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PRAIRIE CREEK PROPERTY

NORTHWEST TERRITORIES, CANADA

TECHNICAL REPORT

for

CANADIAN ZINC CORPORATION

**Prepared by AMC Mining Consultants (Canada) Ltd
In accordance with the requirements of National Instrument 43-101, "Standards of Disclosure for Mineral Projects", of the Canadian Securities Administrators**

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AMC 712017

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1 SUMMARY

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 on behalf of Canadian Zinc Corporation (CZN or the Company) of Vancouver, Canada. It has been prepared 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).

It discloses the results of a Preliminary Feasibility Study (PFS) which has been carried out to assess the viability of starting up the Prairie Creek Mine (the Mine). The report incorporates the Mineral Resources and Mineral Reserves as at 31 May 2012, which are similar to those resources disclosed in an earlier Technical Report entitled *Technical Report, Prairie Creek Mine, Northwest Territories, Canada,* by A B Taylor P.Geo. of Canadian Zinc, dated 28 December 2011 (2011 Technical Report). The 2011 Technical Report was in turn based on a 2007 Technical Report entitled *Technical Report on the Prairie Creek Mine, Northwest Territories, Canada*, prepared by David M.R. Stone, P.Eng. and Stephen J. Godden, FIMMM., C.Eng. of MineFill Services, Inc (2007 Technical Report). This is the first disclosure of mineral reserves on the Property.

History, Location and Ownership

The Prairie Creek Property contains a high-grade, silver-lead-zinc-copper vein that was explored since the early 1900s and then extensively explored and developed by Cadillac Explorations Limited (Cadillac) from 1966 to 1983. A mine was developed and the processing plant and surface infrastructure were built in the early 1980s, at a cost of C$64 million (1982 money). The operations were engineered and fully permitted to produce and process mineralized vein material at a rate of 1,000 tons per day. The 1982/83 fall in metal prices necessitated closure of the mine prior to production. This closure led to a change of ownership and eventually to the Company's involvement in 1992. Through a series of agreements between 1992 and 2004 the Company established an increasing interest in the property, plant and equipment resulting in a 100% interest in the Property by 2004.

The Property consists of two surface leases, twelve mining leases and one mineral claim totalling 8,218 hectares. Its assets include the Mine, a processing plant, various mine- and plant-related surface infrastructure, various earth moving and mining equipment and numerous mineralized occurrences that are at various stages of exploration and development.

Since acquiring the property in the 1990s, CZN has invested over $45 million, quadrupling the known mineral resource and moving the project through six environmental assessments of various development stages, including, most recently, the proposed operation of the Mine, and obtaining numerous exploration and development permits and licences.

The Property is situated approximately 500 km west of Yellowknife in the Mackenzie mountain range that has an average relief of approximately 300 m and comprises low mountains with moderate to steep sides and intervening narrow valleys, 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.

Year round access to the Property is provided by aircraft to a 3,000-foot gravel airstrip immediately adjacent to the camp. The Property is also accessible by an access road which extends from the Property to the Liard Highway, a distance of 170 km and which was originally permitted for use in the winter months throughout its full length and for year round use for the first 40 km out from the mine site. The Company has been granted a Land Use Permit by Parks Canada for the use of the portion of the road within Nahanni National Park Reserve and has been granted a Land Use Permit by the Mackenzie Valley land and Water Board for the remainder of the road. The road needs to be re-established, and the Company successfully rehabilitated approximately 30 km out from the mine site in the summers of 2008 and 2009. The Liard Highway #7 is the major north-south transportation route, which connects Fort Simpson, Northwest Territories to Fort Nelson, British Columbia.

The primary objectives of the Company is to bring the Mine into production at the earliest opportunity and in pursuit of that objective, to rehabilitate, upgrade and modernize the Mine, inclusive of the processing plant and related site infrastructure, and secure the necessary operating permits.

Geology and Mineralization

The Property geology is dominated by Siluro-Devonian stratigraphy that formed in a paleo-basin adjacent to the ancient North American Platformal sediments. The east-dipping and west-dipping thrusts, define the present margins of the Prairie Creek paleo-basin in which sediments accumulated.

Units within the Prairie Creek paleo-basin underwent structural deformation in the form of folds and faults during regional Laramide deformation. The prevalent regional structural trend is approximately north-south; the Prairie Creek paleo-basin is broken into a series of north-south trending, five to 20 kilometre fault blocks. Canadian Zinc's mineral claims and leases overlie two major fault blocks of sediments: the Prairie Creek Block and western Gate Block.

The Prairie Creek block is located on the southern part of the Property. It is outlined by a one to two kilometre wide, doubly plunging antiform with a north-south trending fold axis. It is underlain by a conformable sedimentary sequence ranging from Lower Ordovician to Silurian in age. The antiform plunges at about 15 degrees to the north, so the geological units young in age to the north.

The mine site is situated on the western flank of the Prairie Creek antiform, referred to as the Main Zone. It is Main Zone mineralization that was and is the focus for mine development and exploitation. The Ordovician, Upper Whittaker Formation is the oldest

geological formation in this area. It is composed of interbedded cherts and dolomites that form the core of the Prairie Creek antiform. The Whittaker Formation is in turn overlain by a large exposure of the carbon-rich graphitic-shales/dolomites of the Road River Formation. The iron-bearing Cadillac Formation shales overly the Road River Formation and are located immediately adjacent to the mine site. The bluff-forming rocks immediately to the west of the mine site are formed by the cherty Arnica Formation which overlies the Cadillac Formation and forms the more resistant hilltops in the immediate vicinity of the Mine site.

The Gate Block is located to the west of the main mining leases and overlies similar type rock assemblages to those found on the Prairie Creek Block. Grassroots exploration was completed on this ground to test for mineralization similar to that found in the Prairie Creek Block.

Four main styles of base metal mineralization have been identified on the Property: vein mineralization, stratabound sulphides (SMS), stockwork (STK) and Mississippi Valley type (MVT) mineralization. Base metal mineral showings occur along the entire 16 kilometre north to south length of the Property.

The most significant mineralization on the Property is the vein 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 of severity, yielding mainly cerussite (lead oxide) and smithsonite (zinc oxide). Silver is present in solid solution with tennantite-tetrahedrite and to a lesser extent with galena. Vein widths vary between less than 0.1 metre and more than 5 m. Overall averages indicate a horizontal thickness, but not a true thickness, of approximately 2.7 m.

The most extensively developed vein is the Main Quartz Vein (MQV). Underground development has proved 940 m of strike length and diamond drilling to date has indicated its continuance for a further 1.2 km. The MQV trends approximately north-south and dips between the vertical and 40 degrees east with an average dip of 65 degrees east. It remains open to the north and evidence from a surface showing suggests it continues for a further four km to the south. Diamond drilling to depth has indicated its continuance, but little information is currently available below an elevation of 600 m amsl (i.e. about 250 m below the Mine Site elevation).

Stockwork (STK) mineralization occurs as a series of narrow massive sphalerite-tennantite veins developed at about 40 degrees difference in strike to the average trend of the MQV. This mineralization has developed in sub-vertical tensional openings formed by primary movement along the main vein structure. The sulphide mineral assemblages are similar to those outlined for MQV material. To date, the STK mineralization has only been located in around the exposure on the 930mL underground workings.

Stratabound (SMS) mineralization occurs at depth beneath the trend of the Prairie Creek Vein System over a strike length of more than 3 km. SMS mineralization occurs close to both the vein system and the axis of the Prairie Creek anticline. An apparent thickness of

28 m of SMS mineralization has been intersected in Main Zone drillholes where it occurs approximately 200 m below 870 metre Level. The vein structure cuts through the SMS indicating the vein mineralization to be younger.

SMS mineralization is generally fine-grained, banded to semi-massive and comprises massive fine-grained sphalerite, coarse-grained galena and disseminated to massive pyrite and very little copper. SMS contains only half as much galena, but substantially more iron sulphide/pyrite than typical vein material. Silver is contained in solid solution within galena.

The MVT mineralization found on the Property is comprised of colliform 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.

Exploration and Data Management

The Company has been involved with semi-continuous exploration activity across the Property since 1992. Limited exploration drilling had been undertaken prior to the Company's initial involvement in 1992. Up to November 2011, the Company had completed a total of 261 surface and underground exploration diamond drillholes totalling 68,744 m of coring. 251 channel samples over 322 m were also completed.

The main focus of exploration and underground development work has been on Main Zone mineralization. A deep drilling exploration program was carried out during 2010/11 seasons to test for the northerly extension of the defined resource. This program successfully intercepted what appears to be the same mineralized structure one and a half km north along plunge of the resource bearing geology.

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. This is reduced to BQ size (36.5 mm) if difficult downhole conditions are experienced. In 2010 a new higher capacity HTM-2500 diamond drill rig was airlifted to the property for use in the deep drilling program. Average core recoveries are variable and in 2007 ranged between 97% recovery in the SMS mineralization to 80% for the MQV mineralization. Core recoveries have been consistently recorded since 2006. Bulk density measurements have been obtained through direct measurement and calculations. While acceptable for the purposes of this report, AMC recommends that this area be reviewed prior to the next block model.

Drill samples and underground channel samples are air-freighted, in charter aircraft, from the Mine Site airstrip to Fort Nelson, B.C., from where they are transported by Greyhound bus to the assay laboratory (Acme Labs, ISO 9001-2000 accredited) in Vancouver, B.C. Acme Labs (ISO 9001-2000 accredited) has carried out the majority of the sample assays since the Company's first involvement in 1992. It is currently the only laboratory used by the Company for purposes of sample assaying. 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 finish.

Quality Assurance/Quality Control (QA / QC) samples are submitted by CZN with the regular samples for analysis, including blanks, duplicates and blind standards. Blank and duplicate samples are taken every ten drillcore samples; standards are sent, at the supervising geologist's discretion, after a mineralized zone is sampled. AMC believes that the data collection and handling followed normal industry practice and the data is fit for purpose. However, while QA/QC samples were inserted and the results have been observed in the historical assay certificates, it is not clear if any analysis of the historical data was carried out. This has been remedied in the recent programs but these do not influence the current Mineral Resource.

Mineral Resources and Mineral Reserves

The existing Mineral Resource discussed in the 2011 Technical Report, which was based on the 2007 Technical Report, was reviewed and restated by AMC. This estimate is based on the same data as that of 2007. Minimal work has been carried out within the Main Zone resource area since 2007. A reclassification from the Indicated to Inferred category of a multiple high-grade drill intercept at the northern end of the resource model has resulted in a slight overall reduction in the tonnes and grade of the Indicated category and a slight increase in the Inferred category.

The database comprises both underground and surface drill holes in addition to channel samples collected by the Company since 1992. Channel samples are treated as drillholes. A total of 224 drillholes and 943 channel samples were used in the resource estimate.

Two block models were created: 1) encompassing the MQV and STK solids and 2) for the SMS solid. Block values were computed by the inverse distance to the second power (ID2) in all cases. Three passes were performed for zinc, lead and silver for SMS, and included copper in the case of MQV and STK. Specific gravity was interpolated from collected data for the MQV and SMS, while the STK was assigned a value of 3.31.The summary results of the estimate for the three zones combined, at a cut off of 8% Zn equivalent (Zn Eq) are shown in Table 1 below.

Table 1 Mineral Resources at 31 May 2012

Classification	Tonnes (M)	Zn (%)	Pb (%)	Ag g/t	Cu (%)
Measured	1.700	12.1	9.7	155	0.28
Indicated	3.731	10.2	10.5	162	0.32
Measured + Indicated	5.431	10.8	10.2	160	0.31
Inferred	6.239	14.5	11.5	229	0.57

Notes:
1. Mineral Resources are stated as of 31 May 2012.
2. Mineral Resources include Mineral Reserves.
3. Stated at a cut-off grade of 8% Zn-Eq based prices of $1.30/lb for both zinc and lead, and $35/oz for silver.
4. Average processing recovery factors of 78% for Zn, 89% for Pb and 93% for Ag.
5. Average payables of 85% for Zn, 95% for Pb and 81% for Ag.
6. $ Exchange rate = 1 CD/USD.

The Prairie Creek deposit contains a significant amount of Inferred Resources which, upon further delineation, has the potential to double the life of the mine.

A portion of the Mineral Resources was converted to Mineral Reserves through application of suitable dilution factors in stoping blocks (averaging 22% for MQV and 10% for SMS) utilizing the cut-and-fill mining method for MQV and room and pillar for SMS. A Mineral Reserve of 5.2 million tonnes, grading 9.4% Zn and 9.5% Pb, with 151 g/t Ag has been estimated.

Due to the high grade nature of the deposit, the majority of the vein resource will be mined, allowing for 97% of all of the Measured and Indicated vein resources to be converted to Mineral Reserves and 57% of all Measured and Indicated Resources within the SMS mineralization to be converted to Reserves.

Table 2 Mineral Reserve Estimate for Prairie Creek Mine

Zone	Class	Tonnes (M)	Zn (%)	Pb (%)	Ag (g/t)
Main Quartz Vein	Proven	1.278	10.8	9.4	172
	Probable	3.140	8.7	10.5	165
	Proven and Probable	4.418	9.4	10.2	167
Stratabound	Probable	0.803	9.5	5.7	62
Total Mineral Reserves		5.222	9.4	9.5	151

Notes:
1. Mineral Reserves are stated as of May 31, 2012.
2. Mining cut-off grade of 10% Zn-Eq based upon total variable operating cost of $162/t including mining, processing and transportation.
3. Metal prices assumed are Zn = $1.10/lb, Pb = $1.10/lb and Ag = $28/oz.
4. Average processing recovery factors of 75% for Zn, 88% for Pb and 92% for Ag.
5. Average payables of 85% for Zn, 95% for Pb and 81% for Ag.
6. $ Exchange rate = 1 CD/USD.

Mining

The Mine will be an underground operation primarily based on the MQV. Three levels of underground adits (970 mL, 930 mL, 880 mL, collectively known as the upper mine) have already been established and these are targeted to be mined early in the life of the proposed operation. As mining progresses to depth, mining feed generated from the MQV will be supplemented by the deeper SMS deposit, both deposits being also accessed by a single ramp development.

Mining will be conducted primarily using mechanized cut-and-fill on the narrow vein structure, with potential use of the room-and-pillar mining method on the SMS material. The previously developed shrinkage stopes will be converted to the cut-and-fill method. Paste backfill will be used and the aim is to use 100% of flotation tailings in the backfill. An average mining rate of 1,350 tonnes per day of ore is targeted. During full production, approximately 500,000 tonnes of ore per year will be mined over an 11 year life of mine.

Access to the mine will be primarily through the existing 870 portal. Underground development, existing from the 1980s will be fully utilized to help minimize the amount of pre-development required to achieve mine operation. Limited geotechnical work within the mine has been conducted to date. Ground conditions in existing development underground are good and the existing workings have stood unsupported for close to 30 years with minimal bolting. A geotechnical program has been planned for the summer of 2012 to confirm the geotechnical characteristics of the lower mine.

The MQV structure constitutes the main conduit for water to access the mine, and significant quantities of water pass through the vein. The Mine will be wet and managing groundwater will be a significant aspect of the operation. Presently natural groundwater drains out of the 870 mL portal during the summer season at an average of 20 litres per second (L/s). When in full production, the Mine is estimated to produce up to 100 L/s of water. All water discharged from the mine will either be sent to the mill as feed water or be pumped into the currently existing tailings pond facility which will be revised and converted into a Water Storage Pond and then sent to the proposed new water treatment plant.

Underground development prior to production will comprise of tunnel enlargement and extension at various points of the existing levels. Stope access development and installation of service utilities will also be conducted during pre-production. The pre-production phase is expected to take approximately 12 months and be performed by a contractor. All remaining major development within the mine will be finished by the end of year 5 (after four years of production) and will be conducted by the owner.

In order to achieve 100% disposal of tailings underground, no development rock will be left underground. Only dense media separation float material waste rock will be placed back in the mine, as aggregate for paste backfill running surfaces. There will be a need to temporarily store tailings during times when there are limited stopes available for backfill. Although the waste rock is considered Non-Acid Generating (NAG) due to its high content

of carbonate material, appropriate precautions will be taken to prevent and mitigate any leaching that occurs from surface runoff through the waste rock pile.

Currently on site there exists close to 50,000 tonnes of oxidized mineralized material stockpiled at surface. Further assessment of this stockpile is warranted before any revenue can be allocated towards it since it has been broken and exposed on surface for over 30 years.

Metallurgy, Processing

Metallurgical tests conducted to date proved positive and generated satisfactory simulated results of anticipated actual operations in the production of mineral concentrates at the Mine. Good metal recoveries can be achieved in both sulphide and oxide material, with a reagent suite that does not include cyanide products. The test results showed marketable concentrates can be produced although penalty elements, including antimony, arsenic and mercury, would unavoidably report to the final concentrates.

The test results indicate that the anticipated overall grade of the blended lead sulphide/oxide concentrate assayed 67% lead, with an 88% recovery of total lead in the plant feed, and the zinc sulphide graded 58% Zn with a 75% recovery of the total zinc in the plant feed. An average of 92% of the total silver values in the plant feed was recovered within the lead and zinc concentrates.

As the Mine was fully permitted, though never achieved production, existing infrastructure is substantial. It includes a processing plant that was 90% complete at mine closure in 1982, and a 1.5 million tonne capacity tailings impoundment, power plant, and water treatment plant. There are plans to rehabilitate and upgrade the processing plant, power plant, and water treatment plant.

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

A new dense media separation (DMS) circuit, at 85 tph capacity, will be installed into the crushing circuit to process -1/2” sized material. Indications from metallurgical testing are that the DMS plant will reject an average of 27% of the waste at minimal metal losses, hence mining input at production rates will be 1,350 tpd and, after passing through DMS plant, will produce approximately 1,000 tpd of material to be processed in the grinding/flotation circuit of the mill.

Infrastructure

Five new 1.5 MW diesel powered generator units will provide power and heat for the site. These self-contained, pre-commissioned power generator units will be located adjacent to the mill. Maximum power load for the site is estimated at 4,674 kW and diesel fuel will be the primary energy source required to operate the generators. 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.

