longitudinal downhole stoping and lateral development

Description	Ore Tonnes (t)	Explosives (kg)	Powder Factor (kg/t)
Longitudinal Downhole Stope	17,200	8,471	0.49
Lateral Development (per round)	227	234	1.03

16.8 Development and production schedule

16.8.1 Production rate

The aggregate production rate for a block of 60 m high longhole stopes is 284 tpd. The steady state mine production rate of 1,350 tpd will therefore be achieved by having a minimum of five active stoping areas. AMC's scheduling of development and production has demonstrated that 1,350 tpd is readily sustainable for most of the projected mine life.

16.8.2 Pre-production development

A pre-production period will occur over a timeframe that includes completion of mill and site infrastructure. During this period, the focus will be on driving the ramps towards the nearest mining blocks and developing the ore drives so that stoping can commence as soon as possible and ramp-up to full mine production is expedited. It is projected that a waste development crew will achieve 120 m/month of lateral advance, and an ore development crew will achieve 160 m/month. Approximately 50% of the total development metres will be for ore drives. Table 16.7 gives a breakdown of LOM lateral and vertical development.

Table 16.7 Lateral and vertical development for LOM.

Description	Metres
Main Access Slashing	643
Stope Access	4,293
Development Through-Fill - 3.5 x 3.5	574
Development Through-Fill - 4.5 x 4.5	1,737
Ore Drifting - 4.5 x 4.5	7,521
Ore Drifting - 3.5 x 3.5	18,302
Pastefill Stations	315
Return Air Drives	1,335
Remucks	2,879
Ramps	6,342
Sump Development	752
Electrical Substations	279
Waste Drifting within Stopes 4.5 x 4.5	1,313
Waste Drifting within Stopes 3.5 x 3.5	3,076
Total Lateral Development	**49,362**
Total Vertical Development	**802**

16.8.3 Sustaining development

In order to sustain targeted production and achieve the desired grade profile, access to targeted mining areas is of priority and this is reflected in the amount and location of sustaining development metres. As in the pre-production development, two-thirds of the development metres up until the completion of waste development will be in ore drifting. Two development crews will continue to advance in ore until the final four years of mine life. Waste development also continues throughout the mine life. The projected LOM development schedule is shown in Table 16.8.

Table 16.8 Projected development schedule over the LOM

Description		Total	PY 2	PY 3	PY 4	PY 5	PY 6	PY 7	PY 8	PY 9	PY 10	PY 11	PY 12	PY 13	PY 14	PY 15	PY 16	PY 17	PY 18
Main Access Slashing	*km*	**0.64**	0.56	0.09															
Stope Access	*km*	**4.29**	0.37	0.74	0.86	0.52	0.03	0.56	0.46	0.07	0.07	0.01	0.07	0.01	0.18	0.09	0.26		
Development Through-Fill - 3.5 x 3.5	*km*	**0.57**													0.34	0.24			
Development Through-Fill - 4.5 x 4.5	*km*	**1.74**													0.61	1.12			
Ore Drifting - 4.5 x 4.5	*km*	**7.52**	0.80	1.07	0.59	0.57	0.85	0.40	0.29	0.47	0.51	0.23	0.22	0.43	0.31	0.05	0.01	0.31	0.42
Ore Drifting - 3.5 x 3.5	*km*	**18.30**	1.40	2.20	1.02	1.09	1.71	0.54	0.95	1.57	1.12	1.33	1.49	1.18	0.66	0.32	0.72	0.79	0.22
Pastefill Stations	*km*	**0.31**	0.06	0.04	0.07	0.07		0.02	0.04		0.01								
Return Air Drives	*km*	**1.34**	0.19	0.29	0.42	0.22		0.15	0.04			0.03							
Remucks	*km*	**2.88**	0.26	0.53	0.43	0.20	0.20	0.26	0.23	0.20	0.07	0.07	0.15	0.09	0.11		0.01	0.06	
Ramps	*km*	**6.34**	0.80	1.71	1.63	0.60		0.87	0.58	0.05					0.12				
Sump Development	*km*	**0.75**	0.05	0.14	0.17	0.18		0.07	0.14										
Electrical Substations	*km*	**0.28**	0.05	0.04	0.05	0.07		0.02	0.04						0.01				
Waste Drifting within Stopes 4.5 x 4.5	*km*	**1.31**	0.12	0.05	0.05	0.29	0.08	0.11	0.07	0.06	0.08	0.01	0.01	0.06	0.03	0.04		0.07	0.19
Waste Drifting within Stopes 3.5 x 3.5	*km*	**3.08**	0.04	0.14	0.12	0.30	0.43	0.24	0.07	0.27	0.11	0.50	0.10	0.06	0.09	0.19	0.24	0.19	
Total Lateral Development	*km*	**49.36**	**4.68**	**7.04**	**5.41**	**4.10**	**3.30**	**3.23**	**2.92**	**2.69**	**1.96**	**2.18**	**2.04**	**1.82**	**2.44**	**2.06**	**1.25**	**1.41**	**0.83**
Total Vertical Development	*km*	**0.80**	**0.13**	**0.11**	**0.18**	**0.17**		**0.10**	**0.07**			**0.03**							

16.8.4 Life of Mine production schedule

The steady-state production of 1,350 tpd will be achieved by Project Year 4 and sustained through to Project Year 18, with a final year projected at around 1,000 tpd. An ore stockpile of about 10,000 t is generated in the first year of ore development, the mill becoming available the following year. Table 16.9 shows the projected life of mine production schedule and metal grades.

Table 16.9 LOM production tonnes and grade

Mine LOM Production and Ore Grades					
Project Year	**Tonnes**	**ZnEq**	**% Zn**	**% Pb**	**g/t Ag**
2	108,722	23.63	8.56	8.56	132.75
3	376,144	27.79	10.17	9.78	165.61
4	485,517	30.72	12.69	10.04	167.79
5	484,950	27.54	10.11	9.60	167.18
6	487,229	28.18	8.97	10.85	171.98
7	485,858	28.97	9.93	10.81	167.94
8	488,103	31.08	10.93	11.43	177.91
9	487,940	23.93	8.61	8.91	125.40
10	485,655	28.76	10.87	10.18	156.86
11	485,619	22.40	8.42	8.13	114.53
12	489,451	20.86	8.76	6.88	106.54
13	485,551	22.22	7.35	8.50	128.60
14	487,568	20.23	6.73	7.98	104.82
15	487,020	20.31	7.74	7.58	90.50
16	467,475	14.54	7.67	4.06	53.27
17	449,738	13.78	7.28	3.51	65.49
18	361,049	12.36	6.05	3.50	59.50
Total	**7,603,590**	**23.54**	**8.93**	**8.33**	**127.58**

Figure 16.20 shows the LOM production profile of ore tonnes and zinc equivalent grade from each zone and from the Mine as a whole.

Figure 16.20 LOM production profile

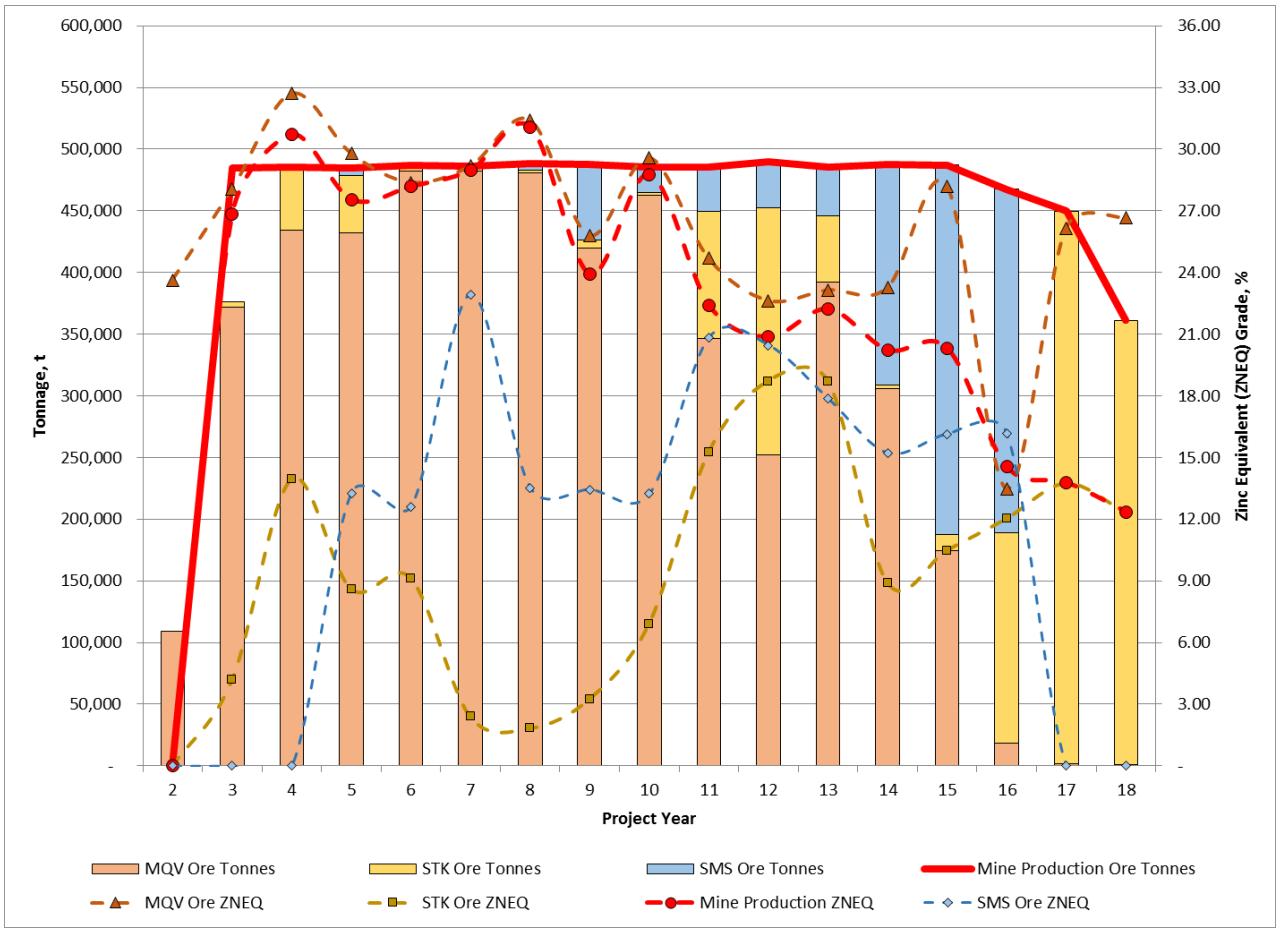
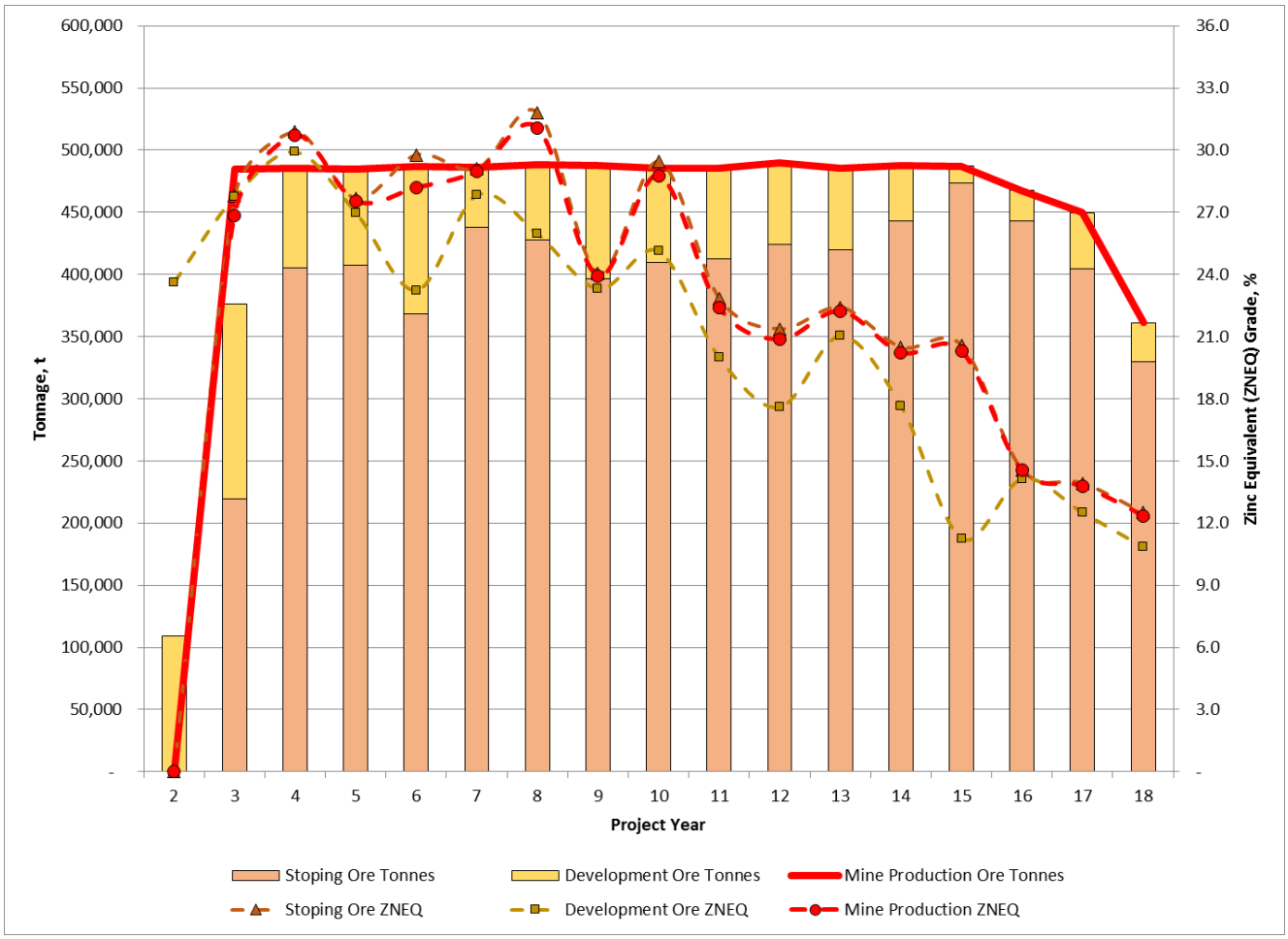


Figure 16.21 shows the split between stoping ore and development ore over the LOM.

Figure 16.21 Split of LOM ore tonnes and grade by stoping and development



16.9 Mobile equipment requirements

For the pre-production period, a contractor will supply the personnel and mobile equipment to execute the initial capital waste development program and mine the early stoping ore tonnes. At the completion of the contract, the ownership of the contractor-supplied mobile equipment may transfer to the Owner. To meet the steady state development and production schedule, the procurement of additional equipment will be required. Table 16.10 shows the total mobile equipment numbers and types that are projected to be required to meet the development and production schedule for the LOM plan. The total equipment fleet will be acquired over the initial three year period.

Table 16.10 Mine equipment requirements

				Year 1	Year 2	Year 3
Item	**Description**	**Purpose**	**Qty**			
Jumbo	2 boom jumbo	Waste development	2	1	1	
Jumbo	1 boom jumbo	Ore/waste mucking	2	1	1	
LHD	Cat R1700G	Ore/waste mucking	1	1		
LHD	Wagner ST6C	Ore/waste mucking	1	1		
LHD	JS220	Ore/waste mucking	1	1		
Battery LHD	Muckmaster 300EB	Ore/waste mucking	3	1	2	
Haulage Truck	Sandvik D40	Ore and waste haulage	2	1	1	
Haulage Truck	Wagner MT2000	Ore and waste haulage	2	1	1	
Long hole drill	Stopemate	Production drilling	1		1	
Long hole drill	Stopemaster	Production drilling	1			1
Bolter	Small section bolter	Bolting headings	1	1		
Shotcrete Sprayer	Reed Lova/Aliva arm	Shotcrete as required	1	1		
Scissor Lift	Maclean type	Bolting & services	3	1	1	1
Telehandler	Cat TL943	Consumables transport	1	1		
Personnel Carrier	Landcruisers	Crew transport	6		4	2
Boom Truck	Maclean BT3	Consumables transport	1		1	
Lube/Fuel Truck	Maclean FL3	Transport fuel and lube	1		1	
Grader	Case 845B	Grade decline	1			1
Tractor	Kubota or similar	Crew and materials transport	2	1	1	
Total			33	13	15	5

The Muckmaster 300EB is a battery powered scoop, all other mucking and haulage equipment is diesel powered

16.10 Emergency preparedness

In development of 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 (883L 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 930L portal, noting that means to bypass the installed primary exhaust fan on the 930L will be implemented.

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 883L or 930L portals. The primary exhaust raise to surface is for ventilation only and will not be used as a second means of egress.

16.10.1 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 in areas where secondary egress is not, or has not yet been, established, or is not able to be safely accessed, and will be sited relative to the active working areas to 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 0.5L 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 24L 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 & 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 information used to estimate the projected mine dewatering requirements is based on Robertson Geoconsultants Inc. (Robertson) (draft) report “Project No. 148005: 2014-2015 Mine Dewatering, Prairie Creek” issued on 16 October 2015. The report is derived from experimental data obtained during large-scale dewatering tests at the mine during the 2014/2015 season and summarizes predicted inflows based on four probabilistic cases. The dewatering plan is based upon what is deemed the most probable (best estimate) case in the report.

Average estimated inflows vary significantly between a low of 31.1 L/s and a high of 207 L/s. AMC cautions that due to the uncertainties and ranges of the estimates, inflows could be higher or lower than anticipated. The dewatering system is designed to be easily expandable should inflows prove to be higher than currently anticipated.

The inflow information provided by Robertson contains estimates from the 840 to the 640 mL elevations; as the mine is now to be excavated to the 480 mL elevation, the data is extrapolated to this depth. Note that the data provides estimated water inflows reporting to a given level. For example, if the mine is at the 640 m Level then inflows at higher levels are projected to be negligible.

A key criterion of the system is to separate non-contact water, which has been drawn from the MQV before it comes into contact with oxidized ore and other contaminants, and contact water. There are, therefore, two dewatering systems: one for non-contact water and one for contact water.

Contact water volumes were estimated by adding together projected seasonal inflows from the area immediately above the mine, host rock inflows to open stopes, and a projected flow of one litre per second from mining operations. Non-contact water is estimated to be the volume needed to be pumped in order to maintain the local water table below the active mining levels.

The contact and non-contact estimates used for this study are shown in Table 16.11 and Table 16.12. The first four lines of Table 16.11 are taken directly from the Robertson report.

Table 16.11 Estimated monthly mine inflow (non-contact water) (L/s)

Dewatering Level	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Avg.
840	16.4	16.2	13.6	11.1	9.8	24.9	33.2	32.4	29.8	23.4	20.4	17.8	20.8
800	24.7	22.0	18.9	17.0	16.1	32.3	44.6	40.8	35.9	29.8	27.9	24.1	27.8
720	41.4	33.6	29.5	28.9	28.7	47.0	67.5	57.5	48.2	42.5	42.8	36.8	42.0
640	58.1	45.2	40.1	40.7	41.3	61.7	90.3	74.3	60.5	55.3	57.7	49.4	56.2
560	74.8	56.8	50.7	52.5	53.9	76.4	113.1	91.1	72.8	68.1	72.6	62.0	70.4
480	91.5	68.4	61.3	64.3	66.5	91.1	135.9	107.9	85.1	80.9	87.5	74.6	84.6

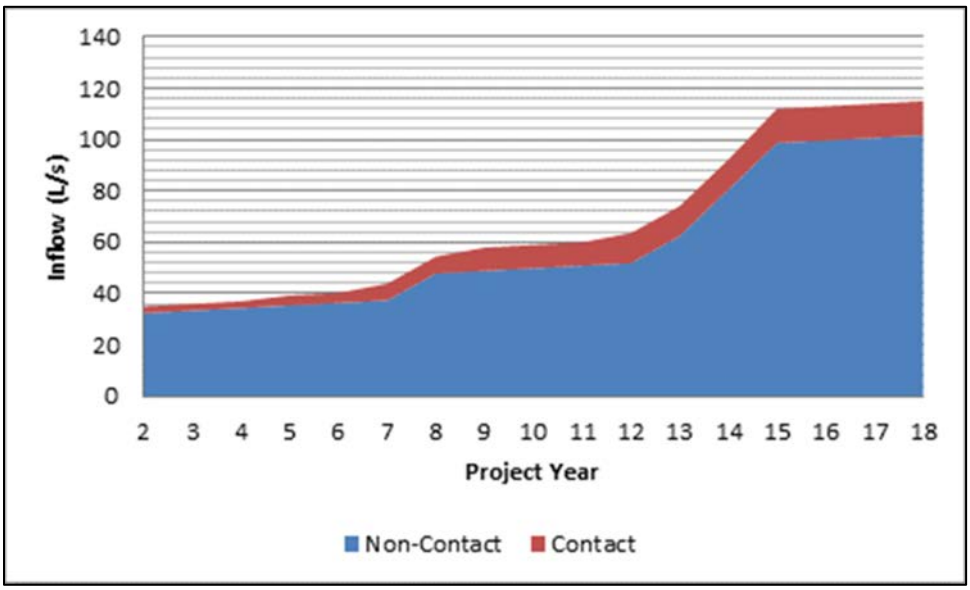
Data for levels 840, 800, 720, and 640 quoted directly from Robertson report.

Table 16.12 Estimated monthly mine inflow (contact water) (L/s)

Dewatering Level	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Avg.
840	2.3	2.3	2.3	2.3	3.4	3.4	2.9	2.7	2.6	2.3	2.3	2.3	2.6
800	3.6	3.6	3.6	3.6	4.7	4.7	4.2	4.0	3.9	3.6	3.6	3.6	3.9
720	6.2	6.2	6.2	6.2	7.3	7.3	6.8	6.6	6.5	6.2	6.2	6.2	6.5
640	8.8	8.8	8.8	8.8	9.9	9.9	9.4	9.2	9.1	8.8	8.8	8.8	9.1
560	11.4	11.4	11.4	11.4	12.5	12.5	12.0	11.8	11.7	11.4	11.4	11.4	11.7
480	14.0	14.0	14.0	14.0	15.1	15.1	14.6	14.4	14.3	14.0	14.0	14.0	14.3

The above estimates project water inflows when the mine is operating down to a given level. The flow rate into the mine will increase as the mine deepens and peak as the mine reaches full depth (projected to take place in Year 7). Relative to the projected development and production schedule, sumps will be positioned in the mine to bring the non-contact water table down below the active mining horizon before significant stoping in that area commences. Figure 16.22 shows the contact and non-contact inflows by year for the extrapolated best estimate case.

Figure 16.22 Annual water inflow by year for extrapolated best estimate



The predicted annual average inflows are used to estimate power consumption and pump utilization. However, the design for pumping capacity must take into consideration the peak inflows that may occur.

To determine maximum potential inflows (over the life of mine), monthly inflow estimates were extrapolated down to the 480 m level from the Robertson data. Monthly estimates for total mine inflows are shown in Tables 16.11 and 16.12. Based on this data, the annual peak inflows occur during the month of July. The non-contact pumps will be sized to handle the peak flow at the lowest point of the mine, providing surplus capacity during other months.

Should actual experience during mining indicate a potential for fault connection to an aquifer (a low probability in the Robertson report) additional pump capacity could be added during the mine life. The mine life includes several early years at low initial water inflow rates with corresponding high surplus installed pumping capacity. This will provide time to learn about and mitigate potential high inflows at greater depths.

16.11.1 Contact water system

The contact water system will consist of multiple sumps located at major mining intervals. Smaller sumps located at sub-levels will drain into the next lower main sump through drain holes. Water collected in the lower main sump will be relayed up to the next higher main sump. Low-flow, high-head submersible pumps in small dirty water sumps would be used for this purpose. These pumps would discharge through a 4” line.

16.11.2 Non-contact water system

The non-contact water system envisages five sumps: one below each major mining horizon. Diamond drillholes would be drilled out into the ore below the horizon in order to direct non-contact water into the sump. This would lower the water table in the ore below the mining activities and prevent the water from picking up additional dissolved metals. Two sumps would be in operation at any given time (one in each of the north and south declines) and, in the later stages of the mine life, one sump (with three pumps) would be operated at the bottom of the mine.

When mining is due to proceed below an intermediate sump, a new sump would be excavated below the new (lower) projected mining level. The pump would then be relocated down to the new sump, the old sump abandoned and the drain holes grouted off before mining activity proceeds below the intermediate dewatering level.

During the early years of mine operation one pump will be installed in each sump. When the final sump is installed at the bottom of the mine, two pumps will be relocated to it, and an additional pump installed. A total of three pumps will be required for the final sump to provide redundancy and surplus capacity. During normal operations, an extra pump is allowed for as an insurance spare in inventory. The system would tolerate a few days of single-pump operation due to the recharge time of the water-bearing zones.

An 8” steel line for dewatering would be run along the 883L adit and down each of the north and south declines. When the final bottom sump is installed, a second parallel eight-inch line would be run up the ventilation raise in each zone and along the initial pipeline route. This approach minimizes the length of steel line required, provides redundancy, and avoids demolition and rework of the dewatering system.

One insurance spare pump should be stocked (but not necessarily installed) at the Mine site.

16.12 Compressed air

Considering the cost and inefficiency of mine-wide compressed air systems, compressed air will be supplied by local portable compressors.

The portable compressors will be electrically-powered, using the same jumbo plugs and jumbo boxes as other mobile equipment. AMC recommends that three compressors of 1000, 900 and 350 cfm capacity be available on site to accommodate the following potential demands:

- Elevated pressure requirements for the ITH drill activities

- Powering mechanized raise climbers
- Miscellaneous activities such as spot bolting with jacklegs, powering air tools

All mobile drilling equipment, including jumbos, long-hole drills and bolters will be equipped with on-board compressors.

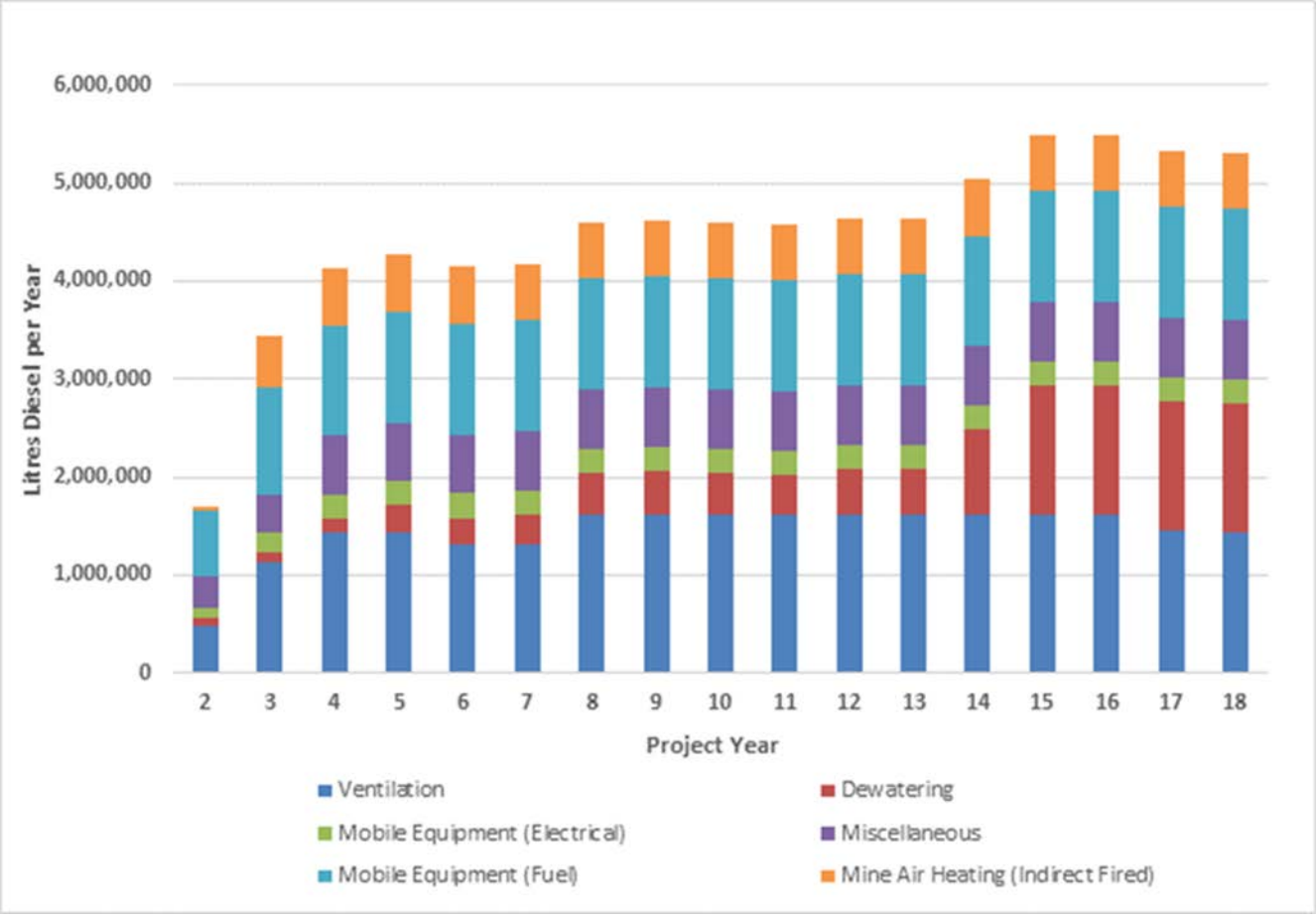
16.13 Mine water supply

The mill has a fresh water source from existing wells. Underground water consumption for drills and wetting the muck at the faces has been estimated at 1 L/sec and is to be sourced from the current sump on 883L. The water will be gravity fed to the lower portions of the mine. Pipes and pressure-reducing valves or a tank and valve system will be required. A small freshwater pumping system will be required for the mining levels above 883L.

16.14 Fuel supply

Estimated annual underground fuel consumption is shown in Figure 16.23. The estimated peak fuel consumption is 1.5 M litres per year, with mobile equipment being the biggest user, followed by mine air heating. Daily fuel consumption of the mobile equipment fleet is estimated to be about 3,000 L. Vehicles that come to surface regularly will 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 16.23 Underground Fuel Consumption Profile



16.15 Workshop facility

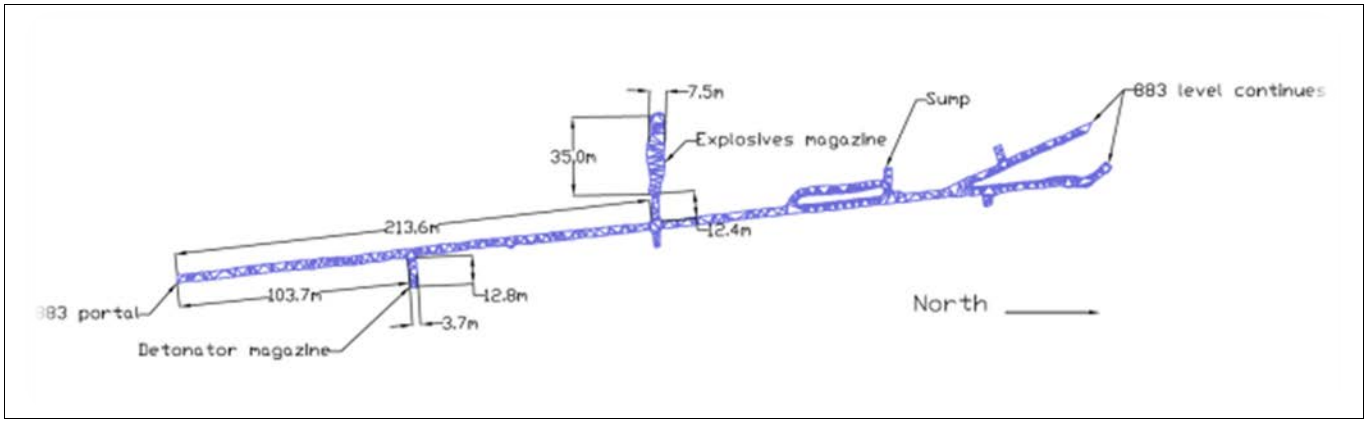
The main maintenance area will be located on surface. All major scheduled preventive maintenance and rebuilds will take place in this surface shop. One small service bay will be located underground, centrally located on the north ramp. Equipment working in the south ramp area will be taken directly to the surface shop for all maintenance. The service bay 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 Explosives magazines

There will be one powder magazine and one cap magazine on the 883 Level – see Figure 16.24. The powder magazine will store the packaged emulsion, boosters and all other cartridge explosives. The cap magazine will store the detonators. The entrance to these explosives magazines will be controlled with a permanent fence/wall and a robust double locked gate. Detonators will be stored on wooden shelves. Further details of explosives storage are recommended to be assessed at the next study stage.

Figure 16.24 Prairie Creek magazine locations



16.17 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 pre-determined 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.

16.18 Electrical distribution

Estimated annual underground power consumption is shown in Figure 16.25. The highest average demand in any one year is estimated at 1.6 MW. The short-term peak demand is estimated at 2.8 MW. The main users of power are ventilation, dewatering and mobile equipment. Electrical power at 4160V will be supplied to underground via both the 883L and 930L access portals using separate 4160V feeder circuits connected to the site main substation. Each of the two feeders (2 X 3 conductors, 350 MCM each) will continue independently down the declines.

As development progresses, the first of the 350 MCM feeders will be connected to a fused 4160V switchgear unit located on the level where mining activity will first occur. This 4160V switchgear unit will supply power to a permanently placed Power Distribution Centre (PDC) unit located on that same level. This PDC will step down

the voltage to 600V and distribute power to any mining equipment, pumps, fans, small motors and lighting demands necessary to permanently service that level and nearby sublevels.

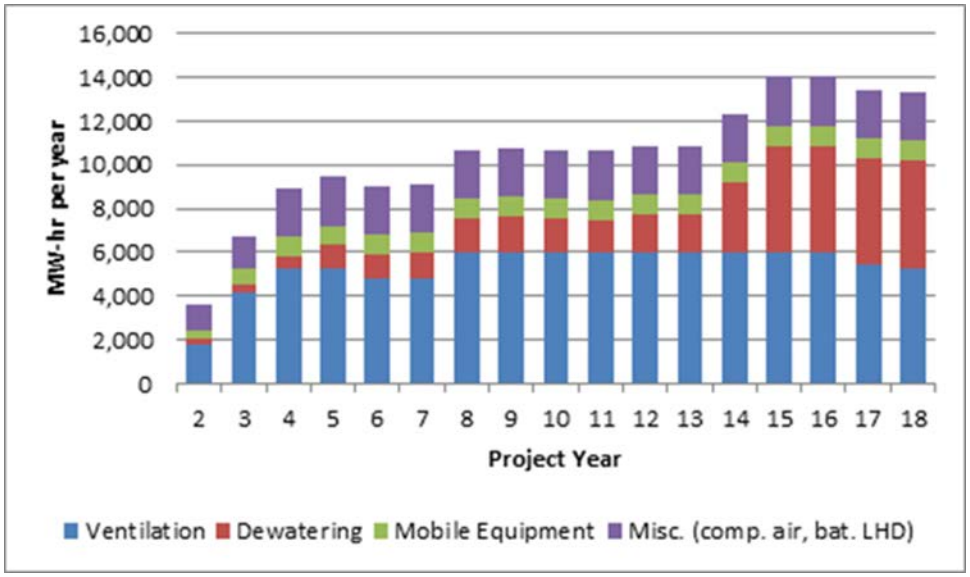
The 4160V switchgear unit will also supply portable Mine Power Centres (MPCs) located throughout the level. Each of the portable MPCs will be fed from the permanent 4160V switchgear via trailing cables and will step down the voltage to 600V to supply power to any mining equipment, pumps, fans, small motors and lighting demands required on a temporary basis to develop and mine the level.