100% of the tailings from the mill will be placed permanently underground in a form of paste backfill mix generated from the new paste backfill plant. The remainder of the DMS reject and mine development waste will report to a Waste Rock Pile Facility, located 700 m behind the mill off the Prairie Creek floodplain.

A detailed transportation plan and schedule has been developed incorporating use of winter road access and transfer facilities. New storage facilities will be built at site to temporarily store concentrate when the winter road opens. The Tetcela Transfer Facility will be established at a mid-point along the access road as a temporary storage area for concentrate prior to the ice bridge being established each winter over the Liard River. The Liard Transfer Facility located on the NWT highway system will act as an inbound/outbound storage area for both supplies and concentrate and for all-season access to railhead in Fort Nelson, B.C., where a rail siding facility is planned. All building costs have been incorporated into the PFS.

The 184 km long winter road with two transfer facilities will provide temporary surface access to the site for a minimum of 60 days of the year.

Formal smelter arrangements have not been agreed to at the present time; however, normal course treatment charges and penalties for deleterious elements have been applied.

Environmental and Social Issues

In December 2011, the Mackenzie Valley Environmental Impact Review Board (the Review Board), the primary authority responsible for all environmental assessment and review throughout the Mackenzie Valley in the Northwest Territories, approved the proposed operation of the Mine. The Review Board issued its Report of Environmental Assessment and Reasons for Decision for the Company's proposed Prairie Creek Mine (the EA Report) and submitted the Report and Decision to the Federal Minister of Aboriginal Affairs and Northern Development.

The Review Board concluded that the proposed development of the Mine, including the list of commitments made by the Company during the proceedings, is not likely to have any significant adverse impacts on the environment or to be a cause for significant public concern. The Review Board therefore concluded that an environmental impact review of this proposed development is not necessary and that the Mine project should proceed to the regulatory phase for approvals.

In a Decision dated June 8, 2012, the Minister of Aboriginal Affairs and Northern Development, on behalf of the responsible Ministers with jurisdiction, including the Minister of the Environment, the Minister of Fisheries and Oceans, the Minister of Environment and Natural Resources, the Minister of Transport Canada and the Minister of Environment and Natural Resources of Government of the Northwest Territories, advised the Review Board

of the Decision that the Ministers will not order an environmental impact review of the proposed development of the Prairie Creek Mine, nor will they refer the proposal to the Minister of the Environment for a *Canadian Environmental Assessment Act* joint panel review.

In May 2012, the Mackenzie Valley Land and Water Board (MVLWB) also issued a work plan indicating that a draft "A" Water Licence for Prairie Creek should be issued for review by end of 2012.

A detailed socio-economic assessment was completed 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 confirmed economic prospects. The majority of the economic and social impacts will be generated through the participation of local labour and business 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 than 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.

Project Metrics

Table 3 Key Project Metrics

Parameter	Unit	Metric
Mine type	-	Underground
Total mined	Mt	5.2
Average grade milled		
Zinc	%	9.4
Lead	%	9.5
Silver	g/t	151
Mining rate	tpd	1,350
Milling rate	tpd	1,000
Project life	years	11
Estimated recoveries		
Zinc	%	75
Lead	%	88
Silver	%	92
Average annual metal production		
Production of zinc concentrate	t	60,000
Production of lead concentrate	t	60,000
Zinc	M lbs	76
Lead	M lbs	90

Silver	M oz	2.2

Capital and Operating Costs

The PFS is based upon capital pricing as of the second quarter of 2012. The level of accuracy of the capital cost estimates is +/-20% for the PFS. The general capital cost breakdown for Prairie Creek is indicated in Table 4.

Table 4 Capital Cost Estimates

Description	Total ($M)
Pre-Production Capital	192.87
Working Capital	41.15
Sustaining Capital	11.34
Total Capital Cost	235.36

Operating costs are summarized in Table 5.

Table 5 Operating Cost Estimates

Total Operating Cost	Year 1	Year 5	Year 1	Year 5
	($/t)	($/t)	($M/year)	($M/year)
Processing	$37.25	$37.17	$7.93	$18.57
Mining	$81.22	$72.10	$17.29	$36.03
G&A	$10.59	$10.83	$2.26	$5.41
Site Surface	$30.72	$23.67	$6.54	$11.83
Transportation	$51.31	$60.30	$10.92	$30.13
Total	$211.10	$204.07	$44.94	$101.97

Economic Analysis

An economic analysis with a +/- 10% sensitivity factor centering on the Base Case outlines the average annual EBITDA, NPV, IRR payback period and are shown on a pre-tax and pre-finance basis in Table 6. The base case shows a Pre-tax Net Present Value, using an 8% discount, of $253 M, with an internal rate of return of 40.4% and payback period of three years. Metals prices used were US$1.20 /lb in the short term and US$1.00 /lb in the long term for both lead and zinc, and US$28.0 /oz in the short term and US$26.0 /oz in the long term for silver.

Table 6 Economic Analysis

	Low Case	Base Case	High Case
Metal Price Scenario	90%	100%	110%
Average Annual EBITDA* $M	$47	$66	$84
Pre-Tax NPV (undiscounted) $M	$303	$493	$683
Pre-Tax NPV @ 8% discount $M	$140	$253	$366
Pre-Tax IRR	27.4%	40.4%	52.8%
Pre-Tax Payback Period (years)	3.8	3.0	2.5

* Annual average EBITDA does not include year 1 of production

Within the cash flow model, revenue is recognized as the concentrate is generated and does not account for any time delay between shipment and payment for concentrate.

The pre-tax IRR is sensitive to zinc, lead and silver prices as well as to capital and operating costs on a percentage basis.

Risks and Opportunities

Major Risks:

- Significant reduction in metal prices.
- Increase in fuel cost significantly in excess of offsetting increases in metal prices.
- A shortened winter road hauling season that could affect the ability to complete the annual concentrate removal and mine re-supply.
- Periods of abnormally cold weather over extended periods of time creating surface-related operating problems.
- Paste delivery sequencing problems creating surface storage issues inconsistent with operating permits.
- Water treatment plant disruption which may cause effluent quality outside compliance limits necessitating the temporary suspension of operations.

Major Opportunities:

- Continued road upgrades and bridge installations that would reduce winter road installation and maintenance costs and also decrease transport costs.
- Cycloning of the DMS feed screen undersize to upgrade feed to the grinding circuit.
- Copper / lead separation to produce a Cu/Ag concentrate that could be air-shipped all year around from the site.

- Use of a form of longhole / sublevel stoping rather than cut and fill in zones of wider mineralization which could reduce operating costs, increase mine productivity and allow for more tailings to be stored underground (less cement required during backfill).
- Use of higher capacity underground equipment to increase efficiency and productivity Reduction in mine dilution in the next stage of design.

Other Relevant Data

The Project is located close to (but outside of) the Nahanni National Park Reserve. In 2009, the Nahanni National Park Reserve 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.

Recommendations (with estimated cost where relevant)

Project optimization

There are a number of recommendations listed in the PFS including:

- Examine opportunities to improve efficiencies in transport, scheduling and logistics on the winter road.
- Consider financial alternatives to purchasing of significant equipment and other procurement.
- Review opportunities for early completion of construction, engineering and mine development programs to reduce start-up times required.
- Consider financial arrangements targeted to further reduce Working Capital needs.
- Undertake additional drilling programs, particularly towards the north end of the deposit, to increase the confidence level in the estimated resources and reserves and to identify additional resources. $2 million
- Modify the mine plan to include increased resources and identify areas of the mine amenable to lower cost bulk mining methods. Optimization of mine schedule and equipment utilization should follow.
- Undertake further studies aimed at upgrading the zinc oxide concentrate to a commercial grade and producing a copper / silver concentrate to maximize potential future revenues. $100,000

Data management and resource modelling

- Review bulk density measurement methods.

- Resolve slight discrepancies between drillhole data and wireframes and lack of extrapolation of the MQV wireframe beyond the southernmost drilled section.
- Restrict rotation of the block model to no more than orientation or use an unrotated model.
- Estimate antimony, arsenic and mercury in the next resource update, as these metals report to the final concentrates.
- Model and estimate the percentage oxide component in the MQV mineralization.
- Review the high grade capping policy.
- Composite chip samples to equal lengths and decluster the data.

The inclusive cost for all modelling is estimated at $50,000.

Mining

- Undertake currently planned geotechnical drilling program in the summer of 2012 to confirm ground support requirements and stability control during operations. $50,000
- Incorporate more detail into the dump pocket design for run-of-mine ore. $10,000
- Prepare a mine dewatering plan to ensure safety at the face during operations. $10,000
- Review and refine equipment selection to identify if there is any merit in allowing for variations in the size of the drills and scoops (smaller and larger) or if standardization of the equipment size (as is currently planned) optimizes efficiencies.

Environment and social issues:

- Continue consultation activities with aboriginal groups, government agencies (e.g., Parks Canada) and other interested stakeholders to maintain positive working relationship.
- Implement environmental studies as required to address information requests from the MVLWB as per its Directive and Work Plan received on 11 May 2012.

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FIGURES

2 INTRODUCTION

2.1 General and Terms of Reference

This Technical Report on the Prairie Creek Property (the Property), located approximately 500 km west of Yellowknife in the Northwest Territories, Canada, has been prepared by AMC Mining Consultants (Canada) Ltd (AMC) of Vancouver, Canada on behalf Canadian Zinc Corporation (CZN or the Company) of Vancouver, Canada. It has been prepared 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).

It discloses the results of a Pre Feasibility Study (PFS), which has been carried out to assess the viability of starting up the Prairie Creek Mine (the Mine). The report incorporates the Mineral Resources and Mineral Reserves as at 31 May 2012, which are similar to those resources disclosed in an earlier Technical Report entitled *Technical Report, Prairie Creek Mine, Northwest Territories, Canada,* by A B Taylor P.Geo of Canadian Zinc, dated 28 December 2011 (2011 Technical Report). The 2011 Technical Report was in turn based on a 2007 Technical Report entitled *Technical Report on the Prairie Creek Mine, Northwest Territories, Canada*, prepared by David M.R. Stone, P.Eng. and Stephen J. Godden, FIMMM., C.Eng. of MineFill Services, Inc (2007 Technical Report). This is the first disclosure of mineral reserves on the property.

CZN is the 100% owner of the Mine that is located in the approximate centre of the Property, which consists of two surface leases, twelve mining leases and one mineral claim. The Property assets include the Mine, a processing plant, various mine and plant-related surface infrastructure, various earth moving and mining equipment, and numerous mineralized occurrences that are at various stages of exploration and development.

2.2 The Issuer

CZN is a publicly traded mining exploration company that is based in Vancouver, Canada and with offices in Toronto and Fort Simpson (NWT). The Company is listed on the Toronto Stock Exchange under the trading symbol: CZN and on the Over the Counter Bulletin Boards, OTCOB 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 Mine, as well as in more general exploration across the Property.

CZN was originally incorporated in British Columbia, Canada, on 16 December 1965 under the Corporations Act of British Columbia. The Company changed its name to San Andreas Resources Corporation (San Andreas) on 29 August 1991 and to Canadian Zinc Corporation on 25 May 1999.

2.3 Report Authors

The names and details of persons who prepared, or who have assisted the Qualified Persons in the preparation of this Technical Report, are listed in Table 2.1. The Qualified Persons meet the requirements of independence as defined in NI 43-101.

Table 2.1 Persons who Prepared or Contributed to this Technical Report

Qualified Persons Responsible for the Preparation of this Technical Report						
Qualified Person	**Position**	**Employer**	**Independent of CZN**	**Date of Last Site Visit**	**Professional Designation**	**Sections of Report**
Mr JM Shannon P.Geo	Geology Manager Principal Geologist	AMC Mining Consultants (Canada) Ltd	Yes	25 April 2012	BA Mod, MA P.Geo.	Section 1-12, Parts of 14, 23-26
Ms D. Nussipakynova P.Geo.	Geology Manager Principal Geologist	AMC Mining Consultants (Canada) Ltd	Yes	none	B.Sc., M.Sc. P.Geo.	Part of Section 14
Mr JB Hancock P.Eng.	Consulting Mining Engineer	Barrie Hancock & Assoc. Inc.	Yes	July 2008	B.Sc., P.Eng.	Section 15, 16 and 19
Mr B MacLean P.Eng.	Principal Consultant	SNC-Lavalin Inc.	Yes	March 2008 June 2008 July 2011	B.A.Sc M.A.Sc., P.Eng.	Section 13, 17, 18, 21, 22 and parts of, 25-26
Other Experts who assisted the Qualified Persons						
Expert	**Position**	**Employer**	**Independent of CZN**	**Visited Site**	**Sections of Report**	
Mr PR Stephenson P.Geo	Principal Geologist	AMC Mining Consultants (Canada) Ltd	Yes	No visit	Peer Review for all sections	
Mr R Pope	Partner	Dillon Consulting Limited	Yes	No visit	Section 20	
Mr A Taylor P.Geo.	COO and VP Exploration	Canadian Zinc Corporation	No	Numerous	Liaison for CZN	

Inspections of the property were undertaken by Qualified Persons J.M. Shannon, Principal Geologist with AMC, J.B Hancock, independent mining consultant and B MacLean of SNC-Lavalin. The scope of the visits covered the data collection, geology and mining aspects of the project, and included inspections of drill core, underground exposures, mining infrastructure, surface facilities, and mill building. As drilling was not ongoing when J.M. Shannon visited site, core handling, logging and sampling was not observed.

Much of the background information on the Property, such as the history, past exploration, past drilling, past sampling and past assaying is reported in the Company's January 2001 Mine Scoping Study report that is filed on SEDAR. This in turn was used in the 2007 and 2011 Technical Reports, where the information was updated. This report uses some of this information in addition to reporting on the PFS findings.

3 RELIANCE ON OTHER EXPERTS

AMC has reviewed the CZN file in regard to legal ownership or title to the Property. However, the Qualified Persons are not legally qualified to assess the validity of the Company's Property leases or claims.

For Section 20 the environmental review was carried out by Richard Pope of Dillon Consulting Limited, (Dillon). To the extent permitted under NI 43-101, the Qualified Persons disclaim responsibility for this section of the report.

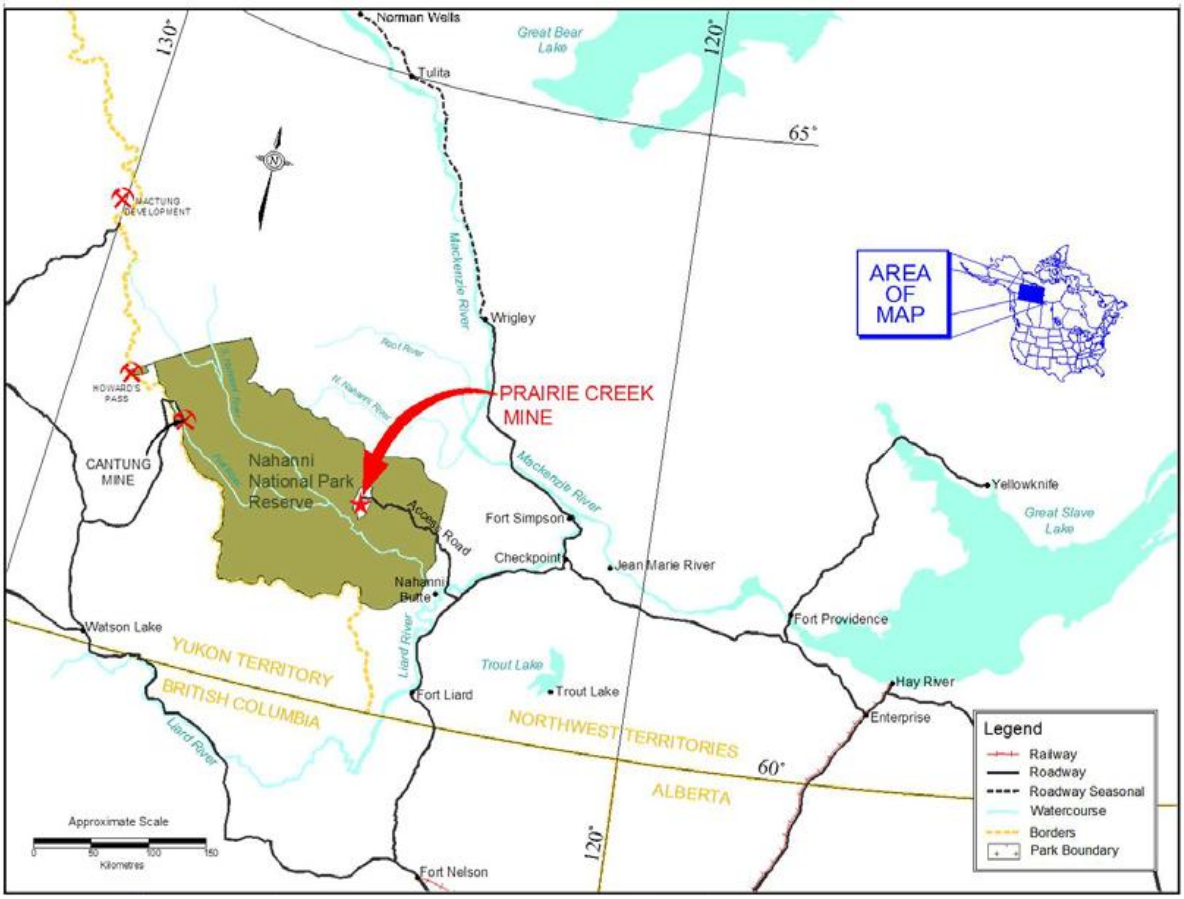
4 PROPERTY DESCRIPTION AND LOCATION

4.1 Property Location

The Property is located in the Northwest Territories mining district of the Northwest Territories, Canada (NWT), near the Yukon border, at latitude 61° 33' North and longitude 124° 48' West. The nearest NWT communities include the Nahanni Butte that is approximately 90 km to the southeast, Fort Liard that is approximately 170 km to the south and Fort Simpson that is approximately 185 km to the east. Yellowknife, the NWT capital and administrative center, is some 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.

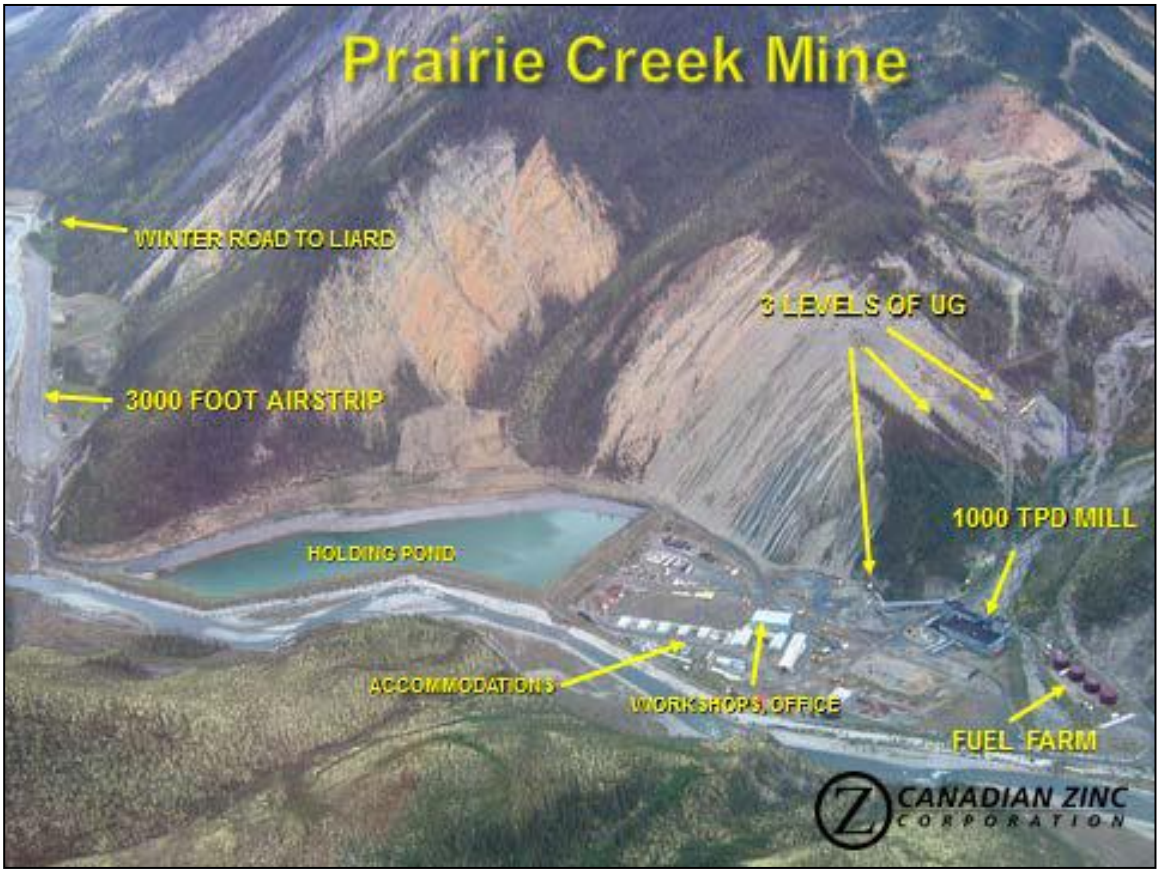
The location of the site within the Northwest Territories and relative to the nearest centers is shown in Figure 4.1.

Figure 4.1 Location of the Prairie Creek Mine Property



The Mine site which is highlighted in Figure 4.1, is located in the Mackenzie Valley, within the watershed of the Naha Dehé (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 seven km downstream, and eighteen km upstream, of the Mine. Since the expansion of NNPR, the Prairie Creek mine is located in an approximate 300 km2 area of crown owned land that is now surrounded by, but not included in, the expanded Nahanni National Park Reserve. A photograph giving a view over the site is included below as Figure 4.2.

Figure 4.2 The Prairie Creek Mine Site

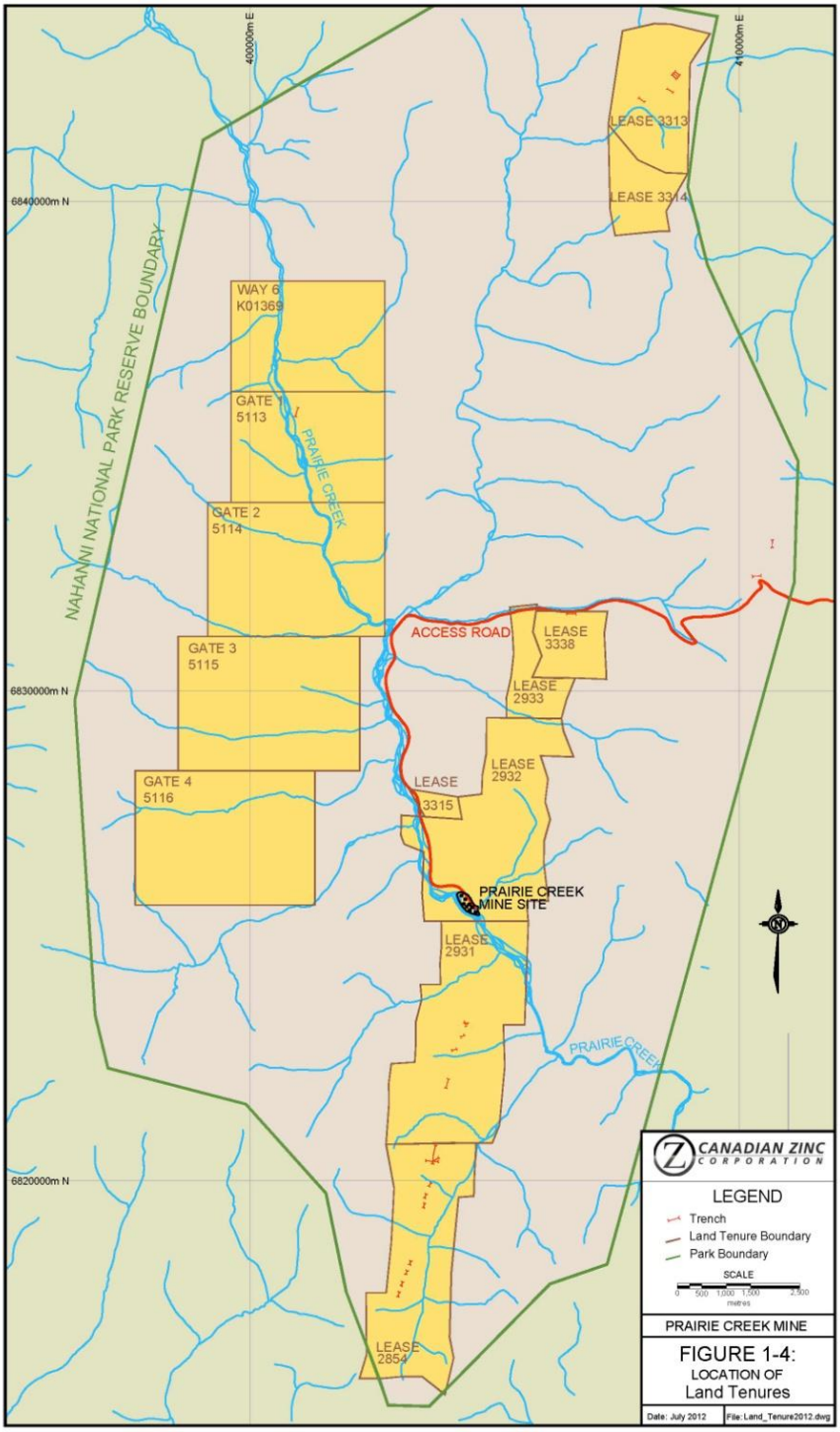


4.2 Property Description and Ownership

The Property consists of a 100% interest in the mining leases, surface leases and staked mineral claims which are described in Section 4.3.

The grand total of land holdings, including mining leases, surface leases and mineral claims at Prairie Creek, is now 8,218 hectares. All of the above leases and claims are in good standing at the date hereof. These claims lie within Crown Land which is in turn surrounded by the Nahanni National Park Reserve 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 and Surface Leases

The mining leases are renewed on a 21 year basis and presently show expiry dates ranging from August 2019 to July 2032.

The surface leases, which are presently held in a care and maintenance basis, are held from the Department of Indian Affairs and Northern Development (now referred to as Aboriginal Affairs and Northern Development Canada) on a renewable ten year basis. The most recent term ended on 31 March 2012, but in a letter dated 22 March 2012 Aboriginal Affairs and Northern Development Canada placed the surface leases in an overholding tenancy for one more year pending the Ministers' acceptance of the Mackenzie Valley Environmental Impact Review Board (MVEIRB) report. In a Decision dated June 8, 2012, the Minister of Aboriginal Affairs and Northern Development, on behalf of the responsible Ministers with jurisdiction, advised the MVEIRB of the Decision that the Ministers will not order an environmental impact review of the proposed development of the Prairie Creek Mine, nor will they refer the proposal to the Minister of the Environment for a *Canadian Environmental Assessment Act* joint panel review.

The application for permits for operations is presently being processed by the MVLWB and it is indicated that new surface leases for production purposes at the Prairie Creek Mine will be issued.

The four gate mineral claims were staked in 1999. In August 2010 a perimeter land survey was completed on the gate mineral claims resulting in an adjusted total surface area for the new gate mining leases of 2,776 hectares. New mining leases for the gate areas were received 16 February 2011 and are dated 9 September 2009 and have a term of 21 years up to 9 September 2030.

Way 5 mineral claim is the only such claim current as the Way 1, 2, 3, 4 and 6 mineral claims were allowed to lapse due to the lack of significant anomalies worthy of further exploration.

The Prairie Creek Mine is located on land claimed by the Nahanni Butte Dene Band of the Dehcho First Nations (DCFN) as their traditional territory. The DCFN are 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 the Company's Leases and Claims

Property Type	File Number	Name	Expiry Date	Area (ha)
Surface Leases	95F/10-5-5	Minesite	See above	113.6
	95F/10-7-4	Airstrip	See above	18.2
Total Area	-	-		131.8
Mining Le as e s	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 Area	-	-		7,354.94
Mineral Claims	K01369	WAY 5	1 November 2013	731.57
Total Area	-	-		731.57
Grand Total	-	-		8,218.31

4.4 Existing Environmental Liabilities

Existing environmental liabilities at the Prairie Creek Property are covered through various security deposits posted to government agencies. There is a reclamation and closure plan with an attached security deposit associated with the two surface leases which contain all the existing and proposed infrastructure. These surface leases are in the process of being converted from a care and maintenance status to a production status and it is expected that a new reclamation and closure plan with a requirement for a new security deposit / bond will be required to be posted prior to operations.

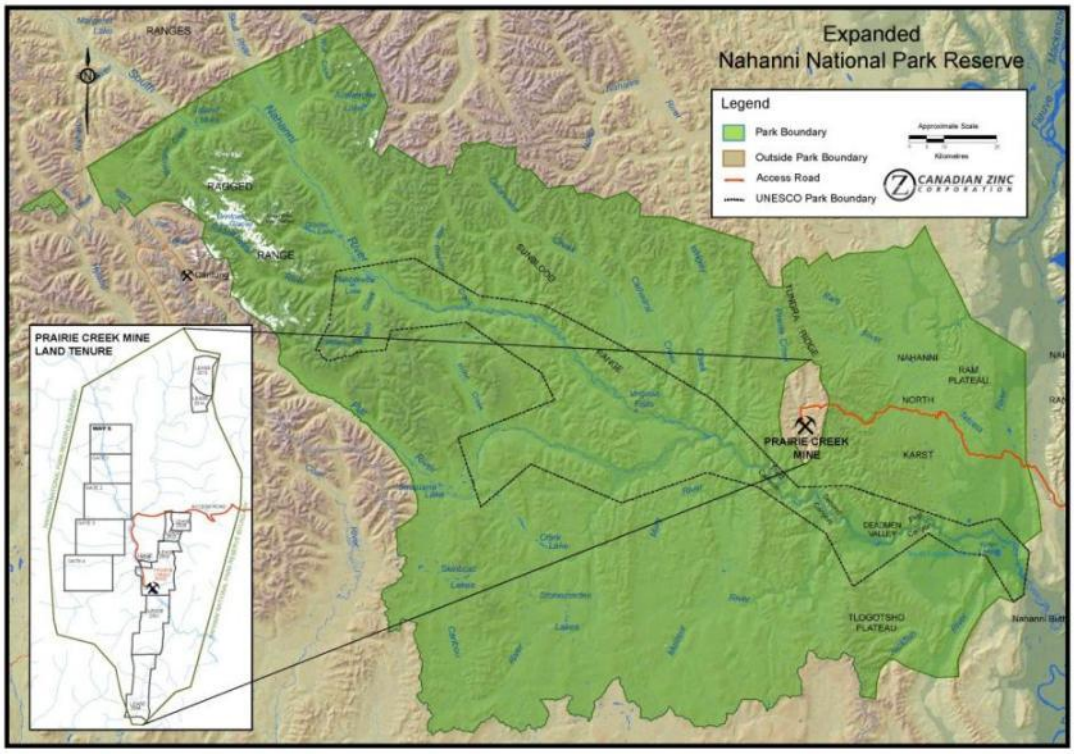
Existing land use permits and water licences issued by the MVLWB also have security deposits associated with them to ensure reclamation is carried out. There is also a letter of credit posted to Department of Fisheries to facilitate some local fish compensation resulting from re-establishment of the main road bed.

4.5 Nahanni National Park Reserve

The Nahanni National Park Reserve was created in 1972, specifically for the purpose of setting aside the South Nahanni River for wilderness recreational purposes. Exploration activity at Prairie Creek had been ongoing for many years prior to 1972 and underground development was well advanced at that point in 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 NNPR. 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 (see Figure 4.4).

Figure 4.4 Claims 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 Canadian Zinc entered into a Memorandum of Understanding (MOU), valid for three years, with regard to the expansion of the Nahanni National Park Reserve 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 Canadian Zinc agreed to work collaboratively to achieve their respective goals of managing Nahanni National Park reserve and an operating Prairie Creek Mine. The details of this are discussed in Section 24.1.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Physiography and Vegetation

The Mine site is located in the Mackenzie Mountain Range that has an average relief of approximately 300 m and comprises low mountains with moderate to steep sides and intervening narrow valleys. The Mine site is located at an elevation of 850 m above mean sea level. The 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 in the upper parts of the mountains.

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, where towns are serviced by scheduled commercial airlines. A 1,000 metre gravel airstrip is located on the flood plain of Prairie Creek, approximately one kilometre north of the Mine site, and is shown in Figure 5.1.

Figure 5.1 Prairie Creek Mine Airstrip

The Liard highway is the closest major transportation route to the Project area, which connects Fort Nelson, B.C. to Fort Simpson, NWT. A 170 kilometre long winter road from the Blackstone crossing on the Liard highway was constructed in 1980. In 1981 and 1982 the road was utilized to transport the bulk of the building materials, supplies and equipment onto 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 successfully transported to the Mine Site during this period.

A detailed transportation plan and schedule have been developed incorporating the use of winter road access and transfer facilities. New storage facilities will be built at site to temporarily store concentrate when the winter road opens. The Tetcela Transfer Facility will be established at a mid-point along the access road as a temporary storage area for concentrate prior to the ice bridge being established each winter over the Liard River. The Liard Transfer Facility located on the NWT highway system will act as an inbound / outbound storage area for both supplies and concentrate and for all-season access to railhead in Fort Nelson, B.C. where a rail siding facility is planned.

The 184 km long winter road with two transfer facilities will provide temporary surface access to the site for a minimum of 60 days of the year.

5.3 Climate

The climate in the general project area is sub-Arctic, insofar as it 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 station measures precipitation, temperature, wind speed and wind direction. A mean annual temperature of minus 2.8 degrees Celsius was recorded during 2005 / 06 (maximum 13 degrees Celsius, minimum minus 25 degrees Celsius), along with an annual rainfall of 350 millimetres.

5.4 Local Resources and Infrastructure

There is a thorough discussion of all the infrastructural items with a great number of photographs in 2007 and 2011 Technical Reports. These are not repeated here, particularly as these issues are addressed later in this report.

5.4.1 Local Resources

The hamlet of Nahanni Butte is the closest settlement to the Mine Site (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, which communities 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. In the past, Potable water was extracted from fresh water wells. A sewerage treatment plant exists on-site.

Processing water was, in the past, available from the Prairie Creek Valley aquifer, at a maximum rate of 1,150 cubic m per day (420,000 cubic m a year), as allowed by the Mine's Water Licence N3L3-0932 that was issued by the Department of Indian Affairs and Northern Development on 1 July 1982. The original Licence expired; future water extraction from the Prairie Creek Valley aquifer will be covered by a Class A Water Licence.

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 earlier outlined. A total of nine alternative locations were evaluated prior to recommending the current location adjacent to Prairie Creek

The tailings impoundment design was originally approved by, and 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.

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.