As development continues downward, the additional 350 MCM feeder cable will be installed with a permanently placed 4160V switchgear unit and a PDC unit installed on each of the main levels (882L, 807L, 747L and 687L). With both ramps being developed at the same time, the electrical equipment can be powered from either ramp.

When the 627L is reached, a 4160V switchgear unit and PDC unit will be permanently placed along with two tie switches that connect the two independent 350 MCM power feeds coming down each ramp. This will provide some redundancy and allow power to be supplied via either feed.

Further development down from the 627L will see an additional permanently placed 4160V switchgear unit and PDC unit installed on each of the 567 and 507 Levels. These units can be fed from either of the 350 MCM power feeds.

Figure 16.25 Estimated annual LOM underground power demand



16.19 Underground mine personnel requirements

During the pre-production period, a contract labour force has been assumed. The labour force is currently projected to transition from contractor to Canadian Zinc in Project Year 2.

Personnel will be scheduled on a regular, fly-in-fly-out rotation during operations. Most positions in operations will require a day and night shift, while technical positions typically only require a day shift. Table 16.13 lists the underground personnel requirements projected for life of mine operations (maintenance personnel are included in the surface listing). A maximum of 111 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 16.13 Underground Mine LOM personnel requirements

Project Year	Year 2	Year 3	Year 4	Years 5 - 6	Year 7	Years 8 - 11	Years 12 - 13	Year 14	Years 15 - 16	Year 17	Year 18
Yearly Tonnage →	**108,722**	**376,144**	**485,517**	**484,950**	**485,858**	**488,103**	**489,451**	**487,568**	**487,020**	**449,738**	**361,049**
Chief Engineer	1	1	1	1	1	1	1	1	1	1	1
Mine Planning/Ventilation	2	2	2	2	2	2	2	2	2	2	2
Mine Technologist	2	2	2	2	2	2	2	2	2	2	2
Chief Geologist	1	1	1	1	1	1	1	1	1	1	1
Sr Geologist	1	1	1	1	1	1	1	1	1	1	1
Grade Control/Beat	2	2	2	2	2	2	2	2	2	2	2
Mine Superintendent			1	1	1	1	1	1	1	1	1
Project Superintendent	1	1									
General Foreman			1	1	1	1	1	1	1	1	1
UG Supervisors	4	4	4	4	4	4	4	4	4	4	4
Sub-total supervisory & technical	**14**	**14**	**15**	**15**	**15**	**15**	**15**	**15**	**15**	**15**	**15**
Miners - Lateral Development	38	57	42	34	26	24	20	36	36	13	8
Miners - Vertical Development *		1	1	1		1	1				
Longhole Drillers		3	5	5	5	5	5	3	2	5	4
Longhole Blasters		4	8	8	8	8	8	6	4	4	4
Ore/Waste Flow - Scoop	3	7	8	9	9	9	9	9	9	6	5
Ore/Waste Flow - Truck	4	9	10	9	9	9	9	9	9	7	6
Backfill		2	4	4	4	4	4	4	4	4	3
Construction/Fill Barricades/Pumping		4	7	7	7	7	7	7	2	4	4
Labour	1	1	8	8	8	8	8	8	8	8	6
Sub-total U/G Workforce	**46**	**88**	**94**	**85**	**75**	**74**	**70**	**81**	**72**	**51**	**40**
Total Dedicated U/G Personnel	**60**	**102**	**109**	**100**	**90**	**89**	**85**	**96**	**87**	**66**	**55**

* Average number of raise miners required
All other personnel such as maintenance, safety, dry, etc. are included in surface manpower

17 Recovery methods

17.1 Introduction

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. A copper concentrate may be produced from lead sulphide concentrate and a zinc oxide concentrate may be produced from the tailings of lead oxide flotation, if warranted by economic conditions.

The process design is based mainly on the results from various metallurgical test programs, including flotation, heavy liquid separation, ore hardness, and dewatering tests.

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 flotation circuits and 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/flotation circuits.

17.2 Summary

The Prairie Creek mill design has been modified to treat approximately 470,000 tpa run-of-mine (ROM) ore. The mill will operate approximately 345 dpa, with a planned maintenance shutdown period of approximately 20 days. The target daily mill feed will be approximately 1,200 to 1,400 tpd depending on head grades. The designed grinding and flotation circuit feed rate is 960 tpd at an availability of 92%.

At LOM average feed grades of 8 to 9% lead, 8 to 9% zinc, and 130 g/t silver, the plant is anticipated to produce, on average, approximately 46,600 tpa lead sulphide concentrate grading 67% lead, and 7,500tpa lead oxide concentrate grading 48% lead. Zinc concentrate product is projected to be 59,900 tpa of zinc sulphide concentrate at 59% zinc.

The proposed plant will consist of crushing, DMS pre-concentration, grinding, sequential flotation, concentrate dewatering, tailings dewatering and backfill paste preparation units. Figure 17.1 shows a simplified process flowsheet. The processing units are detailed below:

- Crushing the ROM mill feed to approximately 80% passing 10 mm (refurbished)
- DMS pre-concentration to reject coarse gangue minerals at approximately 20 to 30% of mill feed (new plant)
- Grinding circuit to further reduce the concentrate from the DMS plant to 80% passing approximately 80 µm (refurbished)
- Lead sulphide flotation producing a lead sulphide concentrate (refurbished)
- Zinc Sulphide flotation producing a zinc sulphide concentrate (refurbished)
- Lead Oxide flotation producing a lead oxide concentrate (refurbished)
- Concentrate dewatering and load-out systems (refurbished)
- Tailings dewatering (refurbished)
- Backfill paste plant (new plant)

ROM ore will be delivered by trucks to the crushing circuits, consisting of a jaw crusher and a cone crusher in closed circuit with a screen. The closed-circuit secondary crushing stage is to ensure the desired particle size for the downstream DMS and grinding circuits.

Crushed ore will be conveyed from the fine ore bin to the DMS plant, consisting of a de-sliming screen and a DMS circuit with ferrosilicon media. The DMS plant will produce three products:

- Slimes or fines, reporting to the grinding circuit

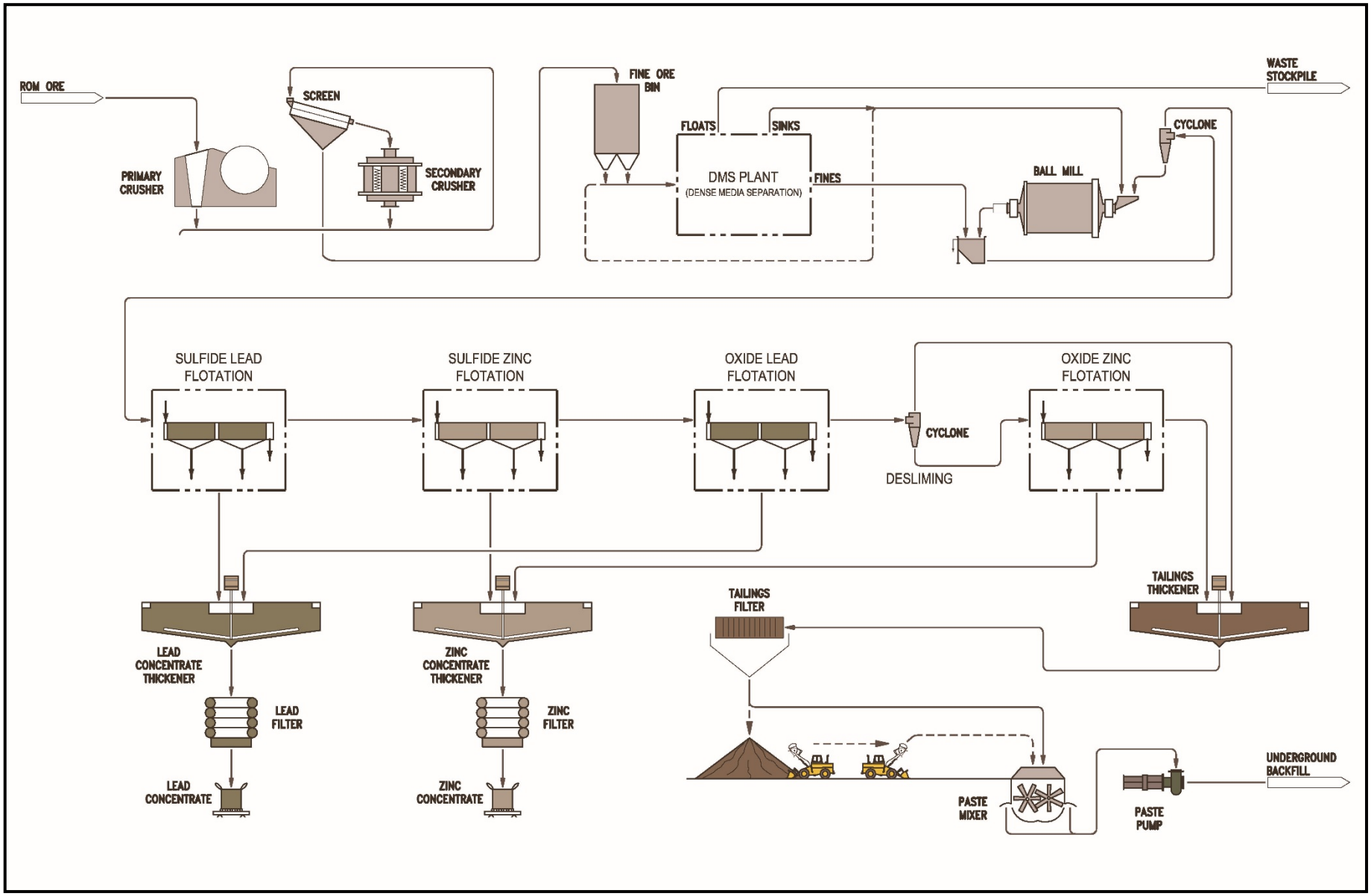
- Heavy fraction (DMS cyclone underflow), reporting to the grinding circuit, where it is joined with the slimes
- Light fraction (DMS cyclone overflow), which mainly consists of gangue material and is discarded as rejects to the waste storage facility

Both slimes and cyclone underflow will be ground in a conventional ball mill circuit. 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 separate rougher/scavenger flotation and multiple-stage cleaner flotation to upgrade the rougher and scavenger concentrates. The rougher scavenger tailings from the lead oxide flotation will be discharged as the final tailings. A detailed description of the flotation circuits is given in Section 17.4.4. 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 filtration circuits. The dewatered concentrates will be stored in the on-site storage facility prior to being transported to off-site smelters.

The option exists to install a zinc oxide flotation circuit to extract a zinc oxide concentrate from lead oxide flotation tailings. Market conditions and the anticipated grade of the zinc oxide concentrate do not currently justify the cost of this plant.

The final flotation tailings will be pumped to the backfill paste plant, where they will be thickened and filtered to the required moisture level. The filtered tailings will either be mixed with binding materials to produce backfill paste or stored in the backfill plant and later reclaimed for the same purpose. Backfill paste system utilization is projected at approximately 50%.

Figure 17.1 Simplified process flowsheet



17.3 Major design criteria

Major processing design criteria are outlined in Table 17.1.

Table 17.1 Major Processing Design Criteria

Criteria	Unit	Value
Annual Rate	t/a	470,000
Operating Days per Year	d	345
Availability/Utilization - Crushing	%	67
DMS Plant	%	92
Grinding and Flotation	%	92
Paste Plant	%	50
Nominal Rate - Crushing	t/h	87
DMS Plant	t/h	58
Milling and Flotation Rate	t/h	43
Paste Plant	t/h	53
Crushing Feed Size, 100% Passing	mm	300
Crushing Product Size, 80% Passing	mm	10
Ball Mill Product Size, 80% Passing	µm	80
Ball Mill Circulating Load	%	250
Bond Ball Mill Work Index, Design	kWh/t	12.5
Bond Abrasion Index	g	0.78
ROM Head Grades Pb	%	8.3
ROM Head Grades Zn	%	8.9
ROM Head Grades Ag	g/t	130
Metal Recovery Method	-	DMS + Sequential Flotation
Mass Recovery - DMS	%	74.0
Lead Recovery to Sulphide Lead Concentrate	%	79.1
Sulphide Lead Concentrate Grade	%, Pb	67.2
Lead Recovery to Oxide Lead Concentrate	%	9.2
Oxide Lead Concentrate Grade	%, Pb	48.0
Silver Recovery to Lead Concentrate	% Ag	73.5
Zinc Recovery to Sulphide Zinc Concentrate	%	82.8
Sulphide Zinc Concentrate Grade	%, Zn	58.7
Silver Recovery To Zinc Concentrate	**% Ag**	**13.5**

17.4 Process plant description

The proposed processing plant will consist of crushing, DMS pre-concentration, grinding, sulphide and oxide mineral sequential flotation, concentrate dewatering, and tailings dewatering/paste preparation units.

17.4.1 Crushing

The existing crushing circuits, consisting of a primary crushing unit and a secondary crushing unit in a closed circuit with a vibrating screen, will be refurbished for the project. ROM ore will be reduced to a particle size of 80% passing 10 mm.

Major equipment and facilities in this area include:

- ROM ore dump pocket, 40-st live capacity, with a fixed grizzly and a vibrating feeder
- Coarse ore surge bin (150 st), with an apron feeder fitted with grizzly bars

- Kue-Ken 36" x 24" jaw crusher
- Secondary crushing feed surge bin (50 st), with a belt feeder
- Double deck screen, apertures of 25 mm and 15 mm
- Symons Nordberg 5.5' cone crusher
- Conveyors, including a metal detector and a magnetic separator
- Fine ore bin (2,000 st), with a reversible belt feeder
- Dust collection systems

17.4.2 DMS plant

A DMS plant is envisaged to reject gangue material and to increase feed grades to the grinding and flotation circuits. The crushed ore will be processed by a de-sliming washing (material finer than 1.4 mm removed), followed by a dense media (ferrosilicon) cyclone separation at a proposed separation specific gravity of 2.8.

Major equipment and facilities in this area will include (all new):

- De-sliming screen, apertures of 1.4 mm
- Heavy media cyclone
- Two sieve bends
- Two 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 conveyed to the ball mill feed surge bin, from where it will be sent to the grinding circuit at a controlled rate. A bypass system ahead of the DMS circuit to reduce the impact of any system interruptions is included in the design. The bypass system will direct the ore from the fine ore bin directly to the ball mill feed surge bin in the event of any malfunctions or maintenance of the DMS system.

The DMS plant will be installed in a new pre-engineered building adjacent to the existing mill building.

17.4.3 Grinding and classification

The grinding circuit will consist of a conventional ball mill in closed circuit with classifying hydrocyclones. The existing 10’ by 11’ wet ball mill, driven by a 700-hp motor, will be refurbished and put into service.

Major equipment and facilities in this area include:

- 10' x 11' wet ball mill, with a 700-hp motor
- Classifying hydrocyclone pack
- Ball mill discharge pump box
- Two hydrocyclone feed pumps, one for operation and one on standby
- Particle size analyzer (new)
- Ball mill feed conveyor
- Accessories, including a steel ball storage bin and a ball bucket

17.4.4 Flotation concentration

Sequential flotation will be employed to separate lead and zinc sulphides and oxides. Three flotation circuits have been designed, in order: sulphide lead, sulphide zinc, and oxide lead flotation. A flotation plant is partly installed. The existing flotation cells will be refurbished, reconfigured or replaced as necessary.

Lead sulphide flotation

The lead sulphide flotation circuit will consist of rougher and scavenger flotation, cleaner and scalper flotation on the scavenger concentrate, and three stages of cleaner flotation. Four products will be produced from the lead sulphide flotation circuit, namely:

- Final lead sulphide concentrate, to be sent to the lead dewatering thickener
- Scavenger tailings feeding to the zinc sulphide flotation circuit
- Scalper flotation tailings feeding to the zinc sulphide flotation circuit
- Scalper flotation concentrate to go back to the grinding circuit

Major equipment proposed for lead sulphide flotation will include:

- Two rougher conditioning tanks, each equipped with one mechanical agitator
- Two rougher flotation mechanical cells, each with a volume of 5.1 m^3
- Three scavenger flotation mechanical cells, each with a volume of 5.1 m^3
- Two scavenger cleaner flotation mechanical cells, each with a volume of 5.1 m^3
- One scavenger scalper flotation mechanical cell, with a volume of 5.1 m^3
- One 1st cleaner conditioning tank, equipped with one mechanical agitator
- Three 1st cleaner flotation mechanical cells, each with a volume of 5.1 m^3
- Three 2nd cleaner flotation mechanical cells, each with a volume of 2.8 m^3
- Three 3rd cleaner flotation mechanical cells, each with a volume of 2.8 m^3
- Accessory equipment, including pump boxes, pumps, and samplers

Zinc sulphide flotation

The zinc sulphide flotation circuit will consist of rougher/scavenger flotation followed by three stages of cleaner flotation. Three products will be produced, namely:

- Final zinc sulphide concentrate, to be sent to the zinc dewatering thickener
- Scavenger tailings feeding the lead oxide flotation circuit
- Zinc sulphide cleaner scavenger tailings feeding the lead oxide flotation circuit

Major equipment and facilities proposed for the zinc sulphide flotation circuit include:

- Two rougher conditioning tanks, each equipped with one mechanical agitator
- Four rougher flotation mechanical cells, each with a volume of 5.1 m^3
- Two rougher scavenger flotation mechanical cells, each with a volume of 5.1 m^3
- One 1st cleaner conditioning tank, equipped with a mechanical agitator
- Four 1st cleaner flotation mechanical cells, each with a volume of 5.1 m^3
- One 1st cleaner scavenger flotation mechanical cell, with a volume of 5.1 m^3
- Three 2nd cleaner flotation mechanical cells, each with a volume of 5.1 m^3
- Two 3rd cleaner flotation mechanical cells, each with a volume of 5.1 m^3
- Accessory equipment, including pump boxes, pumps, and samplers

Lead oxide flotation

The lead oxide flotation circuit will consist of one stage of rougher/scavenger flotation and two stages of cleaner flotation. Two products will be produced:

- Final lead oxide concentrate, to be sent to the lead thickener for dewatering
- Oxide lead scavenger tailings as the final tailings to the backfill paste plant

Major equipment proposed for the lead oxide flotation includes:

- Two rougher conditioning tanks, each equipped with one mechanical agitator
- Four rougher flotation mechanical cells, each with a volume of 8.5 m^3
- One scavenger flotation mechanical cell with a volume of 8.5 m^3
- One 1st cleaner flotation column, 1.3 m diameter by 7.0 m high
- One 2nd cleaner flotation column, 0.85 m diameter by 6.0 m high
- Accessory equipment, including pump boxes, pumps, and samplers

17.4.5 Concentrate dewatering and load out systems

The three flotation concentrates from the flotation circuits will be dewatered to the designed moisture level via a combined dewatering process of thickening and pressure filtration. In the present configuration, lead oxide and sulphide concentrates will go to the lead thickener, resulting in a blended lead concentrate. A separate lead oxide concentrate would require an additional thickener. The filtered concentrates will be stored on site prior to being hauled off-site for shipping to smelters. The thickeners and pressure filters used for dewatering were installed in the 1980s; they will be refurbished prior to use.

Lead concentrate dewatering and load-out system

The lead sulphide and oxide concentrates will be pumped to the lead concentrate thickener where a solid density of approximately 65% w/w is expected to be achieved. An existing conventional thickener approximately 35' in diameter will be used for the dewatering. Flocculant will be added to the thickener to improve the settling process. The thickened concentrate will be pumped to a concentrate surge tank equipped with a mechanical agitator. The surge tank will be capable of holding the thickened concentrate for approximately eight hours to offset any minor maintenance required for the filter and load-out system.

The thickened concentrate will be further dewatered to a moisture level of 9% by a tower-type press filter. The dewatering filtrate will return to the lead concentrate thickener. The concentrate will then be sent to the lead concentrate stockpile prior to shipping, as noted above.

The lead concentrate thickener overflow will be collected in a dedicated process water tank and reused in the grinding circuit and lead sulphide flotation circuit.

Major equipment used in the lead concentrate dewatering circuits includes:

- Lead concentrate thickener, 35' (10.7 m) diameter
- Lead concentrate surge tank, equipped with a mechanical agitator
- 22 m^2 tower-type filter press
- Concentrate filter discharge conveyors
- Accessory equipment, including pump boxes and pumps

Zinc concentrate dewatering and load-out system

The zinc sulphide flotation concentrate will be pumped to a conventional thickener, where a solid density of approximately 60% w/w will be achieved. Flocculant will be added to the thickener to improve the settling process. Thickened concentrate will be pumped to a concentrate surge tank equipped with a mechanical agitator. The surge tank will be capable of holding the concentrate for approximately seven hours to offset any minor maintenance required for the filter and load-out system.

The thickener overflow will be collected in a dedicated process water tank and reused in the sulphide zinc and oxide lead flotation circuits, as well as in the paste plant. Any excess water will be discharged into the process water reservoir, where water will be pumped to the water treatment plant or reused as process water after the reagents in the water have degenerated naturally.

Thickened concentrate will be filtered in a tower-type press filter to a moisture content of approximately 9%. The dewatering filtrate will return to the zinc concentrate thickener. The filter cakes will be sent to the zinc concentrate stockpile prior to shipping as noted above.

- Major equipment and facilities in the zinc concentrate dewatering circuit include:
- Zinc concentrate thickener, 35' (10.7 m) diameter
- Zinc concentrate surge tank, equipped with a mechanical agitator
- 25 m^2 tower type filter press
- Concentrate conveyors
- Accessory equipment, including pump boxes and pumps

CZN reviewed various options for concentrate shipment, including bulk bags and boxes, and commissioned a transportation study by Allnorth, an engineering consultant. Both concentrates will be shipped off-site in bulk trucks and trailers. Trucks will be loaded by front-end loader and unloaded on a tipple at Fort Nelson for loading into rail cars. These will transport concentrates either to a smelter in Canada or to Vancouver for onward shipment by sea.

17.4.6 Tailings handling

Two tailings streams will be generated from the proposed processing plant: the rejects from the DMS plant and the rougher/scavenger tailings from the lead oxide flotation circuit. The rejects from the DMS plant will be trucked to the waste rock storage facility. The flotation tailings will be pumped to the tailings thickener and then to the backfill plant to produce paste for backfilling.

The flotation tailings will be pumped to a high-rate tailings thickener where a solids density of approximately 60% w/w will be achieved. Flocculant will be added to the thickener to facilitate the thickening process. The thickened tailings will be pumped to a surge tank equipped with an agitator. The design capacity of the surge tank is two hours of tailings production.

Tailings thickener overflow will be collected in the dedicated process water tank along with water from the zinc thickener. Major equipment in the tailings handling area includes:

- 7.5 m diameter high-rate tailings thickener
- Tailings surge tank, equipped with an agitator
- Accessory equipment, including pump boxes and pumps

17.4.7 Tailings paste plant

There will be a new paste plant and paste delivery system. This has been discussed in Section 16.5.2.

17.4.8 Reagent preparation and delivery

Various chemical reagents will be added to the flotation circuits to facilitate mineral recovery. Specific reagent requirements for the Prairie Creek processes have been identified, along with packaging envisaged 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
- Day holding tank
- Metering pumps

Liquid reagents will be diluted prior to delivery to the flotation circuits or pumped by metering pumps directly to the flotation circuits without dilution.

To ensure containment in the event of a spill, the reagent preparation and storage facility will be located within a containment area designed to accommodate 110% of the contents of the largest tank. Each reagent will also be prepared in its own bounded area in order to limit spillage and facilitate its return to its respective mixing tank.

Storage tanks will be equipped with level indicators and instrumentation to ensure that spills do not occur during normal operation. 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.

17.4.9 Assay and metallurgical laboratory

A stand-alone assay laboratory with HVAC and safety stations will be constructed. It will be equipped to conduct all routine assays for the mine, concentrator and environmental department. Key instruments will include:

- Microwave plasma-atomic emission spectrometer (MP-AES)
- Two graphite atomic absorption spectrophotometers (AAS) or equivalent
- X-ray fluorescence spectrometer (XRF)
- UV/VIS spectrophotometer
- This laboratory will be housed in a free-standing pre-engineered building remote from vibration and dust caused by mill machinery.

A separate metallurgical laboratory in the mill will undertake test work to monitor metallurgical performance and facilitate improvement of process unit operations and efficiencies. It will be equipped with equipment that is relatively insensitive to vibration and dust, including laboratory crushers, ball and rod mills, particle size analysis sieves, flotation cells, filtering devices, balances, and pH meters.

17.4.10 Mill water supply and distribution

Both fresh and process water will be required in the operation. Two separate water supply systems for fresh and process water have been designed for the processing plant.

Fresh water

Fresh water will be supplied from on-site boreholes or the water treatment plant to a fresh/fire water storage tank. Fresh water will be used primarily for:

- Fire water for emergency use
- Cooling water for mill motors and mill lubrication systems
- Gland services for the slurry pumps
- Reagent make-up

The fresh/fire water tank will be equipped with a standpipe that will ensure that the tank is always holding at least a two-hour fire water supply.

Potable water will be from an on-site potable generation system. The water will be bottled and delivered to various service points.

Process water

As described in Sections 17.4.5 and 17.4.6, thickener overflows from the concentrate and tailings thickeners will be re-used in the process circuits. Water reclaimed from the mine water reservoir will also be pumped to the process water tank that will be used for the DMS and sulphide lead flotation circuits.

Excess water from the mine and process water storage reservoirs will be separately pumped to, and treated at, a new water treatment plant to meet discharge criteria before being released to the environment or reused in the process circuits.

17.4.11 Compressed air supply

Compressed air will be supplied for various applications, such as general plant use, instrumentation, pressure filters in the concentrate and tailings filtration circuits, and flotation cells.

Two plant air compressors, one in operation and one on standby, will provide high-pressure air for general plant use, flotation columns, pressure filters and instrumentation. For instrumentation, an air dryer will be installed to remove any contained moisture. The dried compressed air will be stored separately in a dedicated air receiver.

Air for flotation cells will be at low pressure and will be provided by two blowers.

17.4.12 On-stream sample analyzers

For process control, the processing plant will be equipped with an on-stream analyzer capable of analyzing 24 process slurry streams. Shift samples will also be taken for metallurgical accounting purposes. These samples will include feeds to the 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 performance.

17.5 Process plant instrumentation and controls

17.5.1 Plant control

A programmable logic controller (PLC)-based main control system will provide equipment interlocking, process monitoring and supervisory control functions for the entire process plant. 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 by the control system. Operator interface to the plant control system will be via PC-based operator workstations.

The process plant control room will be staffed by trained personnel 24 hours per day. Operator work stations will be capable of monitoring the entire plant site process operations, viewing alarms, and controlling plant equipment. Supervisory work stations will be located in the offices of the plant superintendent and mill maintenance superintendent.

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

Field instruments will be microprocessor-based “smart” devices, grouped by process area and wired to local field instrument junction boxes. Signal trunk cables will connect field instrument junction boxes to control system I/O cabinets located in area electrical rooms.

New/refurbished 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.

17.5.2 Control philosophy

To control and monitor all mill building processes, three PC work stations will be installed in the central control room. 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
- Vendors' instrumentation packages

An automatic sampling system will collect samples from various process streams for online analysis and daily metallurgical balance accounting.

17.6 Annual production estimate

Annual metal production shown in Table 17.2 is projected from the mine production schedule shown in Section 16 and the metallurgical performance outlined in Section 13. For the LOM annual average, the process plant is estimated to produce approximately 54,000 t of lead concentrate with an average grade of 64.5% lead and 60,000 t of zinc concentrate with an average grade of 58.7% zinc. Average silver grades are projected at approximately 820 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 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 0.3% for arsenic, 0.3 to 0.8% for antimony and 0.01 to 0.04% for mercury. For zinc concentrate, on a yearly basis, average mercury contents are estimated to fluctuate from 0.02 to 0.30%. 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	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	Total/LOM Average
Mill Feed Tonnage/Grade																	
Tonnage, kt	423	458	471	466	467	446	488	515	510	495	528	565	493	467	433	378	7,604
Grade																	
- Ag, g/t	162	177	179	173	169	180	145	151	112	105	122	95	90	53	66	60	128
- Pb, %	9.6	10.7	10.3	10.9	10.9	11.6	10.0	9.8	7.9	6.8	8.0	7.3	7.6	4.1	3.5	3.5	8.3
- Zn, %	10.1	13.0	10.7	9.0	10.0	11.0	9.0	10.6	8.3	8.6	7.1	6.9	7.8	7.7	7.3	6.1	8.9
- Pb(Oxide), %	2.3	1.9	2.2	1.2	1.2	1.1	1.1	1.0	0.9	0.8	1.5	1.3	0.7	0.5	0.3	0.3	1.1
- Zn(Oxide), %	2.5	2.2	2.0	0.7	0.8	0.7	0.7	0.5	0.4	0.6	0.9	0.6	0.2	0.1	0.1	0.2	0.8
Lead Concentrate Tonnage/Grade/Recovery																	
Tonnage, kt	54	66	65	70	69	71	66	69	56	48	58	55	50	25	24	21	866
Grade																	
- Pb, %	63.3	64.8	63.8	66.2	66.2	66.4	66.3	66.0	64.7	62.8	63.5	64.7	66.4	62.3	55.5	55.5	64.5
- Zn, %	7.5	7.5	7.4	7.4	7.4	7.4	7.4	7.4	7.6	7.7	7.4	7.5	7.4	7.8	8.4	6.8	7.5
- Ag, g/t	927	917	960	879	863	864	805	855	753	770	810	724	665	664	713	651	824
Recovery																	
- Pb, %	84.5	87.5	85.8	90.5	90.3	91.2	89.9	89.9	89.5	89.1	86.5	85.7	89.2	83.2	86.6	86.6	88.3
- Ag, %	73.6	74.8	74.2	75.8	75.7	76.0	75.5	75.2	73.1	70.4	73.0	74.1	75.0	67.5	59.5	59.5	73.5
Zinc Concentrate Tonnage/Grade/Recovery																	
Tonnage, kt	50	77	63	60	67	72	63	81	62	63	50	55	58	54	48	35	958
Grade																	
- Pb, %	4.0	4.1	4.1	4.2	4.2	4.3	4.1	4.0	3.2	2.6	3.3	3.5	3.4	2.5	1.0	1.0	3.5
- Zn, %	59.0	59.0	59.0	59.0	59.0	59.0	59.0	58.9	58.5	58.0	58.6	59.0	59.0	58.4	57.0	57.0	58.7
- Ag, g/t	158	174	174	158	159	165	143	157	125	131	121	79	76	83	119	119	137
Recovery																	
- Zn, %	68.3	76.7	74.5	84.3	84.2	86.2	84.6	87.5	86.1	85.1	79.3	83.1	89.5	88.7	86.2	86.0	82.8
- Ag, %	11.4	16.6	13.1	11.7	13.4	14.7	12.7	16.3	13.6	15.7	9.5	8.0	10.0	18.0	20.1	18.3	13.5

17.7 Stockpile

A stockpile of approximately 10,000t currently exists outside the mill building. Initial ore from vein development will be added to this pile until the mill is ready to handle it. During the life of the mine, the stockpile will be used to even-out mine production against mill capacity.

18 Project infrastructure

The Prairie Creek Mine is a remote, isolated site, with infrastructure that Canadian Zinc must upgrade, expand or replace where necessary. Figure 18.1 shows a recent (2015) photograph of the site.

Figure 18.1 The present-day Prairie Creek Mine site infrastructure

The mine site lies on the flood plain of Prairie Creek. 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 never been 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

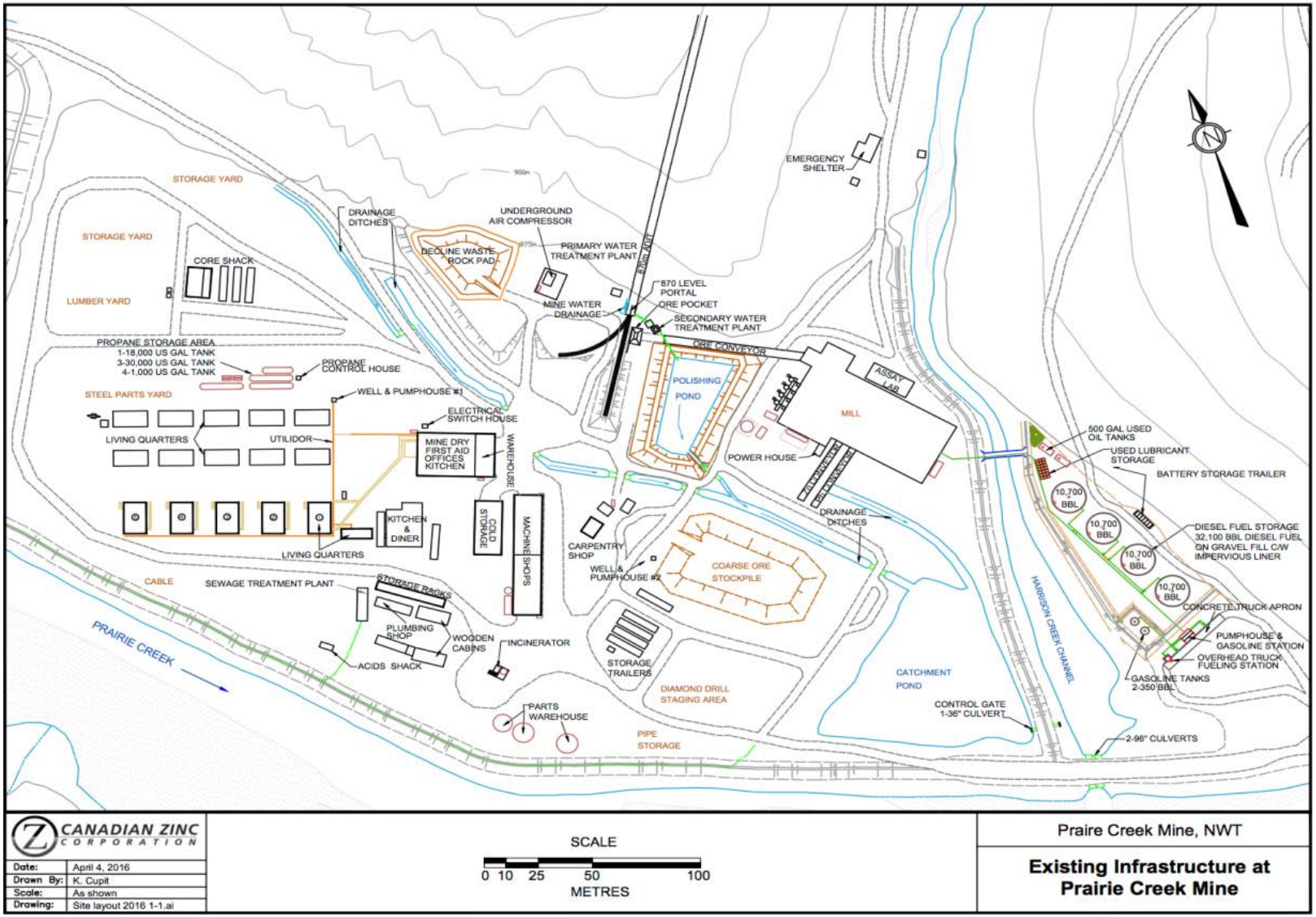
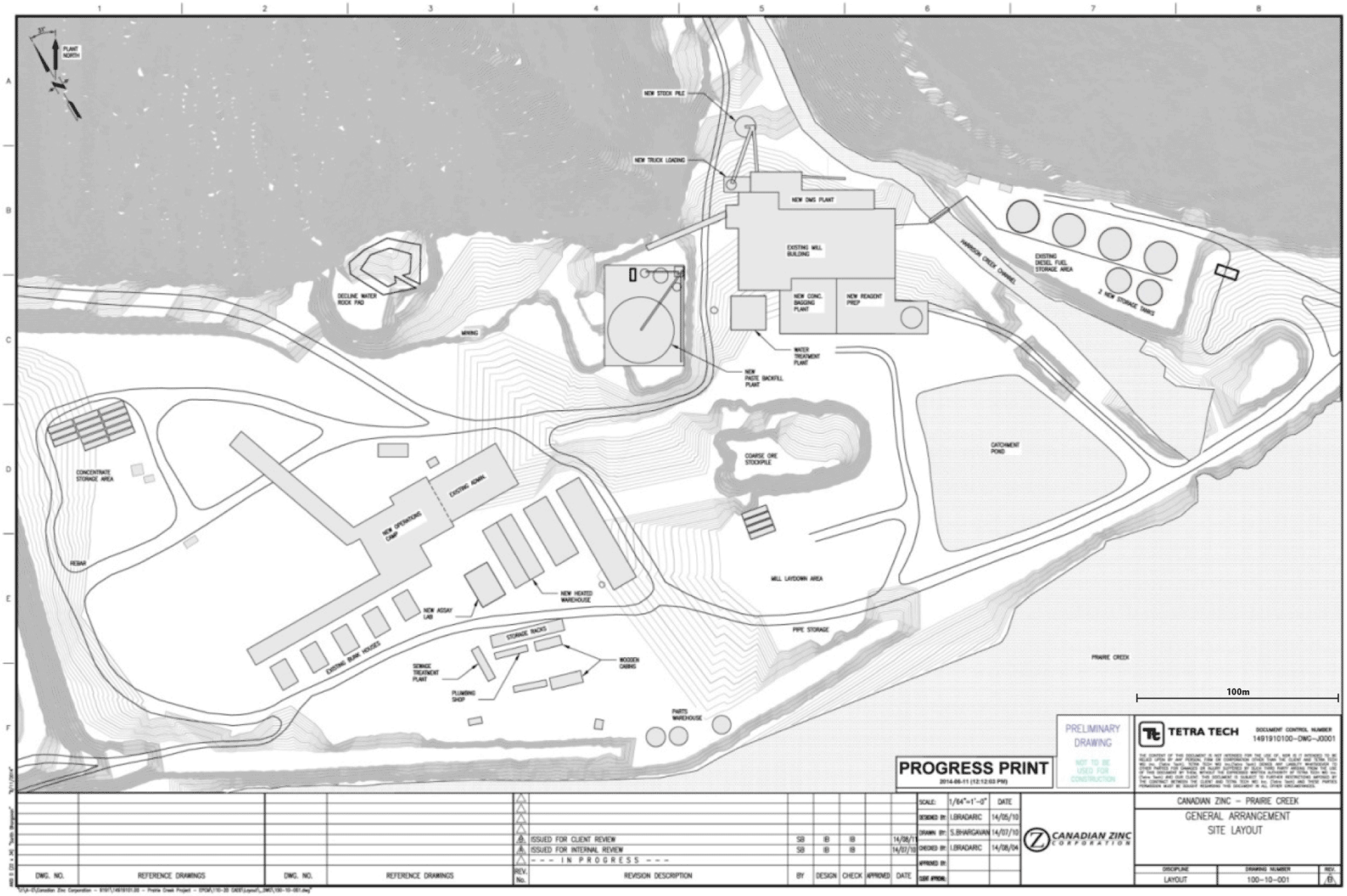


Figure 18.3 Proposed general arrangement of the modified Prairie Creek Mine site infrastructure



18.1 Camp

The current camp accommodation is adequate for up to 50 people, with cookhouse facilities in the administration building. The former owners installed trailer accommodation and cookhouse facilities for a further 100 people, which are currently in a state of disrepair and will be demolished.