5.4.4 Communications

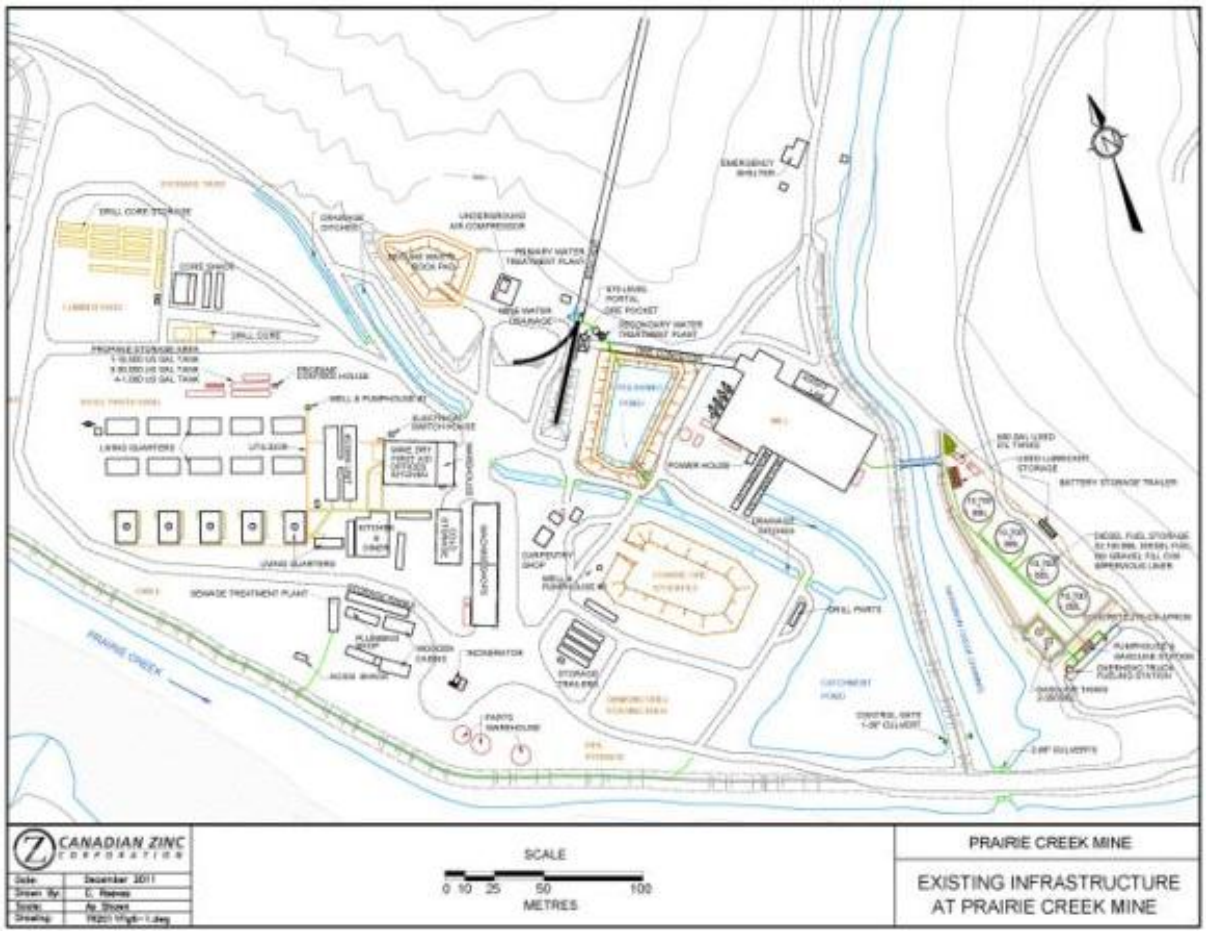
All outside communications from the Mine Site are via satellites. There is a main telephone and fax machine, handheld phones, and an internet facility. On site, staff members use radios that link the various work crews on surface and an underground Femco telephone system has been installed.

5.4.5 Mine Buildings

Most of the Mine Site surface facilities were established when the Mine was first constructed in 1982, and a prefabricated administration building was completed. This contains full office mine dry, first aid and warehouse facilities. Trailer accommodations and kitchen facilities to support a 200 man construction crew was built, along with full shop facilities. A tank farm to store diesel was engineered and constructed onsite along with a large pond impoundment, originally intended as a tailings containment area but which was never used as such. The various buildings and tank farm outlined are in a good

condition and only regular maintenance is required to keep them in good order. The layout of the entire existing surface infrastructure is shown in Figure 5.2.

Figure 5.2 Site Plan



5.4.6 Processing Plant

A processing plant also constructed pre 1982 exists on site, and consists of a crusher rated to handle 1,500 tons per day of material, a grinding and flotation circuit is rated at 1,000 tons per day, where separate lead and zinc concentrates are planned to be produced. 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.

A diesel powerhouse which contains four Cooper Bessemer 1.1 MW generators and switching facilities was constructed but is not operational and will be replaced.

The mill building and, covered primary crusher feed conveyor (to the left of the plant complex) that extends from the 870 mL portal is shown in Figure 5.3.

Figure 5.3 Processing Plant Complex

On Mine closure in 1982, the processing plant was approximately 90% complete and it has never been used for Prairie Creek processing.

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 for purposes of facilitating production at the planned rate of 1,000 tons per day. In 2006 / 2007 the Company completed some additional Main Zone development driving a new decline, to facilitate underground resource definition drilling. This work included the installation of a new ventilation system and electrical sub-stations, a track upgrade and general rehabilitation. A new water treatment facility was installed and a new mine water polishing pond was constructed near the 870 mL portal.

Main Zone mineralization is the primary target for underground mining as it is adjacent to the processing plant and it contains the most extensive underground workings. It is presently accessed by three levels that are referenced to m above mean sea level and which are known as 970 mL, 930 mL and 870 mL. The level portals have been located by

surveying from base stations within the Mine Site; underground surveys were completed by Cadillac Explorations Limited (Cadillac) prior to 1982.

In October 2006, the Company contracted McElhanney Associates to check-survey 870 mL and the level portals, using a total station to tie into known surface stations. Minor (less than one metre) discrepancies were found between Cadillac's and McElhanney's survey results. Further underground surveying was completed in 2007 upon completion of the decline.

The levels consist of the following development:

- 970 Metre Level: 220 m of footwall drift with six crosscuts at 30 metre intervals. This level is not connected to either the 930 mL or the 870 mL.
- 930 Metre Level: 940 m of trackless footwall haulage drift with 32 crosscuts at ten metre 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 allow production at a rate of about 500 tons per day using trackless methods. Approximately 40,000 tons of vein material was mined from 930 mL in 1981 / 82, which material is currently stockpiled next to the processing plant.
- 870 metre Level: 610 metres of tracked footwall haulage drift, 380 metres of vein drifting and approximately 150 metres of other development. The portal for the 870 mL is adjacent to the mill feed conveyor.

A small amount of underground drilling was completed on 930 mL in the early 1980s. At the time of Mine closure in 1982, preparations were underway to create a small backfill plant on 930 mL, but this work remains uncompleted. Limited workshop storage facilities were, however, completed and a mine air heater, indirectly fired by propane, was installed. Limited services were available and concrete pads for substations were installed.

In preparation for the 2006 / 07 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 horse power ventilation fan was installed at the portal. The fan forces air down a manway to 870 mL where a new decline has been excavated by the Company.

5.5.2 Production Equipment

A single boom drilling jumbo and a scoop tram were airlifted onto site to complete the underground development. The locomotive and mine cars and a variety of key mining and surface equipment already exists at the Mine, including an earth moving excavator, various dozers and rippers, scoop trams / load-haul-dump units, articulated trucks, a haulage rock trucks and various service trucks.

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, which is a mineralized vein exposed in the bank of Prairie Creek. 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 they were re-staked and subsequently conveyed to Cadillac Explorations Limited in 1966. Cadillac also acquired a 182,590 acre, regional Prospecting Permit.

Between 1966 and 1969, trenching was carried out on a number of mineralized zones, during which time underground exploration also commenced in the main zone and Zones 7 and 8 as follow up to trench results. The underground workings were established in 1969 in Zone 7 and consist of a 280 metre south drive collared approximately 325 vertical m 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 at this time. Similarly in Zone 8 a 240 metre 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. There was reportedly a vein that had been exposed in the drive but only reported to carry low metal values. The portal in Zone 8 is presently blocked by debris and is inaccessible. Although high-grade mineralization has consistently been identified, Zone 7 and Zone 8 mineralization has not been explored in sufficient detail to estimate any mineral resource at this time.

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 was also carried out. Penarroya discontinued their 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 Mine Site was used in the mid-1970s to transport equipment and supplies to the Mine Site.

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 output of 1,000 tons per day.

Between 1980 and 1982, extensive mine development took place. Cadillac acquired a 1,000 ton per day mill and concentrator from Churchill Copper, equipment for which was dismantled and transported to the Mine 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. The Mine received a Class A Water Licence and Land Use Permit and was fully permitted for production in 1982. It was at this time that the silver price 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 money) 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 the bankruptcy proceedings in 1984.

6.3 Ownership post-1991

In 1991, Nanisivik Mines Limited (Nanisivik), an unaffiliated third party, acquired the Property from Procan. Pursuant to a 23 August 1991 Option Agreement, 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, the Company acquired a 100 percent interest in the mineral properties and a 60 percent interest in the plant and equipment, subject to a net smelter royalty of two% in favor of Procan. In January 2004, the Company acquired all of Procan's (which had become Titan Pacific Resources Limited) interest in the plant and equipment, including the two percent net smelter royalty, thereby securing a 100 percent interest in the Property.

6.4 Historical Resource Estimates

A number of historical estimates have been compiled and reported for the Main Zone deposits. Initially these were for the vein only but were later incorporated the stratabound and stockwork mineralization. The chronology is shown in Table 6.1, but the results are shown and discussed in the Minefill Services Technical Report of October 2007.

Table 6.1 History of Estimates

Year	Company	Zone Estimated		
		Vein	Stratabound	Stockwork
1970	Behre Dolbear & Company	Yes	-	-
	for Pennarroya Canada		-	-
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

These estimates were made using different resource and reserve categories and are not NI 43-101 compliant.

Stratabound mineralization was not discovered until 1992; thus there are no estimates before that time.

The resource estimates compiled by MRDI Canada (MRDI), a division of H.A. Simons Limited included Main Quartz Vein, stratabound and stockwork mineralization. A summary of MRDI's report is contained as an appendix to the Company's January 2001 Scoping Study, which has been filed on SEDAR and is available under the title Technical Reports filed on 24 April 2001.

At the time of the estimation, NI 43-101 was not in place and the estimate was prepared in accordance with the Australasian JORC Code.

Table 6.2 Summary of MRDI's January 1998 Mineral Resource Estimate

Mineralization Type	Category	Tonnes	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)
Vein	Measured	542,000	197	0.4	13	12.5
	Indicated	1,434,000	190	0.4	12.8	11.2
	Inferred	7,412,000	174	0.4	11	12.7
Stockwork	Measured	79,000	294	0.7	15	31.1
	Indicated	228,000	134	0.4	5.6	14.5
	Inferred	742,000	145	0.4	5	14.6
Stratabound	Measured	500,000	51		5.4	10.5
	Indicated	785,000	59		5.1	10.6
	Inferred	124,000	26		2.7	7.9
Total	Measured	1,121,000	138	0.3	9.8	12.9
Total	Indicated	2,447,000	142	0.3	9.7	11.3
Total	Measured +Indicated	3,568,000	142	0.3	9.7	11.8
Total	Inferred	8,278,000	169	0.4	10.3	12.8

A Qualified Person has not done sufficient work to classify the historical estimate as a current mineral resource, and CZN is not treating the historical estimate as a current mineral resource.

MRDI's estimates were based on a combined total of 89 drillholes, which included some older data that was usable. In addition a total of 926 channel samples from underground development were utilized. The database comprised 1,529 MQV sample assays, 39 stockwork sample assays, and 282 sample assays from stratabound mineralization. The silver grades were cut to 600 grams per tonne, which affected a total of three anomalously high, MQV silver composites.

A three-dimensional block model was developed with a block size of 10m by 10m by 30m. One metre composites were created from the data set and interpolation was by Inverse distance weighting to the power of three (ID^3).

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

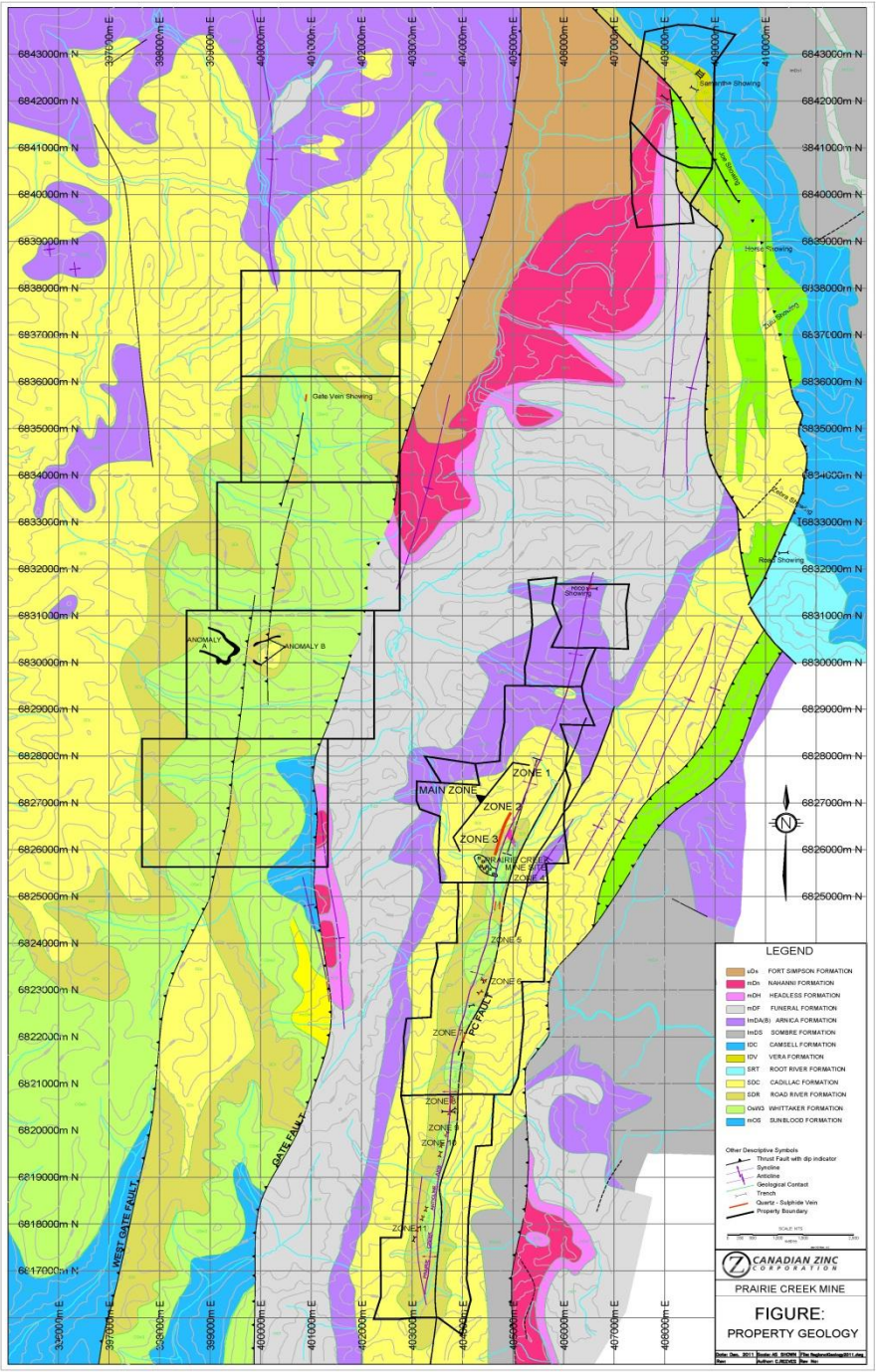
The *1987 Geological Survey of Canada Memoir 412* by Morrow and Cook provides probably the best description of the regional geological setting of the Property. Morrow and Cook describe the stratigraphy that accumulated during Siluro-Devonian time and formed in a paleo-basin adjacent to the ancient North American Platformal sediments. The east-dipping Tundra Thrust that is located on the Property and, 30 km to the west, the west-dipping Arnica Thrust, define the present margins of the Prairie Creek paleo-basin in which accumulated a thick Devonian sequence of sediments, including the Cadillac and Funeral Formations.

Units within the Prairie Creek paleo-basin underwent structural deformation in the form of folds and faults during regional Laramide deformation. The prevalent regional structural trend is approximately north–south; the Prairie Creek paleo-basin is broken into a series of north–south trending, five to 20 kilometre fault blocks.

7.2 Property Geology

Canadian Zinc's existing mineral claims and leases overlie two major fault blocks of sediments: the Prairie Creek Block and western Gate Block, see Figure 7.1. The north-eastern part of the Property also includes some of the marginal platformal sequence of rocks that are relatively undeformed by the faulting and folding that is apparent within the Prairie Creek paleo-basin sequence. The focus of exploration and development has been within the Prairie Creek Block. This block hosts the Main Zone mineralization which was and is the focus for Mine development and exploitation.

Figure 7.1 Property Geology



7.2.1 Marginal Platform

The northern part of the Company's claims, from Lease 3313 south to the Way 2 mineral claim, straddles the Tundra Thrust, which separates the Prairie Creek paleo-basin sequence to the west from the platformal series of sedimentary formations to the east. 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). Mississippi Valley-type mineralization is hosted in biohermal reefs of the Root River Formation, or facies equivalent.

In the southern part of the Company's claims, a reverse fault continuation of the Tundra Thrust separates the Prairie Creek Block from the marginal platform, approximately two km east of the Mine Site. The platformal sequence in this area is dominated by a thick assemblage of Sombre Formation dolomites.

7.2.2 Prairie Creek Block

Overall, the southern part of the Property is outlined by a one to two kilometre wide, doubly plunging antiform with a north-south trending fold axis that is referred to as the Prairie Creek Block. It is bordered to the west by the so-called Gate Fault and to the east by the Tundra Thrust. It is underlain by a conformable sedimentary sequence including the Lower Ordovician Whittaker Formation dolomites, Silurian Road River Formation shales and the thinly bedded, limy shales of the Cadillac Formation. Lower to Middle Devonian Arnica and Funeral Formation dolomites and limestone overlie this assemblage on the northern part of the Property.

Structurally, the longitudinal arch of the antiform occurs approximately five km south of the Mine Site. A local fault, referred to as the Prairie Creek fault, offsets the eastern flank of the antiformal fold and juxtaposes Cadillac stratigraphy against the Road River Formation. Erosion of the antiformal structure has resulted in windows of older Road River shales, cored by the Whittaker Formation dolomites. The antiform plunges at about 15 degrees to the north, so the geological units young in age to the north, which is also the case on the underground levels.