CZN plans to install a new, self-contained, modular camp with accommodation for 150 people. The current 50-person accommodation (the five trailer units shown in the foreground of Figure 18.4), will be used for short-stay and overspill accommodation.

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. This well was tested in July 2014 and showed no draw-down after four hours at a pumping rate of 46 litres per minute. This will suffice to supply 300 litres per person per day to 200 people. CZN will install water treatment for domestic water.

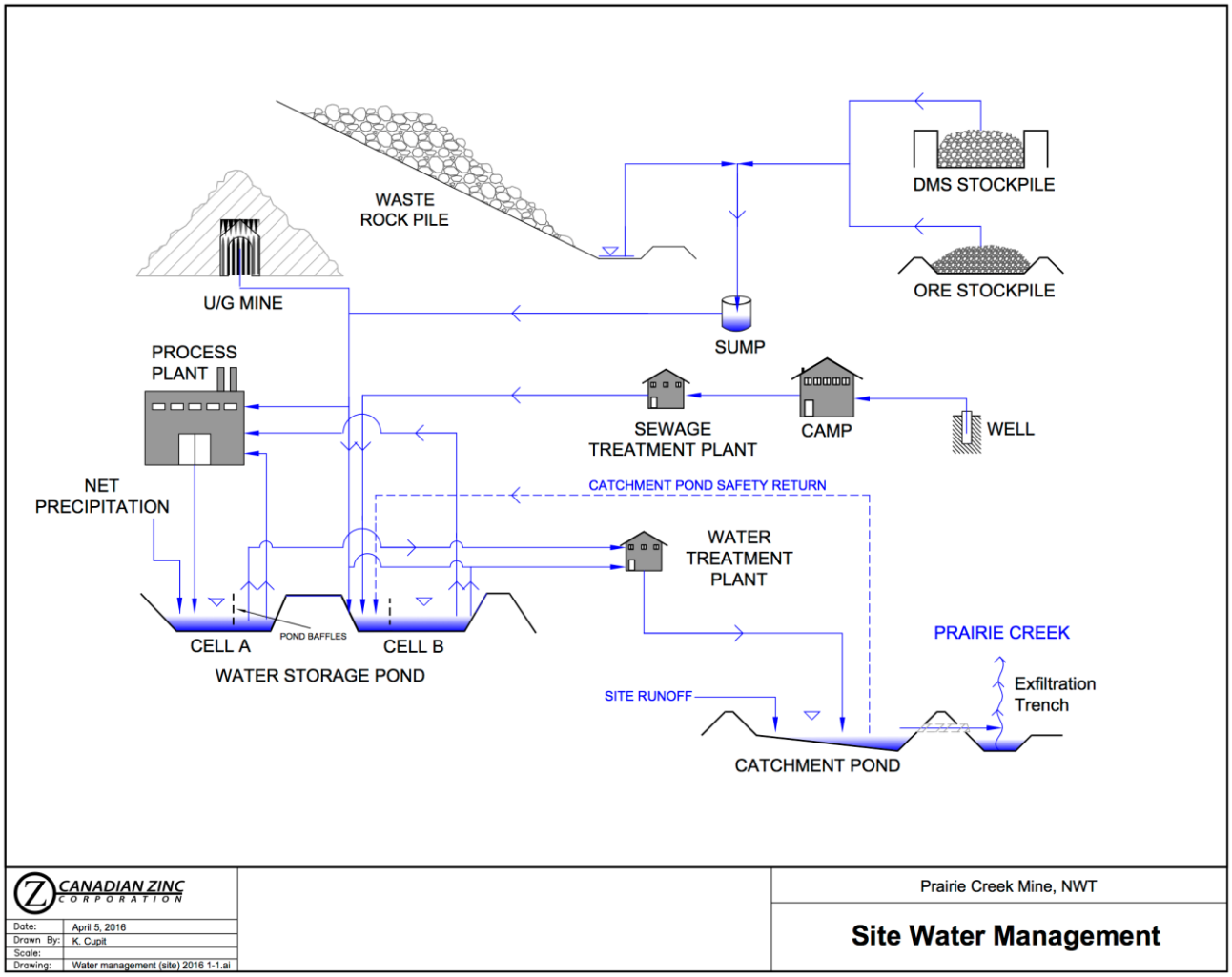
18.2.2 Site water management

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 grey water. All site waters will report to the new water treatment plant if there is a need for treatment, prior to discharge to the environment.

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 would be converted into an area for the new paste backfill plant.

Figure 18.5 Prairie Creek proposed new site water balance



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, but at a much lower volume (~20 L/sec) than mine water (seasonally ~15 - 100 L/sec).

A large pond was originally built on site with dykes and a clay lining, 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, one for mine water and similar site drainage, and the other for mill process water. 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.

During operations, a 65% process water/35% mine water blend will supply feed water to the mill. Process water will be fed from the WSP after aging. Mine water will be fed from both the Mine and WSP. CZN plans to intercept most groundwater before it flows into the Mine, thus preventing its contamination with oil, mud and blasting residues (refer to Section 16.11). This non-contact water will be sent to the WSP. Water that flows into the Mine and becomes contaminated will be sent directly to the Mill as feed water, with any excess being sent to the Water Treatment Plant (WTP). 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. Excess process water will also be treated in the WTP, but in a separate circuit, although there would be no treatment in winter (January-March) when it can be held in storage in the WSP. All water, treated and untreated, will be released to Prairie Creek according to 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’.

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 staff are listed in the organization chart. They 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 acccommodated 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 950GB 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 Company's existing C-Band satellite dish may also be set up for redundant fail-over communications.

18.5 Administration building

The original owner installed a large, two storey steel building to house the Mine dry, a warehouse and offices, known as the Administration Building (see Figure 18.6). This building has been maintained as the site operations base and will retain this function. The building will need some refurbishing to bring it up to current building codes. Offices and training rooms will be in this building.

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 in a poor state of repair. A new, 30 m x 15 m sprung-structure, fabric-covered warehouse will be erected on a concrete pad adjacent to the existing administration building. The existing cold storage building will be refurbished and will continue in that function.

18.7 Workshops

The existing heavy-equipment workshop will be refurbished and will be used for the maintenance of both surface and underground mobile equipment (see Figure 18.7). The mill will be self-sufficient for maintenance.

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.

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 Roosedahl Engineering Enterprises showed that only minor repairs are needed to restore all four fuel tanks to serviceable condition. The fuel farm containment system meets the required Environment Canada regulations. As the current CZN operating plan is for access by all-season road, two tanks will suffice for ongoing site fuel storage needs.

18.10 Sewage treatment

The existing Sewage Treatment Plant (STP) is a secondary-level, extended aeration treatment plant – see Figure 18.10; this 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. The effluent will be pumped to the Water Storage Pond. 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 (BOD5) 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 BOD5 : <20 mg/L, and TSS: <20 mg/L.

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.

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. CZN has determined that these would be inefficient and are beyond reasonable repair and has tendered for a replacement to provide 6 Megawatts at 4,160 Volts, comprising four 1.5 Megawatt diesel generators with a fifth generator on standby. Existing power generation capability on site, totalling 1.15 MW, will supplement the 6 MW system.

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 mill building and provide mine air heating.

The site electrical system has been designed to PFS level. 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 major power cables underground on the 883L are in acceptable condition. The contractor that worked on site in 2014-15 replaced the two sub-stations and re-established the underground electrical system. A power cable runs up the south service raise from the 883L to the 930L and was used in 2014-15 to power a 75-horsepower fan at the 930L portal.

Temporary electric power for the mine will be provided by diesel generators, supplied by the contractor, on surface near the 883L portal. CZN will replace these generators with a feed from the new diesel generators when they become available.

Electric power underground will be used for 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 site completion contractor will be responsible for construction power.

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 and kilowatt-hours per year.

Figures 18.11 and 18.12 are single-line electrical drawings for the site.

Table 18.1 LOM average power demand

LIFE OF MINE AVERAGE ELECTRICAL LOAD				
Area	**Connected Load (kW)**	**Running Load (kW)**	**Demand load (kVA)**	**Annual kWh**
Mine				
Ventilation	616	610	610	5,343,000
Dewatering	1,218	250	845	2,166,000
Mobile Electric-Hydraulic Eqpt	505	100	364	864,000
Infrastructure	658	240	452	2,137,000
Sub-Total Mine	**2,997**	**1,200**	**2,271**	**10,510,000**
Mill				
Mill Building	94	67	72	500,248
Crushing	405	244	257	2,136,853
DMS Plant	507	374	394	3,277,502
Grinding	606	534	562	4,673,345
Lead-Copper Flotation	350	280	294	2,449,528
Lead Oxide Flotation	202	144	152	1,261,276
Zinc Sulphide Flotation	324	190	199	1,659,634
Dewatering	187	131	137	1,143,626
Concentrate Handling	9	9	10	69,079
Paste/Backfill Plant	1,437	332	350	2,907,373
Reagent Handling	57	33	35	289,157
Metallurgical Laboratory	41	17	18	151,275
Tailings Thickening	34	17	18	149,339
Instrument Air	600	177	186	1,549,137
Fresh Water	149	104	110	554,642
Process Water	377	274	289	2,403,898
Sub-Total Mill	**5,379**	**2,927**	**3,083**	**25,175,912**
Plant				
Power Generation Service	183	87	92	760,363
Water Treatment	405	185	195	1,622,905
Administration Building	21	13	14	116,153
Camp	1,100	834	878	6,390,189
Sub-Total Plant	**1,709**	**1,119**	**1,179**	**8,889,610**
Total Site	**10,085**	**5,246**	**6,533**	**44,575,522**

"Connected Load" is the total of all electrical devices connected to a power supply.
"Running Load" is the long-term average power draw.
"Demand Load" is the load for which the generators may be required to provide power at any one time.

Figure 18.11 Overall single line electrical drawing – sheet 1

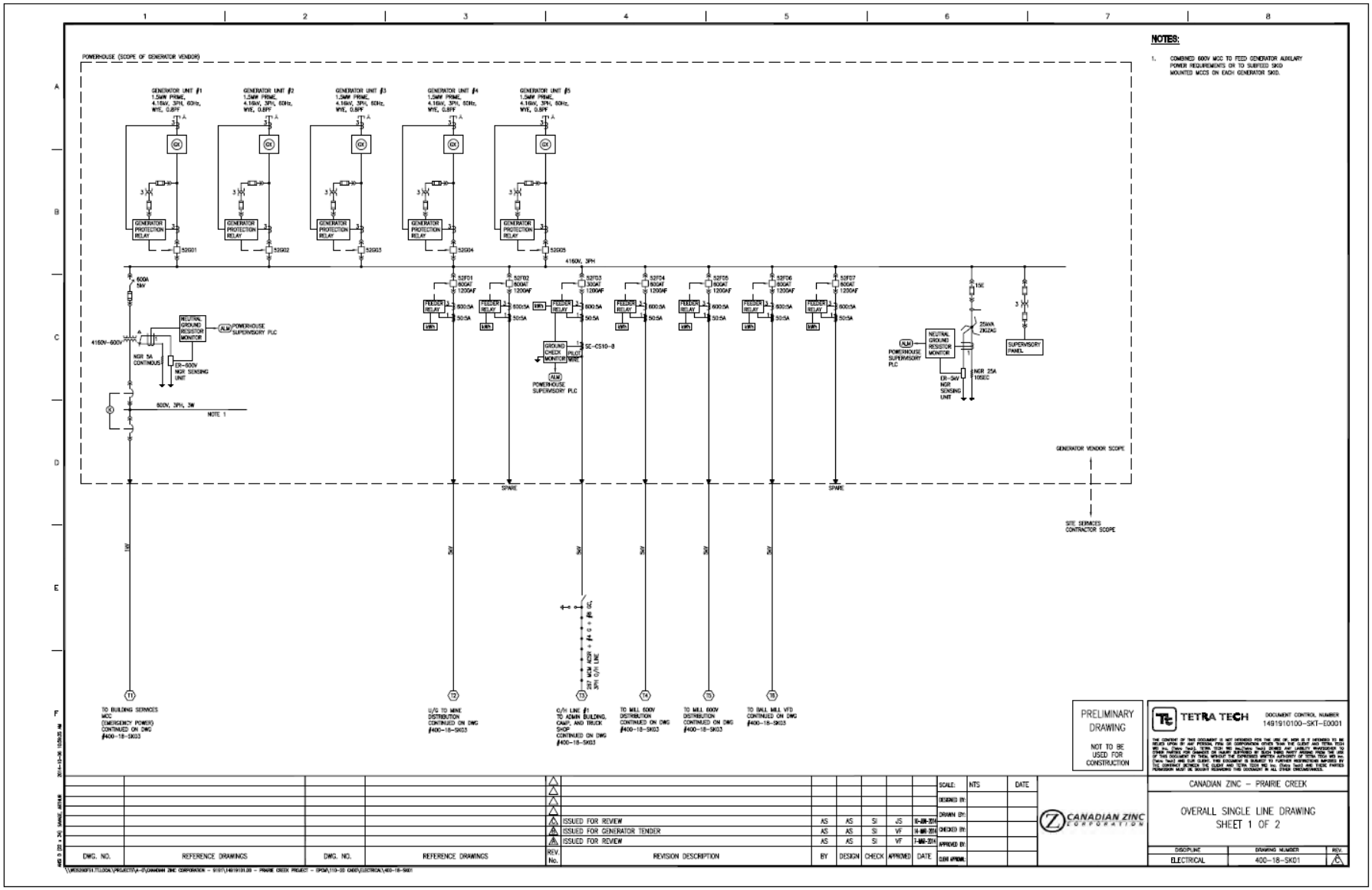
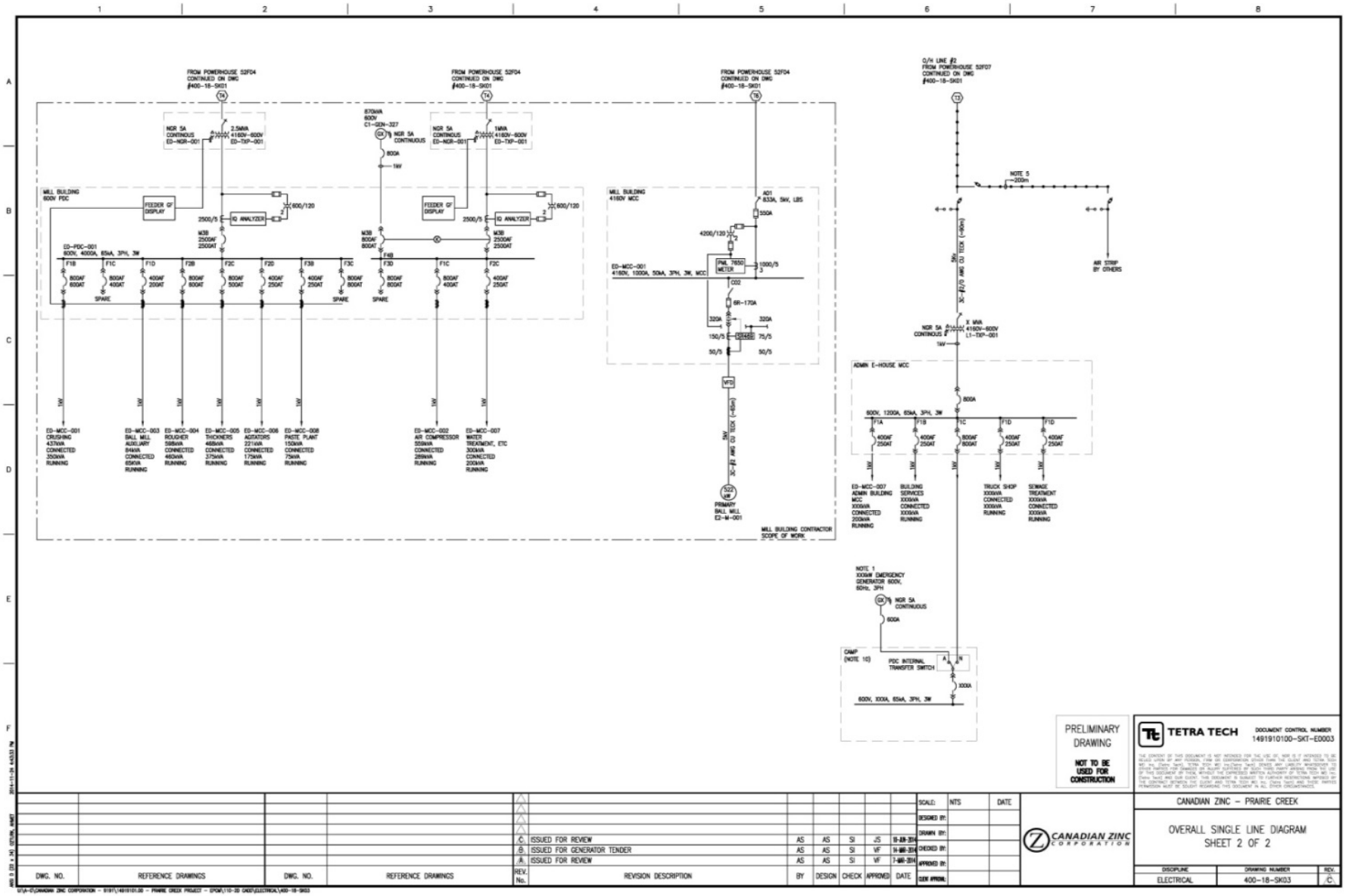


Figure 18.12 Overall single line electrical drawing – sheet 2



18.13 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.

18.14 Fire detection and suppression systems

The fire detection and suppression system is partly installed. The existing hardware will be reused to the extent possible to operate with a modern fire control panel. The existing main control panel is located in the administration building. This panel will be retained to serve the offices, and will be integrated with a new mill fire control panel. Each plant facility and arctic walkway will include a fire detection and/or suppression system appropriate to the application and use of the facility, and conforming to National Fire Protection Association requirements.

Water for fire suppression will be taken from a water ring main to be installed around the plant. The water main will be laid within buildings and utilidors. The fire suppression system will operate as a dry system whereby, on the demand of the fire detection system, the mains will be filled with water.

The fire detection and suppression facilities will be as follows:

- The offices are equipped with fire detection and sprinklers.
- Camp modules will be purchased with installed detection and suppression devices. These will be connected to the water main and the fire control panel.
- The assay lab will be equipped with installed detection and suppression devices. These will be connected to the water main and the fire control panel.
- Conveyors will be equipped with linear heat cable detectors and sprinklers around the drive ends. The enclosed conveyors will be furnished with sprinklers over the whole length of the belt.
- Local manual stations will be placed at various locations in the process building, with hose stands located appropriately.
- The power plant will include a high-fog system over the engines and the pipe hose stands within the building.
- CO_2 handheld extinguishers will be located throughout the buildings.

18.15 Water Treatment Plant

The Water Treatment Plant (WTP) will treat excess water in the Water Storage Pond (WSP) prior to discharge to the environment (refer to Figure 18.5). The WTP will consist of two treatment circuits, one for mine water and one for process water. The treated water streams will join for the final clarification step.

The design of the treatment plant was based upon test work by SGS-CEMI. The primary conclusions of the test work were as follows:

- Mine water treated with hydrated lime to a pH>9 will be sufficient to meet the effluent quality.
- Mill process water will require a more elaborate treatment strategy involving sulphide treatment at pH 5 followed by ferric treatment at pH 9.
- There is a coagulation benefit to mixing the two effluent streams prior to clarification in the reactor clarifier.

The mine water will be mixed with hydrated lime to a pH of 9.3 and held for 60 minutes. The treated water will then be pumped to a reactor clarifier for the addition of flocculant, Magnafloc E10 or equivalent, at a dosage rate of 0.5 mg/L to aid solids settling. Space has been reserved in the layout for two additional lime treatment tanks and an associated clarifier to increase the mine water treatment rate from 96 m^3/hr to 200 m^3/hr if required.

The mill process water will not meet the Metal Mining Effluent Regulations requirements with lime treatment alone, specifically for cadmium and lead, so it requires a different treatment strategy. Treatment will first involve the addition of 1N sulphuric acid to reduce the pH to 4.5. Sodium sulphide will then be added at a dosage rate of

40 mg/L and allowed to react for 30 minutes. Following this, 80 mg/L of ferric sulphate will be added to precipitate zinc and arsenic over a reaction time of 60 minutes, with hydrated lime then added to raise the pH to 9 with 120 minutes of reaction time. Magnafloc E10 will be added as flocculant at 2 mg/L prior to clarification.

Sludge from the clarifier will be collected. A portion may be recycled into the process to act as seed, the remainder will be transferred to the paste plant for blending in the underground backfill mix. The capacity of the process water treatment circuit will be 22 L/sec and any need for expansion is not anticipated at this stage.

The capacity of the mine water circuit has been initially set at 120 L/sec 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 double 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.

18.16 Explosives

LOM explosives storage and use has been previously discussed in Section 16.16. CZN has obtained WSCC approval to use the existing magazines on surface until such time as enlargement of the 883L adit has progressed 60 m past the entrance to the planned underground explosives magazine location, at which time the magazine can be constructed.

18.17 Mine services – compressed air and communications

Section 16.12 discusses the supply of compressed air to underground.

Section 16.17 discusses the communications strategy for underground.

18.18 Dewatering

The underground dewatering system has been described in Section 16.17.

18.19 Mine escape and rescue

Refer to Sections 16.10 and 24.1.2 for details on emergency preparedness and refuge stations.

18.20 Surface mobile equipment

CZN has a fleet of mobile equipment onsite (refer to Figure 18.13), a portion of which, upon refurbishing, would be capable of supporting operating requirements.

Figure 18.13 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

Item	Quantity
Pick-up Trucks	4
Fuel Truck	1
Flat Deck Truck fitted with a Hiab	1
Grader	1
Forklifts	2
D6 Dozer	1
Front End Loaders 966 Cat (typical)	2
Dump Truck	1
Mobile Crane 20 tonne	1
Ambulance	1
Fire Truck	1

18.21 Water storage pond

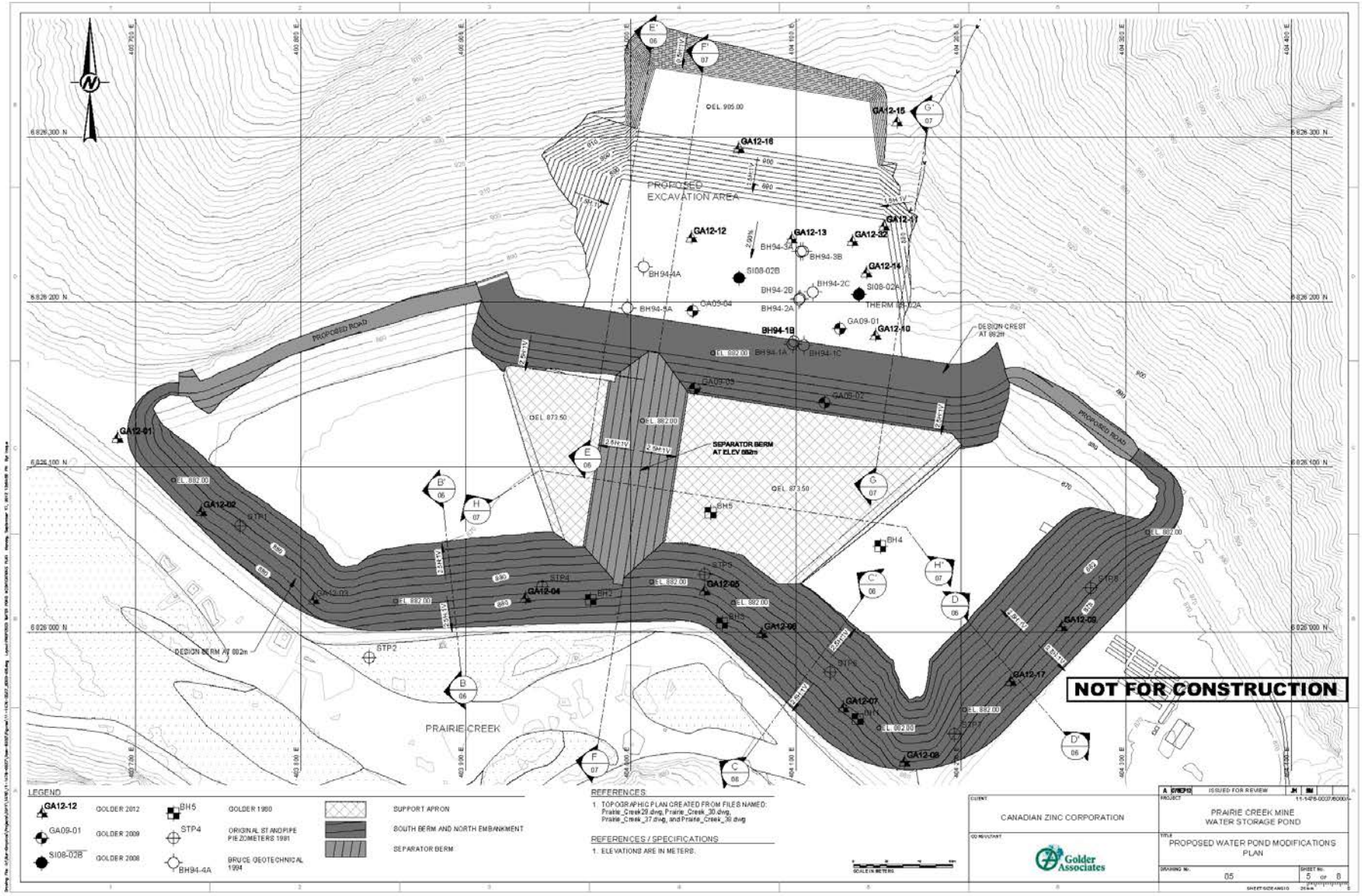
Three types of water are to be managed at the Prairie Creek Mine site: run-off from snowmelt and rainfall, water from the mine, and water from the mill. Runoff management will be essentially the same as at present using installed components, including ditches, routing all site runoff into a final pond (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 will be to Prairie Creek using an exfiltration trench, the details of which are discussed below.

The site is protected from flooding associated with Prairie Creek by the dyke of a large pond and a flood protection berm which is connected to it. Both structures are armoured with rip-rap. 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 a probable maximum flood.

The large pond on site (see Figure 18.14) 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 backslope slumped, probably due to a combination of permafrost thaw and slope movement along a weakness in an underlying clay layer. For operations, the plan is to convert the pond into a water storage pond (WSP), divided into two cells by the construction of a divider berm (see Figure 18.15). The divider berm will buttress the slumped section of the backslope.

Figure 18.14 Tailings impoundment facility - to be converted into a Water Storage Pond

Figure 18.15 Proposed Water Storage Pond layout



The WSP will be the key structure of CZN's water management plan. All contaminated water streams will be sent to the pond. The pond will supply water to the mill for processing, and to the water treatment plant (WTP) for treatment and discharge to the environment. Golder Associates has produced a design for the converted WSP. The design provides for two cells and remedial works to stabilize the backslope.

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 Golder Associates to undertake further analyses to optimize WSP reconstruction and increase storage capacity. Total storage capacity is planned to be increased from 590,000 m^3 to 666,000 m^3.

As described in Section 17, process water from the mill will be sent to Cell A, and all mine and runoff water to Cell B. Process water will be recycled to the mill after several months of residence time in the WSP to allow organic reagents to degrade, in a blend with mine water. 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.

Discharge from the site will consist of treated water from the WTP and natural site drainage during the open water 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.

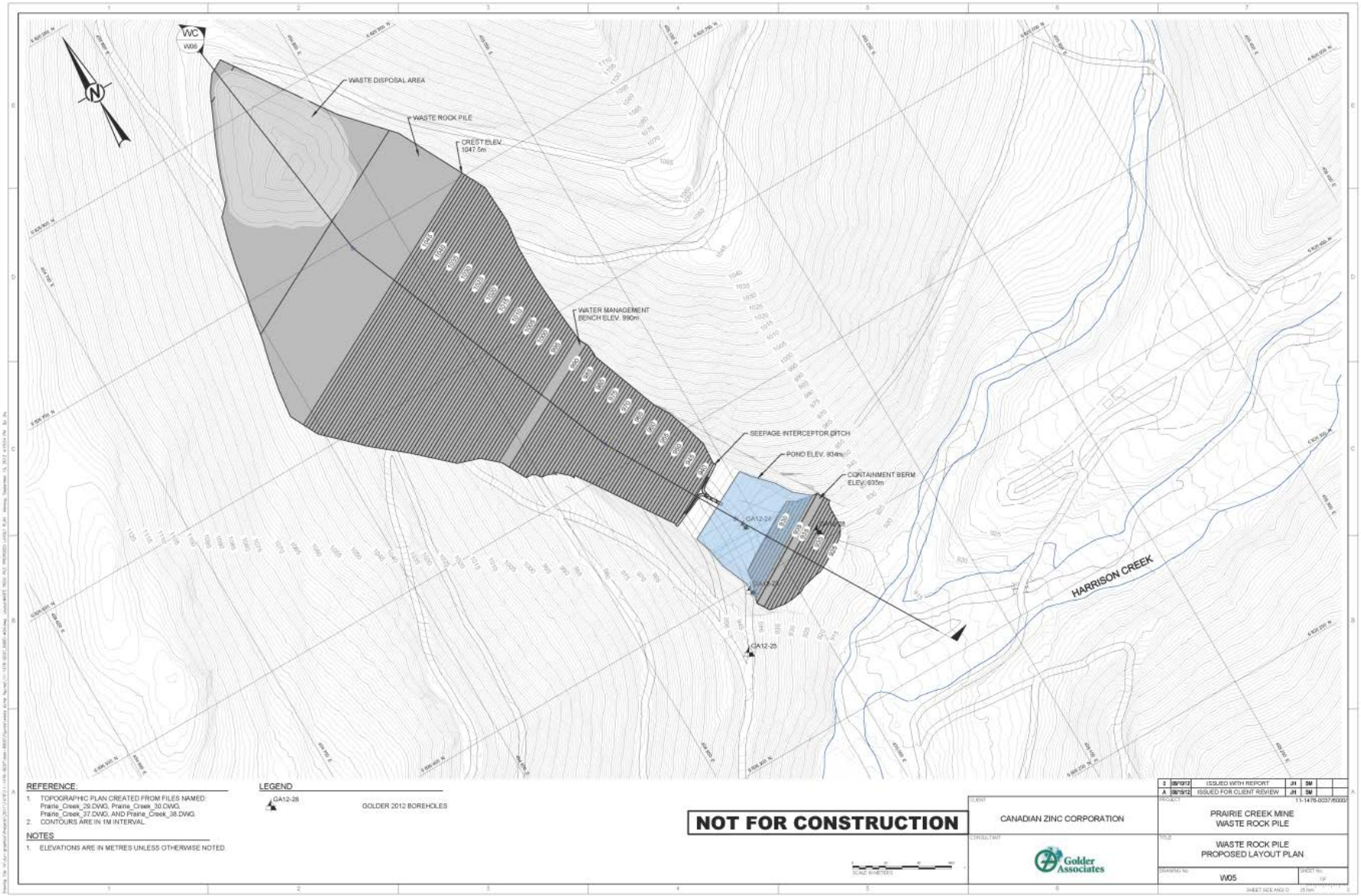
The exfiltration trench will contain two perforated pipes of different lengths. The longer pipe will be used in summer when the channel is wider, and the shorter pipe will be used in winter. Having two pipes also provides redundancy in the event that one pipe is unusable for any reason.

18.22 Waste rock pile

Waste rock will be stored in a ravine about 1 km north of the plant site. The site and deposition plan have been engineered by Golder Associates. Some preparatory earthworks will be needed, but the site is estimated to suffice for the storage of all waste rock and DMS float material for the life of the Mine. Figure 18.16 shows a photograph of the proposed site and Figure 18.17 is a plan view of the design.

Figure 18.16 Proposed site of Waste Rock Pile Storage facility.

Figure 18.17 Proposed Waste Rock Storage facility layout.



18.23 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.

18.24 Transportation

18.24.1 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 Waste Rock Pile would have to be upgraded but this could easily be done with the present equipment on site.

18.24.2 All season access road

The existing mine plant equipment 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, which was essentially grandfathered through the environmental assessment process having been previously assessed. 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 has applied for permits to construct an all season road, substantially following the winter road alignment as shown in Figure 18.18. CZN plans to tender a design-build contract to pre-qualified contractors.

Figure 18.18 Proposed route of the all season access road into Prairie Creek Mine

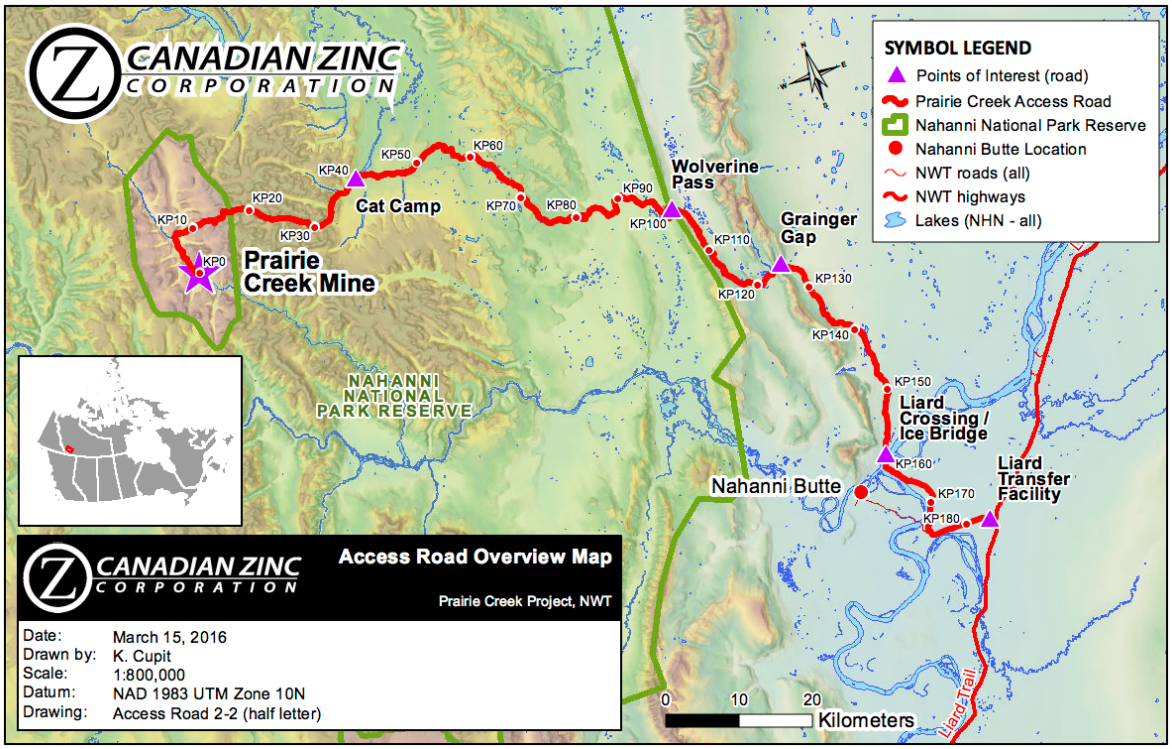
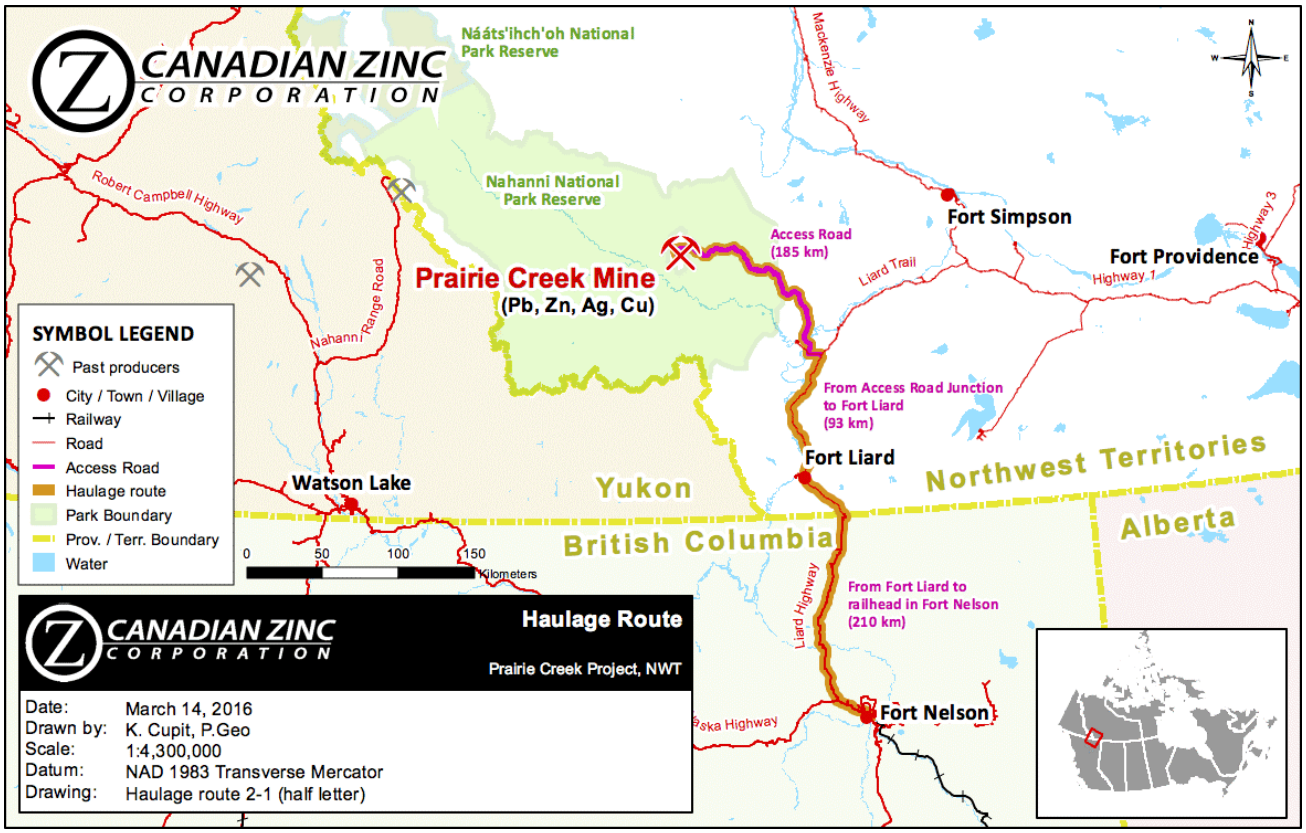


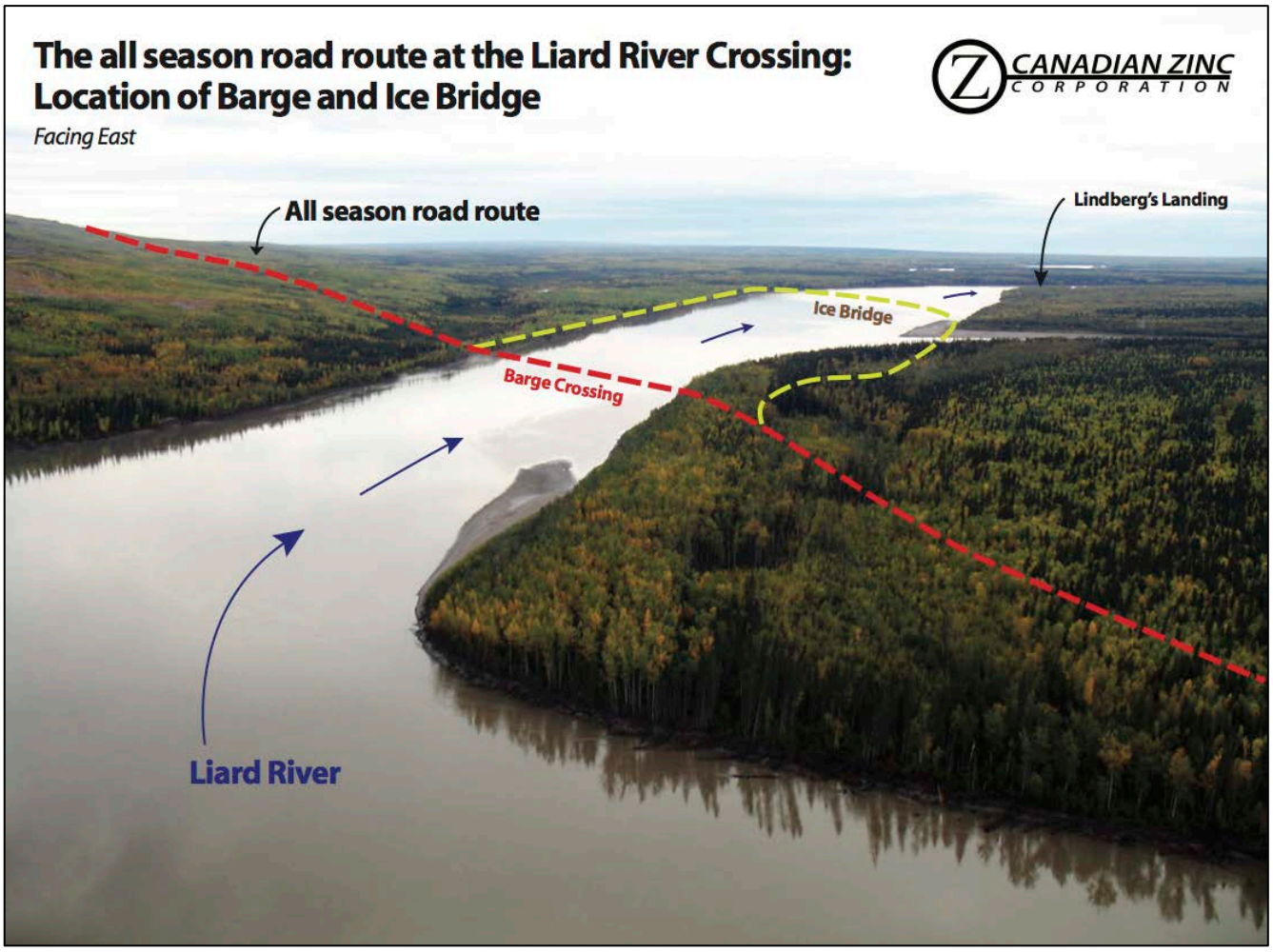
Figure 18.19 shows the total proposed trucking route from the Mine to Fort Nelson.

Figure 18.19 Proposed Trucking route Prairie Creek Mine to Fort Nelson



The Liard River Crossing is one of the greatest challenges and limitations to the transportation infrastructure. The 700 m crossing of the Liard River will utilize a barge during the summer hauling and an ice bridge during winter hauling (see Figure 18.20). Historical data collected for numerous years by the Government of the Northwest Territories Department of Transportation on adjacent similar ice bridge crossings at Nahanni Butte (5 km upstream) and Fort Simpson crossing (50 km downstream) provide valuable insight into the duration and capacity of the ice bridge. Hauling will not occur during freeze-up and break-up and the road will be temporarily closed. Capital and operating costs at the Liard Crossing by barge or ice bridge were estimated by Allnorth using information from experienced operators in the north who are operating similar crossings.

Figure 18.20 Proposed route of the all season access road at the Liard River Crossing



A staging area at the Liard Transfer Facility, near the junction of Highway 7 and the Nahanni Road, would consist 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 30 km north of the border in the Northwest Territories.

A transfer facility will be required at Fort Nelson to unload concentrate from trucks, store it, and then load onto gondola rail cars for delivery to the Port of Vancouver. The transfer facility is anticipated to include a rail spur with capacity for 50 cars. A suitably sized piece of land will have to purchased or leased in order to build this facility. The transfer facility layout concept incorporates a full covered facility with a truck “tilt-deck” unloading at one end, rail loading at the opposite end, and with a seven day concentrate storage capacity.

The current project master schedule envisages site completion over winter roads, followed by concentrate haulage over a new all-season road.

18.25 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.

18.25.1 Personnel movement

The site workforce will work on a regular fly-in-fly-out rotation to be determined. Each person will probably rotate to and from the site eight or nine times per year, 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 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 VFR conditions; some weather delays may be anticipated.

18.25.2 Inbound freight

CZN may construct one or more winter roads for initially moving construction freight to the site and to position equipment for the construction of the all-season road. Otherwise, CZN anticipates moving freight over the all-season road for the life of the mine. This road will be available for use seven to eight months of the year, interrupted by the freeze-up and break-up of the Liard River. In the winter, traffic will cross the river on an ice bridge; in summer, there will be a barge service. Allnorth estimated an operational window of 105 days for the ice bridge crossing and 115 days for operation of the barge service. The bulk of the inbound freight would consist of food, diesel fuel, equipment, spare parts, mining consumables, mill reagents and general supplies to support the operation.

The proposed truck configuration 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 50,000 litres/day during hauling operations.

18.25.3 Outbound concentrate

The 2012 Pre-Feasibility Study envisaged moving ~120,000 tonnes 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; the study did not investigate onward movement of concentrate beyond the CN rail head at Fort Nelson.

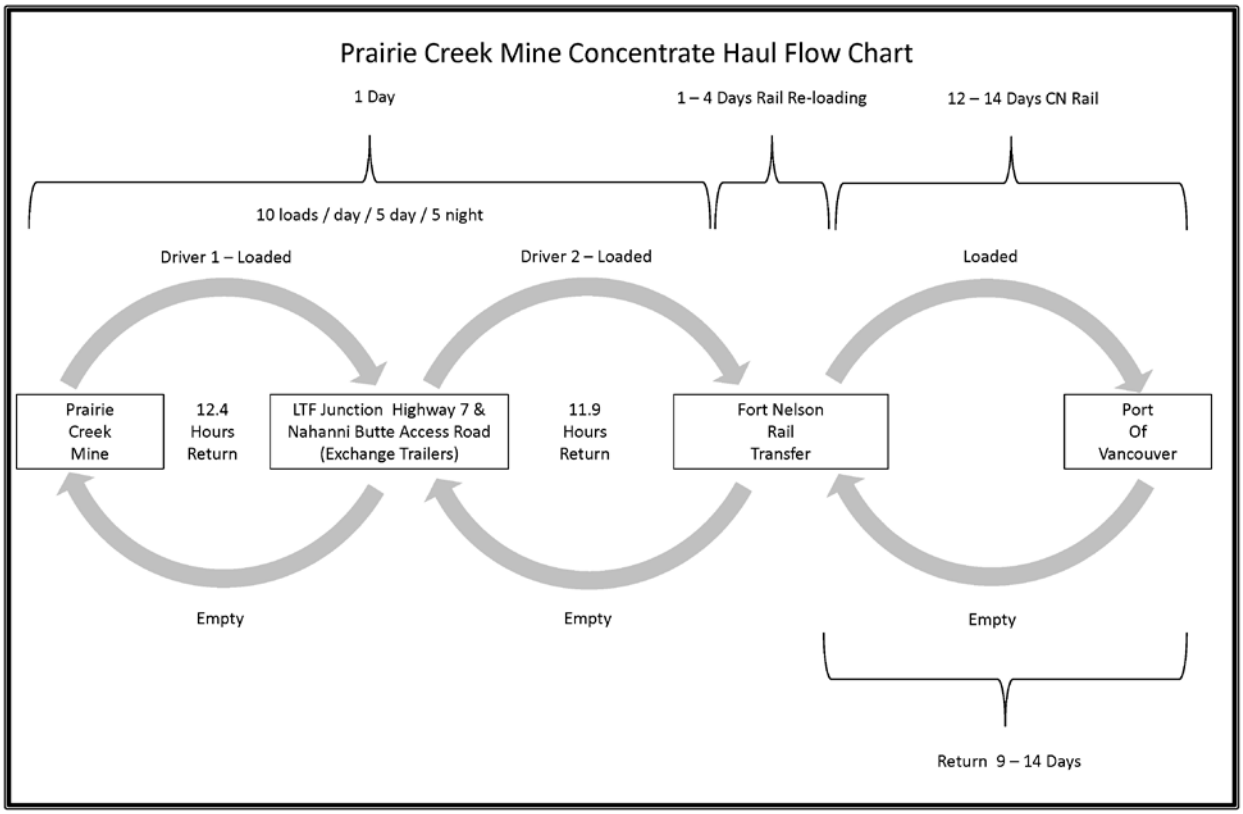
More detailed analysis showed that moving a year's concentrate production to market during the two to three months of winter road availability would impose significant costs for transfer facilities and rehandling as well as substantial funds tied up in concentrate inventory. The current operating plan envisages moving concentrate over an all season road, reducing concentrate storage to two two-month periods in late fall and late spring. CZN applied for an all season road Land Use Permit in 2014 and is presently in the Environmental Assessment process.

In 2015, CZN engaged Allnorth, an engineering consultant, to produce a comprehensive transportation plan with estimated capital and operating costs and to advance the design of the all-season road. The transportation plan included looking at all possible routes and decided that the most viable route was to truck to Fort Nelson and rail to Vancouver. The initial 180 km of road, on the new all season road, would be limited to active operations for 218 days of hauling during the year due to restrictions and limitations of spring and fall road bans and the Liard River crossing. It is proposed to utilize covered Tridem or Tandem Class 8 Super B-train with payloads of 41 or up to 51 tonnes per load depending on road conditions and permits. An 8% moisture factor has been added to convert dry metric tonnes to wet metric tonnes for the purpose of estimating transportation costs.

The trucks would deliver the concentrate to the Fort Nelson truck/rail Transfer Facility where it would be loaded onto rail cars and shipped to the Port of Vancouver. Canadian National Railways (CN) owns the rail line to Vancouver and is proposing gondola-style rail cars with a capacity of 90 tonnes. For projected average concentrate production, over 1,200 rail car deliveries will be made annually to the Port of Vancouver. This will need to be coordinated with the trucking schedule.

Figure 18.21 shows the conceptual concentrate haul flow sheet.

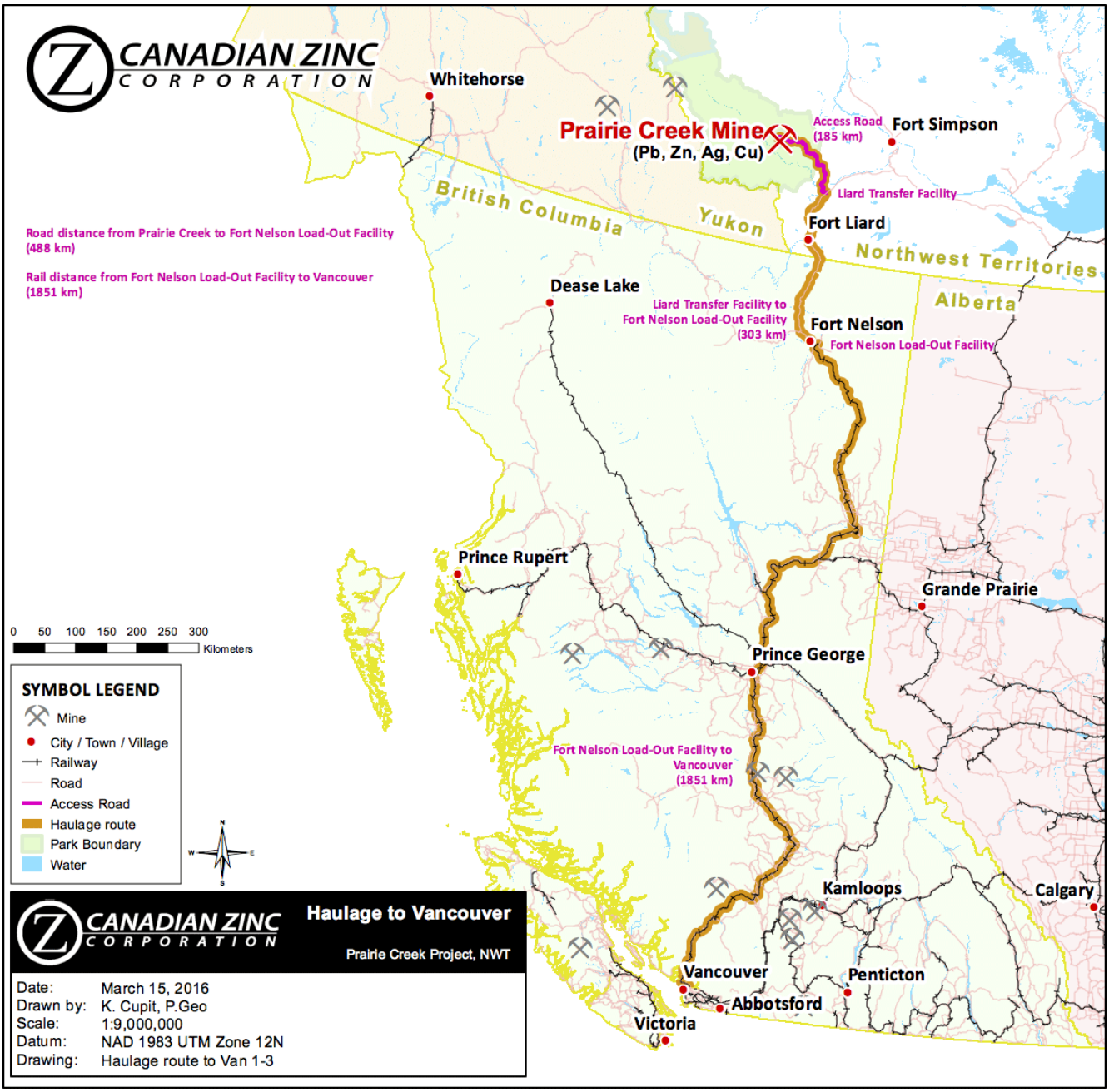
Figure 18.21 Conceptual Concentrate Haul Flow Chart



The total numbers of rail cars required will depend on the projected turn-around time provided by CN Rail. CN Rail serves Fort Nelson twice per week with an expected turnaround of cars ranging from 23 to 28 days. Given that six cars are anticipated to be loaded per day, a total fleet of 150 cars would be required to maintain delivery of concentrates from Fort Nelson to the Port of Vancouver.

Figure 18.22 shows the total proposed haulage route from the Mine to Vancouver.

Figure 18.22 Total haulage route to Vancouver



In late 2014, 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.

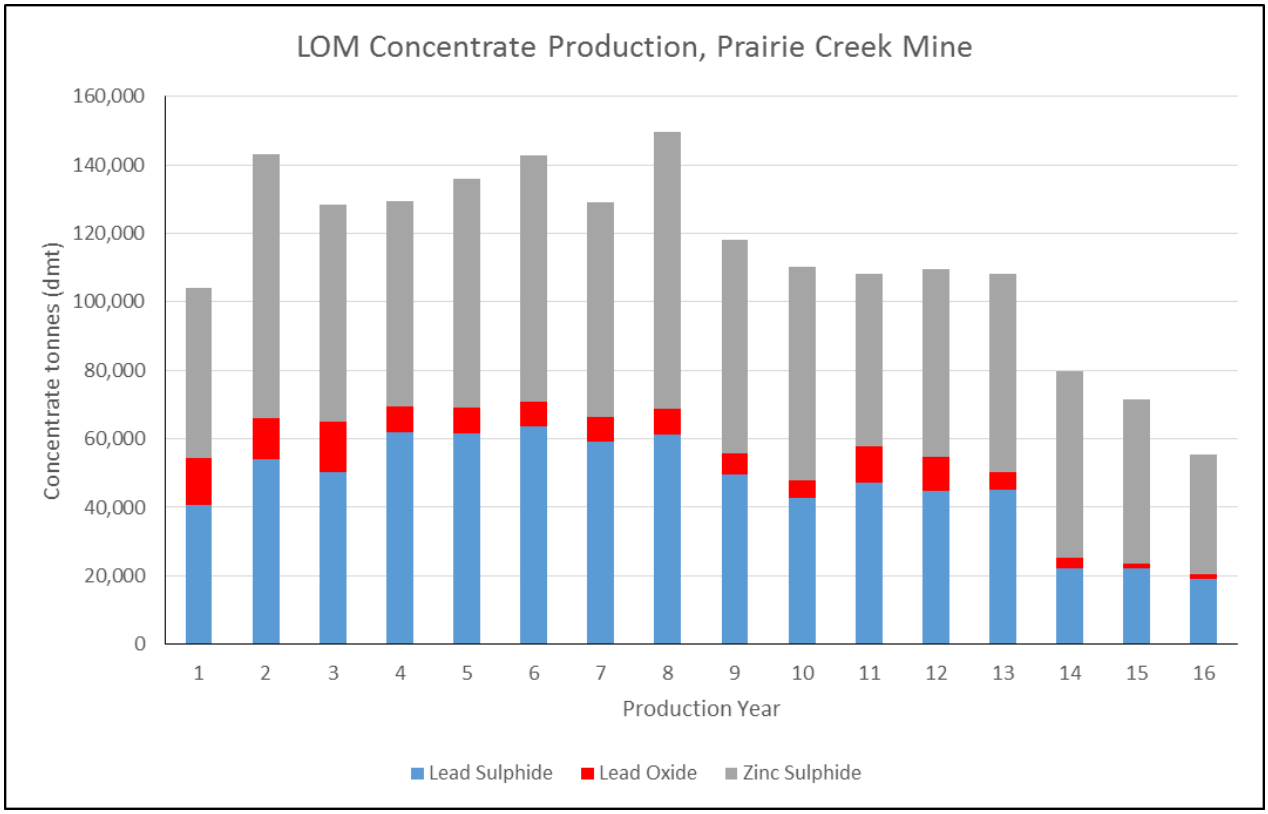
Bench scale locked cycle tests produced samples and specifications of each of five different concentrates which included copper sulphide, lead sulphide, lead oxide, zinc sulphide and zinc oxide concentrates. In 2014, the Company engaged Cliveden Trading AG, an international metal trading and advisory 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 2015 Mineral Resource Estimate resulted in an increase in LOM Mineral Reserves, at envisaged production rates, from 11 to 17 years (17 years of ore production, 16 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:

- Tonnage feed available from the mine
- Source of mill feed, MQV, STK, SMS (all have slightly different compositions)
- Grinding/Flotation capacity of the mill
- Latter years are running lower on defined MQV Mineral Reserves

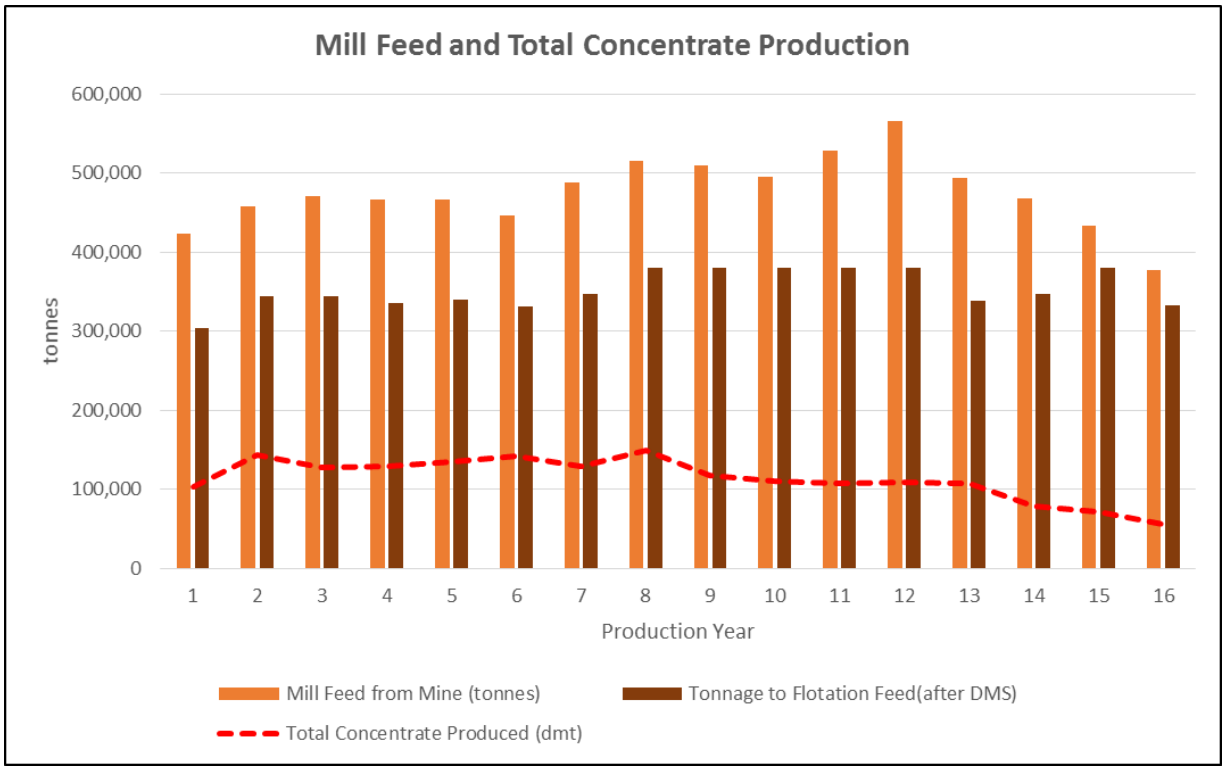
In the initial years, sufficient mining faces will be accessed and developed to reach targeted steady state production in Production Year 3. The present mine plan is designed for the first nine years to access mainly the high grade MQV-type ore. Years 10-14 are mining a blend of MQV and SMS (lower lead content), which is followed by the subsequent years blending in 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. This is a function of reduced MQV material, as the presently defined MQV Mineral Reserve is depleted by mining at that time. However, it is anticipated that, during future operations, further exploration definition drilling of the presently defined 5.0 million tonne MQV Inferred Resource would be completed in tandem with mining to maintain the MQV Mineral Reserves ahead of mining, and that this will 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 rate, at this time, is determined by the volume the mill can process in its current configuration. The existing 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 new Dense Media Plant on the front end of the milling circuit provides capacity to mine at rates of 1,350 tpd which, as indicated, is predetermined by the mill capacity. The mill process capacity (grinding/flotation) 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 higher than ideal levels of impurities that will have smelter penalty implications. The main impurity of interest is mercury, and the Company continues to study processes to reduce the mercury component occurring in the concentrates through on-site or near-site leaching, roasting, pressure oxidation and bio-leaching techniques. The reduction of other deleterious elements, which would cause additional economic penalties at the smelter, is also being studied with respect to further optimization of the milling process.

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/t[1]	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

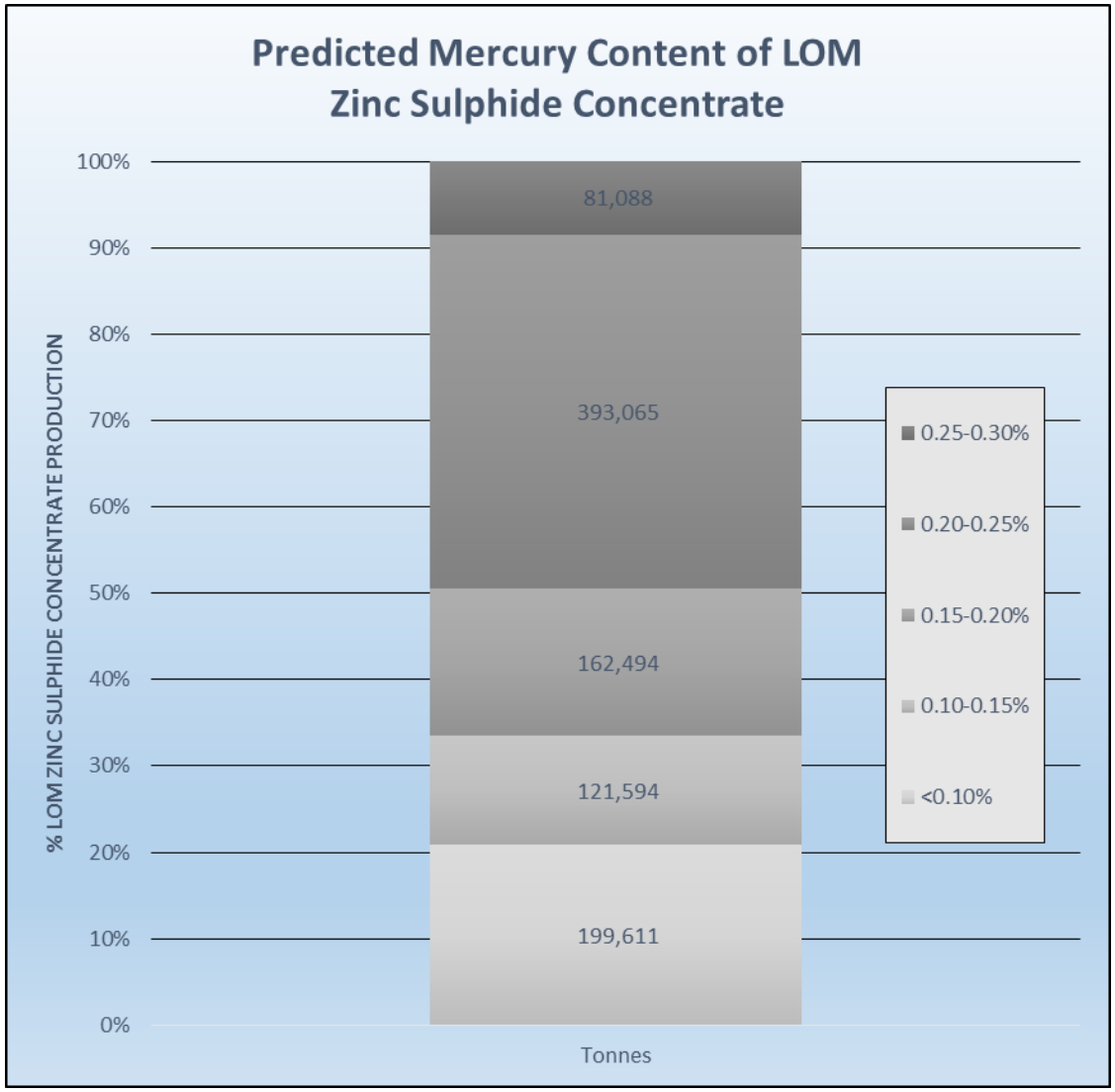
Variations in the grade of concentrate and grade of deleterious elements occur throughout the projected LOM, depending on the composition of the feed product from the Mine and the stockpile. Table 19.2 shows this variability over time for the four types of concentrates produced in relation to the key 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	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16
Mill Feed (tonnes)	423,057	457,559	470,575	466,257	466,938	446,447	487,508	515,477	509,867	495,170	528,187	564,935	493,352	467,475	433,029	377,758
Tonnage to Flotation Feed (after DMS)	303,694	343,596	343,596	335,296	339,469	331,123	347,679	380,791	380,791	380,791	380,791	380,791	338,862	346,384	380,791	332,298
Lead Sulphide Concentrate (dmt)	40,549	54,109	50,361	61,834	61,401	63,714	59,171	61,148	49,677	42,629	47,118	44,659	45,268	22,222	22,016	19,157
Concentrate Grade																
- Pb, %	68.50	68.50	68.43	68.50	68.50	68.50	68.49	68.20	66.73	64.61	67.10	68.46	68.35	64.31	56.08	56.06
- Zn, %	7.40	7.40	7.37	7.40	7.40	7.40	7.40	7.41	7.55	7.73	7.31	7.40	7.39	7.76	8.49	6.69
- Ag, g/t	1,088	1,010	1,100	918	903	896	840	892	781	795	894	802	687	698	720	658
- Cu, %	1.93	1.73	2.01	1.77	1.63	1.63	1.48	1.63	1.21	1.39	1.34	1.05	0.60	0.61	1.16	1.16
- As, %	0.175	0.186	0.193	0.230	0.232	0.236	0.261	0.320	0.213	0.180	0.172	0.194	0.132	0.164	0.151	0.151
- Sb, %	0.480	0.556	0.597	0.731	0.732	0.707	0.661	0.798	0.548	0.604	0.483	0.427	0.290	0.301	0.600	0.601
- Hg, %	0.022	0.025	0.015	0.026	0.026	0.027	0.028	0.036	0.026	0.008	0.018	0.031	0.018	0.012	0.002	0.002
- Cd, %	0.034	0.037	0.036	0.043	0.044	0.043	0.044	0.047	0.042	0.042	0.037	0.039	0.036	0.029	0.047	0.037
Lead Oxide Concentrate (dmt)	13,847	11,890	14,733	7,673	7,769	7,141	7,254	7,467	6,037	5,063	10,763	10,206	4,825	3,110	1,673	1,451
Concentrate Grade																
- Pb, %	48.0	48.0	48.0	48.0	48.0	48.0	48.0	48.0	48.0	48.0	48.0	48.0	48.0	48.0	48.0	48.0
- Zn, %	7.7	7.7	7.7	7.7	7.7	7.7	7.7	7.7	7.7	7.7	7.7	7.7	7.7	7.7	7.7	7.7
- Ag, g/t	455	493	482	568	552	584	522	557	517	559	442	382	456	418	616	563
- Cu, %	0.74	1.03	0.89	1.84	1.67	1.88	1.58	1.74	1.32	1.56	0.78	0.61	0.75	0.61	2.10	2.10
- As, %	0.067	0.110	0.085	0.240	0.238	0.272	0.278	0.341	0.233	0.202	0.100	0.115	0.174	0.168	0.274	0.275
- Sb, %	0.183	0.329	0.265	0.761	0.751	0.817	0.704	0.850	0.598	0.676	0.280	0.247	0.362	0.298	1.090	1.094
- Hg, %	0.013	0.012	0.008	0.016	0.015	0.014	0.018	0.019	0.017	0.006	0.016	0.031	0.014	0.012	0.002	0.003
- Cd, %	0.019	0.017	0.020	0.027	0.026	0.023	0.028	0.025	0.029	0.030	0.031	0.037	0.027	0.023	0.044	0.041
Lead Blended Concentrate (S+Ox)(dmt)	54,396	65,998	65,094	69,507	69,171	70,855	66,426	68,615	55,714	47,692	57,881	54,864	50,093	25,332	23,689	20,608
Concentrate Grade																
- Pb, %	63.3	64.8	63.8	66.2	66.2	66.4	66.3	66.0	64.7	62.8	63.5	64.7	66.4	62.3	55.5	55.5
- Zn, %	7.48	7.45	7.45	7.43	7.43	7.43	7.43	7.44	7.57	7.73	7.39	7.46	7.42	7.76	8.44	6.8
- Ag, g/t	926.5	917.1	960.0	879.0	863.2	864.3	805.4	855.2	752.7	769.8	809.5	723.7	664.6	663.7	712.5	651
- Cu, %	1.624	1.605	1.755	1.779	1.633	1.656	1.494	1.644	1.224	1.412	1.238	0.966	0.615	0.610	1.223	1.222
- As, %	0.147	0.172	0.168	0.231	0.233	0.239	0.263	0.322	0.215	0.182	0.159	0.180	0.136	0.164	0.160	0.160
- Sb, %	0.404	0.516	0.522	0.735	0.734	0.718	0.665	0.804	0.553	0.612	0.445	0.393	0.296	0.300	0.635	1.636
- Hg, %	0.020	0.023	0.014	0.025	0.025	0.026	0.027	0.034	0.025	0.008	0.017	0.031	0.018	0.012	0.002	0.002
- Cd, %	0.030	0.033	0.032	0.041	0.042	0.041	0.042	0.044	0.040	0.040	0.036	0.039	0.035	0.028	0.047	0.037
Zinc Sulphide Concentrate (dmt)	49,638	77,181	63,424	60,074	66,603	71,849	62,734	81,088	62,404	62,698	50,452	54,624	58,170	54,447	47,697	34,769
Concentrate Grade																
- Pb, %	4.03	4.14	4.11	4.24	4.20	4.28	4.13	4.00	3.23	2.58	3.34	3.47	3.41	2.45	0.99	0.98
- Zn, %	59.0	59.0	59.0	59.0	59.0	59.0	59.0	58.9	58.5	58.0	58.6	59.0	59.0	58.4	57.0	57.0
- Ag, g/t	157.7	174.3	174.2	157.6	159.2	164.8	143.2	157.1	125.1	130.6	121.5	78.8	76.3	82.5	119.3	119
- Cu, %	0.234	0.189	0.262	0.321	0.244	0.246	0.206	0.195	0.111	0.108	0.150	0.107	0.053	0.016	0.035	0.041
- As, %	0.021	0.020	0.025	0.042	0.035	0.036	0.036	0.038	0.019	0.013	0.019	0.017	0.009	0.004	0.005	0.005
- Sb, %	0.058	0.061	0.078	0.132	0.110	0.107	0.091	0.095	0.050	0.046	0.053	0.043	0.026	0.008	0.018	0.022
- Hg, %	0.179	0.202	0.123	0.209	0.210	0.216	0.226	0.282	0.167	0.060	0.155	0.250	0.120	0.087	0.017	0.019
- Cd, %	0.269	0.296	0.289	0.340	0.354	0.342	0.351	0.368	0.315	0.315	0.290	0.246	0.200	0.198	0.315	0.315

With the more detailed mine plan model and process schedule, more accurate predictions can be made of the quality and quantity of concentrate material that will be produced from year to year during the life of the mine. In particular reference to mercury content in zinc sulphide concentrate as shown in Figure 19.3, over 50% of the predicted zinc sulphide concentrate produced will have less than 0.20% mercury and over 90% of the zinc sulphide concentrate tonnage will contain less than 0.25% mercury. Once again, the variation in mercury content relates primarily to the type of ore being mined and the delivery profile to the mill, with the MQV ore being highest in mercury content and the STK and SMS ores being significantly lower.