7.2.3 Gate Mineral Claims

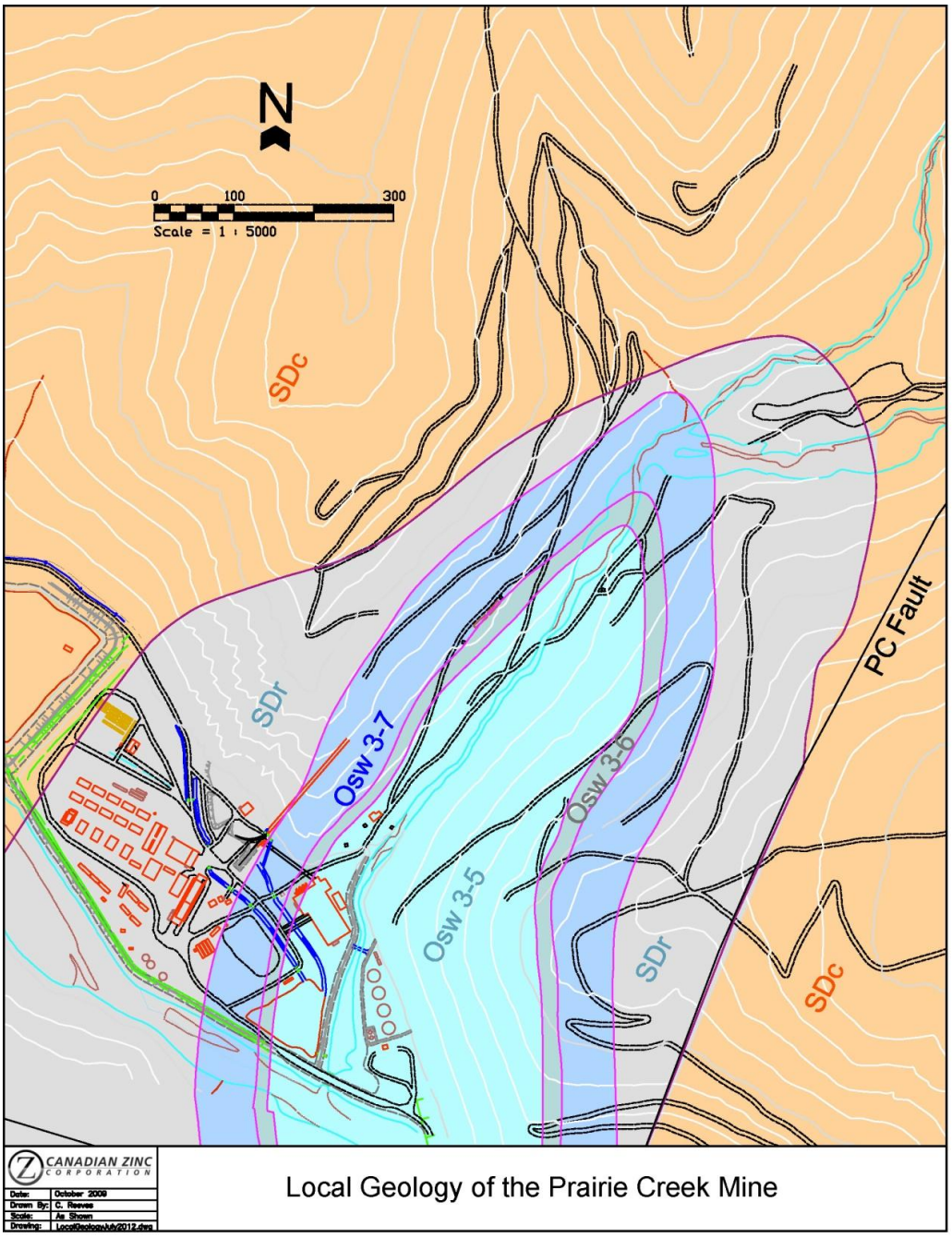
The six contiguous gate claims, which together define what is termed the Gate Block, are located to the west of the main mining leases and overlie similar type rock assemblages to those found on the Prairie Creek Block. Grassroots exploration was completed on this ground to test for mineralization similar to that found in the Prairie Creek Block.

The geological formations of the Whittaker and Road River Formations are known to occur within the Gate Block, as relatively flat-lying to gently dipping units. Compared to the Prairie Creek Block, there is much more exposure of the prospective Whittaker Formation in the Gate Block.

7.2.4 Main Zone Geology

The Mine Site is situated on the western flank of the Prairie Creek antiform (Figure 7.2), adjacent to the historic mineral resource that was previously referred to as Zones 1, 2 and 3 but was later redefined as Zone 3 and then the Main Zone. As earlier outlined, it is Main Zone mineralization that was and is the focus for Mine development and exploitation.

Figure 7.2 Main Zone Geology Plan

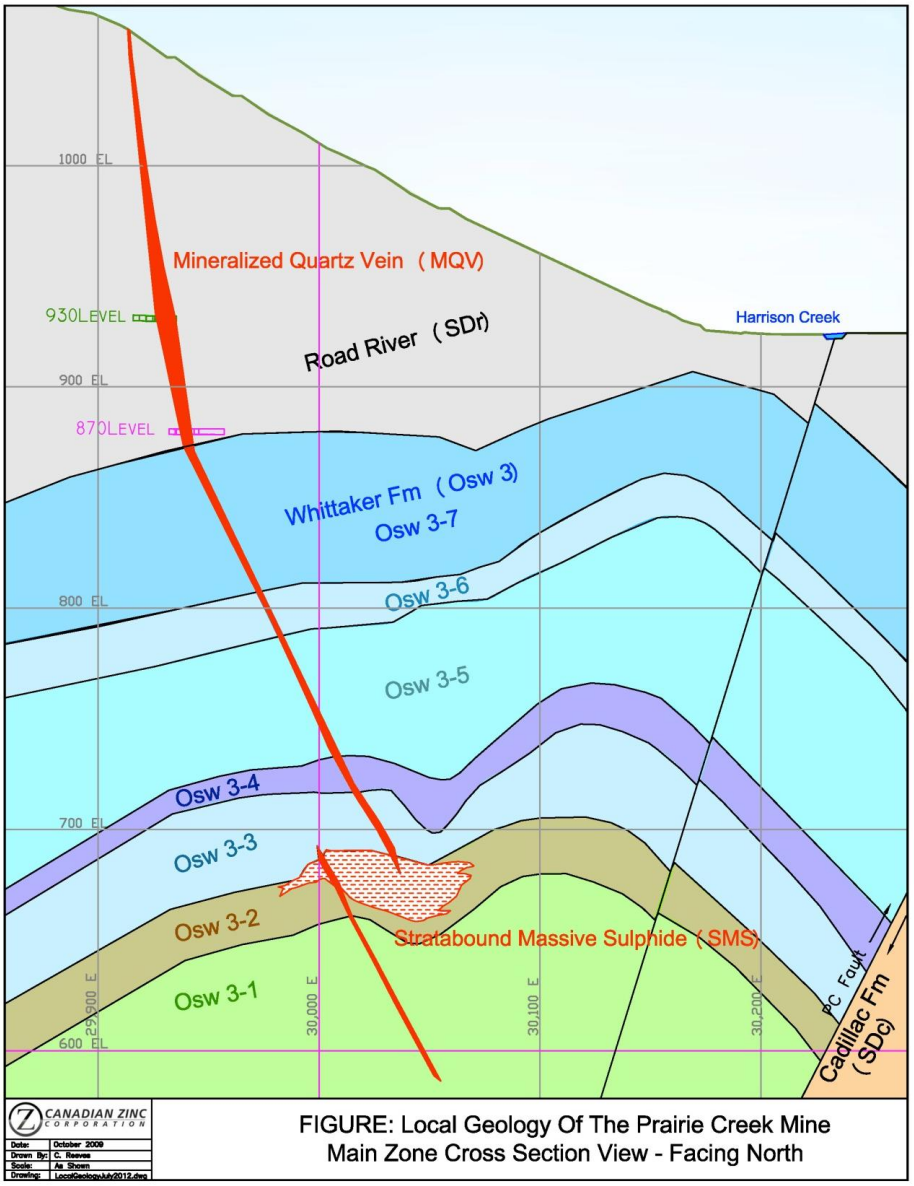


SDc= Cadillac Fm., SDr = Road River Fm, OSW=Upper Whittaker Fm, PC=Prairie Creek Fault. Additional details are in the text.

The three levels of available underground development assist in identifying the detail of Main Zone geology:

- 870 mL is collared in the Ordovician, Upper Whittaker Formation, which is the oldest geological formation in the Main Zone area and which forms the core of the Prairie Creek antiform (Figure 7.3).
- The Whittaker Formation is in turn overlain by a large exposure of the carbon-rich graphitic shales / dolomites of the Road River Formation.
- The iron-bearing, hence brown / orange weathered, Cadillac Formation shales overlie the Road River Formation and are located immediately adjacent to the Mine Site.
- The bluff-forming rocks immediately to the west of the Mine Site are formed by the cherty Arnica Formation which overlie the Cadillac Formation and form the more resistant hilltops in the immediate vicinity of the Mine Site.

Figure 7.3 Typical Cross Section



The bulk of diamond drilling carried out to date on the Property has been concentrated within the Lower Road River Formation and Upper Whittaker Formation. It is as a result of this work that the Upper Whittaker has confidently been divided into seven lithological sub-units (Table 7.1). The seven sub-units are referenced with the prefix OSW3, denoting Whittaker Formation, and by a number hence, from top to bottom, 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 in sometimes broad ranges because the contacts are generally gradational.

The assumed geological formation underlying the Whittaker Formation, known as the Sunblood Formation, has not been intersected by any holes drilled on the Property. The deepest hole that has thus far been drilled penetrates what is thought to be a Mid-Whittaker gritty dolomite unit.

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 dolomite 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 dolomite (graptolites common, occasional crinoids and brachiopods). Marker horizon near base – possible debris flow.
Upper Whittaker	OSW3-7	50 to 55	Interbedded chert-dolomite unit. Well-bedded, black to mid-grey cherts interbedded with dolomite. 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 dolomite with white spar-filled cavities. Bioclastic material is fine grained and comminuted (crinoids, brachiopods).
	OSW3-5	55 to 100	Upper chert nodule-dolomite unit. Massive to poorly bedded weakly bioclastic, fine- to medium- grained dolomite. 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-dolomite unit (similar to Upper chert-nodule dolomite unit).
	OSW3-2	20 to 30	Mottled dolomite unit. Fine grained dolomite with spheroidal mottled texture and chert. Unit is host to SD-1, SD-2 stratiform sulphide deposits. Disseminated fine-grained pyrite common.
	OSW3-1	20 to 30	Grey massive dolomite with minor chert nodules.
Middle Whittaker	MuOw2	40 to 50	Grey gritty dolomite with some sand size grit units with greenish, shaley partings.
Lower Whittaker	MuOw1	+50	Chert Nodule dolomite.

7.3 Mineralization

Exploration has revealed many base metal mineral showings on the Property. Historical exploration on the Property has led to referencing some of these surface mineral showings by numbers, and some by name:

- Quartz vein mineralization occurs in a north-south trending, 16 kilometre long corridor in the southern portion of the Property where the occurrences are exposed on surface.
- The mineralized vein showings are referred to as sequentially numbered Zones, some of which are known to contain sub-surface stratabound mineralization.
- The subsurface area above the underground workings is referred to as Zone 3 (as earlier noted, Zones 1 and 2 were identified as separate showings but they are now incorporated into, and considered to be part of, Zone 3, which is now referred to as the Main Zone).
- Extending for about ten km to the south of the Mine Site is a semi-continuous pattern of other vein exposures referred to as Zones 4 to 12, inclusive.
- A further expression of vein mineralization, known as the Rico showing, is located approximately four km to the north of the Main Zone.
- Mississippi Valley type (MVT) showings in northern section of the Property are developed over a distance of approximately ten km. They are referred to, from north to south, as the Samantha, Joe, Horse, Zulu, Zebra and Road showings.

Stockwork and stratabound mineralization are not exposed on surface. They have only been intersected in drillholes. These mineralized bodies have not been individually named.

7.3.1 Vein 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 of severity, yielding mainly cerussite (lead oxide) and smithsonite (zinc oxide); minor oxidation only of tetrahedrite-tennantite has been found. Silver is present in solid solution with tennantite-tetrahedrite and to a lesser extent with galena. Vein widths vary between less than 0.1 metre and more than five m; overall averages indicate a horizontal thickness (i.e. not true thickness) of approximately 2.7 m.

The most extensively developed vein is the Main Quartz Vein (MQV). Underground development has proved 940 m of strike length, and diamond drilling to date has indicated its continuance for a further 1.2 km. The MQV trends approximately north-south and dips between the vertical and 40 degrees east (average = 65 degrees east). It remains open to the north and is expected to continue for a further four km to the south, evidence for which is the Rico showing. Diamond drilling to depth has indicated its transverse continuance, but little information is currently available below an elevation of 600 m amsl (i.e. about 250 m below the Mine Site elevation).

Vein mineralization developed within the cherty dolomites of the Ordovician-Silurian, Upper Whittaker Formation and shaley dolomites of the lower Road River Formation. It apparently formed in axial plane of weakness within the Prairie Creek structural antiform.

It is thought that the more competent units of the Lower Road River and Whittaker Formations more readily formed tension features in which vein sulphide mineralization is hosted; and the rock type changes to a much more graphitic shale in the mid- and upper-parts of the Road River Formation, the units of which are less competent and provide a poor host for the vein-type formation.

For example, at the end of 930 mL the MQV can be seen to dissipate into the mid-Road River shales. The vein does not appear to be well developed in either the upper shales of the Road River and Cadillac Formations.

Figure 7.4 MQV Exposed Underground

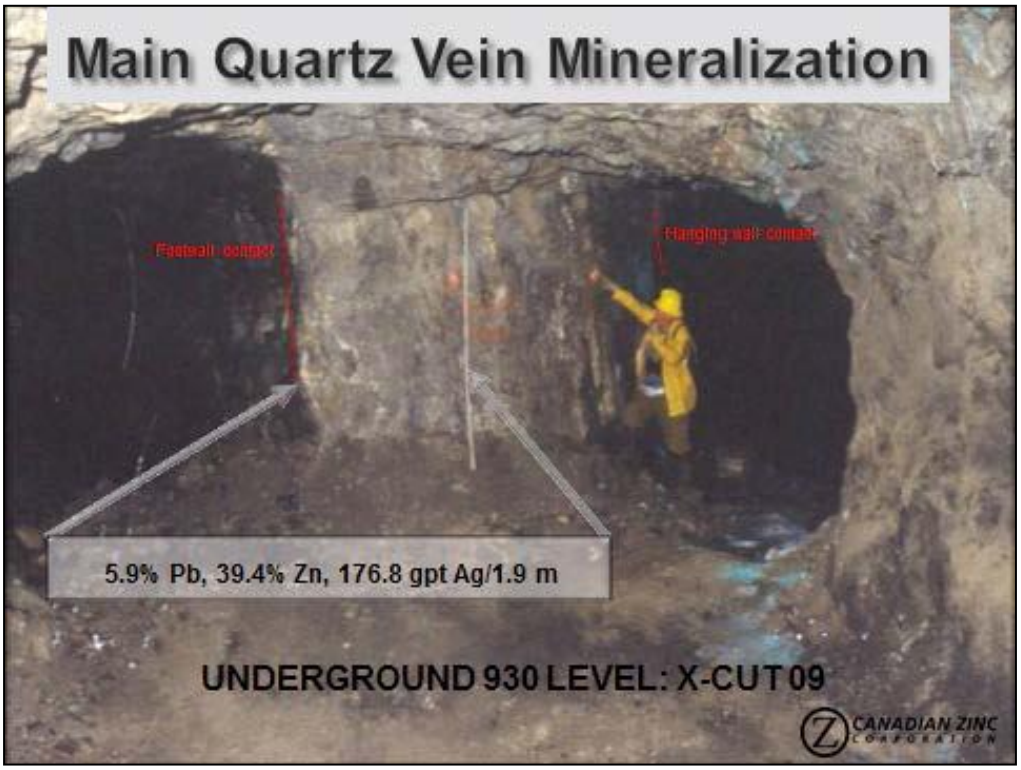


Figure 7.5 MQV Mineralization in Drill Core

Preliminary structural evidence suggests that the various mineralized vein showings might be structurally linked as a series of en-echelon segments comprising a single, but nevertheless structurally complex, mineralized vein structure. The presence of an en-echelon vein structure might go a long way to explaining the apparent off-sets between the various vein showings.

7.3.2 Stockwork Mineralization

Towards the end of 930 mL (at Crosscut 30) a series of narrow (average 0.5 metre wide), massive sphalerite-tennantite veins are developed at about 40 degrees to the average trend of the Main Quartz Vein. The sub-vertical veins range from 0.1 to 0.5 m thick, have no apparent alteration halo, and are separated from each other by unmineralized dolomite. The veins can be seen to locally offset and be cut off by fault planes and are difficult to correlate without more detailed drilling.

Figure 7.6 Stockwork Mineralization

This mineralization is referred to as the (vein) stockwork though it does not appear to represent a true stockwork but a series of splays off the MQV. So far, stockwork style mineralization has only been located in the immediate area surrounding the exposure in the 930mL workings.