Figure 19.3 LOM predicted mercury content in zinc sulphide concentrate



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 world-class 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 biggest 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 biggest 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.

In order to better determine capital costs of significant equipment or upgrades required for the future operation of the Prairie Creek mine, a contract tendering process was arranged by Tetra Tech. This tendering process resulted in the Company receiving hard pricing for most of the significant capital items contemplated for operations, including power systems, water treatment, paste backfill, mill upgrades, surface facilities and roads. This contract pricing contained cost escalation factors that adjusted and confirmed the quotes until the end of 2016. While the Company awarded a number of bids it is not committed to the awarded parties until financing is secure, when further negotiations of contract details can be concluded. The tendering process also resulted in an award to a mining contractor for the underground development of the Prairie Creek mine.

The Project contemplates the leasing of major mining equipment. This is common practice in the industry as it reduces initial capital expenditures and it assists the Company in maintenance of the equipment, as leased equipment is also often subject to maintenance agreements. As part of the contract tendering process mentioned above, requests for an option for leasing or lease to purchase were included.

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 2016 Pre-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., were reviewed for the purpose of determining a consensus pricing for the 2016 Pre-Feasibility Study Base Case. The Consensus Long-term Price Forecasts (2021-2025) for each of the metals to be produced at the Prairie Creek Mine are: lead US$1,990/tonne (US$0.90/lb); zinc US$2,416/tonne (US$1.09/lb) and silver US$19.10/oz.

As discussed below, the long-term outlook for both lead and zinc is considered very positive and these fundamentals, with the continuing Canadian-US dollar exchange rate which improves metal prices expressed in Canadian dollars, impact favourably on the Prairie Creek Project. Long-term base case metal prices used in the financial analysis in the 2016 Pre-Feasibility Study are shown in Table 19.3. Based on the evaluation of consensus forecasts and review of published information from various market commentators, 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)	$1.00
Lead ($/lb)	$1.00
Silver ($/oz)	$19.00

Prices above are in U.S. dollars
C$:US$ exchange rate: C$1.00:US$1.25

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 2015 Annual Report; Teck Corporation 2015 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, as compared to the previous two years, and are expected to remain the same in 2016. However, in 2017, many financial analysts expect GDP growth to recover, with US growth trending moderately higher and China, while not experiencing the growth levels of the past decade, still forecast to have strong steady growth.

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 2013/14) both show sustained decline since the highs experienced in 2011. However, many financial analysts suspect the indices have reached the bottom and agree that moderate growth will be experienced over subsequent years.

The prices of lead and zinc, expressed in US dollars, were generally lower in 2015 than in 2014. Metal prices rose during the first half of the year, but declined later in the year as economic indicators signaled lower growth rates in China. Precious metal prices were also generally lower, reflecting a stronger USD and low rates of inflation. The prices of gold and silver were negatively affected during the year by a stronger USD and by expectations of continued low global rates of inflation. The prices were, on average, lower than 2014.

The USD strengthened significantly against the majority of the world's currencies in 2015, largely due to the relatively stronger performance by the US economy and expectations of US interest rate rises, and resulted in declining local costs in producing countries measured in USD.

19.2.2 Price of zinc

Zinc's primary use is in the production of galvanized steel, accounting for approximately 50% of the metal's demand currently. Hence, zinc's fundamentals remain largely underpinned by the automotive sector, with infrastructure development and emerging market considerations, particularly in China and India, being key to the metal's long-term outlook.

The zinc market continues to face a considerable medium-term supply issue, as there are, arguably, no new large advanced-stage development projects poised to replace production capacity lost from a number of recent mine closures. Mine closures or reductions since 2013 have removed approximately 4.5 billion pounds of annual zinc production from the market, equivalent to approximately 15% of global supply and outpacing foreseeable zinc mine additions. The impact of this anticipated shortfall stands to be amplified by projected global zinc demand growth, currently estimated at about 4% per annum.

It has been suggested by commentators that the market requires over $5 billion (USD) in new mine development now to fill the anticipated medium-term supply gap, which is not happening, and is compounded by a lack of significant new zinc deposit discoveries. Recognizing that mines take years to develop bodes well for a 'higher for longer' medium-term zinc price rally. The last zinc price rally, which began in early 2006, saw the metal spike to an all-time high of $2.09 (USD) per pound in late 2008.

Skeptics may point to China, the world's largest zinc producer, to fill the anticipated supply gap; however, the country's higher cost mines will arguably dictate future zinc floor pricing. Anecdotal evidence is readily available of continuing pressure on some zinc mines in south China to curtail production for both environmental and economic reasons. China continues to be the leading producer and consumer of zinc.

Closures of the large Century and Lisheen mines in Australia and Ireland respectively, along with announced output cuts from Glencore and Nyrstar, have raised prospects of a market deficit in 2016. In the medium to long-term it is expected that mine supply will struggle to keep pace with the anticipated growth in demand for zinc.

The zinc metal market is already in a structural deficit, which is being addressed by stock draws to meet the current shortfall. In the future, however, shortages in both metal and concentrates will squeeze the market, and with China being less able to respond due to credit and environmental pressures, it is projected that a strong upside in prices is required to avert the implied dip in metal stocks to less than one week of consumption by 2019.

In its review of the outlook for trends in world supply and demand for lead and zinc for 2016, the International Lead and Zinc Study Group ("ILZSG") noted that world usage of refined zinc metal is expected to increase by 3.5% to 14.33 million tonnes in 2016, primarily driven by a further 4.5% increase in China, where demand is expected to benefit from continued infrastructure investment.

The ILZSG is expecting a sharp fall in zinc mine production outside China of 9.4%, due to a combination of mine closures and announced production cutbacks. A significant predicted 46% reduction in Australian zinc mine production is a consequence of the closure of MMG's Century mine in August 2015, cutbacks at Glencore's operations at Mount Isa and McArthur River, and announced reductions at CBH Resources Endeavour mine and Perilya's Broken Hill operation.

Zinc mine production is also expected to be lower in Ireland, where the Lisheen mine closed in November 2015, and in the United States, mainly as a consequence of the suspension of operations at Nyrstar's mid-Tennessee mines. Chinese output, which is reliant on production from a large number of small mines, is forecast to grow by 12.4%.

In the United States, rising demand for zinc from the automotive and construction sectors is expected to result in increase in usage of 3.1% in 2016. After rising 3.2% in 2015, it is anticipated that European growth will remain stable in 2016.

According to the ILZSG, overall global zinc mine output is expected to fall by 1.4% to 13.27 million tonnes. Chinese imports of zinc contained in zinc concentrates are expected to be significantly lower than the 1.37 million tonnes recorded in 2015.

The ILZSG expects that global demand for refined zinc metal will exceed supply by 352,000 tonnes in 2016. Wood Mackenzie believes that 2016's global zinc mine production will fall 2.1% over 2015 to 13.1 million tonnes. Mine production in the rest of the world is expected to contract by about 8% in 2016 following the closure of the Century and Lisheen mines and the cutbacks by other operators, including Glencore which, in October 2015 announced a reduction in mine production by 500,000 tonnes. Despite some mine production increases, closures of large long-life mines and production curtailments announced by major suppliers are expected to reduce annual global mine production by more than 1.2 million tonnes of contained zinc in 2016. Wood Mackenzie is also forecasting an increase in global zinc refined metal demand in 2016 of 3.6% to 14.6 million tonnes, exceeding current estimates for global supply, keeping the refined market in deficit and further reducing global stockpiles of zinc metal.

In the medium term this anticipated increase in demand coupled with the reduction in supply is 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 batteries, can be related to that of zinc as the metals are usually 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. China continues to be the leading producer and consumer of lead.

The demand for lead metal continues to be moderately strong. Demand is currently led by increased need for batteries in Asia. The International Lead and Zinc Study Group is forecasting increased usage in the automotive and industrial battery sectors in the near-term and continued decline in the world production of refined lead metal in the near-term.

Strong global auto sales were experienced in 2015 as US car sales increased significantly, Chinese sales hit a new record and European sales rose over 9%.These trends should help the lead price to move higher. Global demand for lead totaled 11.1 M tonnes, corresponding to an increase during 2015 of just under 1%. There was a modest increase in demand for batteries, both for new vehicles and for the replacement market. Lead demand in China remained virtually unchanged, year on year. Chinese imports of lead contained in lead concentrates in 2015 increased by 5.9% compared to 2014, to total a record 1.03 million tonnes.

Global demand for refined lead metal is forecast to increase by 2.6% to 11.11 million tonnes in 2016, primarily due to increased usage in the automotive and industrial battery sectors. In China, increased usage in the automotive and telecommunications sectors will be partially balanced by a reduction in demand in the e-bike market, resulting from slower sales growth and increased competition from lithium-ion batteries.

The ILZSG anticipates that global demand for refined lead metal in 2016 will rise by 2% to 10.83 million tonnes. Mine production outside China is expected to fall 6.1%, primarily due to reductions in Australia resulting from the closure of MMG's Century mine and cutbacks announced by Glencore, CBH Resources and Perilya.

After reaching a record 1.03 million tonnes in 2015, Chinese imports of lead contained in lead concentrates are forecast to fall to just under 900,000 tonnes in 2016. An anticipated 2.3% rise in global refined lead metal production to 10.90 million tonnes will be principally influenced by increased output in China and Korea, and the ILZSG anticipates that increases in refined metal supply, most notably in Korea, will result in a global refined lead metal surplus of 76,000 tonnes in 2016.

The ILZSG is forecasting global demand for refined lead metal to increase due to increased usage in the automotive and industrial battery sectors and Metals Bulletin Research suggested that lead could become one of the leading metals in 2016 as supply fundamentals continue to tighten.

19.2.4 Price of silver

Over the past decade, significant changes have occurred in the consumption pattern of silver. For 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. In 2015, industrial fabrication demand accounted for an estimated 54% of total physical silver demand.

Silver's use in photovoltaics for solar energy is projected to rise in 2016 and surpass the previous peak of 75.8 Moz (million ounces) set in 2011, as global solar panel installations are expected to grow at a high single-digit pace. Moreover, silver's use in this application may account for more than 13% of total silver industrial demand in 2016, up from 1.4% a decade ago.

Global mine supply production is projected to fall in 2016 by as much as 5% year-on-year. This would represent the first reduction to global silver mine production since 2002. Looking further ahead, many analysts expect global silver mine production to fall through 2019 as primary silver production from more mature operations begins to drop.

The silver market deficit (total supply less total demand) is expected to widen in 2016, drawing down on above-ground stocks. The larger deficit is expected to be driven by a contraction in supply.

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 just 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. The gold/silver price ratio ended the year 2015 at 76 (compared with an average of 66 the past three decades).

19.2.5 Longer term metal price outlook

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.

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 been suffering from overcapacity 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, resulting, eventually, in production cutbacks or smelter closures. This, 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 slowdown in the growth in demand for metals and the increased supply has resulted in falls in the price of metals in 2015, coupled with a rise in treatment charges as the supply of concentrates increased. In weaker economic markets, metal prices have often reached a low point when they equate to the cash cost level for high-cost mines.

The long-term outlook for both lead and zinc is positive. As indicated above, with the Century mine in Australia and the Lisheen mine in Ireland now closed, almost 1.5 billion lbs. of annual zinc production, representing almost 5% of global production, has been removed from supply. Wood Mackenzie is forecasting an increase in global zinc refined metal demand in 2016, exceeding current estimates for global supply, keeping the refined market in deficit and further reducing global stockpiles of zinc metal.

As global demand weakens and commodity prices fall, Wood Mackenzie is seeing a challenging environment in the metals and mining industry. Mining companies are increasingly coming under pressure to reduce operating costs, causing many to shift their strategic planning, delay new investments and look to long-term future growth.

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. This structural issue has been exacerbated by the retreat in the zinc price, which has forced major mine production cutbacks and threatens to force smelters to cut production also.

On the supply side, the depleting nature of ore reserves, difficulties in finding new orebodies, the permitting processes, the 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.

CRU noted that closures, contractions and cutbacks occurring outside of China are expected to lead to a loss in zinc production of approximately 1.6 million tonnes between 2015 and 2020. Additional closures are also expected in light of the decline in the zinc price, and given the current zinc price and exchange rates, zinc production from Chinese mines is seen as being especially vulnerable. Taking into account only the anticipated closures, total zinc stocks are expected to plummet, and given the expected modest growth in demand zinc supply, CRU expects the zinc market to stay tight for the foreseeable future.

(Sources: Published information extracted from: International Lead and Zinc Study Group; World Bank Group; CRU; Wood Mackenzie; Metals Bulletin Research; Boliden 2015 Annual Report; Teck 2015 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 mining in the order of 1,350 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 and possible construction of an additional 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 Naha Dehe Dene Band (NDDB) and the Liidlii Kue First Nation (LKFN) of the Dehcho First Nations. The nearest community is Nahanni Butte, home of the NDDB, 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 1980s 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 Corporations' 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 are 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 in 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. Based on the water quality predictions (including toxicity testing), effluent discharge via an exfiltration trench installed below the bed of Prairie Creek and only part-way across the channel from the Mine operation, the treated mine water effluent is expected not to impact the aquatic environment.

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, the following documents will need to be submitted:

- 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, Tetcela Transfer Facility, bridges
- Engineering designs for crossings at km 24, 26.4, 36.8, 28.7 & 43
- Final engineering designs
- Construction drawings showing cut and fill locations & 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 will 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. At mine start-up, up to 50,000 tonnes of flotation tailings may also be placed in the process effluent cell on a temporary basis. This is because mine openings will not yet be available to receive paste. The tailings will be reclaimed from the pond at a later date and placed underground. Underground stopes will not be available for paste backfill until approximately five months into the operating period.

During operations, the WSP will supply feed water to the mill from both cells as a 65% process effluent/35% mine water blend. The majority of mine water will be treated year-round in a new treatment plant to reduce metal concentrations, 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). If CZN is able to intercept groundwater before it flows into the Mine, thus preventing its contamination, 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 rates of both mine water and process effluent 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, sulphuric acid and 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.

Bags of concentrate being transported will be frozen, 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 will include 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 7 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	
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, and to build and operate an airstrip connected to the road. An all season road will enable the haul of concentrates from the Mine site to rail year-round for at least 215 days each year, with temporary closures relating to freeze-up and break-up on the Liard River, and greatly reduces the risks of haul requirements over the winter period. 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. At that time, 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 is currently in the process of responding to Information Requests, which is the precursor to a Technical Session to be held and is part of the normal EA process.

20.5 Aboriginal background: Dehcho First Nations

The Prairie Creek Mine is located on land claimed by the Nahanni Butte Dene Band and the Lidlii Kue First Nation, which are both part of the Dehcho First Nations (Dehcho or DCFN). The Nahanni Butte (Nahaahdee) 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 are 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, and the Metis people within the DCFN territory. The Dehcho have had their own system of laws, religion, economy, customs, traditions and language since time immemorial. 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 all of the Dene and Metis peoples in the Dehcho territory of the Northwest Territories, which comprise thirteen separate communities. 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 people 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 the negotiations of the 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. It is expected that the negotiations towards a Dehcho Final Agreement will take many years to complete.

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 or the permitting thereof.

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 provides for the subsequent negotiation of an Impact Benefits Agreement regarding mining operations. Nothing within the MOU is intended to define, create or extinguish any rights of the Band or Canadian Zinc and the MOU is not legally binding on the parties.

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 Company believes that the separate goals of the DCFN in achieving political sovereignty and economic self-sufficiency, whilst protecting the environment are compatible. 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.

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 has 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 metres high, Virginia Falls which falls over 90 metres, 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.

In August 2012, Canadian Zinc and the GNWT Department of Transportation 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 to maintain and enhance these roads effectively and 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.

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.

20.7 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 $4.3 million, 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 mill process water circuit in the WTP will be shut down in a scheduled manner in order to empty all liquid reagent mixing and storage tanks. All reagent distribution lines will be blown down with compressed air to prevent freeze damage. All isolation valves at the acid storage tank will be closed and all acid lines between the acid storage tank and the WTP will be drained.

The mine water circuit in 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 the Liard Transfer Facility (LTF).

The LTF 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 the fuel tank, sewage tank and waste bin will have been emptied.

No temporary closure activities are contemplated for the winter road 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 has been designed by Golder Associates to store up to 500,000 m^3 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 transfer facilities and road construction 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 tent frame structures, 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; and,
- Final reclamation of the WRP

The access road connecting the Prairie Creek Mine site to the Liard Highway was accommodated for in separate Land Use Permits and Type B Water Licences issued by both MVLWB and Parks Canada since the road runs through both jurisdictions.

The total amount of security deposits are as described in the issued permits associated with future operations and summarized in Table 21.6 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 2016 Preliminary Feasibility Study capital and operating cost estimates for developing and commissioning the Prairie Creek mine and concentrator were compiled from a number of sources by CZN, based on information and inputs provided by AMC, Tetra Tech and other specialist consultants, and include vendor quotes and leasing proposals.

Following completion of the 2012 PFS, CZN carried out a number of optimization studies and advanced the project design level in many areas. The capital cost estimate for mine pre-development in the 2016 PFS update study is derived from the unit prices indicated by the selected mining contractor, Procon Mining and Tunnelling, in 2015, multiplied by the quantities estimated by AMC. Tetra Tech developed tender packages for various capital items and firm pricing has been obtained from qualified vendors and contractors. Where such information has not been available, estimates have been provided by Tetra Tech, derived using their in-house estimating procedures and database, with specialty input from:

- Golder Associates: Waste Rock Pile and Water Storage Pond;
- Allnorth: transportation and road construction/maintenance costs;
- Trealmont Trade Lane, Kinder Morgan and Cliveden: port and shipping costs.

The engineering design detail for site completion and construction work was not considered sufficiently advanced to seek competitive tenders and this cost has been estimated to a pre-feasibility level of accuracy by Tetra Tech, using their in-house estimating procedures and database.

Approximately 35% of the capital cost estimates, excluding contingency, is based on firm pricing obtained through competitive tendering by pre-qualified vendors and contractors, binding through December 2016. This cost estimate provides a higher level of accuracy than that presented in the 2012 PFS.

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	TM/AMC
Mill Capital Cost Estimate	TT
Surface Facilities Capital Cost Estimate	CZN/TT/TM
Mine Workforce Cost Estimate	TM/AMC
Mill Workforce Cost Estimate	TT
Surface & G&A Workforce Cost Estimate	TM/CZN
Mine Operating Cost Estimate	TM/AMC
Mill Operating Cost Estimate	TT
Surface Facilities & G&A Operating Cost Estimate	CZN
Power Requirements	TT
Power Cost Estimate	TM
All Season Road Installation & Maintenance Cost Estimate	CZN/Allnorth
Owner's Costs	CZN
Reclamation Security Estimate	CZN

AMC = AMC Mining, CZN = Canadian Zinc Corporation, TT = Tetra Tech Inc., TM=Tom Morrison

21.2 Capital Cost estimate

Pre-Production Capital Cost refers to capital costs incurred until the first processing of mined ore, and has been estimated to a total of $216 million, excluding contingency, and $244 million including a contingency of $28 million, excluding working capital.

The 2016 PFS estimate of pre-production capital cost is based substantially on fixed price supply contracts, (FOB Fort Nelson and valid within contractual escalation limits until December 2016), contracted mine development unit prices, and a higher level of design detail than was available in 2012. Contingencies have been applied, at various levels considered appropriate, between 5% and 25% as identified below, with a blended average of 12.7%. The 2012 PFS was estimated to an accuracy of -20%/+20%.

The Capital Cost estimate is broken down into pre-production capital and sustaining capital, and is presented as a summary outlined in Table 21.2 below. The sustaining capital is carried over operating years 1 through 16.

Table 21.2 Summary of capital costs

Description	Total ($M)
Pre-Production Capital (incl. contingency of $27.5 M)	243.6
Sustaining Capital	70.4
Total Capital Cost (Life of Mine)	**314.0**

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 summary

Description	Project Year 1 ($M)	Project Year 2 ($M)	Total Cost ($M)
Mine development	-	34.5	34.5
Process plant[1]	6.7	12.5	19.2
Support Infrastructure[2]	11.5	30.9	42.4
All season road[4]	16.3	41.9	58.2
Site completion[3]	14.8	25.6	40.4
Owner's costs	4.5	4.4	8.9
Reclamation security (Pre-production)	4.8	7.7	12.5
Total (excluding contingency)	**58.6**	**157.5**	**216.1**
Contingency	14.7	12.8	27.5
Total Pre-Production Capital	**73.3**	**170.3**	**243.6**

1. Includes dense media separator, structural upgrading, instrumentation, flotation circuit upgrade, reagent handling and piping.
2. Includes power plant, paste plant, water treatment plant, water storage pond, waste rock pile, camp and housing accommodation, warehousing and facility upgrades.
3. Includes engineering and construction of surface facilities, freight and logistics, initial fills and spares.
4. Includes Liard River crossing and Fort Nelson load-out facility.

Based on proposals or quotations received, a number of capital items will be supplied on a lease-to-purchase basis. The main items included on a capital lease basis are: diesel generators; water treatment plant; paste plant; and some process equipment. The lease costs of such items incurred in the pre-production period are included in Pre-production Capital costs, and lease costs incurred after production start-up are included in Sustaining Capital.

21.2.2 Salvage value

Some of the capital equipment is expected to have minor salvage value at the end of mine life. Table 21.4 presents a summary of the purchase price of the equipment and the expected salvage value after considering the costs of disassembly and transportation off site.

Table 21.4 Capital salvage value

Description	Capital Cost ($M)	Residual Value (%)	Salvage Value ($M)
Underground mining & electrical equipment	23.4	15	3.5
Process plant equipment	1.8	15	0.3
General site equipment	19.5	15	2.9
Total	**44.7**		**6.7**

21.2.3 Capital cost exclusions

The following cost items have not been included in the Pre-Production Capital Cost Estimate:

- Any scope changes;
- 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;

The following recommended additional engineering studies and preliminary works (Table 21.5) are not included in the Pre-production Capital Cost Estimate and are regarded as Project sunk costs. Further development of the estimated capital costs to a definitive feasibility study level will require upgrading the existing engineering. Additional engineering items are those that would be required to upgrade the current level of engineering to definitive feasibility level and deliver the Project in a form ready for construction and to obtain fixed pricing from construction contractors.

Table 21.5 Additional engineering and preliminary works

	Cost Estimate ($000)
Early Works Programs	
Site clearing (1)	900
Foundations, WSP, WRP (2)	700
Feasibility Level Engineering and IFC design drawings (3)	4,000
Mine	
Hydrology (4)	100
Paste backfill (5)	300
Milling/Metallurgy	
Flotation tests on MQV material (6)	200
Variability tests (7)	250
Transportation	
Permitting for all season road (8)	500
All season road construction estimate (9)	100
Procurement and logistics studies	600
Feasibility level EP	1,400
Sub-total, critical items	**9,050**

1: Demolition of derelict plant and buildings, e.g. old generators and accommodation trailers.
2: Site work for building foundations, the Water Storage Pond and the Waste Rock Pile, which are impacted by seasonal weather conditions.
3: Allowance to bring engineering design up to a definitive feasibility level of detail to obtain fixed pricing from construction contractors.
4: Further groundwater hydrology modelling is recommended to produce a definitive estimate of the infrastructure and costs needed to deal with predicted groundwater inflows over the LOM.
5: Tests on the strength obtainable from paste backfill have so far produced preliminary results. Further testing is recommended to better predict paste backfill strengths, optimum binder dosage and resulting definitive costs.
6. Further locked cycle flotation testing is recommended to optimize the flotation process, improve recoveries and produce cleaner concentrates.
7. Variability tests on samples from various parts of the deposit.
8: Environmental assessment of the permit application for the all season road is ongoing in 2016.
9: The cost of constructing and maintaining the all season road is currently included in Capital Costs based on estimates received from contractors. A definitive feasibility level of engineering design is recommended to obtain fixed pricing from construction contractors.

The following Recommendations and Optional items (Table 21.6) are those that may further optimize Project economics and are not included in the Pre-production Capital Cost Estimate. See Section 26: Recommendations.

Table 21.6 Optional recommendation items

Optional Recommendation Items	Cost Estimate ($000)
Resource/Reserve Development	
Diamond drilling STK (1)	1,000
Diamond drilling MQV (1)	1,000
Structural Geology Study (2)	100
Milling/Metallurgy	
Bioleach study (3)	200
Mill capacity study (4)	100
Metal-sensing (5)	50
Site Infrastructure	
Energy alternatives (6)	50
Water treatment (7)	100
Additional optimization (8)	200
Total, optional items	**2,800**

1: Ongoing diamond drilling is recommended to improve understanding of stockwork (STK) mineralization and follow the down-plunge extension of the MQV.
2: Further studies of structural geology are desirable to improve wall-rock characterization, enabling more efficient stope design and a better ability to predict dilution.
3: In 2014, CZN initiated experiments on bioleaching zinc sulphide concentrate to remove mercury; these experiments were promising but are incomplete. Further work is recommended to upgrade the experiments to plant scale and estimate the capital and operating costs of an on-site or near-site leach plant.
4: The assumed mill capacity of 1,350 tpd is based on the milling rate of 1,350 tpd recommended by SNC-Lavalin in the 2012 Prefeasibility Study. Further studies are recommended to evaluate the possibility of increasing the mill throughput by 10-15% without major changes to the mill. It is believed that the Prairie Creek resource would support a substantial increase in mining/milling rates and an expanded or new mill.
5: Technology is under development, enabling the individual fragments in a stream of broken ore to be accepted or rejected, based on direct sensing of their metal content. This technology has the potential to reduce energy crushing and grinding waste rock.
6: The potential exists to reduce power generation costs and emissions by using Liquid Natural Gas (LNG). This would entail some capital expenditure and, hence, needs review.
7. Water treatment will be a substantial cost. Further studies on water quality and treatment methods would be beneficial to reduce this cost.
8. This item is a general allowance for trade-off studies with the objective to improve productivity and reduce costs.

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 production by means of longhole mining. On completion of that scope of work, CZN will have the option of extending the contract, tendering a new scope of work, or itself taking over the mine production work.

The cost estimate in this 2016 PFS update study is derived from the unit prices indicated by the successful tenderer, Procon, 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, the unit prices have been factored for increasing mining depth and the take-over of mining operations by CZN.

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. Because of the change in the scope of work, resulting from the new Mineral Resource/Mineral Reserve, initial capital expenditure now includes as capital all work in

the first year of work underground. Mining costs incurred after the first year of work underground are deemed operating costs.

The mine development capital works comprise:

- 883 mL portal reconstruction and enlargement of drift to 4.5 m x 4.5 m, 557 m;
- 4.5 m x 4.5 m ramp, 495 m;
- 4.5 m x 4.5 m vein-access crosscuts, 172 m;
- 4.5 m x 4.5 m remucks, paste fill line access drifts, ventilation drifts, sumps, miscellaneous development, 447 m;
- 3.5 m x 3.5 m electrical cut-outs, miscellaneous development, 89 m;
- 4.5 m x 4.5 m vein drifts, 661 m;
- 3.5 m x 3.5 m vein drifts, 1,396 m;
- Ventilation raises, 132 m;
- General mine rehabilitation and installation of new mine infrastructure.

Overall, mine pre-development and equipment capital costs, including contingency, are estimated at $34.5M.

A 15% contingency was applied to all mining capital expenditures.

Process plant, onsite and offsite infrastructure

Tetra Tech carried out initial design and developed tender packages for the following capital items within the Process Plant and Site Infrastructure:

Process Plant:

- Cyclones;
- Flotation cells;
- Agitators;
- Reagent preparation systems;
- Flocculant system;
- Samplers and analyzers;
- Air compressors;
- Concentrate handling system; and
- DMS plant.

Surface Infrastructure:

- Modular camp;
- Generators;
- Sprung-structure, heated warehouse;
- Water treatment plant;
- Assay laboratory;
- Paste fill plant;
- Instrumentation and control systems;
- Pre-engineered buildings; and
- Winter road construction.

Tetra Tech received tenders, carried out technical and commercial review and notified the successful tenderers. Pricing in all cases is firm, within prescribed inflation terms, until December 2016.

Civil, concrete, structural steel, piping, electrical and instrumentation material and installation costs were developed by Tetra Tech engineers, based on the company's in-house database, in early 2015. In March 2016, CZN confirmed that costs for wages and construction materials had not changed significantly since early 2015.

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 21.7.

Table 21.7 Reclamation and closure costs for Prairie Creek Mine

Location (Licence)	Security Required ($)	Trigger
Minesite "A" WL	1,550,000	Security posted per May 22/2015 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	
Minesite 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	
Liard Transfer Facility LUP	90,000	Prior to any activities under this permit
(MV2008T0012)	75,000	Upon completion of first con storage shed
	75,000	Upon completion of second con storage shed
	75,000	Upon completion of third con storage shed
	315,000	
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
(Parks2012_L001)	397,721	Prior to commencement of Tetcela Transfer Facility
	1,072,097	
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	**20,395,334**	
Previously remitted	**1,800,000**	
Ongoing requirements	**18,595,334**	

Reclamation costs of $12.5M incurred prior to mine start up are included in Pre-production Capital costs with the balance 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 & 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. The proportion of this cost attributable to finalize design and produce IFC drawings is considered a sunk cost. This EPCM item may be subdivided approximately as 50% for engineering and procurement and 50% for site construction management.

Construction indirects

Costs for temporary construction facilities and services were prepared by Tetra Tech.

Flight costs are based on commercial flights between Edmonton and Yellowknife and charter flights between Yellowknife and site.

Catering costs per person per day are based on CZN's actual 2014 camp operating costs at $75/person-day seven days a week calculated from total on-site man-hours for direct and indirect labour.

Freight cost, spare parts, vendor representatives

The equipment packages are priced FOB Fort Nelson. Tetra Tech has estimated the cost of delivering bulk materials and construction equipment from source to Fort Nelson. Tetra Tech obtained estimated costs to haul equipment and materials from Fort Nelson to site over a winter road from a trucking contractor.

Tetra Tech estimated the cost of spare parts and vendor assistance, if not already provided as part of the equipment tenders.

21.2.7 Contingency

Contingency was estimated by Tetra Tech in each area based on the level of engineering definition and method of pricing.

Contingencies range from:

- 5% for equipment and materials with a large price database and firm-price, new mechanical equipment;
- 10% for freight from source to Fort Nelson;
- 15% for freight from Fort Nelson to site;
- 15% for contract mine development;
- 20% for bulk earthworks;
- 25% for refurbishment of existing steel and concrete.

The blended rate is 12.7%.

21.2.8 Sustaining capital

Sustaining capital costs have been stated in constant 2016 Canadian dollars without any allowance for escalation or inflation. Sustaining capital costs (Table 21.8) have been developed as follows:

- Sustaining capital for mining, including the purchase and rebuild of mobile equipment, is estimated at $31.0 M over the remainder of the mine life, including buy-out of mine contractor equipment, fleet additions and renewals and extensions to underground infrastructure.
- 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 SwedeVent; and from AMC's database.
- Leasing costs of capital items of $29.1M are included in the first six years of sustaining capital costs.
- Reclamation security in the amount of $8.3M is included in the first year of sustaining capital.

- Process and infrastructure sustaining capital was based on an estimated cost of $1.15/tonne mined ore (SNC-L), including a contingency of 15% for a total of $8.7M over the life of the mine.
- Salvage value of $6.7M has been estimated and is a reduction of sustaining capital taken at the end of the mine life.

Table 21.8 Total Sustaining Capital costs

Sustaining Capital	LOM $M
Mining	31.0
Capital leases	29.1
Reclamation security	8.3
Infrastructure	8.7
Salvage	(6.7)
Total	**70.4**

21.2.9 Working capital

Working capital includes accumulation of operating expenses prior to receipt of revenue from sales and includes the full operating cost of mining and processing operations for three months and has been estimated at $30 million, excluding contingency, and $36 million including a contingency of $6 million.

21.3 Operating cost estimate

21.3.1 Total operating cost

The operating cost estimate was initially expressed in 2016 Canadian dollars and then developed to a pre-feasibility study estimate level of +20/-20% overall intended accuracy, over the life of the mine.

Operating costs per tonne milled and delivered to the smelter are shown in Table 21.9 for the life of the mine.

Table 21.9 Total operating cost summary

Total Operating Cost	LOM
Tonnes	7,603,590
	($/t)
Mining	78.58
Processing	40.75
G&A	22.58
Site Surface	21.96
Sub-total	163.87
Transportation*	65.10
Total	**228.97**

* 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

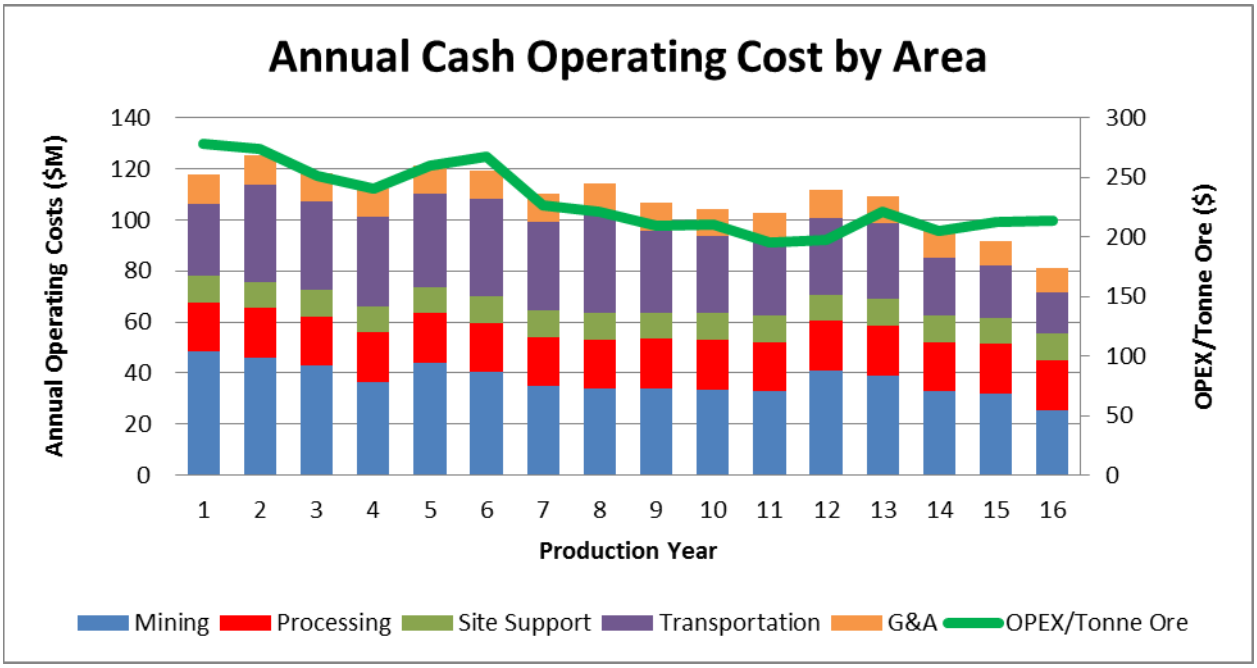
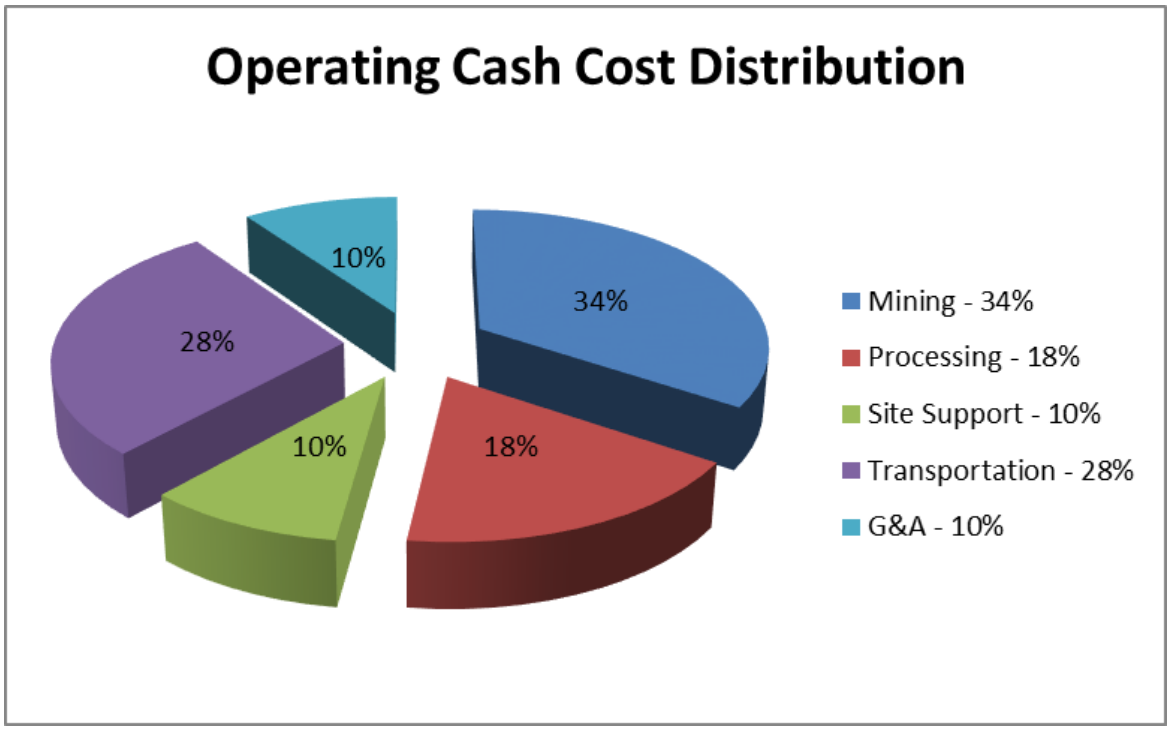


Figure 21.2 shows the split of cash operating costs.

Figure 21.2 Total operating cash cost distribution



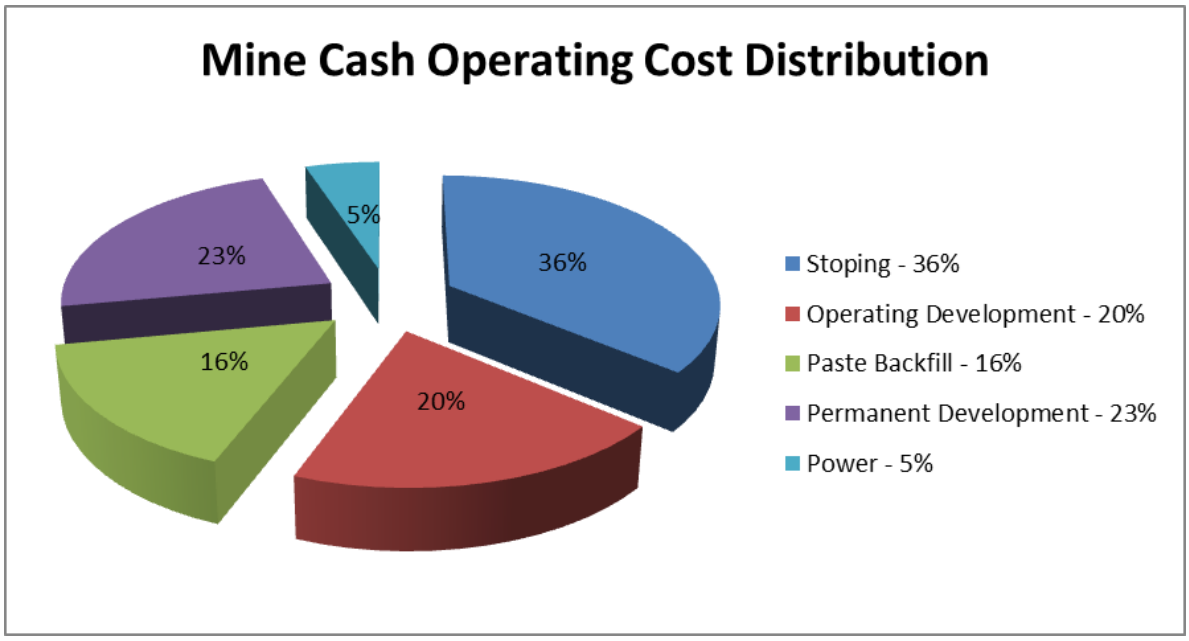
Mine operating cost

Mine operating costs are estimated according to unit prices tendered by the selected mining contractor, Procon, multiplied by annual quantities estimated by AMC, factored for increasing mining depth and the later take-over of the mining operations by CZN. 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, diesel fuel, open flame, over and above generator waste heat.

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 G&A operating cost

Processing, site support service and G&A operating costs have been estimated based on staffing levels agreed between CZN, AMC and Tetra Tech, labour rates obtained from Infomine wage surveys, and materials costs provided by Tetra Tech. The actual cost, year by year, will vary somewhat according to the tonnage accepted by the mill. Surface Support includes centralized maintenance for mine and plant mobile equipment, power tools, camp and fabrication. The mill is to be self-contained for maintenance.

Figures 21.4, 21.5 and 21.6 show the respective split of costs in the processing, site surface support and G&A areas.

Figure 21.4 Milling cash operating cost distribution

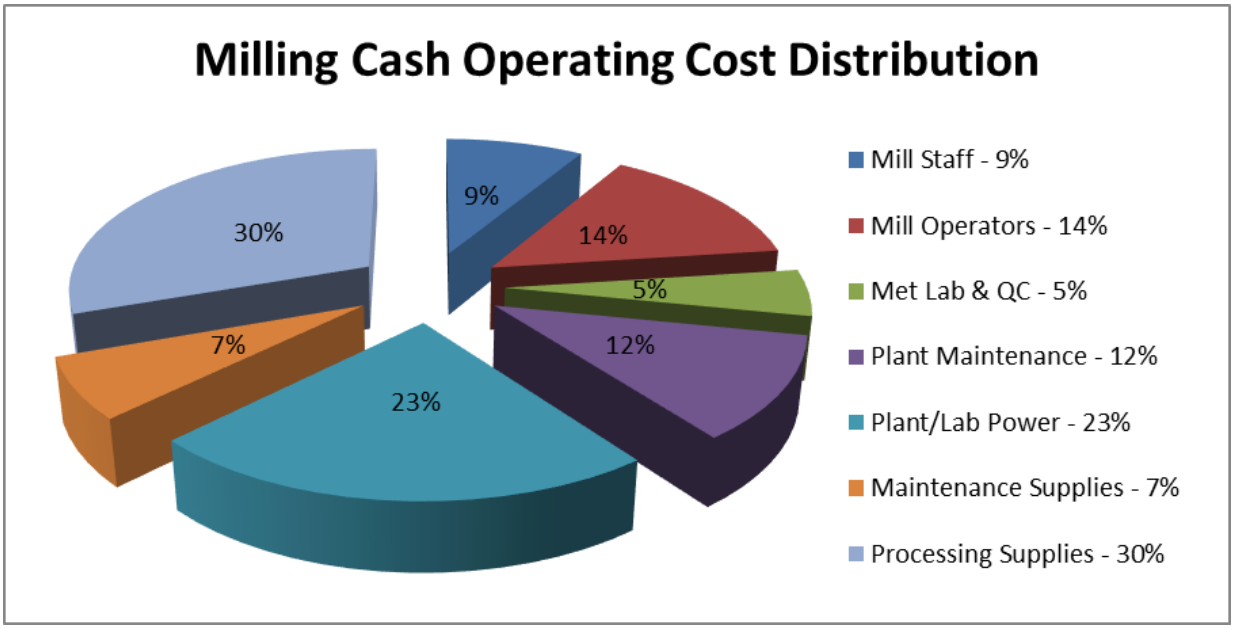
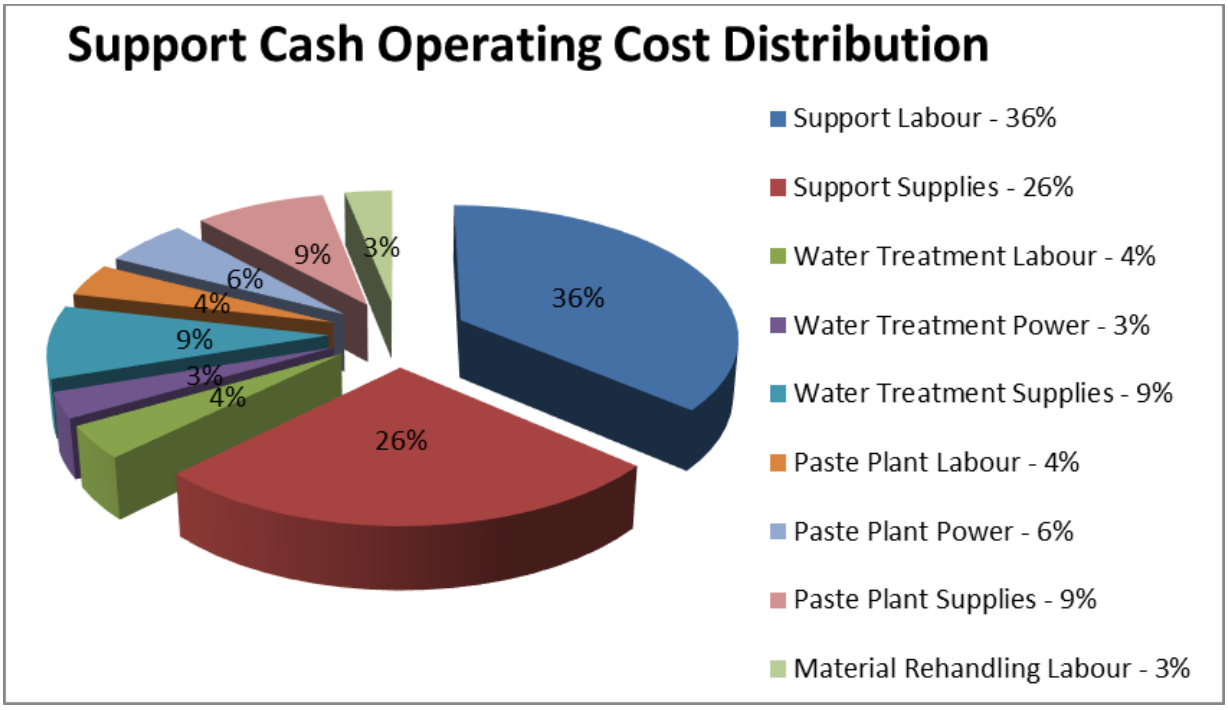
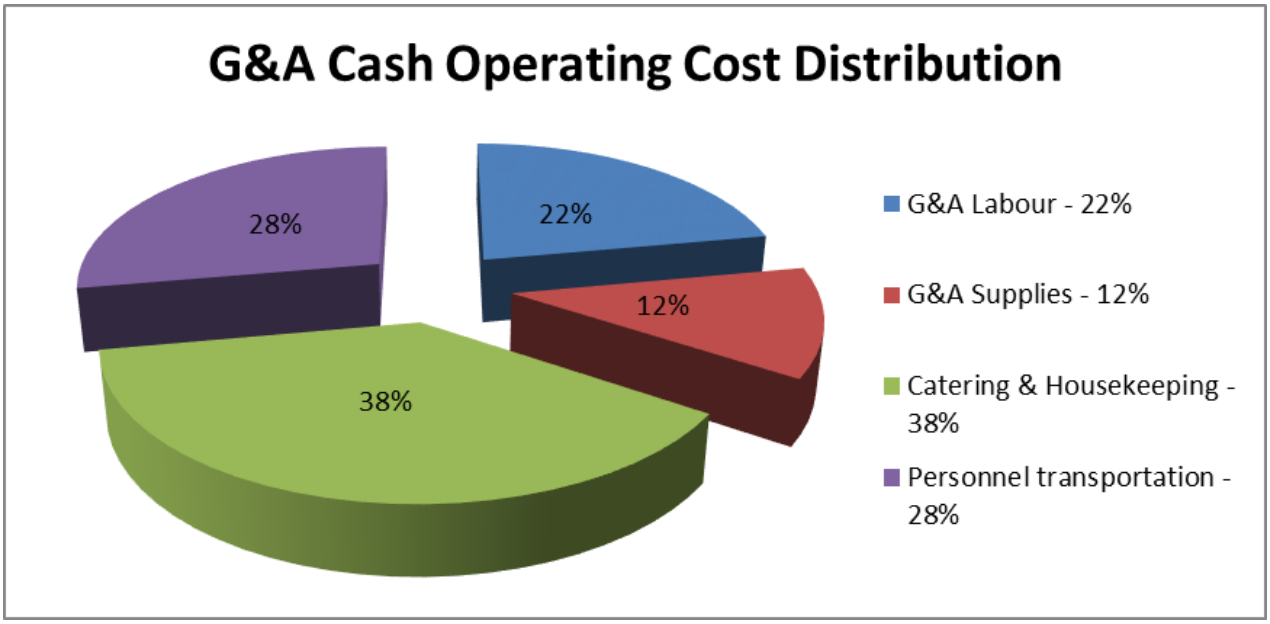


Figure 21.5 Support cash operating cost distribution



Power to run the camp is included in Support Supplies.

Figure 21.6 G&A cash operating cost distribution



21.3.2 Annual operating manpower costs

Labour costs are derived from the 2014 and 2015 Canadian Mine Salaries, Wages, and Benefits Survey, published by InfoMine USA Inc.

The manpower costs exclude camp accommodation and transportation of employees to and from the mine site. These costs are included in the G&A operating costs. Labour rates and payroll burdens of 46.5% were used for estimating manpower costs and are based on current rates provided by CZN.

The cost of mining is based on contractor unit pricing and the labour component is not separated. The labour cost for site services includes all site maintenance costs. Catering manpower costs are included in the camp-person day rate.

The manpower for G&A includes all positions required for general and administrative support at the Prairie Creek site – see Table 21.10. Head office (Vancouver and Toronto) positions are excluded.

Table 21.10 Staffing level on payroll (site)

Production Year	**-1**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**	**15**
Project Year	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
Mine	60	102	109	100	100	90	89	89	89	89	85	85	96	87	87	66
Mill	71	71	71	71	71	71	71	71	71	71	71	71	71	71	71	71
Surface Service	38	38	38	38	38	38	38	38	38	38	38	38	38	38	38	38
G&A	21	21	21	21	21	21	21	21	21	21	21	21	21	21	21	21
Sub-total, full-time	**190**	**232**	**239**	**230**	**230**	**220**	**219**	**219**	**219**	**219**	**215**	**215**	**226**	**217**	**217**	**196**
Camp staff, full-time	26	27	26	25	25	25	25	24	24	24	24	25	24	24	22	21
Total, full-time	**216**	**259**	**265**	**255**	**255**	**245**	**244**	**243**	**243**	**243**	**239**	**240**	**250**	**241**	**239**	**217**
Trucking (212 days/yr)	25	33	30	30	31	33	30	34	28	26	26	26	29	20	18	15
Camp staff for trucking	3	4	3	3	4	4	3	4	3	3	3	3	3	2	2	2
Total site	**244**	**296**	**298**	**288**	**290**	**282**	**277**	**281**	**274**	**272**	**268**	**269**	**282**	**263**	**259**	**234**
Ft. Nelson	29	38	35	35	36	38	35	40	32	30	30	30	34	23	21	18
Total payroll	**273**	**334**	**333**	**323**	**326**	**320**	**312**	**321**	**306**	**302**	**298**	**299**	**316**	**286**	**280**	**252**

The manpower cost for road installation, maintenance and concentrate haul is included in the total operating cost for transportation.

21.3.3 Annual supplies costs

Processing supplies costs are based on all consumables and supplies for the process plant, water treatment plant and DMS plant, including equipment wear parts, grinding balls, reagent chemicals and power.

The cost of mining is based on contractor unit pricing and the material cost component is not separated out. Mining materials costs are included in unit rates. Major categories of G&A costs have been estimated by CZN and include costs of office and general, professional fees and air flights.

“Support supplies” includes the cost of providing power to the camp. These costs also include the day-to-day operation and maintenance of the surface facilities and infrastructure that support the mill and mine operations.

Power cost & diesel fuel consumption

The steady-state, site-wide average power demand is estimated at 5.2 MW. The cost of generating this power is estimated by multiplying standard specific fuel consumption of 0.27 litres of diesel fuel per kWh by a current landed cost, obtained from an Edmonton bulk supplier through Tetra Tech procurement, of $0.72/litre, and adding $0.02/kWh for generator servicing and overhaul, for a total cost of $0.215/kWh.

Table 21.11 shows the estimated average power consumption by main area of use.

Table 21.11 Power consumption

Area	**Average MWh/year**	**%**
Mine	10.5	23.5
Mill	25.2	56.5
Surface Facilities	8.9	20.0
Total	44.6	100.0

Estimated annual diesel fuel consumption is as indicated in Table 21.12 (LOM average).

Table 21.12 Annual diesel fuel consumption

Area	million litres	% of total
Power generation	12.0	85.1
Mobile Equipment	1.6	11.3
Mine Air Heating	0.5	3.5
Total	14.1	100

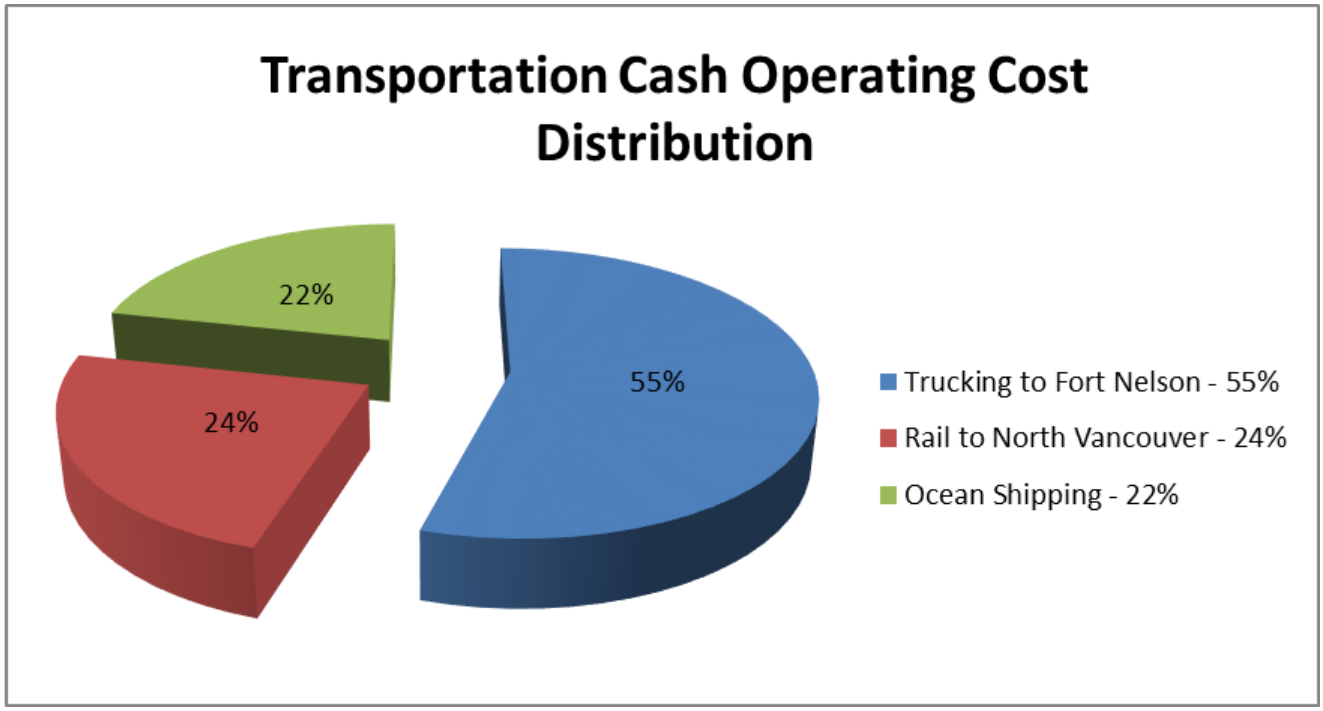
21.3.4 Transportation cost

The cost of hauling concentrates from Prairie Creek Mine site to Fort Nelson was estimated by Allnorth to be approximately $137 per wet metric tonne (wmt) of concentrate, and includes the cost of trucking, road maintenance and operating a transload facility at Fort Nelson.

CZN obtained cost estimates from CN Rail and Kinder Morgan, which indicated that the cost to transport concentrate from Fort Nelson to North Vancouver and transfer to an ocean freighter would be approximately $85/wmt of concentrate. Cliveden estimated ocean freight and insurance to Asia would be approximately $29/wmt of concentrate.

The total transportation cost is estimated as $65.00/dmt of mill feed, which includes approximately $33/t for road/truck transportation, $24/t for rail and trans-loading and $8/t for ocean freight with the major cost components are indicated in Figure 21.7.

Figure 21.7 Transportation cash operating cost distribution



For uniformity, transportation costs included are as estimated in early 2015, based on shipping from the Port of Vancouver to eastern Asia. These costs do not reflect volatility in shipping rates or differences in shipping rates to Australia or Northern Europe. Transportation costs have declined since early 2015, largely as a result of lower fuel costs.

22 Economic analysis

The key parameters and assumptions used in the base case economic model and the highlight production and economic results are summarized in Table 22.1.

Table 22.1 Highlights of the 2016 PFS Update

Mine and Mill Parameters		Concentrates	Average Tonnes/Year	Average Grade	Payability
Total ore mined (million tonnes)	7.6	Zinc concentrate	60,000	Zinc: 59%	Zinc: 85%
Mining rate (tonnes per day)	1,350			Silver: 136 g/t	Silver: 70%
Milling rate (tonnes per day) after DMS	960	Lead concentrate	55,000	Lead: 65%	Lead: 95%
Life of Mine (years)	17			Silver: 824 g/t	Silver: 95%

Life of Mine Statistics				
	Ore Grade Initial 10 Years	Ore Grade LOM	Mill Recoveries	Average Annual Contained Metal
Zinc	10.0%	8.9%	83%	86M lbs***
Lead	9.8%	8.3%	88%	82M lbs***
Silver	154 g/t	128 g/t	87%	1.7M oz***

Project Assumptions Base Case			
Zinc price	US$1.00/lb	Exchange Rate	$1.25CDN:$1.00US
Lead price	US$1.00/lb	Discount Rate	8%
Silver price	US$19.00/oz		

Operating and Capital Costs			
Operating Costs**	LOM $/t ore mined	Capital Costs	$M
Mining	79	Pre-production capital	216
Processing	41	Contingency	28
Site Services	22	Total Pre-production Capital	244
G&A	23	Sustaining Capital	70
Total On-site Costs	165	Working Capital	36
Transportation*	65		
Total Operating Costs**	230		

* Includes truck, rail, handling and ocean shipping ** Does not include treatment, refining charges or royalty	*** Metal contained in both lead and zinc concentrates

Economic Results	Pre-tax	Post-tax
Cash Flow Undiscounted ($M)	710	431
NPV @ 8% ($M)	284	155
IRR (%)	22.5	17.9
Payback period (years from first revenue)	4	5
Average annual EBITDA ($M)	64	

Note: Rounding of numbers may influence totals.

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 2016 dollars.

Inputs into the financial model were primarily the responsibility of CZN 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. CZN collaborated with Tom Morrison to generate surface operating cost, mine workforce, mill and surface workforce and mine operating and capital cost estimates. TT was responsible for metallurgical balance, mill and surface capital cost estimates and mill operating estimates. AMC completed the mine schedule production and part of the mine operating cost estimate in collaboration with Tom Morrison.

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 2016 Q1 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)	$1.00
Lead ($/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 2016 dollars.
- Exchange rate equal to CDN$1.25 to US$1.00
- Life of mine equal to 17 years (16 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 $30 million, not including a contingency of $6 million, is calculated to be the equivalent of three months of operating costs 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 $243.6 million, including a contingency of $27.5 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 Year 1 ($M)	Project Year 2 ($M)	Total Cost ($M)
Mine development	-	34.5	34.5
Process plant[1]	6.7	12.5	19.2
Site services[2]	11.5	30.9	42.4
All season road[4]	16.3	41.9	58.2
Construction indirects[3]	14.8	25.6	40.4
Owner's costs[5]	4.5	4.4	8.9
Reclamation security (Pre-production)	4.8	7.7	12.5
Total (excluding contingency)	**58.6**	**157.5**	**216.1**
Contingency	14.7	12.8	27.5
Total Pre-Production Capital	**73.3**	**170.3**	**243.6**

1. Includes dense media separator, structural upgrading, instrumentation, flotation circuit upgrade, reagent handling and piping.
2. Includes power plant; paste plant, water treatment plant, water storage pond, waste rock pile, camp and housing accommodation, warehousing and facility upgrades.
3. Includes engineering and construction of surface facilities, freight and logistics, initial fills and spares.
4. Includes Liard River crossing and Fort Nelson load-out facility.

22.3 Sustaining capital

Sustaining capital costs (Table 22.4) over the remainder of the mine life are estimated at $70.4 million, including the purchase and rebuild of mobile equipment, estimated at $31.0M, leasing costs of capital items of $29.1M, and reclamation security in the amount of $8.3M. Process and infrastructure sustaining capital was estimated at a total of $8.7M over the life of the mine.

Table 22.4 Sustaining capital summary

Sustaining Capital	LOM $M
Mining	31.0
Capital leases	29.1
Reclamation security	8.3
Infrastructure	8.7
Salvage	(6.7)
Total	**70.4**

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	**LOM**
Tonnes	7,603,590
	($/t)
Mining	78.58
Processing	40.75
G&A	22.58
Site Surface	21.96
Sub-total	**163.87**
Transportation	65.10
Total	**228.97**

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 2016 PFS includes mercury penalties ranging from $700,000 to $4 million per year, with an average of approximately $2 million 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, the Company has generated estimates for indicative “world smelter terms” with respect to treatment charges, penalties, and other terms – see Tables 22.6 and 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 2016 PFS has been prepared assuming average blended indicative treatment charges of US$212 per tonne for zinc sulphide concentrates and US$195 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.

Table 22.6 Zinc concentrates indicative terms used in Economic Model

Zinc Sulphide Concentrate Indicative Terms used in Economic Model	
Payables	
Zinc	Pay for 85% of the final zinc content, subject to a minimum deduction of 8%.
Silver	Deduct 3 ounces per dry metric tonne and pay for 70% of the balance.
Deductions	
Treatment charge	US$212 per dry metric tonne of zinc concentrates
Penalties	Per dry metric tonne
Lead	US$1.50 for each 1% above 3%
Arsenic	US$1.50 for each 0.1% above 0.3%
Antimony	US$1.50 for each 0.1% above 0.3%
Fluorine and Chlorine combined	US$1.50 for each 100 ppm above 200 ppm
Magnesium oxide	US$1.50 for each 0.1% above 0.3%
Silica	US$1.50 for each 1% above 3%
Cadmium	US$2.00 for each 0.1% above 0.25%
Mercury	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	Pay for 95% of the final lead content, subject to a minimum deduction of 3%.
Silver	Pay for 95% of the final silver content subject to a minimum deduction of 50 grams.
Deductions	
Treatment charge	US$195 per dry metric tonne of lead concentrates
Refining charge	Silver: US$1.50 per troy ounce payable.
Penalties	Per dry metric tonne
Zinc	US$0.00 for each 1% above 4%
Arsenic	US$1.50 for each 0.1% above 0.3%
Antimony	US$1.50 for each 0.1% above 0.3%
Chlorine	US$1.50 for each 100 ppm above 300 ppm
Bismuth	US$1.50 for each 10 ppm above 100 ppm
Mercury	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 mining tax. 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.

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 will typically be 9.5% of the value of the output of the mine, 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.

Federal & 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.

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.

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	$64 M	-
NPV (8% discount rate)	$284 M	$155 M
IRR	22.5%	17.9%
Payback period	**4 years**	**5 years**

The 2016 PFS indicates average annual production of approximately 60,000 tonnes of zinc concentrate and 55,000 tonnes of lead concentrate with a combined metal content of approximately 86 million pounds of zinc, 82 million pounds of lead and 1.7 million ounces of silver. The 2016 PFS indicates average annual earnings before interest, taxes, depreciation and amortization of $64 million per year and cumulative EBITDA earnings of $1.024 billion over an initial mine life of 17 years, using Base Case metal price forecasts of US$1.00 per pound for both zinc and lead and US$19.00 per ounce for silver, and an exchange rate of CDN$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$1.00/lb zinc, US$1.00/lb lead, and US$19.00/oz silver, and an exchange rate of CDN$1.25:US$1.00. These long-term price assumptions were selected based on consensus price forecasts published by Consensus Economics Inc. as at February 2016, 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.

Based on the evaluation of consensus forecasts and review of published information from various market commentators, 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 expectations.

The Base Case indicates positive economic results with a pre-production capital expenditure of $244 million and a pre-tax NPV of $284 million at an 8% discount rate (post-tax NPV $155 million).

22.8.2 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%.

It is anticipated that the slightly negative cash flow projections beyond Production Year 13 will be addressed as part of the next project planning stage.