7.3.3 Stratabound Mineralization

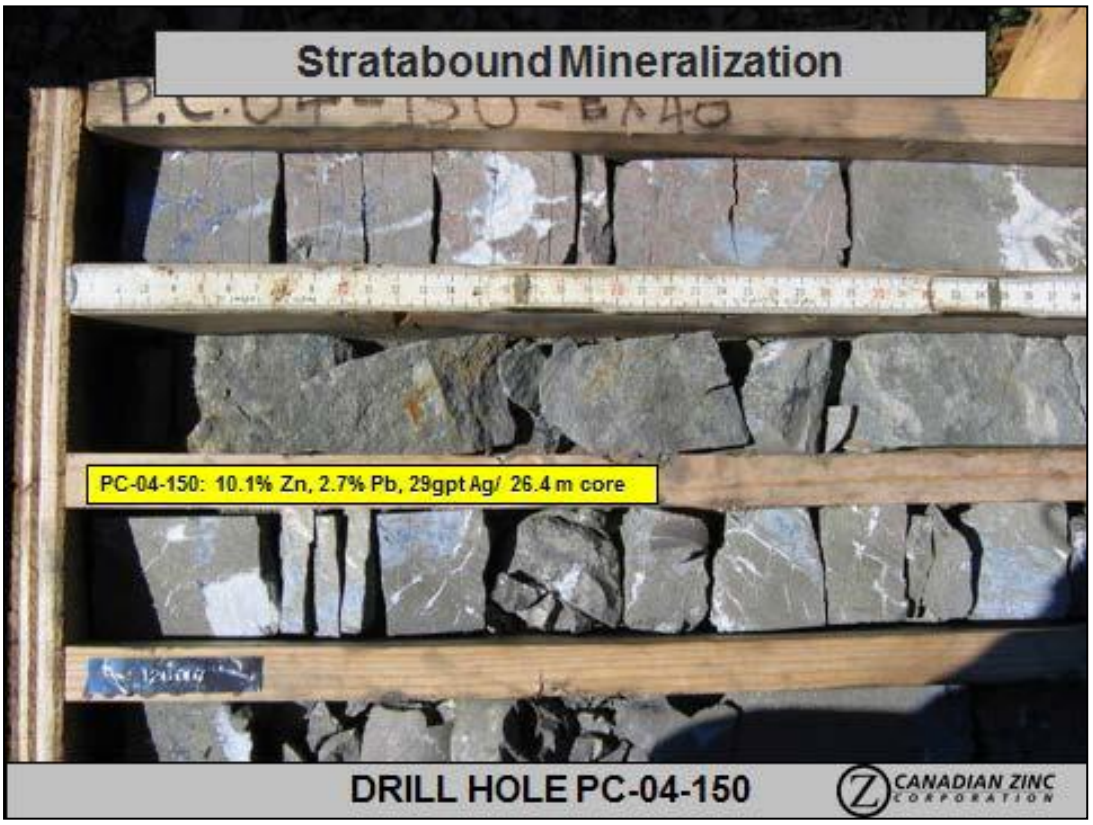
Stratabound mineralization was discovered by CZN in 1992, while surface drilling was being carried out to extend the then known Main Quartz Vein resources at depth. So far, indications of stratabound mineralization have been found by drilling over a strike length of more than three km; it has thus far been located by drillholes in the Main Zone, as well as in Zones 4, 5 and 6.

Oxidation is not apparent in stratabound mineralized material, and the sulphide 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).
- Contains no tennantite-tetrahedrite and very little copper.
- Contains only half as much galena as, but substantially more iron sulphide / pyrite than, typical vein material.
- Fragments of stratabound material can be seen to be in vein material indicating that the stratabound material predates the vein material.

The majority of stratabound massive sulphides occur mainly within Mottled Dolomite unit of the Whittaker Formation (OSW3-7 – see Table 7.1), which the mineralization totally replaces without any significant alteration. The stratabound sulphides are developed close to both the vein system and the axis of the Prairie Creek anticline; they cut across the vein structure and they are probably older than the vein deposits (Figure 7.5). An apparent thickness of 28 m of stratabound mineralization has been intersected in Main Zone drillholes, where it occurs approximately 200 m below 870 mL.

Figure 7.7 Stratabound Mineralization in Drill Core

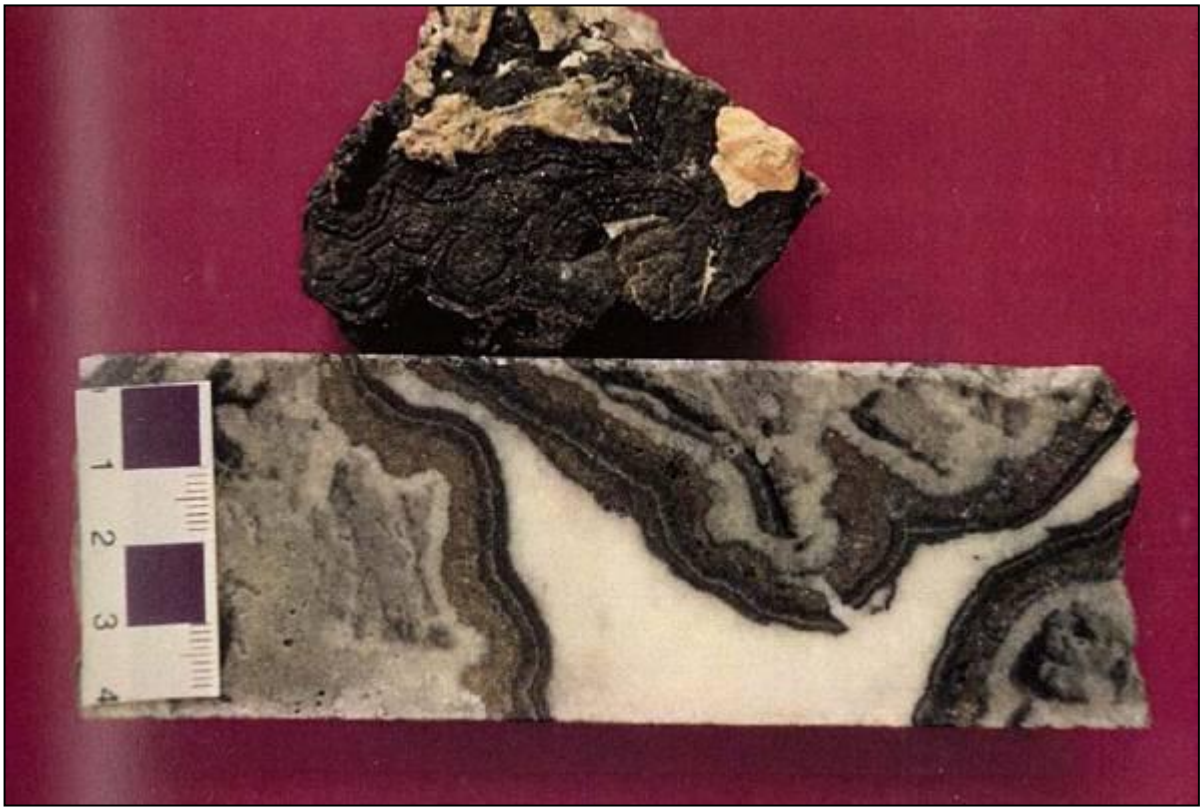


7.3.4 Mississippi Valley Type Mineralization

The 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 drillcore from the Zebra showing is shown in Figure 7.6.

Figure 7.8 MVT Style Mineralization form Zebra Showing



8 DEPOSIT TYPES

The mineralization at the Mine is hosted in well bedded cherty dolomites of the Ordovician-Silurian, Upper Whittaker Formation and shaley dolomites of the lower Road River Formation. Three main styles of base metal mineralization have been identified on the Property:

Hydrothermal quartz vein mineralization (sulphide with secondary oxide towards surface), termed the Main Quartz Vein, (MQV). Thus includes the Stockwork (STK), though it does not contain as much quartz as the MQV.

Stratabound sulphides (SMS) and MVT type mineralization the MQV apparently formed in axial plane of weakness within the Prairie Creek structural antiform, and is a very long linear feature which is controlled at the top by a change in the stratigraphy. Minor hydrothermal vein stockwork-type mineralization is locally developed in association with the Main Zone vein deposit.

Generally a model along the lines of some of the Irish carbonate-hosted, lead-zinc deposits (e.g. Lisheen, Galmoy and Silvermines) may be the most appropriate analogy for the stratabound mineralization.

The style and textures of the MVT mineralization appears to be similar to some of the deposits mined at Pine Point, NWT.

9 EXPLORATION

After initial discovery, in 1928, of a high grade base metal vein exposed on the south side of Prairie Creek, the focus of early exploration occurred to the south of the mine site area, on large well exposed surface vein base metal showings. It was not until the 1960s that significant mineralization was located in the Main Zone which shifted the focus of exploration.

Exploration on the Property to date has since been focused on Main Zone mineralization, with the secondary focus on a series of vein mineral showings exposed on the surface along the strike of the Main Zone. As the known MVT showings occur in a more remote part of the Property and report somewhat lower grades than either vein or stratabound mineralization, they have not been the focus of any significant exploration to date.

9.1 Zone Nomenclature and Early Work

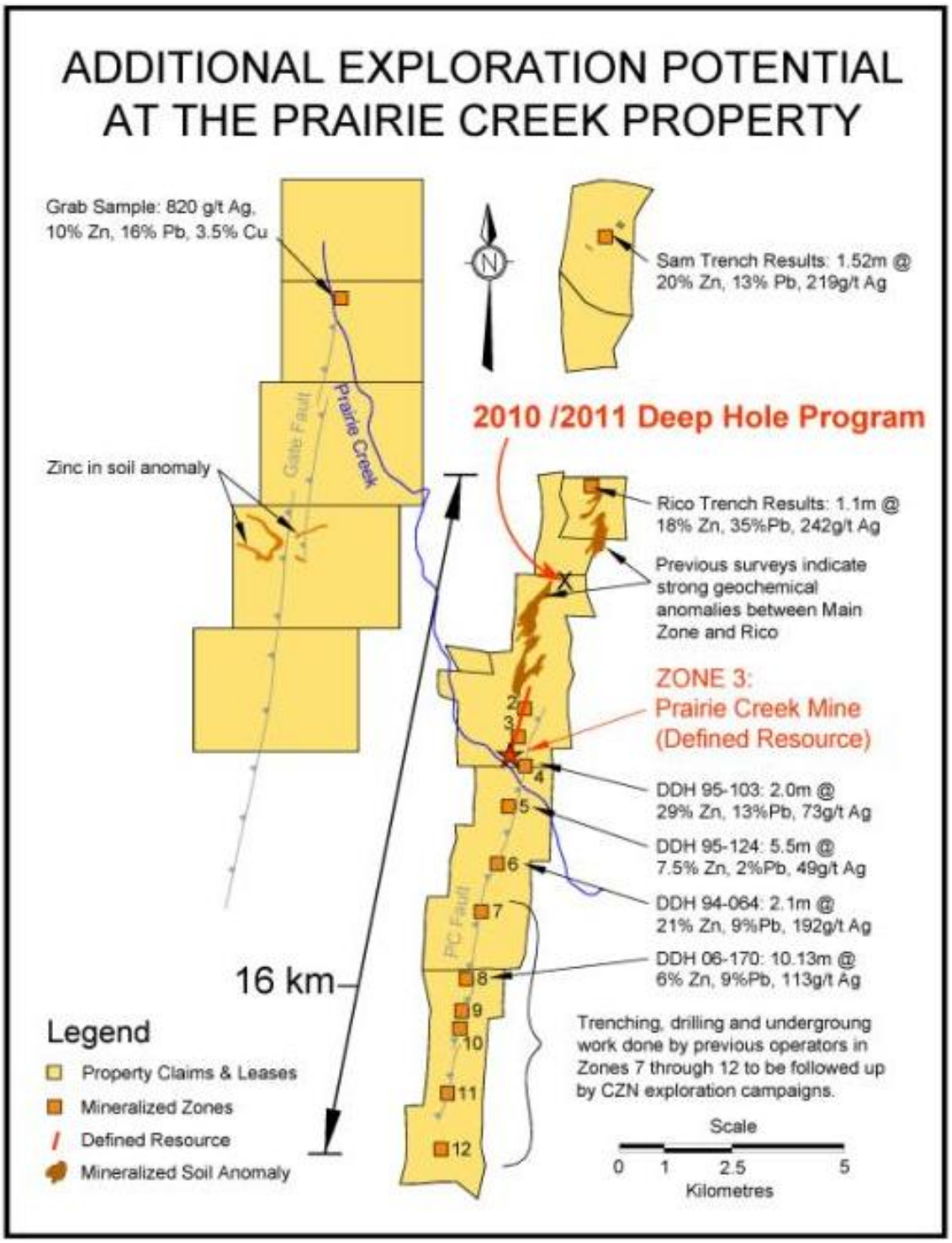
Twelve zones where base metal vein mineralization is exposed on the surface are numbered 1 to 12 from north to south in Figure 9.1. In addition, eight named surface mineralized showings and one surface soil anomaly have been identified on the Property. The results of surface drilling, trenching and underground channel sampling have shown the mineralization to be consistently high-grade. The metals with economic potential include zinc, lead, silver, and copper.

The main focus of exploration and underground development work to date has been on Zone 3 mineralization, which is adjacent to the Mine Site and which today includes the Zone 1, Zone 2 and Zone 3 mineralized showings. As such, it is commonly called the “Main Zone”, within which exploration work has shown to contain three different styles: vein, stratabound and stockwork.

The term Main Zone is used in this technical report, as is the term Main Quartz Vein, which is commonly used to describe the Main Zone vein occurrence (as distinct from Main Zone stratabound (SMS) and stockwork (STK) mineralization). The resource estimates presented in Section 14 consider exclusively mineralization of the Main Zone, where these estimates are based on all verified surface and underground drilling and underground channel sampling data up to the July 2007 completion date of Phase I of the Main Zone underground drilling program.

The secondary focus of exploration work on the Property has been in the other mineral zones outside the Main Zone, which do not at this time contain any defined mineral resource. Zone 7 and Zone 8 mineralization, where trenched mineral showings are approximately 3.5 km and five km to the south of the Main Zone, respectively, was explored in 1969 and is discussed in Section 6.1.

Figure 9.1 Location of Zones with Highlights



Most of the exploration and development work prior to CZN's involvement with the Project was focused on the Main Quartz Vein, in particular by means of the underground development.

9.2 Exploration Work 1991 to 2011

The exploration work carried out in this period has been described in the 2007 and 2011 Technical Reports. In this report, the drilling component is summarized in a tabular form in Section 10, and non-drilling activities are discussed in this section. Table 9.1 is a simplified table which summarizes the work in this period with highlights. A full discussion with tables of results is contained in the earlier reports.

Table 9.1 Summary of Exploration Work, 1992 to 2011

Year	No of Holes	M	Highlights
1992		6,535	Discovery of previously unknown stratabound mineralization by diamond drilling. Discovery hole (PC-92-008) ran 10.60% Zn, 5.29% Pb, 44.37 g/t Ag, over 28.40
1993		8,352	Tested for further Stratabound mineralization. Extended MQV by intersecting 18m of strong vein 170m below workings Trench samples from Rico showing, in north showed grades of 18% Zn, 35% Pb, 242 g/t Ag in a vertical mineralized
1994 and 1995	37	13,276	Extension of Main Zone, more stratabound lenses Rico Zone and Zebra showing (MVT play)
1997	-	-	Channel sampling of previously un-sampled underground drift development, see below
1999	-	-	Gate Claims 1 to 4 were staked and geological mapping, soil and rock sampling, was carried out Discovery of a vein in outcrop, with some grades similar MQV
2001	5	1,711	Diamond drilling program designed both to increase confidence in 1998 resource estimate and to identify high-grade Possibility of high grade shoots recognized
2004	7 7 11	2,167 1,149 2,383	MQV drilling which intersected significant mineralization Step out on the vein hit narrow but high grade intersections Stratabound exploration outside Main Area
2005			Rehab of underground workings
2006	9 11	1,610 595	U/G drilling commences on MQV Channel and round sampling which is discussed below Drilling of Zone 8 mineralization investigated
2007	41 12	9,014 1,671	Phase 1 U/G program confirms vein grades Gate claims drilling and Zone 8,9 and 11 show poor results
2010	3	2,703	Deep drilling and proximal to resource drilling
2011	30	5,638	Deep drilling and proximal to resource drilling.

9.3 Channel Sampling

In 1997, a program was undertaken to complete sampling information on the two accessible underground levels. Following some rehabilitation work, this was carried out on 870 mL and 930 mL, where a series of 231 channel samples was collected from 294 m of previously un-sampled underground drift development, over a weighted average true width of 1.78 m. These samples gave a weighted average grade of 17.2% Zn, 16.0% Pb, 330 g/t Ag, 0.8% Cu.

The assay results outlined are for Main Quartz Vein material only. The program brought the total of verifiable channel samples available from the Main Zone workings to 1,072, inclusive of channel samples collected by Cadillac Mines between 1980 and 1982. The channel samples together form 393 composite channel samples comprising 14 channel samples from 970 mL, 273 channel samples from 930 mL and 106 channel samples from 870 mL.

In 2006 internal access to the new decline ramp was provided by a new crosscut (Crosscut 870-07) that was driven as part of the 2006 underground exploration program. The Main Quartz Vein was intersected about 12 m from the crosscut collar, the walls of the 10 metre intersection was channel sampled and a true vein thickness of 6.5 m was estimated. A summary of these assay results is available in the MineFill Services, Technical Report of October 2007.

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 kilograms of representative material was collected from each round, which was subsequently crushed to less than one centimetre in size, split into two kilogram 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
    - 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 has been called chip sampling in some reports. Regardless, physical sampling of the exposed deposit, which is essentially a fault constrained zone with some gouge that is oxidized should be a good sample.

9.4 Gate Claims

Gate Claims 1 to 4 were staked in 1999 and subsequently converted to mining leases in 2008. During 2001 a small exploration program, comprising geological mapping, soil and rock sampling, was carried out over areas that contain geology similar to that found on the Prairie Creek Block. The work resulted in the discovery of a vein in outcrop, with select grab samples reporting grades similar to those previously established for the Main Zone vein (820 g/t Ag, 3.5% Cu, 16% Pb and 10% Zn). A large, 1,000 parts per million zinc-in-soil anomaly was also located over favorable geology on the Gate 3 claim.

During 2007 the Company carried out a surface 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.

9.5 Post 2007 Exploration

The 2010 / 11 / 12 deep diamond drill exploration program was designed to test for extensions of the Inferred vein Mineral Resource within the same lithological units to the north of the Mine Area. Here the host geology and structure are projected to continue at depth, approximately 1.5 kilometres north of the last drill hole within the currently defined mineral resource.

During 2010 a new drill rig, that could drill deeper, was brought to site. Subsequent drilling defined the location of the host stratigraphy. A continuation of the program in 2011 returned significant mineral intersections (refer to Section 10.3) in the anticipated structural zone projected from the known defined resource.