Table 22.9 Prairie Creek Mine Project cash flows

Prairie Creek Mine
Economic Model - AMENDED - September 30, 2016
March 31, 2016 - CAD$ Millions (unless otherwise noted)

Time Periods	Total	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19
Project year		-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
Production *(000's dmt)*																				
Mill Feed	7,603.59			423.06	457.56	470.57	466.26	466.94	446.45	487.51	515.48	509.87	495.17	528.19	564.94	493.35	467.47	433.03	377.76	-
Lead - %	8.33			9.63	10.69	10.28	10.91	10.86	11.56	10.05	9.78	7.90	6.79	8.05	7.33	7.55	4.06	3.51	3.49	-
Silver - g/t	127.58			162.0	176.7	179.1	173.0	168.9	180.4	145.4	151.3	112.5	105.3	121.5	94.9	90.0	53.3	65.5	59.7	-
Zinc - %	8.93			10.14	12.98	10.68	9.01	9.99	11.02	8.97	10.59	8.31	8.63	7.06	6.86	7.77	7.67	7.28	6.10	-
Copper - %	0.25			0.32	0.35	0.37	0.40	0.36	0.39	0.31	0.33	0.20	0.21	0.21	0.14	0.10	0.05	0.11	0.11	-
Total Mill Concentrate Production (000's dmt)	1,823.79			104.03	143.18	128.52	129.58	135.77	142.70	129.16	149.70	118.12	110.39	108.33	109.49	108.26	79.78	71.39	55.38	-
Total Metal Contained in Concentrates *(000's dmt except silver in metric tonnes)*																				
Copper	13.94			1.00	1.21	1.31	1.43	1.29	1.35	1.12	1.29	0.75	0.74	0.79	0.59	0.34	0.16	0.31	0.27	-
Lead	592.24			36.42	45.96	44.14	48.58	48.59	50.15	46.60	48.53	38.07	31.59	38.47	37.37	35.24	17.12	13.62	11.78	-
Silver	844.40			58.23	73.98	73.54	70.56	70.31	73.08	62.48	71.42	49.75	44.90	52.99	44.01	37.73	21.31	22.57	17.55	-
Zinc	626.71			33.35	50.46	42.26	40.61	44.44	47.66	41.95	52.89	40.72	40.05	33.83	36.31	38.02	33.76	29.19	21.22	-
Operations *(CAD$ Millions)*																				
Gross metal value	4,119.15			245.02	332.17	305.08	311.50	320.74	336.48	301.00	344.66	261.31	237.85	246.24	241.52	233.49	157.82	137.76	106.52	-
Smelter deductions	806.29			48.19	66.49	60.73	61.88	63.52	67.15	59.10	69.21	50.48	46.73	46.39	45.38	43.10	30.47	27.13	20.35	-
Gross smelter revenue	3,312.86	0.00	0.00	196.83	265.68	244.35	249.61	257.22	269.33	241.90	275.45	210.83	191.12	199.85	196.14	190.39	127.35	110.62	86.17	
Smelter treatment charges & penalties	547.78			31.46	43.93	38.33	40.02	41.93	44.17	39.80	47.80	35.20	31.38	32.06	33.34	31.00	22.47	19.65	15.24	-
Total Net Smelter Revenue	2,765.08	0.00	0.00	165.37	221.75	206.02	209.60	215.29	225.16	202.10	227.65	175.64	159.74	167.78	162.81	159.39	104.88	90.97	70.93	0.00
Operating Costs																				
Mining	597.48			48.37	45.95	42.88	36.48	44.02	40.34	34.82	33.72	33.95	33.62	32.90	40.96	39.17	32.90	31.91	25.47	-
Processing	309.87			19.37	19.37	19.37	19.37	19.37	19.37	19.37	19.37	19.37	19.37	19.37	19.37	19.37	19.37	19.37	19.37	-
Surface Support	167.00			10.20	10.26	10.32	10.39	10.44	10.49	10.49	10.49	10.49	10.49	10.49	10.49	10.49	10.49	10.49	10.49	-
General & administrative	171.70			11.24	11.43	11.16	11.05	10.95	10.90	10.95	10.80	10.72	10.66	10.61	10.98	10.59	10.48	9.88	9.30	-
Transportation	494.61			28.43	38.22	34.56	34.83	36.37	38.10	34.71	39.84	31.94	30.01	29.50	29.78	29.48	22.34	20.24	16.25	-
Total Operating Costs	1,740.65	0.00	0.00	117.61	125.23	118.29	112.11	121.15	119.19	110.35	114.21	106.48	104.14	102.87	111.58	109.09	95.58	91.89	80.88	0.00
Income before taxes and royalties	1,024.43	0.00	0.00	47.76	96.53	87.73	97.48	94.15	105.97	91.76	113.44	69.16	55.60	64.91	51.23	50.29	9.30	(0.92)	(9.95)	0.00
Taxes and royalties	279.42	-	-	-	1.64	2.20	2.06	7.31	31.64	40.19	34.72	42.98	26.06	21.19	24.63	19.79	19.45	4.06	0.85	0.66
Income from Mining Operations (CAD$ Millions)	**745.01**	**0.00**	**0.00**	**47.76**	**94.88**	**85.53**	**95.43**	**86.84**	**74.32**	**51.57**	**78.71**	**26.18**	**29.55**	**43.72**	**26.60**	**30.50**	**(10.14)**	**(4.98)**	**(10.79)**	**(0.66)**
Total Capital Expenditures (CAD$ Millions)	**314.03**	**73.18**	**170.43**	**30.83**	**14.68**	**9.04**	**8.34**	**2.09**	**1.13**	**1.37**	**0.92**	**1.92**	**1.36**	**1.43**	**0.94**	**0.86**	**0.80**	**0.75**	**(6.04)**	**0.00**
Cash Flows *(CAD$ Millions)*																				
Post-tax Cash Flows - undiscounted (CAD$ Millions) [1]	430.98	(73.18)	(170.43)	(15.86)	66.74	79.84	85.68	84.07	70.56	55.23	71.73	36.62	31.97	40.17	27.62	30.29	1.55	(2.56)	(0.67)	11.59
Discounted Pre-tax Cash Flows (CAD$ Millions)	284.30	(73.18)	(157.80)	(13.60)	54.28	60.30	59.71	57.59	59.64	51.55	53.25	36.87	24.89	24.37	19.21	17.05	6.62	0.44	0.05	3.06
Discounted Post-tax Cash Flows (CAD$ Millions)	154.83	(73.18)	(157.80)	(13.60)	52.98	58.69	58.31	52.98	41.17	29.84	35.88	16.96	13.71	15.95	10.16	10.31	0.49	(0.75)	(0.18)	2.90
Cumulative Post-tax Cash Flows (CAD$ Millions)	**154.83**	**(73.18)**	**(230.98)**	**(244.58)**	**(191.60)**	**(132.92)**	**(74.60)**	**(21.63)**	**19.55**	**49.39**	**85.27**	**102.23**	**115.94**	**131.90**	**142.05**	**152.37**	**152.86**	**152.11**	**151.93**	**154.83**

Project Assumptions & Economics

Lead	US$1.00/lb	Net present value of pre-tax cash flows @8% (CAD$ Millions)	284.30
Silver	US$19.00/oz	**Net present value of post-tax cash flows @ 8% (CAD$ Millions)**	**154.83**
Zinc	US$1.00/lb	Internal rate of return of pre-tax cash flows	22.53%
Exchange rate	$1.25CDN:$1.00US	**Internal rate of return of post-tax cash flows**	**17.88%**

Note 1 - Cash flows are subject to timing differences as discussed in Section 22.1 of this report.

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 $284 million using an 8% discount rate, with an Internal Rate of Return (IRR) of 23%, and a post-tax NPV of $155 million, with a post-tax IRR of 18%, and yielding average annual earnings before interest, taxes, depreciation and amortization (EBITDA) of $64 million per year and cumulative EBITDA earnings of $1.024 billion over an initial mine life of 17 years (16 years of concentrate production) with payback of five years from first revenue.

Figure 22.1 Project cash flow

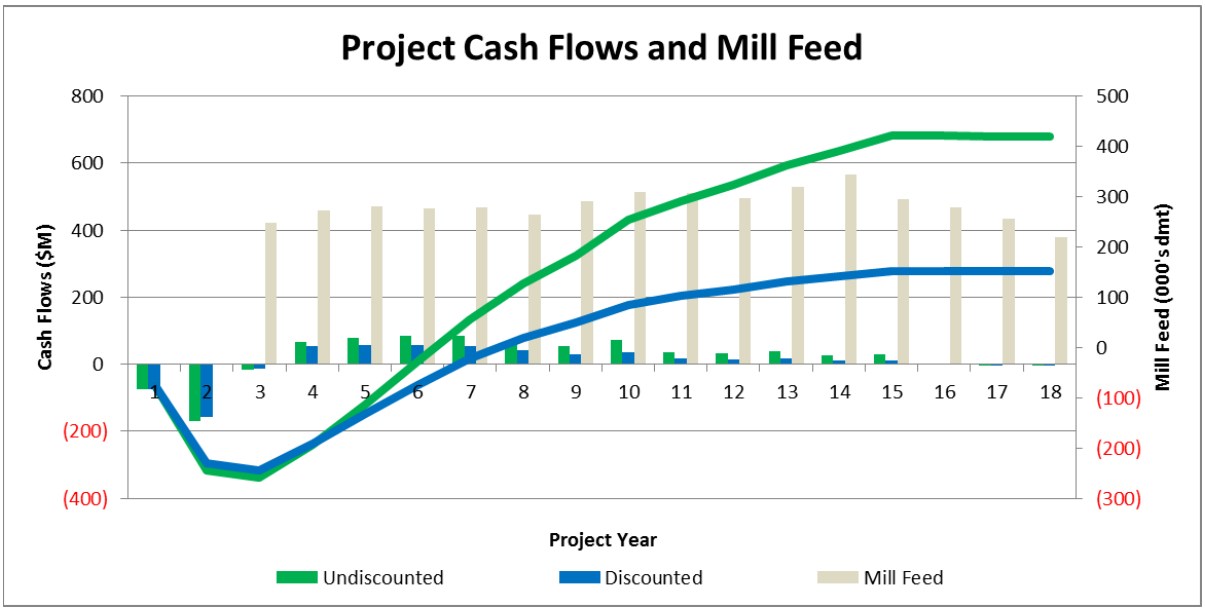
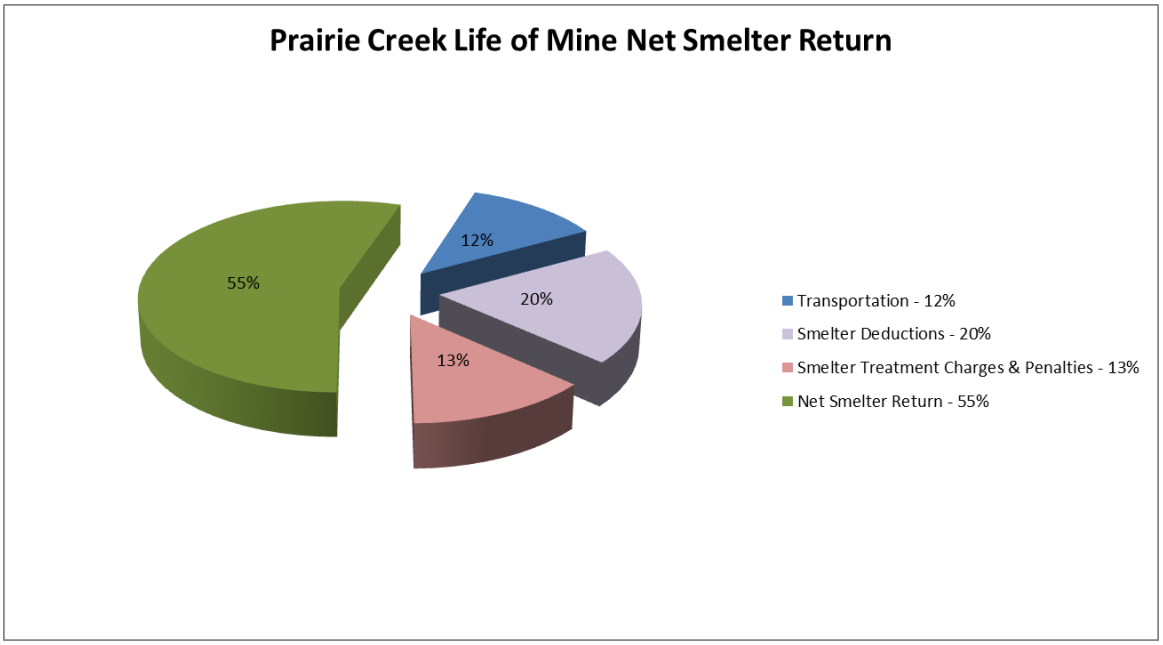


Figure 22.2 shows the net smelter return for the life of mine, which is projected to be 55% of LOM gross metal value after all off-site costs (smelter deductions; smelter treatment charges and penalties; and transportation costs) are incurred.

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 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 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 $272 million and IRR of 24.3%, or a post-tax NPV of $249 million and IRR of 23.1%, respectively.

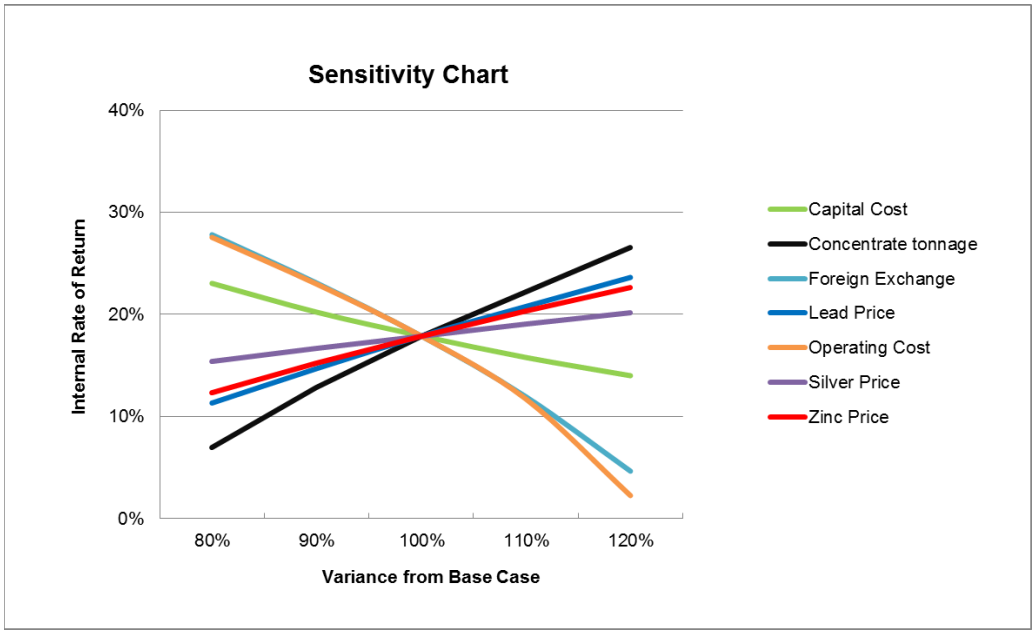
Table 22.10 Sensitivity analysis – Prairie Creek Mine

Metal Price Scenario [1]	90%	100%	110%
Average Annual EBITDA ($M)	43	64	85
Pre-Tax Cash Flow Undiscounted ($M)	379	710	1,041
Pre-Tax NPV @ 8% discount ($M)	107	284	462
Pre-Tax NPV @ 5% discount ($M)	185	405	626
Pre-Tax IRR	14.2%	22.5%	29.7%
Post-Tax Cash Flow Undiscounted ($M)	217	431	639
Post-Tax NPV @ 8% discount ($M)	35	155	272
Post-Tax NPV @ 5% discount ($M)	88	235	377
Post-Tax IRR	10.4%	17.9%	24.3%
Post-Tax Payback Period (years from first revenue)	7	5	3
Exchange Rate Scenario [2]	**$1.125CDN:$1.00US**	**$1.30CDN:$1.00US**	**$1.375CDN:$1.00US**
Average Annual EBITDA ($M)	47	71	81
Pre-Tax Cash Flow Undiscounted ($M)	441	818	979
Pre-Tax NPV @ 8% discount ($M)	140	342	429
Pre-Tax NPV @ 5% discount ($M)	226	477	585
Pre-Tax IRR	15.9%	25.0%	28.5%
Post-Tax Cash Flow Undiscounted ($M)	258	498	598
Post-Tax NPV @ 8% discount ($M)	57	192	249
Post-Tax NPV @ 5% discount ($M)	116	281	349
Post-Tax IRR	12.0%	20.0%	23.1%
Post-Tax Payback Period (years from first revenue)	7	4	3

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 IRR of the Prairie Creek project is most sensitive to operating costs, which is a function of volume of ore mined and metal produced. The project is less sensitive to zinc, lead and silver prices as well as capital costs on a percentage basis. A pre-tax sensitivity chart would show a similar shape to the post-tax scenario.

The 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 the Base Case which is also shown at an exchange rate of 1.375:1). 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			
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	99	-	(42)	5.0	35	(38)	(70)	2.2
0.90	18.00	405	202	121	15.0	233	100	44	11.1
1.00	**19.00**	**710**	**405**	**284**	**22.5**	**431**	**235**	**155**	**17.9**
1.00 [1]	**19.00 [1]**	**979**	**585**	**429**	**28.5**	**598**	**349**	**249**	**23.1**
1.10	20.00	1,016	608	447	29.1	623	366	262	23.8
1.20	21.00	1,322	811	611	35.2	810	493	366	29.0
1.30	22.00	1,627	1,014	774	40.9	1,002	624	473	34.1

1. Foreign Exchange assumed to be $1.375CAD:$1.00US on this line only

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 CDN$1.40:US$1.00, would indicate a pre-tax NPV (8%) of $92 million and IRR 13% (post-tax NPV (8%) $24 million and IRR 10%).

22.10 Summary

The updated 2016 Pre-Feasibility Study is based on optimization work completed over the past three years, including the 2015 underground exploration program at Prairie Creek which increased total Measured and Indicated Resource tonnages by 32%. The updated PFS indicates a Base Case Pre-tax Net Present Value of $284 million using an 8% discount rate, with an Internal Rate of Return of 23%, and a post-tax NPV of $155 million with a post-tax IRR of 18%.

An economic analysis was completed to assess the performance of the Project under the key production, cost and revenue assumptions adopted in the PFS. Sensitivity analyses were performed for variation in metal prices, operating costs, capital costs, and discount 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$1.00/lb zinc, US$1.00/lb lead, and US$19.00/oz silver. These long-term price assumptions were selected based on consensus price forecasts published by Consensus Economics Inc. as at February 2016, 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 (see Section 19). The economic analysis has been run with no inflation (constant dollar basis) and is presented in 2016 dollars.

The 2016 PFS indicates average annual production of approximately 60,000 tonnes of zinc concentrate and 55,000 tonnes of lead concentrate, containing approximately 86 million pounds of zinc, 82 million pounds of lead and 1.7 million ounces of silver. The 2016 PFS indicates average annual earnings before interest, taxes, depreciation and amortization of $64 million per year, and cumulative EBITDA earnings of $1.024 billion over an initial mine life of 17 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 on a year-end basis using a discount rate of 8%.

The net smelter revenues were derived using the 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.

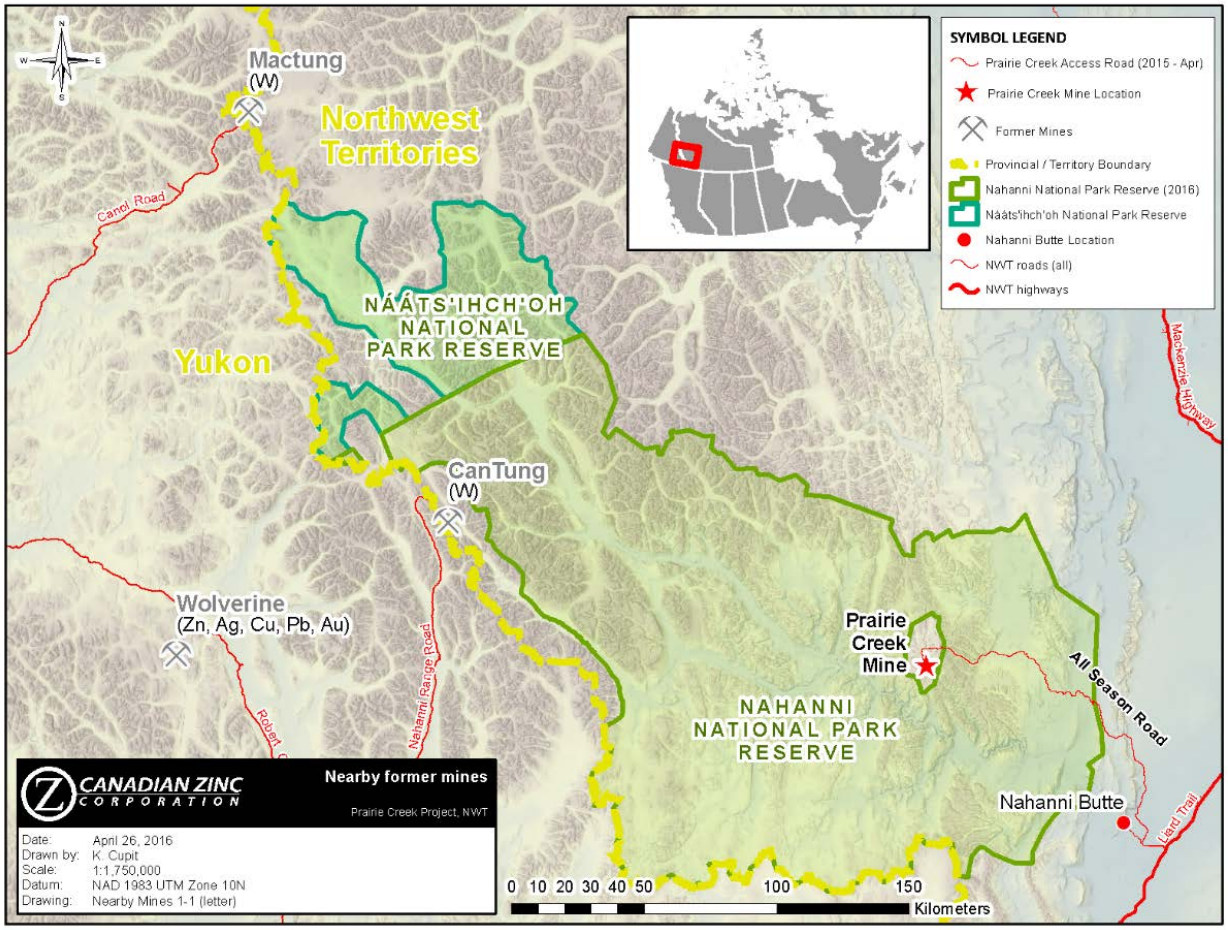
Summary Highlights of the 2016 PFS

- Post-tax Net Present Value, using an 8% discount, of $155M, with a post-tax internal rate of return of 18%, based on base case metal price forecasts of US$1.00/lb for both zinc and lead and US$19.00/oz silver for the Life of Mine production at an exchange rate of $1.25CDN:$1.00US.
- Average EBITDA of $64M per year and cumulative EBITDA of $1,024M over the LOM.
- 17 year mine life based exclusively on a Mineral Reserve of 7.6 million tonnes, grading 8.9% zinc and 8.3% lead, with 128 g/t silver, including a Mineral Reserve in the Main Quartz Vein of 5.2 million tonnes, grading 9.4% zinc, 10.4 % lead and 160 g/t silver.
- Average annual production of 60,000 dmt of zinc concentrate and 55,000 dmt of lead concentrate containing 86M lbs of zinc, 82M lbs of lead and 1.7M ounces of silver.
- Pre-production capital cost is estimated to be $216M, of which $59M will be incurred in Year 1 and $157M in Year 2, with an additional contingency of $28M.
- Average LOM cash operating costs per tonne of ore mined (before transportation costs) are estimated at $165/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 completely 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.



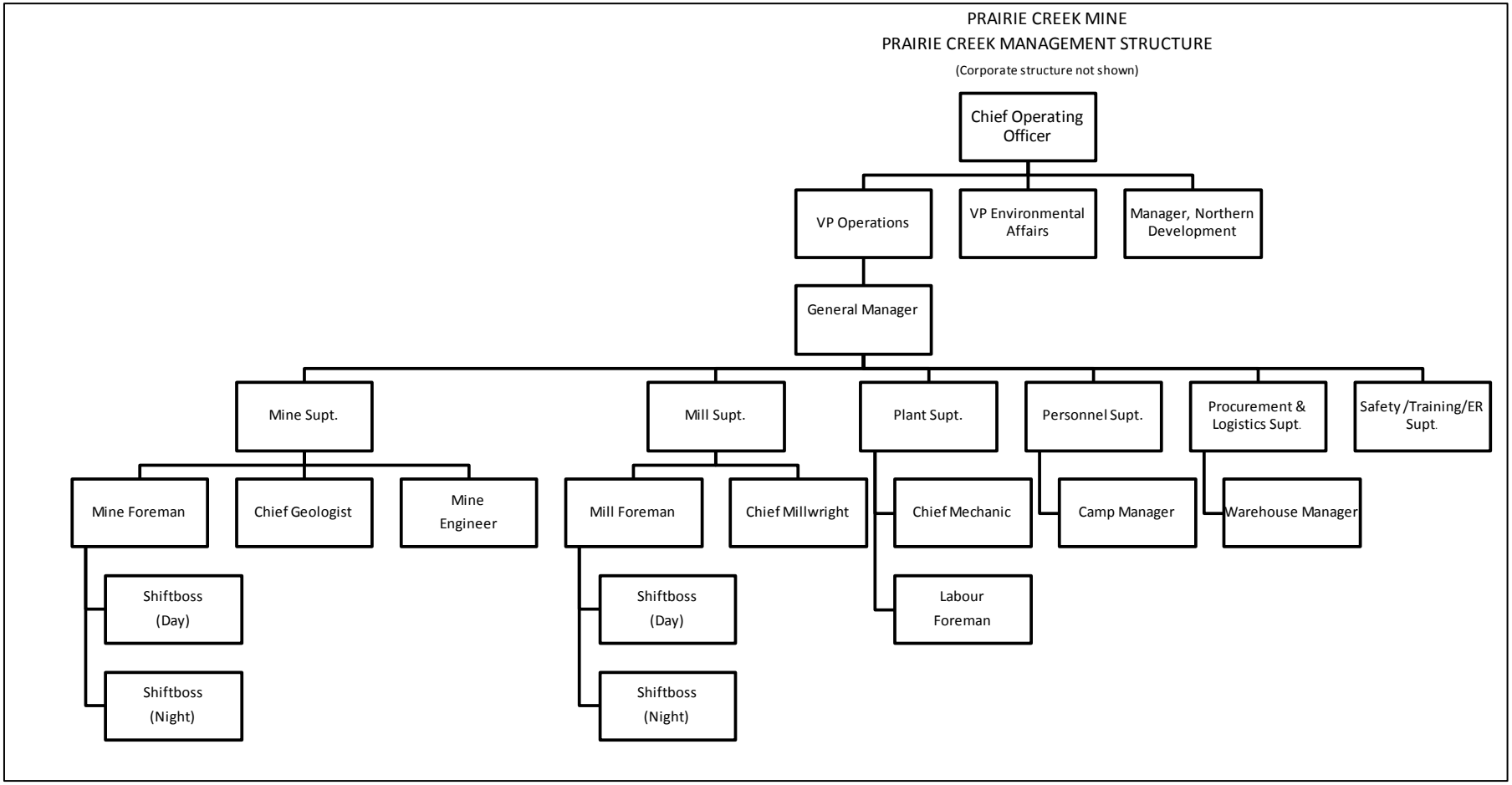
The closest mineral property of significance is the Cantung Mining Property, located 185 km to the west of Prairie Creek. The Cantung Mine was formerly owned by North American Tungsten Corporation Ltd., and operated intermittently since 1968. Mining consisted of both open pit and underground operations 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 November 19, 2015 the GNWT announced that it had acquired the leasehold interests to the mineral rights of North American Tungsten Corporation's undeveloped Mactung Property and the Government of Canada had assumed responsibilities for the Cantung Mine. The Wolverine Mine in the Yukon Territory (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 position will be head office-based.

Figure 24.1 Management structure



Site employees will work a regular fly-in-fly-out rotation to be determined, with travel paid for by CZN.

Organizations of this type comprise: (a) supervisory and technical positions, shared between senior (A Team) incumbents and junior (B Team) assistants, and (b) manual positions, which are occupied continuously. Supervisory and technical positions need effective information transfer between A and B Teams. For manual positions, typically involving shift work, information transfer is less important.

24.1.1 Staffing levels

Currently envisaged average staffing levels are shown in Table 24.1. Additional to the personnel shown below, there will be a trucking workforce largely based in Fort Nelson.

Table 24.1 Staffing levels and camp space requirement

Mine	**A Team**	**B Team**	**Total**
Mine Superintendent	1		1
Mine General Foreman		1	1
Chief Engineer	1		1
Mine Shiftbosses	2	2	4
Mine Engineer	1	1	2
Mine Technologist	1	1	2
Chief Geologist	1		1
Geologist		1	1
Sampler	1	1	2
Development Miners (typical, numbers may vary according to mine plan)	22	22	44
Longhole Drillers	3	2	5
Longhole Blasters	4	4	8
Scoop & Truck Operators	9	9	18
Construction Crew	3	4	7
Backfill Crew	2	2	4
Labourers	4	4	8
Sub-total, Mine	**55**	**54**	**109**
Mill	**A Team**	**B Team**	**Total**
Mill operations			
Mill Superintendent	1		1
Mill General Foreman		1	1
Mill Shiftbosses	2	2	4
Mill Operators	11	11	22
Metallurgical Engineer	1	1	1
Metallurgist	1		2
Mill Clerk	1	1	2
Assayer	3	3	6
Sample & Reagent Preparation Labourer	5	5	10
Sub-total, mill operations	**25**	**24**	**49**
Mill maintenance			
Mill Maintenance Foreman	1		1
Maintenance Planner		1	1
Millwrights	3	3	6
Electricians	2	2	4
Welders	1	1	2
Crane/Equipment Operators	2	2	4
Apprentices/Labourers	2	2	4
Sub-total, mill maintenance	**11**	**11**	**22**
Sub-Total, Mill	**36**	**35**	**71**

Surface Facilities	**A Team**	**B Team**	**Total**
Plant Superintendent	1		1
Chief Mechanic		1	1
Labour Foreman	1	1	2
Plant Engineer	1		1
Maintenance Planner		1	1
Mechanics	5	5	10
Electricians	2	2	4
Welder/Fabricators	1	1	2
Carpenters	1	1	2
Pipefitters	1	1	2
Freight/ concentrate Handlers	2	2	4
Equipment Operators	2	2	4
Labourers/apprentices	2	2	4
Sub-total, Surface Facilities	**19**	**19**	**38**
G&A Group	**A Team**	**B Team**	**Total**
General Manager	1		1
Payroll Supervisor		1	1
Procurement & Logistics Superintendent	1		1
Purchaser		1	1
Warehousemen	2	2	4
Receptionist/Dispatcher	1	1	2
Personnel Superintendent	1		1
IT Technician	1	1	2
Camp Manager	1	1	2
Safety/Training/Emergency Response Superintendent	1		1
Safety/Training/Emergency Response Officer		1	1
Nurse/Paramedic	1	1	2
Environmental Technician	1	1	2
Sub-Total, G&A	**11**	**10**	**21**
Sub-Total, Mine	**55**	**54**	**109**
Sub-Total, Mill	**36**	**35**	**71**
Sub-Total, Surface Facilities	**19**	**19**	**38**
Sub-Total, G&A	**11**	**10**	**21**
Sub-total, Mine, Mill, Surface Facilities, G&A	**121**	**118**	**239**
Camp Operations	14	14	28
Visitors & Contingency	10	10	20
Total	**145**	**142**	**287**

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 & 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 PFS, 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

24.2.1 Strategy

CZN's strategy for project implementation has involved obtaining fixed-price tenders for a number of work packages. These include:

- Mine redevelopment
- Modular camp
- Generators
- Pre-engineered sprung-structure warehouse
- Ball mill motor and drive
- Incinerator
- Cyclones
- Flotation cells
- Reagent system
- Flocculant system
- Air compressors (mill)
- Aeration blowers
- Agitators
- Samplers and analyzers
- Laboratory
- Dense media separation plant
- Paste fill plant
- Water treatment plant
- Pre-engineered buildings
- Site completion general contract
- Winter road construction and maintenance
- Trucking and concentrate haulage
- All-season road design-build

CZN engaged Tetra Tech Inc., a major engineering consulting company, to bring engineering to a stage where firm pricing could be obtained for the above items, with the exception of mine redevelopment, which CZN handled in-house. Tender documents were developed to pre-qualify vendors and contractors, tenders were received and evaluated, and recommendations made on which tenders should be accepted.

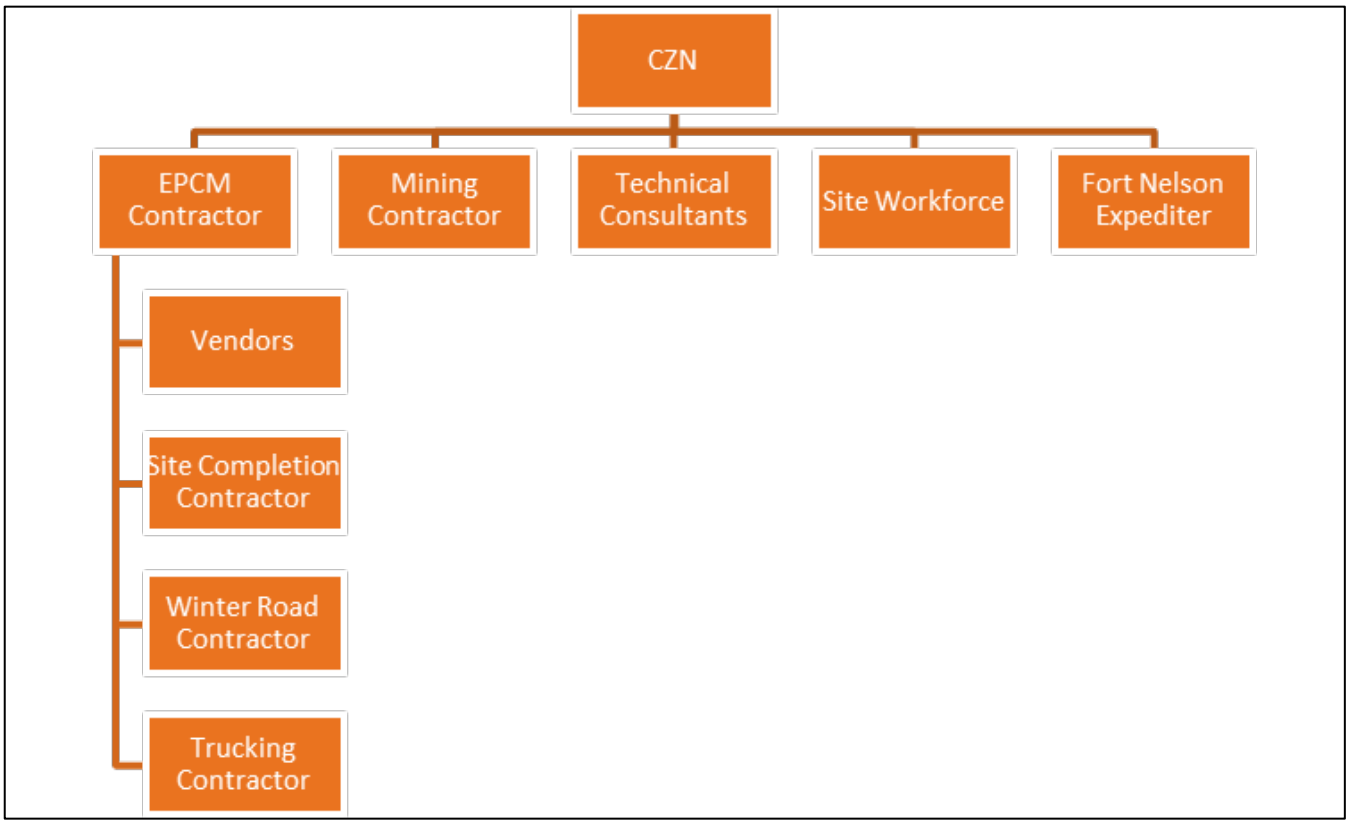
This strategy enabled CZN to advance the project towards production and to obtain capital cost projections carrying greater certainty than could be obtained from engineers' and budget-level estimates. The tendering process was initiated in late 2013 and was substantially complete by the end of 2014. As part of this strategy, vendors were invited to offer financing, which significantly reduced initial capital expenditure. Tenders are valid to the end of 2016.

AMC Mining Consultants provided technical mine design, production scheduling and associated quantities, which could then be put out to tender with pre-qualified mining contractors.

24.2.2 Project execution

Figure 24.2 shows the organization and relationship of the different entities involved in executing the project.

Figure 24.2 Project execution organization chart



CZN will maintain close control over the project implementation process, contracting out only those services that it cannot provide in-house. Specifically, CZN will:

- Carry out site earthworks with its existing equipment fleet and direct-hire workforce.
- Contract mine development to a specialist contractor.
- Contract expediting and laydown services to a qualified company in Fort Nelson.
- Engage technical consultants as needed.
- Engage an EPCM contractor to organize site completion and winter road construction and haulage.