Further exploration was also completed by drilling in the proximity, but not within the defined resource during 2010 and 2011 and further described in Section 10.3.

9.6 Conclusion

The work described above is a mix of early stage, brownfield in terms of extending known zones, having clear targets, and resource definition. In AMC`s opinion, the programs appear to have been well directed, with a clear structural or stratigraphic target in mind. The samples referred to in this section which are channel or round sampling are of a close spacing and directed to evaluating the MQV specifically. The results demonstrate consistency in this structure over a long strike length.

10 DRILLING

10.1 General

The exploration work carried out in the period 1991 to 2007 has been described in the 2007 and 2011 Technical Reports. In this report, any non-drilling activities are discussed in Section 9, with some reference to drilling. Drilling as an exploration tool is of great importance, as the stratigraphy is well known and the targets for testing are clear. This section discusses drilling progress since 2007 and includes commentary on the techniques and the process itself. The drilling meters completed since 1992 are summarized by zone in Table 10.1.

Table 10.1 Summary of Diamond Drill Metres

Zone	1992	1993	1994	1995	2001	2004	2006	2007	2010	2011	Totals
1/2/3/4	6,535	7,028	8,731	9,718	1,711	5,963	1,610	9,014	2,703	5,638	58,651
5	-	362	97	1,062	-	-	-	-			1,521
6	-	650	2,051	-	-	-	-	-			2,701
7 & 8	-	279	-	-	-	-	595	-			874
11	-	33	-	-	-	-	-	1,671			1,704
Rico	-	-	372	193	-	-	-	-			565
Zebra	-	-	2,025	703	-	-	-	-			2,728
Totals	6,535	8,352	13,276	11,676	1,711	5,963	2,205	10,685	2,703	5,638	68,744

In addition to the above, approximately 19,244 m of drilling was carried out on the Property in many phases before 1992. However none of that drilling has been used in the current estimates.

It should be noted that almost 80% of the drilling in the table above was carried out in the Zone 1-4 area.

As discussed in Section 9.5, exploration in 2010, 2011 and 2012 has been designed to test for extensions of the Inferred vein Mineral Resource with the same lithological units to the north of the Mine area. The results of the 2010 and 2011 drilling are discussed below. The 2012 program is ongoing.

10.2 2010 Drill Program

Since the target geology was projected to occur at a depth beyond the reach of the Company's existing drills, a new, higher capacity HTM2500 diamond drill rig was airlifted to the property. A drilling pad was selected at a location next to Casket Creek, a new eight kilometre long access road was constructed and diamond drilling at Casket Creek

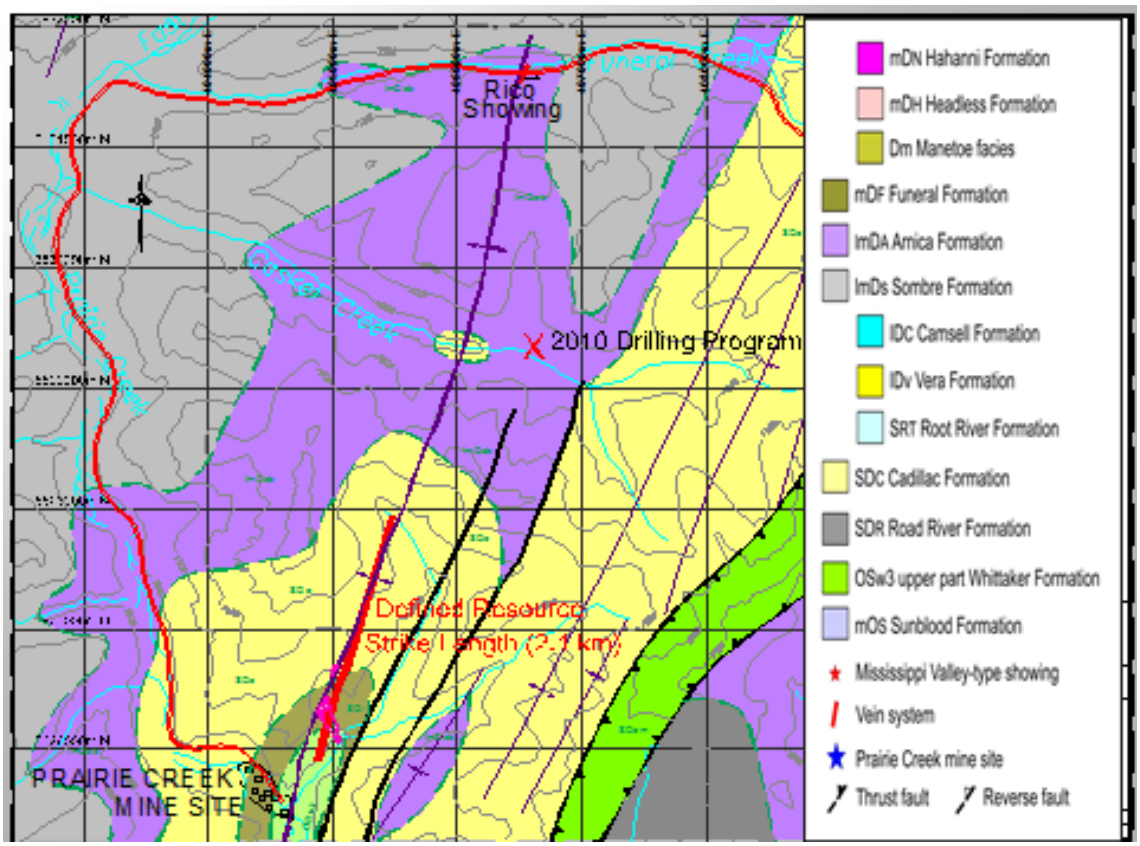
commenced in early August. By mid October 2010, 2,703 m of drilling had been completed in three holes.

Drill hole PC-10-186 was successfully completed to a depth of 1,557 m. This hole intersected and confirmed the presence of the target Whittaker Formation, which hosts the majority of the defined mineral resource at the Prairie Creek mine at a down hole depth of 1,500 m. New stratigraphical information at depth in this hole enabled determination of a more precise target location of the potential vein hosting structure.

A second wedged drill hole, PC-10-186W1, was redirected from the upper part of PC-10-186 and steered to the west toward the revised target location. Down hole technical problems were encountered after about 150 m into the wedged hole, at an estimated depth of about 536 m, forcing abandonment of this hole.

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 caused suspension of drilling activities. In Figure 9.3 the location of the 2010 deep hole drill program is shown in the plan.

Figure 10.1 Plan showing Location of 2010 Deep Hole Drilling Program



This drilling program confirmed the presence of the host Whittaker geological formation at the projected horizon, about four km north of the Prairie Creek Mine portal. The potential vein target, projected to lie at a down hole depth of about 1,500 m, remains untested. The nearest drill hole, PC-95-125, located approximately 1.5 km to the south towards the Mine, drilled in 1995, returned multiple mineralized vein intersections 750 m down the hole, including a 6.3 metre intercept grading 18.7% zinc, 8.5% lead and 239 g/t silver.

10.3 2011 Drill Program

The 2011 program had two objectives: continuation of the deep drilling program in the Casket Creek area to test for extensions of the Mineral Resource 1.5 km to the south; and a program at the Mine property to test for additional high-grade vein structures and for other, wider stratabound deposits adjacent to the resource.

The Casket Creek program consisted of 2,513 m of coring in four holes, including wedges and commenced with the completion of drillhole PC-10-187. This hole intersected significant vein-type zinc and lead mineralization that demonstrated the probable continuation of vein-type mineralization similar to that located within the main zone at the Mine. This hole cut the fault structure above the favourable formations that host the existing resource. Due to intersecting the structure in the less competent shaly rock of the Cadillac Formation, the fault zone was seen to be weakly developed.

A wedge hole, PC-11-187W2 was drilled as an undercut to the initial hole. This cut the structure 50 m below the PC-10-187 intercept. The PC-11-187W2 intercept and that of PC-11-187 are shown in Table 10.2. However, this intersection is located stratigraphically well above the primary host rock units found at the mine.

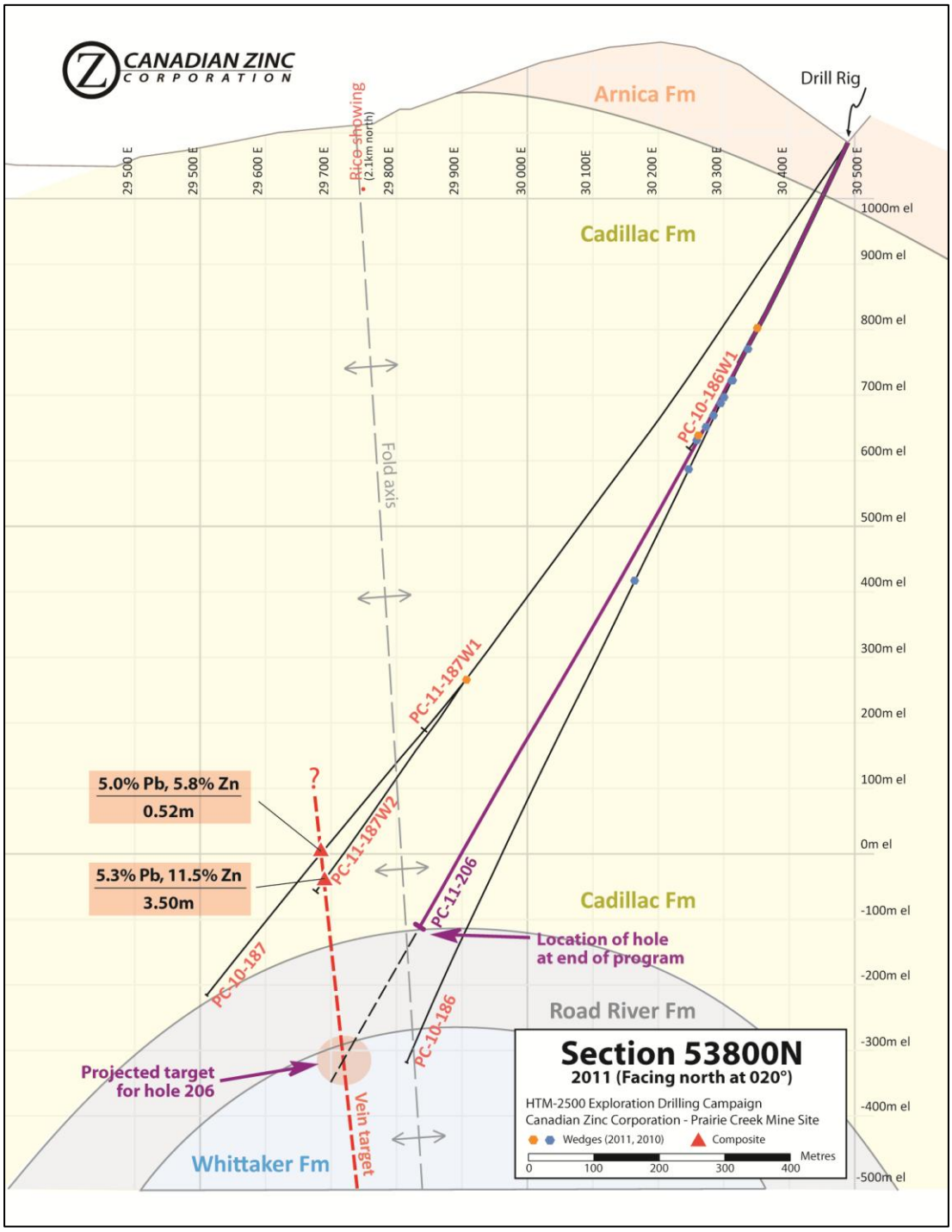
Table 10.2 Results of 2011 Program

Drillhole	From (m)	To (m)	Interval (m)	Pb (%)	Zn (%)	Ag (g/t)	Cu (%)
PC-10-187	1348.36	1348.88	0.52	4.92	5.90	34	0.034
PC-11-187W2	1384.00	1387.50	3.50	5.26	11.47	84	0.176

The drill intercepts are approximately 100 m west of the fold axis of the main regional antiform in an identical structural setting to that of the defined vein resource in the main mine zone.

Based on the model, a subsequent drillhole, PC-11-206, was designed to cut some 250 m lower within the primary host geological formations as found at the mine. This hole had reached a depth of 1,365 m, approximately 230 m short of the target, at the end of October 2011 due to weather conditions. The details of this program are shown in a cross section interpretation facing north, in Figure 10.2.

Figure 10.2 Casket Creek Cross Section Looking North



The program at the Mine consisted of 26 shallow holes for 3,125 using a CZN owned Longyear rig. The MQV structure was intersected and identified in each completed hole, many of which returned high grade base metal values. Due to its close proximity to the surface, the mineralization was highly weathered and 70–90% oxidized, which inhibited core recovery and grade determination.

In addition, a new area of stratabound-type mineralization was intersected in PC-11-190 approximately 150 m below the 870 metre mine level, which is the lowest level of current developed workings at the Prairie Creek Mine.

Drill holes PC-11-205, 207 and 208 tested an area above a previously defined zone, known as “the stockwork zone”, containing a series of narrow high grade veins trending at oblique angles to the main vein strike. Holes PC-11-207 and 208 reported wide moderate grade intercepts resulting from compositing a number of narrow veins together. Hole PC-11-205 did not report any significant assays.

10.4 Drilling Procedure

10.4.1 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. This is reduced to BQ size (36.5 mm) if difficult downhole conditions are experienced. In 2010 a new higher capacity HTM-2500 diamond drill rig was airlifted to the property for use in the deep drilling program. This rig is shown set up at Casket Creek for the deep drill hole program in Figure 10.3.

Figure 10.3 HTM-2500 Drill Rig at Casket Creek

Various different drilling contractors have been engaged to run the CZN drills and complete the programs. Currently Cabo Drilling Corporation of North Vancouver B.C., fill this role. Times when contractors were on site to complete drill programs using their own rigs are discussed below.

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 millim) core, due to the difficult terrain expected at the drilling sites.

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.

10.4.2 Field Procedure

Surface drillholes in the field are located by chain and compass, and alignment is completed by Brunton compass sighting along pickets. Once aligned, the dip of the hole is set using a digital inclinometer placed on the rods. Drillholes are aligned underground by a qualified surveyor using a total station. Shooting to spads in the decline back for a reference line, the surveyor turns perpendicular to the drift and marks the foresight and backsight on the walls with spray paint. The drill mast is aligned parallel to the foresight and backsight lines, usually slightly offset from the intended line. A supervising geologist is present at all times during the drilling of each hole.

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 in feet by the drillers, as the drill rod lengths are also measured in feet. Full drillcore 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.

10.4.3 Surveying

The collars of 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 by the designated surveyor, using mine grid coordinates that are then converted to UTM coordinates using a verified transformation formula.

Downhole surveys on surface and underground are now completed using a FLEXIT SmartTool instrument, prior to which an Icefield MI-3 tool was used and, prior to 1995, a Pajari instrument. Downhole survey shots are now completed every 15 m instead of every 60 m, as was previously the case. The completion of individual surveys is dependent on downhole conditions.

The raw data is processed by the software that comes accompanies the various survey tools. Outputs 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 GEMCOM. The paper and electronic files are stored at the Company's head offices in Vancouver, B.C.

10.4.4 Core Logging

All drillcore 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. drillcore 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 to metre measurements and aluminium tags are stapled to each box-end noting drillhole number and the box-start and end depths. Drillcore 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 drillcore pieces over ten centim, divided by the total length of the run. Rockmass ratings are then calculated for ten metre envelopes around individual mineralized intersections, using industry standard methods.

All drillcore is geologically logged on paper 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. Sample intervals are marked by the geologist responsible for that hole, prior to core photography, which is done two or three boxes at a time. The photographs are archived in the Company's files.

A senior geologist is always in charge of core logging, and the data is transposed by the geologist responsible for the hole into an Excel spreadsheet format for copying into the central database. Error checking is performed while data is entered. No standard procedures are, however, in place and no checking of manually entered data is routinely carried out. Missing or overlapping intervals are checked with the responsible geologist / s, when such errors are spotted.

Due to no drilling being carried out at the time of AMC's site visit, none of the above was observed by AMC. The text is taken from the 2007 Technical Report and is believed to reflect the actual process.

10.4.5 Core Recovery

There are many instances where reference to poor core recovery is made in the 2007 and 2011 Technical Reports, and some of this is noted as being due to drilling problems.

The MQV mineralization at Prairie Creek is hosted within a structural fault zone and this zone was identified in all completed holes. Due to the close proximity to surface, the mineralization was highly weathered and 70–90% oxidized, which inhibited core recovery and grade determination.

Core recoveries have been consistently recorded since 2006. An example of core recovery from the 2007 drilling is shown in Table 10.3. Average recoveries are about 80% for the MQV and 97% for the SMS mineralization. No recovery information was provided for the STK mineralization.