24.2.3 Project sequence

Preliminary works

- Refurbish site mobile equipment
- Refurbish 50 bunkhouse rooms
- Field investigations for:
  - All-season road
  - Waste Rock Pile
  - Water Storage Pond
  - Buildings condition
- Award contracts for:
  - Mine redevelopment and initial production
  - Engineering Procurement and Construction Management (EPCM)
  - Supply of equipment and materials

  - Winter and all season road construction
  - Winter road haulage
  - Design-build all-season road
- Complete detailed engineering and IFC drawings
- Recommission sewage treatment plant
- Upgrade domestic water supply

Procurement phase

- Procurement of long-lead-time items
- Site preparations:
  - Demolitions
  - Foundations for new camp
- Equipment refurbishment
- Award site completion contract
- Repair mill building exterior
- Repair administration building
- Assembly of equipment and materials at Fort Nelson
- Construct winter road
- Site earthworks:
  - Water Storage Pond
  - Waste Rock Pile
  - Foundation excavations

Construction phase

- Construct winter road
- Start all-season road construction
- Haul fuel, materials and equipment over winter road
- Install new 150-person camp
- Mobilize site completion contractor
- Mill building structural examination and repairs
- Construct foundations for new plant
- Complete site construction, including:
  - Power plant
  - Mill completion and upgrade
  - DMS plant
  - Paste fill plant
  - Water treatment plant
  - Reagent handling area
  - Assay laboratory
  - Tank farm refurbishment
  - Instrumentation and telecommunications
- Mine development
- Plant commissioning and integration
- First stope ore production.

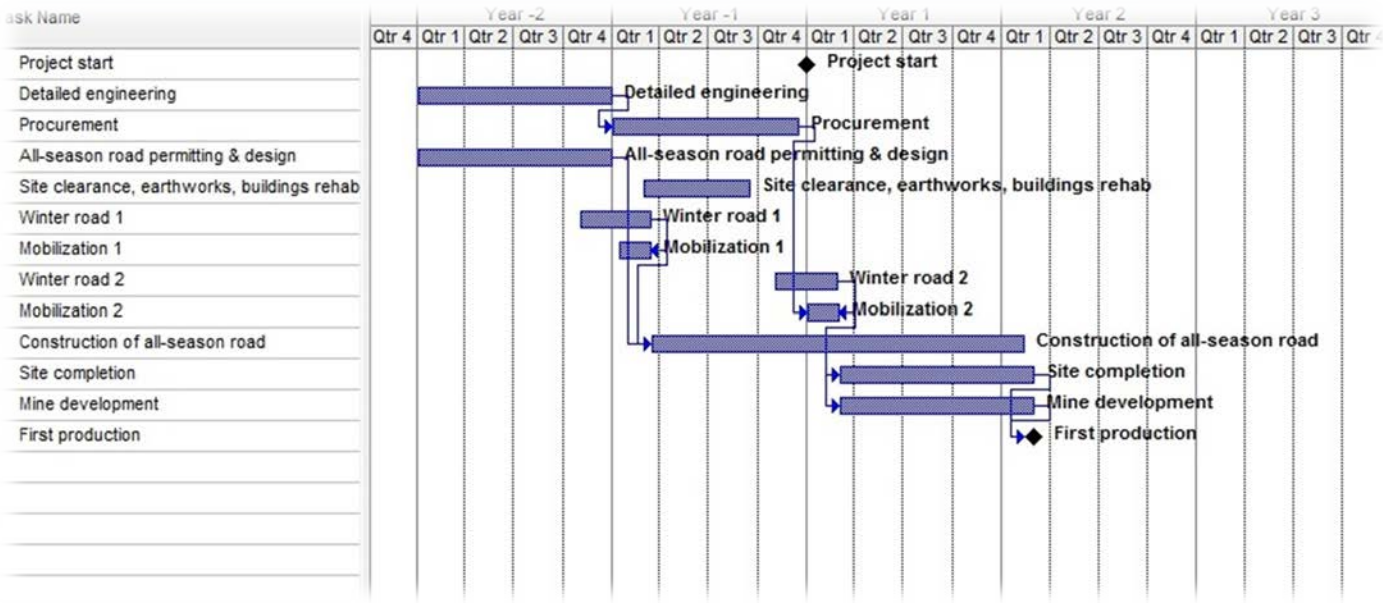
Production phase

- All-Season Road completion
- Start of full-scale concentrate production and shipping
- Inbound haulage of supplies.

Master schedule

The times and dependencies for the above activities may be summarized as follows (Figure 24.3):

Figure 24.3 Project schedule



25 Interpretation and conclusions

Study work on the Prairie Creek Project is at the prefeasibility level of accuracy and detail. The current study results reflect an advancement of the 2012 Prefeasibility Study and are based on updated and increased Mineral Resources and Mineral Reserves and ongoing optimisation projects and studies completed since the date of the 2012 Prefeasibility Study. The key interpretation and conclusions are briefly summarized in the following text.

The Prairie Creek Property contains a high-grade, silver-lead-zinc-copper vein and other lead-zinc deposits and deposit types. The area has been explored since the early 1900s. A mine was developed and processing plant and surface infrastructure built in the early 1980s but low metal prices necessitated closure prior to production. CZN's involvement began in 1992 and it assumed 100% interest in 2004.

The Project is located within, but separate from, the Nahanni National Park Reserve. In 2009, the NNPR was expanded to surround, but exclude, the Prairie Creek Mine, and 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 the operation and development of the Prairie Creek Mine and the management of the Nahanni National Park Reserve.

To date, four main styles of base metal mineralization have been identified on the Property: vein mineralization (MQV), stratabound massive sulphides (SMS), stockwork (STK) and Mississippi Valley type (MVT) mineralization. The MQV, STK and SMS form the basis of the Mineral Resource and Mineral Reserve estimates, with the MQV mineralization being the most significant.

Exploration and associated study work by CZN has previously resulted in a Preliminary Economic Assessment in 2007 and a Prefeasibility Study in 2012 (effective date 15 June 2012, revised date 23 July 2014). The current study is based on the most recent Mineral Resource estimated in September 2015.

This Technical Report discloses the results of an updated Prefeasibility Study based on a re-estimate of the Mineral Resources, and other engineering information developed since the date of the 2012 Prefeasibility Study. The Mineral Resource estimate is based on assays from all underground channel samples and surface and underground drill core collected by CZN since 1992.

Table 25.1 below (see also Tables 14.8 and 14.14) shows total Mineral Resources at 8% ZnEq cut-off grade for this current estimate (September 2015) and those for the estimation conducted in the previous Technical Report of May 2012. There has been a 60% increase in the sum of Measured and Indicated Resources but with a 10% to 15% decrease in average grades. The increase in Measured and Indicated tonnes relates mainly to the successful additional step-out definition drilling completed since 2012 that further extended the detailed MQV resource along strike. The decrease in relative grade of the Measured and Indicated Resource relates mainly to the influence of significantly more tonnage, as Mineral Resources, in the lower grade Stockwork Zone. Similarly, the increase of 13% in Inferred Mineral Resource tonnage from the 2012 study, with grades reduced by approximately 25%, is mainly due to the conversion of almost a million Inferred MQV tonnes into Indicated tonnes, and the reduction in grade due to the incorporation of over 1.6 million new tonnes of lower grade STK. Different methodologies and a more constrained geologic model also accounted for minor differences resulting in reduced grades.

Table 25.1 Mineral Resources estimate comparison – May 2012 and September 2015

SEPTEMBER 2015 RESOURCE ESTIMATE					MAY 2012 RESOURCE ESTIMATE			
TOTAL	**Tonnes**	**Ag (g/t)**	**Pb (%)**	**Zn (%)**	**Tonnes**	**Ag (g/t)**	**Pb (%)**	**Zn (%)**
MEASURED	1,482,000	200	10.8	13.1	1,700,000	155	9.7	12.1
INDICATED	7,222,000	123	8.5	8.7	3,731,000	162	10.5	10.2
MEA & IND	8,704,000	136	8.9	9.5	5,431,000	160	10.2	10.8
INFERRED	7,049,000	166	7.7	11.3	6,239,000	229	11.5	14.5

Mineral Resources were subsequently converted into Mineral Reserves by applying mine planning and scheduling criteria. Primary ZnEq cut-offs were 12% for LHOS and 11% for DAF. 9.7% ZnEq was applied as an incremental stoping cut-off and 7.1% ZnEq was applied as a cut-off for development ore. Table 25.2 shows the 2016 Mineral Reserves estimate and the previous Mineral Reserves estimation from 2012 (see also Tables 15.2 to 15.4).

Table 25.2 Mineral Reserves estimate – 2012 and 2016

MINERAL RESERVES COMPARISON		MINERAL RESERVE 2016			MINERAL RESERVE 2012			TONNES and CONTAINED METAL*		
		ALL	PRV	PRB	ALL	PRV	PRB	ALL	PRV/PRV	PRB/PRB
TOTAL										
Tonnes	t	**7,603,590**	1,373,944	6,229,646	**5,221,000**	1,278,000	3,943,000	146%	108%	158%
Silver	g/t	**127.58**	175.7	116.97	**150.87**	172	144.02	123%	110%	128%
Lead	%	**8.33**	9.41	8.09	**9.49**	9.4	9.52	128%	108%	134%
Zinc	%	**8.93**	12.02	8.24	**9.34**	10.8	8.86	139%	120%	147%

* % Comparison 2016 vs 2012

Total Mineral Reserves have increased by 46% from 2012, with consequent increases in estimated contained metal of 23% (silver), 28% (lead) and 39% (zinc). The increase in Mineral Reserves relates primarily to the successful step-out definition drilling further extending the MQV and, at the same time, adding on new Reserves within the STK zone. The SMS was only reclassified, with very little new data, and remains relatively unchanged in overall tonnage and grade. The STK zone generally carries lower grades than the MQV and this influence has diluted the overall grades by approximately 5% to 18%.

An underground mining operation is envisaged, based primarily on the MQV, but including extraction from the SMS and STK areas. A steady-state ore production rate of 1,350 tonnes per day (nominally 485,000 tonnes per year) is projected, with a 17-year mine life.

The main mining method will be longhole open stoping (LHOS) with paste backfill. Mechanized drift-and-fill (DAF) will be used for the SMS ore, also with paste fill. The aim is to use 100% of flotation tailings as backfill.

Ground conditions are good in existing development underground. The historical workings have stood for about thirty years with minimal ground support. CZN commissioned a geotechnical program at the end of 2013, including mapping and examination of drill core. This program showed that the ground is amenable to longhole mining and was used for rock support design.

Metallurgical tests have proved positive. Dense media separation (DMS) of mine production is envisaged prior to grinding and flotation. Good metal recoveries can be achieved in both sulphide and oxide material, with a cyanide-free reagent suite. According to the test results, the anticipated overall average grade of the blended lead sulphide/oxide concentrate is anticipated to be 67% lead, with an approximately 88% average recovery of total lead in the plant feed. The zinc sulphide concentrate is estimated to be 58% zinc, with an approximately 83% average recovery of the total zinc in the plant feed. An average of 87% of the total silver values 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 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 runoff 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 and a water treatment plant, along with an administration building, a tank farm for diesel fuel storage, accommodations and workshops. CZN plans to rehabilitate and upgrade these facilities.

In particular, five new 1.5 MW diesel powered generator units will provide power and heat for the site. 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 and mine air.

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. The all season road is currently being permitted. Trucking will be interrupted during freeze-up and break-up on the Liard River. The Liard Highway will be used for transport to the railhead at Fort Nelson, whence 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 as backhaul over the same route.

The site workforce will follow a regular fly-in-fly-out rotation to be determined, 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 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. 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 is located in the environmentally sensitive watershed of the South Nahanni River. Particular attention has been paid to potential impacts on water quality that may be caused by Project construction and operations. Prairie Creek itself is a relatively small waterbody that would ultimately receive treated water from the mining operation. CZN has proposed a unique water management plan (flow-weighted load discharge), while at the same time addressing concerns related to other terrestrial and aquatic environments.

A Type “A” Water Licence was issued by the Mackenzie Valley Land and Water Board to CZN on 8 July 2013 that permits mining, milling and processing activities at the Prairie Creek Mine Site, use of local water, dewatering of the mine, and disposal of waste from mining and milling. This Licence and other issued Land Use Permits are part of the overall regulatory approvals for mining, milling, and transport of concentrates and freight on a winter-road basis. Dependence on winter road access only, however, would impose significant limitations and risks on the operation. CZN has, therefore, made an application for an all season road, which is currently in the Environmental Assessment process under the Mackenzie Valley Review Board.

Prior to issuing of main licences and permits in 2013, CZN had been involved in numerous regulatory processes to obtain various Land Use Permits and Water Licences for exploration and development at the Prairie Creek Mine site. Regulatory processes included both normal-course permitting and numerous Environmental Assessments. The ultimately successful regulatory processes entailed significant effort by CZN over an extended period of time.

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 important Impact and Benefits Agreements with each of the Nahanni Butte Dene Band and Liidlii Kue First Nation (Fort Simpson), both part of the Dehcho First Nations. Later that year, CZN negotiated a Socio-Economic Agreement with the Government of the Northwest Territories, 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.

The project capital cost is broken down into pre-production capital and sustaining capital as summarized below in Table 25.3. Pre-production capital will be expended during the first two years of the project.

Table 25.3 Capital cost summary

Description	Total ($M)
Pre-Production Capital	243.6
Sustaining Capital	70.4
Total Capital Cost	**314.0**

Average all-in cash operating costs over the LOM are estimated at $230/t ore mined, as shown in Table 25.4.

Table 25.4 Total operating cost summary

Total Operating Cost	LOM average
Tonnes/ year	7,603,590
	($/t)
Mining	78.58
Processing	40.75
G&A	22.58
Site Surface	21.96
Transportation*	65.10
Total	**228.97****

* includes truck/rail/handling/shipping
** No rounding of values.

The Base Case economic model has been developed using long-term metal price assumptions of US$1.00/lb zinc, US$1.00/lb lead, and US$19.00/oz silver. These long-term price assumptions are based on consensus price forecasts published by Consensus Economics Inc. as at February 2016, 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.

A sensitivity analysis was conducted on the Base Case model to evaluate its robustness against variations in financial parameters, specifically Base Case metal prices +/- 10% and the Base Case foreign exchange rate plus or minus 10%. The financial analysis was conducted with a +/- 10% sensitivity factor 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. 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 $272 million and IRR of 24.3%, or an NPV of $249 million and IRR of 23.1%, respectively.

Tables 25.5 and 25.6 indicate risks and opportunities for the Project.

Table 25.5 Project risks

Preliminary Project Risks		
Risk	**Explanation/Potential Impact**	**Possible Risk Mitigation**
Low commodity prices.	The project is sensitive to low commodity prices negatively affecting economics	A focus on efficiency throughout the operation will minimize the economic impact of lower commodity prices.
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.
Dilution and extraction factors	Excessive dilution of ore is a critical internal risk and can lead to excessive milling costs, lower head grades, lower metal recoveries and increased tailings generation. Inability to extract the planned material from the Reserve is an associated risk.	Mining practices and understanding of ground behaviour will improve with experience, assisted by good supervision and effective ground control. The option exists to cable bolt areas of the hangingwall, reduce stope sizes and implement other dilution control measures.
Ore variability	Bulk sampling has been from accessible workings in the upper part of the MQV in generally high-oxide areas. Existing bulk samples may not be representative of other parts of the deposit.	Systematic sampling of the MQV, STK and SMS vein drifts will provide more representative samples on which to run further metallurgical tests.
Deleterious elements	Higher than expected deleterious components could cause increased smelter penalties and be problematic for marketing concentrates.	Further modeling of deleterious components as a result of vein drift sampling will assist in more confident projections of concentrate qualities. As a backup, mercury leaching experiments and economic evaluation should be continued.
Water management	Unexpected increases in volumes and poorer quality of site waters may increase the amount of treatment required and expansion of facilities, which, in turn, would increase opex/capex.	Further hydrology modeling will give more confidence in predicting site water quantities/qualities and assist in planning for the operation of the load-based water management system.
All season road	Increases in construction costs and decreases in haulage days would have negative effects on capex/opex	Once the road permit is secured and the final route determined, further assessment of construction costs needs to be completed. Competitive design-build tendering will provide the most reliable and lowest costs.
Site completion	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. Building surveys and materials testing will, however, be advisable.
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. Continue with comprehensive training programs for local people and northern residents. Firm but fair management, incentive bonus systems and an understanding of the importance of morale will minimize the effects of this problem.

Table 25.6 Project opportunities

Project Opportunities		
Opportunity	**Explanation/Potential Impact**	**Possible Benefit**
Substantial increase in daily mining/milling rate.	The resource would support a substantial 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 significant 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.
On-site or near-site deleterious element reduction.	A number of possible methods are still to be tested that could reduce the quantity of penalty elements in the concentrate, which would enhance its value.	Reduction in penalty elements would enhance marketability.
Zinc oxide recovery	Recovering and selling the zinc oxide component of the mineral deposit could bring extra revenue to the mine, given sufficiently high zinc prices.	Further studies need to be completed to enhance the grade of the zinc oxide concentrate and reduce reagent costs to enable the sale of zinc oxide concentrates to benefit the mine economics.
Reduction in cement in paste backfill	A reduction in cement dosage or use of an alternate binder in paste fill.	Reduction of a significant mine operating cost.
Alternate energy sources	With an all season road accessing the mine, alternate energy sources such as LNG/CNG can be considered.	Lower power costs, lower emissions, less environmental risk.
Used equipment	Obtain used equipment for surface and mine. Numerous sites are downsizing or closing and have available equipment.	Lower capex.
Maximize dewatering of mine prior to development.	Less water exposed to contamination in the mine will require less treatment and allows consideration of cheaper explosives, such as ANFO.	Lower mining and water treatment costs.
Additional exploration.	Numerous base metal showings occur throughout the property with the potential to host significant deposits, but these require further exploration.	Potentially extend the LOM through expanding Mineral Reserves and increase the scale of the operation.

The key parameters and assumptions used in the base case economic model and the highlight production and economic results are summarized in Table 25.7.

Table 25.7 Highlights of the 2016 PFS Update

Mine and Mill Parameters		Concentrates	Average Tonnes/Year	Average Grade	Payability
Total ore mined (million tonnes)	7.6	Zinc concentrate	60,000	Zinc: 59%	Zinc: 85%
Mining rate (tonnes per day)	1,350			Silver: 136 g/t	Silver: 70%
Milling rate (tonnes per day) after DMS	960	Lead concentrate	55,000	Lead: 65%	Lead: 95%
Life of Mine (years)	17			Silver: 824 g/t	Silver: 95%

Life of Mine Statistics				
	Ore Grade Initial 10 Years	Ore Grade LOM	Mill Recoveries	Average Annual Contained Metal
Zinc	10.0%	8.9%	83%	86M lbs***
Lead	9.8%	8.3%	88%	82M lbs***
Silver	154 g/t	128 g/t	87%	1.7M oz***

Project Assumptions Base Case			
Zinc price	US$1.00/lb	Exchange Rate	$1.25CDN:$1.00US
Lead price	US$1.00/lb	Discount Rate	8%
Silver price	US$19.00/oz		

Operating and Capital Costs			
Operating Costs**	LOM $/t ore mined	Capital Costs	$M
Mining	79	Pre-production capital	216
Processing	41	Contingency	28
Site Services	22	Total Pre-production Capital	244
G&A	23	Sustaining Capital	70
Total On-site Costs	165	Working Capital	36
Transportation*	65		
Total Operating Costs**	230		

* Includes truck, rail, handling and ocean shipping
** Does not include treatment, refining charges or royalty
*** Metal contained in both lead and zinc concentrates

Economic Results	Pre-tax	Post-tax
Cash Flow Undiscounted ($M)	710	431
NPV @ 8% ($M)	284	155
IRR (%)	22.5	17.9
Payback period (years from first revenue)	4	5
Average annual EBITDA ($M)	64	

Note: Rounding of numbers may influence totals.

The Prairie Creek Property contains a high-grade, silver-lead-zinc-copper vein along with other lead-zinc deposits and deposit types. The Base Case economic model indicates a robust project at consensus forecasts for the long-term prices of lead and zinc, generating a pre-tax undiscounted cumulative cash flow of $710 million, a pre-tax NPV (8%) of $284 million with an IRR of 23%, and a post-tax NPV of $155 million with a post-tax IRR of 18%. Additional project optimization, as recommended in this report, would further enhance the economics.

The development of the Prairie Creek mine offers significant economic advantages. 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 unique opportunity to enhance the social and economic well-being of the surrounding communities. There will be approximately 220 direct full time jobs. 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 Project is considered to be a viable project, based on the Mineral Reserves, mine plan, production and economic parameters determined within the 2016 PFS. AMC recommends that Canadian Zinc proceed with the development of the Prairie Creek Project.

26 Recommendations

EPCM Strategy

As part of its assessment in the Technical Report, AMC has recommended that:

- A front-end engineering and design phase to complete detailed engineering and IFC drawings to definitive feasibility study levels to obtain fixed pricing from construction contractors. This could take 6-9 months at an estimated cost of $4 million[1], plus part of EPCM.
- Based on the Prefeasibility Study design, AMC recommends a future detailed feasibility design and implementation stage of the Project focus on:
  - Confirmation of the cost of re-developing the Mine based on final schedules of quantities and rates sourced under competitive bid conditions.
  - The appointment of appropriately qualified and experienced engineering staff or contractors to manage the technical aspects of the final feasibility, construction and operation of the mine.
  - The development of a construction quality assurance plan for use in the implementation stage of the Project to ensure that the works are constructed as specified.
  - The selection of an appropriately qualified and experienced contractor(s) to construct the works required for the establishment of the mine.
  - The selection of an appropriately qualified and experienced contractor(s) to carry out the operation of the Mine, or, alternatively, the recruitment of staff with the requisite expertise to operate the Mine.

Mine

- Further hydrology studies to improve determination of water storage and conductivity through modelling techniques. This will influence the assessment of water storage requirements and the capacity of the water treatment plant on surface. The estimated cost is $100,000[1]. Critical item.
- Paste backfill binder selection trials to identify the best binder type to meet design strengths. Comprehensive rheology and strength testing program to determine the most appropriate fill recipes for the Prairie Creek Mine. Test results will also allow refinement of pump sizing and power demand. These studies would be carried out on tailings flotation products developed during the Locked Cycle Tests mentioned above; cost estimate $300,000[1]. Critical item.
- Data transfer and automation studies. The mine can benefit from ongoing innovations in data transfer and automation technology. Data transfer can result in better equipment serviceability and more efficient mining operations. Automation can range from the automatic control of fans and pumps to remote and/or autonomous equipment operation. These studies will typically take the form of cost-free proposals from equipment vendors.

Milling/Metallurgy

- Creation of a composite bulk sample from the recent, fresh 2015 underground drilling rejects from the MQV on which to carry out Locked Cycle Tests for better definition of the milling process, recoveries and reagent consumption; cost estimate $200,000[1].
- Conduct variability tests on the samples generated from different mineralization zones, various spatial locations and lithological zones; estimated cost is approximately $250,000[1].
- Further bioleach studies, involving tank-scale mercury bioleaching on a bulk sample Locked Cycle Test zinc concentrate, to determine if the mercury content of the zinc concentrate can be substantially reduced and, if so, at what cost in plant capex and opex; cost estimate $200,000[2].
- High-level assessment of substantially increasing the mill capacity with various scenarios of mining and milling, including possible mill expansion (preliminary capex/opex); cost estimate $100,000[2].
- Testing of metal-sensing technology whereby the metal content of a rock fragment can be instantly detected and the particle rejected to waste or retained for milling; cost estimate $50,000[2]. A metal sensor upstream of the crusher could pay dividends in rejecting waste material before energy is spent crushing it. Additionally, a sensor in tandem with the DMS plant could refine the ore/waste cut-off in the mill.

Site preparation prior to project start-up

- Removal of existing generators from the power house prior to installation of new generators. 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 in preparation for upgrading. This work is on the project critical path schedule and would best be done during one summer. The estimated cost is $900,000[1].
- Site preparation before installation of new equipment, specifically buried utilities and foundations for the new camp and heated warehouse. The cost is estimated at $700,000[1]. Summer work. This work is on the project critical path.

Transportation

- Continuation of permitting application for the all-season road; cost estimate $500,000[1]. Critical item.
- Detailed engineering for all-season road; cost estimate $100,000[1]. This is an allowance for the completion of a tender package. CZN believes that the current design status suffices for a design-build competitive tender. Critical item.

Resource/Reserve development

- Additional diamond drilling delineation of the existing STK resource to better define the distribution and geometry of mineralization for mining consideration; Optional cost estimate $1 million[2]. The maximum amount of geotechnical information will be recovered from core, including oriented core. Optional item.
- Additional diamond drilling of the MQV to enhance mine planning; cost estimate $1 million[2]. The maximum amount of geotechnical information will be recovered from core, including oriented core. Optional item.
- A detailed structural geological study of the Prairie Creek deposit. This would further refine the interpretation of the geological model and also be useful to further property exploration; cost estimate $100,000[2]. AMC noted that part of this work should be gathering further geotechnical information from the area below the 883 mL. Optional item.

Note:
[1] Refer to Table 21.5 Additional Engineering and Preliminary Works
[2] Refer to Table 21.6 Optional Recommendation Items

27 References

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Mackenzie Valley Land and Water Board public registry lists all pertinent documents associated with past and present permitting of Land Use Permits and Water Licences for Canadian Zinc Corporation since the year 2000. Go to www.mvlwb.ca and enter into Public Registry for Canadian Zinc Corporation.

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 Certificates

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 179 West Second Street, North Vancouver, British Columbia V7M 1C5 .
2. This certificate applies to the technical report titled “Prairie Creek Property PFS Update, North West Territories, Canada, Technical Report (Amended and Restated)”, with an effective date of 31 March 2016, subsequently amended September 30, 2016, (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 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 part of Section 1 (Mineral Resource Estimate); part of Section 2 (my QP reference); parts of Section 4 (4.1, 4.2); parts of Section 5 (5.1, 5.2, 5.3); parts of Section 6 (6.1, 6.2, 6.3, 6.4); parts of Section 7 (7.1, 7.2); parts of Section 8 (8.1, 8.2); part of Section 9 (9.1); parts of Section 10 (10.1, 10.2, 10.3, 10.4, 10.5, 10.6); part of Section 11 (11.4); part of Section 12 (12.3); Section 14; Section 23; parts of Section 25 (Mineral Resource-related aspects); parts of Section 26 (Mineral Resource-related aspects); Section 28 (my certificate).
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 previously inspected the Property on two occasions but the visits predated 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: 31 March 2016
Signing Date: 30 September 2016

(Original signed by) Gregory Z. Mosher, P.Geo.

_____

Gregory Z. Mosher, P.Geo.
Principal Geologist
Global Mineral Resource Services

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 General Manager / 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 PFS Update, North West Territories, Canada, Technical Report (Amended and Restated)”, with an effective date of 31 March 2016, subsequently amended September 30, 2016, (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 39 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, on 12-15 September 2016.
5. I am responsible for parts of Section 1 (Mineral Reserves, Mining, mining-related aspects of Recommendations and Conclusions); part of Section 2 (AMC personnel references); Section 15; Section 16 (other than Surface management of Waste, Tailings and Heavy media rejects); parts of Section 25 (Mineral Reserves and Mining-related aspects); parts of Section 26 (Mineral Reserves and Mining-related aspects); Section 28 (my certificate).
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 internal reports since 2012.
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: 31 March 2016
Signing Date: 30 September 2016

(Original signed by) Herbert A. Smith, P. Eng.

Herbert A. Smith, P. Eng.
General Manager / Principal Mining Engineer
AMC Mining Consultants (Canada) Limited

CERTIFICATE OF JIANHUI (JOHN) HUANG, PH.D., P.ENG.

I, Jianhui (John) Huang, PhD., P.Eng., of Coquitlam, BC, do hereby certify:

1. I am a Senior Metallurgist with Tetra Tech WEI Inc. with a business address at Suite 1000, 10th Fl., 885 Dunsmuir St., Vancouver, BC, V6B 1N5.
2. This certificate applies to the technical report titled “Prairie Creek Property PFS Update, North West Territories, Canada, Technical Report (Amended and Restated)”, with an effective date of 31 March 2016, subsequently amended September 30, 2016, (the “Technical Report”) prepared for Canadian Zinc Corporation (“the Issuer”).
3. I am a graduate of North-East University, China (B.Eng., 1982), Beijing General Research Institute for Non-ferrous Metals, China (M.Eng. 1988), and Birmingham University, United Kingdom (Ph.D., 2000). I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (#30898). My relevant experience includes over 30 years involvement in mineral processing for base metal ores, gold and silver ores, and rare metal ores, and industrial minerals. I am a “Qualified Person” for purposes of National Instrument 43-101 (the “Instrument”).

   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 on 21 and 22 January 2014 in the capacity of a Qualified Person.
5. I am responsible for parts of Section 1 (Metallurgy and Processing and Process Operating Costs); part of Section 2 (my QP reference); Section 13; Section 17; part of Section 21 (Mineral Processing cost input); parts of Section 25 (Metallurgy- and Processing-related aspects); parts of Section 26 (Metallurgy- and Processing-related aspects); Section 28 (my certificate).
6. I am independent of the issuer as described in Section 1.5 of 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: 31 March 2016
Signing Date: 30 September 2016

(Original Signed by) Jianhui (John) Huang, PhD., P.Eng.

_____

Jianhui (John) Huang, PhD., P.Eng.
Senior Metallurgist
Tetra Tech WEI Inc.

CERTIFICATE OF HASSAN GHAFFARI, P.ENG.

I, Hassan Ghaffari, P.Eng., of Vancouver, British Columbia, do hereby certify:

1. I am a Director of Metallurgy with Tetra Tech WEI Inc. with a business address at Suite 1000, 10th Fl., 885 Dunsmuir St., Vancouver, BC, V6B 1N5.
2. This certificate applies to the technical report titled “Prairie Creek Property PFS Update, North West Territories, Canada, Technical Report (Amended and Restated)”, with an effective date of 31 March 2016, subsequently amended September 30, 2016, (the “Technical Report”) prepared for Canadian Zinc Corporation (“the Issuer”).
3. I am a graduate of the University of Tehran (M.A.Sc., Mining Engineering, 1990) and the University of British Columbia (M.A.Sc., Mineral Process Engineering, 2004). I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (30408). My relevant experience includes 23 years of experience in mining and plant operation, project studies, management, and engineering. I am a “Qualified Person” for the purposes of National Instrument 43-101 (the “Instrument”).

   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 that is the subject of this Technical Report.
5. I am responsible for part of Section 1 (Processing Capital costs); part of Section 2 (my QP reference); parts of Section 21 (Mineral Processing cost input and 21.2.7); Section 28 (my certificate).
6. I am independent of the issuer as described in Section 1.5 of NI 43-101.
7. I have 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: 31 March 2016
Signing Date: 30 September 2016

(original signed by) Hassan Ghaffari, P.Eng.

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Hassan Ghaffari, P.Eng.
Director of Metallurgy
Tetra Tech WEI Inc.

CERTIFICATE OF ALAN B. TAYLOR, P.GEO.

I, Alan B. Taylor, P.Geo., of Vancouver, British Columbia, do hereby certify that:

1. I am currently employed as a Chief Operating Officer and VP Exploration with Canadian Zinc Corporation, with an office at Suite 1710, 650 West Georgia Street, Vancouver, British Columbia V6B 4N9.
2. This certificate applies to the technical report titled “Prairie Creek Property PFS Update, North West Territories, Canada, Technical Report (Amended and Restated)”, with an effective date of 31 March 2016, subsequently amended September 30, 2016, (the “Technical Report”) prepared for Canadian Zinc Corporation (“the Issuer”).
3. I am a graduate of Brock University (B.Sc. Hons., 1975) and University of Western Ontario (M.Sc. 1983). I am a registered member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, Licence #19838. My relevant experience with respect to lead-zinc Mineral deposits extends over 30 years and includes exploration, Mine geology and Mineral Resource estimations. I am a “Qualified Person” for the purposes of National Instrument 43-101 (the “Instrument”).

   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 numerous times in the last 15 years some in the capacity of a Qualified Person.
5. I am responsible for parts of Section 1 (Introduction, Highlights, Location, Ownership & History, Geology and Mineralization, Exploration and Data Management, Access Road and Transportation Plan, Concentrate Marketing, Permitting, Environmental and Community, Project Metrics, Economic Analysis, Non-mining related aspects of Recommendations and Conclusions); part of Section 2 (all aspects other than individual consultant references); Section 3; parts of Section 4 (4.3, 4.4, 4.5); parts of Section 5 (5.4, 5.5); part of Section 6 (6.5); part of Section 7 (7.3); part of Section 8 (8.3); part of Section 9 (9.2); part of Section 10 (10.7); parts of Section 11 (11.1, 11.2, 11.3); parts of Section 12 (12.1, 12.2); parts of Section 18 (18.2, 18.5, 18.9, 18.10, 18.11, 18.15, 18.21, 18.22, 18.24); Section 19; Section 20; parts of Section 21 (21.2.1 [other than Mine development, Support infrastructure, Site completion], 21.2.5, 21.2.8, 21.2.9); parts of Section 22 (22.1, 22.5, 22.6, 22.7, 22.8, 22.9; 22.10); parts of Section 25 (Location, Geology, Licensing, Transportation, Socio-economics, Cost estimates); parts of Section 26 (Transportation, Resource Reserve Development; Section 27; Section 28 (my certificate).
6. I am non-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.
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: 31 March 2016
Signing Date: 30 September 2016

(Original Signed by) Alan B. Taylor, P.Geo.

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Alan B. Taylor, P.Geo.
Chief Operating Officer / VP Exploration
Canadian Zinc Corporation

CERTIFICATE OF THOMAS A. MORRISON, P.ENG.

I, Thomas A. Morrison, P.Eng., of Vancouver, British Columbia, do hereby certify that:

1. I am contracted to, and carried out this assignment for Canadian Zinc Corporation, with an office at Suite 1710, 650 West Georgia Street, Vancouver, British Columbia V6B 4N9.
2. This certificate applies to the technical report titled “Prairie Creek Property PFS Update, North West Territories, Canada, Technical Report (Amended and Restated)”, with an effective date of 31 March 2016, subsequently amended September 30, 2016, (the “Technical Report”) prepared for Canadian Zinc Corporation (“the Issuer”).
3. I am a graduate with the following academic qualifications: B.A. (2.2) Geography Cambridge University, UK 1971, and ACSM (1) Mining Engineering Camborne School of Mines, UK 1976. I am a registered Professional Engineer with the Association of Professional Engineers and Geoscientists of British Columbia (Membership #14007) and am a Licencee with the Northwest Territories and Nunavut Association of Professional Engineers and Geoscientists. I have worked as a mining engineer in the mineral and civil construction industries for 40 years.

   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 for approximately one week at a time on 4 separate occasions between November 2013 and July 2014 in the capacity of a Qualified Person.
5. I am responsible for parts of Section 1 (Site Infrastructure, Project Execution, Employment, Capital and Operating Costs other than for Processing; and related aspects of Recommendations and Conclusions); part of Section 2 (my QP reference); parts of Section 16 (Surface management of Waste, Tailings and Heavy media rejects); parts of Section 18 (18.1, 18.3, 18.4, 18.6, 18.7, 18.8, 18.12, 18.13, 18.14, 18.16, 18.17, 18.18, 18.19, 18.20, 18.23, 18.25); parts of Section 21 (21.1, 21.2.1 [Mine development, Support infrastructure, Site completion], 21.2.2, 21.2.3, 21.2.4, 21.2.6, 21.3.1, 21.3.2, 21.3.3, 21.3.4); parts of Section 22 (22.2, 22.3, 22.4); Section 24; parts of Section 25 (Infrastructure-related material); parts of Section 26 (EPCM Strategy, Mine, Site Preparation); Section 28 (my certificate).
6. I am independent of the issuer as described in Section 1.5 of 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: 31 March 2016
Signing Date: 30 September 2016

(Original signed by) Thomas A. Morrison, P.Eng.

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Thomas A. Morrison, P.Eng.
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Canadian Zinc Corporation

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