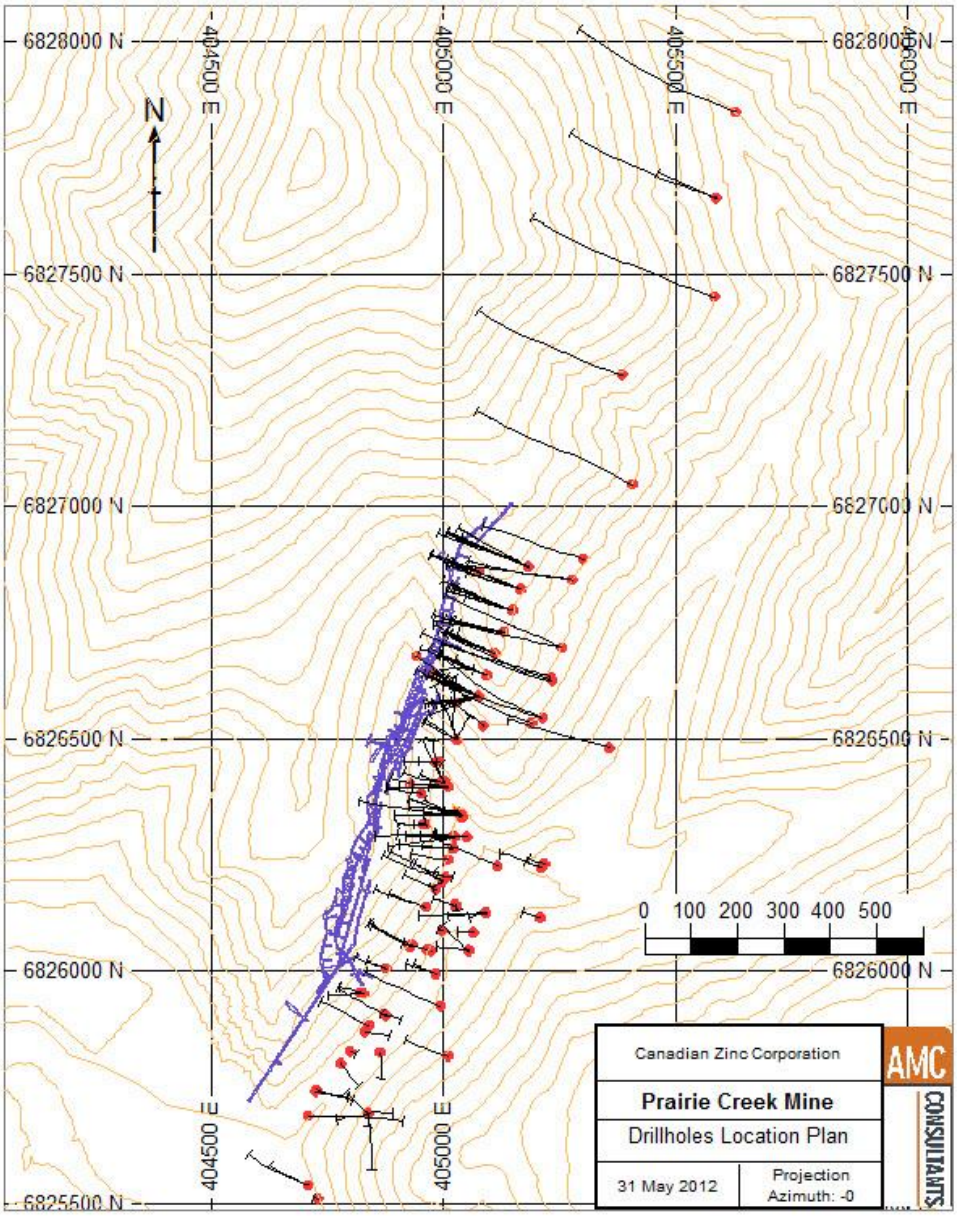
Table 10.3 Core Recoveries from 2007 Drilling

Rock type	From	To	Recovery	n
MQV	500	1390	80.3	176
SMS	-	-	97.1	13

10.5 Drilling Results

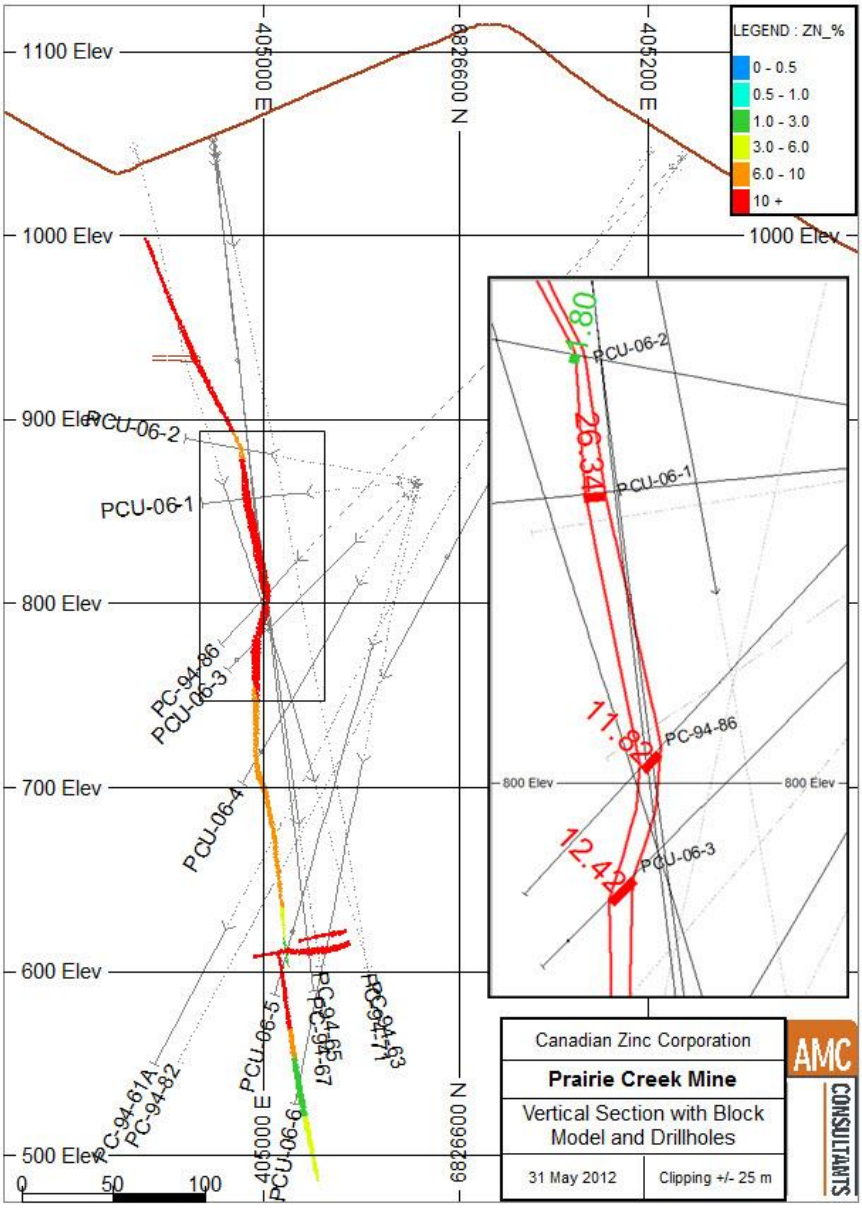
Drilling results are tabulated in the 2007 Technical Report. As this is now an advanced stage property, a plan and a representative section are included below.

Figure 10.4 Drill Plan Showing Drilling Density



Only the surface drilling is shown in Figure 10.4. In the section represented in Figure 10.5 both surface drill traces and those of the underground are shown, some zinc grades are shown in red. The underground drilling was carried out on sections approximately 50 m apart.

Figure 10.5 Cross Section Showing Drillholes



10.6 Bulk Density

Bulk density measurements were performed on drillcore samples of MQV and stratabound material, in preparation for MRDI's 1998 resource estimate. The method used to determine SG values is unknown, but it is likely to have been an industry standard method based on Archimedes Principle.

The database used by MRDI for purposes of the 2007 resource estimation included 231 Main Quartz Vein samples and 22 stratabound samples. Regression analysis was used to define numerical functions, relating assay grades to SG, from which SGs were estimated for the 1,298 Main Quartz Vein, 39 stockwork and 260 stratabound samples for which laboratory determined values were not available.

It is outlined in the 2007 and 2011 Technical Reports that while the lead plus zinc numerical function, applied by MRDI to estimate Main Quartz Vein SGs, is sufficiently robust for purposes of NI 43-101 compliant resource estimation (R-squared = 0.936), there were doubts as to the validity of MRDI's numerical functions for stratabound material. In view of this, CZN contracted Acme Labs to undertake an SG measurement program on 54 pulp samples of stratabound material for which silver, copper, lead, zinc and iron assay results had previously been defined.

In addition, due to correlation being poor, a fixed value was used for the stockwork mineralization, see Section 14.2.1.

AMC did not review the data in detail and recommend that this area is reviewed prior to the next block model.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 Chain of Custody

11.1.1 Underground Channel Samples

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

11.1.2 Drillcore Samples

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

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

11.1.3 Sample Sacks

Individual drillcore sample bags are sealed with plastic ties and placed in rice sacks (50 pounds per bag). Individual rice sacks containing either channel samples or drillcore 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 drillcore and sampling facility, prior to their transport off-site.

11.1.4 Transport

Shipments of sample sacks are air freighted in charter aircraft from the Mine Site airstrip to Fort Nelson, B.C., from where they are transported by Greyhound bus to the assay laboratory (Acme Labs, ISO 9001-2000 accredited) in Vancouver, B.C.

11.1.5 Drillcore Storage

After final tagging, the boxes containing the main mineralized drillcore 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 weathering. The boxes containing

unmineralized drillcore are stacked either on rebar drillcore racks or square piled in stacks at the northeast corner of the mine site yard.

Figure 11.1 Stored Mineralized Drillcore Intersections, Prairie Creek Mine

11.2 Assay Method

Acme Labs (ISO 9001-2000 accredited) has carried out the majority of the sample assays since the Company's first involvement in 1992. Acme Labs is currently the only laboratory used by the Company for purposes of sample assaying.

11.2.1 Sample Preparation

Samples are sorted and inspected for quality of use (quantity and condition), wet or damp samples are dried at 60 degrees Celsius. The samples are then crushed to 70 percent passing ten mesh (2 millim), homogenized, riffle split (250 gram sub-sample) and pulverized to 95 percent 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; 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 finish.

11.3 QA / QC Procedures

Quality Assurance / Quality Control ("QA / QC") samples are submitted by CZN with the regular samples for analysis, including blanks, duplicates and blind standards. Blank and duplicate samples are taken every ten drillcore samples. Standards are sent, at the supervising geologist's discretion, after a mineralized zone is sampled.

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 in the stored core boxes: the stored core half 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 drillcore samples are followed as regards labelling, storage and transport of duplicate samples.

11.3.3 Standard Samples

The Company has established its own assay standard samples, in conjunction with Smee & Associates Consulting Limited of North Vancouver, B.C. (Smee). The standards were compiled from a shipment of mineralized samples sent by the Company to CDN Resource Laboratories Limited in Delta, B.C. (CDN). From these samples CDN prepared three homogeneous pulps suitable for use as standard reference materials, from which pulps were prepared separately and in an identical manner:

- 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), Actlabs 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 the Company for insertion into the sample stream, as earlier outlined. Certificates for each of the Company's three standards (as compiled by Smee) are available and have been seen by MineFill.

11.4 Conclusion

AMC believes that the data collection and handling followed normal industry practice and the data is fit for purpose. However, while QA / QC samples were inserted and the results have been observed in the assay certificates, it is not clear if any analysis of the data was carried out. This has been remedied in the recent programs but these do not influence the current Mineral Resource.

12 DATA VERIFICATION

Raw and final assay data prepared by Acme Labs undergo final verifications by a British Columbia Certified Assayer, who signs each Analytical Report before they are released to the Company.

12.1 Historical Drillcore Data

None of the surface holes completed prior to the Company's involvement on the Property in 1992 were included in any recent estimates. This includes MRDI's 1998 estimates, MineFill's 2007 estimates or the current estimate. The reasons for their exclusion include 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 the 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 the manual and digital comparison of original laboratory datasets and the Company's Excel spreadsheet database:

- The sample numbers and assay values on the certificates were called out by an individual, as another individual located the corresponding sample numbers within the database and verified the assay values
- Assay values were deemed verified when the original signed assay certificate or photocopy was present and the database reflected the assay certificate values accordingly
- Each data point was marked as verified, corrected or unverified, as appropriate

The results of MineFill's data verification program are summarized in Table 12.1. Verified assay data only was used for purposes of the resource estimates summarized in Section 14.

Table 12.1 Summary of Results, MineFill's 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

No data collected since the MineFill verification has been used in the 2012 estimate as any recent holes are outside the resource model extents. However, a random check, on approximately 5% of the drillholes results used in the dataset used in the estimation were checked with the assay results traced from the certificates to the data base. No discrepancies were found.

Data was verified in Datamine software. This included checks on duplicates, overlaps, sample intervals beyond the end of the hole and collar coordinates. Datamine creates a summary table of these results.

AMC considers that the data is fit for the purpose of estimating a resource for a PFS.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Metallurgical Samples

A number of bulk metallurgical samples were collected by CZN for the various metallurgical testing that has been completed to date. The samples consisted of 50 to 80 kg of ore material collected from a number of different crosscuts in the underground 930 and 870 levels from which to composite a representative blend for overall mine ore feed.

In the SGS Phase 1 and 2 programs, heavy liquid separation testing was carried out on two composited vein samples from Level 930 and Level 870. These samples were crushed to minus ½ inch then wet screened at 14 mesh. The minus 14 mesh fraction was the fines, which bypassed the heavy liquid separation process.

The main flotation development test work was carried out on a heavy liquid preconcentrated Master composite. The composite was prepared by SGS Lakefield from 11 individual samples containing dilution material. Additional test work was also conducted by SGS Lakefield on unpreconcentrated Master composite and two preconcentrated sub-composites, including (a) a low oxidation composite, and (b) a high oxidation composite.

In the SGS Phase 3 program the heavy liquid separation testing was carried out on two composited vein samples from Level 930 and Level 870. The Level 930 composite was prepared from 6 individual samples. The Level 870 composite was prepared from seven individual samples.

In the SGS Phase 4 program the main development test work was carried out on a preconcentrated Master composite. The composite was prepared from 11 individual samples containing dilution material. Additional test work was also conducted on unpreconcentrated Master composite and two preconcentrated sub-composites, including (a) a low oxide composite, and (b) a high oxidation composite.

In the SGS Phase 5 program the testwork was done on a composite ore sample consisting of nine individual samples. The principal objective of the test program was to produce a quantity of process water for environmental purposes.

13.2 Metallurgical Testing

13.2.1 Summary of Previous Work

A considerable amount of test work was completed on the metallurgical samples including original testing by Lakefield Research dating back to June of 1980 as part of the original Kilborn Feasibility Study. Numerous reports and tests were completed on the Prairie Creek ores throughout the 1990’s and early 2000’s and these reports are available in the Canadian Zinc technical files.

The majority of recent work was completed by SGS at their Lakefield facilities (SGS Lakefield) in five phases. The recent studies focused on utilizing DMS separation ahead of flotation and also targeted the oxide component of the ore to determine recoveries. The resulting detailed ore processing flow sheet is incorporated into the proposed future operations. The following sections are summary overviews of this work:

SGS Lakefield Research Limited Phase 1 & 2 – an investigation into the recovery of lead, zinc and silver from Prairie Creek sulphide / oxide ores. – April 2005

Summary of Objectives:

- To develop a treatment process for the beneficiation of both vein and stratabound ore types from Prairie Creek
- Confirm if the two types of ore can be blended in the milling process
- Develop a reagent scheme that would eliminate the need for cyanide in the process
- Determine if separate sulphide and oxide concentrates can be developed
- Determine if Run-of-mine ore can be pre-concentrated in a heavy media circuit

Summary of Results:

- Development of a reagent scheme and beneficiation process produced:
  - Lead Concentrates (sulphide & oxide combined) of 56-72% lead at a recovery of 88-84% and 750-860 gpt silver at a recovery of 74-78%
  - Zinc Concentrates (sulphide and oxide combined) of 51-57% zinc at a recovery of 82-77%
- Development and confirmed process flowsheet for Stratabound ore produced:
  - Lead concentrate (sulphide) of 60% lead at recovery of 90% and 400gpt silver at a recovery of 62%
  - Zinc Concentrate (sulphide) of 54% zinc at a recovery of 91%

All ore types were blended with encouraging results. The Upper and Lower Vein Areas and Stratabound ore types were blended and produced:

- Lead Concentrate (sulphide & oxide combined) with 60% lead at a recovery of 80% and 860 g/t silver at a recovery of 73%
- Zinc Concentrates (sulphide & oxide combined) with 55% zinc at a recovery of 79%

Heavy media separation indicates 31% of ROM ore was rejected and had low metal contents.

SGS Lakefield Research Limited Phase 3 – Development Testwork – the recovery of lead, zinc and silver from Prairie Creek sulphide / oxide ore samples. – January 2006

Summary of Objectives:

- To examine if the ore can be pre-concentrated using a heavy media liquid
- To conduct optimization work on the pre-concentrated ore using the flowsheet and reagent scheme developed in the Phase 1 and 2 test programs
- To produce four separate concentrates of lead and zinc sulphide and oxide for evaluation
- To perform dewatering tests on the flotation products

Summary of Results:

- Crushed ROM ore can be pre-concentrated using a heavy liquid (SG 2.8) with a relatively small loss (less than 4% of sulphides and less than 7% oxides report to the float reject).
- In general the reagent scheme developed in the Phase 1 and 2 testwork on ROM ore samples was applicable to the pre-concentrated ore with slight modification.
- Results on Level 930 resulted in excellent recoveries of 91% for lead, 87% for zinc and 98% overall for silver. This was obtained even with the ore containing higher quantities of oxide lead and zinc.
- Results on Level 870 resulted in excellent recoveries of 94% for lead, 82% for zinc, and 98% for overall silver. The zinc sulphide concentrate grade was 57.5% zinc and the zinc oxide grade was only 29.7% zinc and when combined into an overall zinc concentrate a grade of 46.4% zinc was achieved.

SGS Lakefield Research Limited Phase 4 – Development Testwork – the recovery of lead, zinc and silver from Prairie Creek sulphide / oxide ore samples. – June 2007

Summary of Objectives:

- To examine the possibility of improving zinc depression during lead sulphide and oxide flotation.
- To improve zinc oxide concentrate grade.
- To confirm the applicability of heavy media preconcentration on the new master composite.

Summary of Results:

- Between 19% and 27% of the total ore was rejected in the HLS process.
- Improved lead concentrate grade and reduced zinc content in the lead concentrate.

- Improved zinc concentrate grade while maintaining the same zinc recovery as for the 930 and 870 level ores.
- The final reagent scheme developed in Phase 3 work was slightly modified to (a) compensate for the difference in processing characteristics, and (b) improve concentrate grade and selectivity.
- In follow-up to test 30 confirmation tests on two sub-composites (sulphide Pb/Zn flotation only) but selectivity problems resulted from sample contamination with heavy liquid (was not washed) test (38). To address this lab problem SGS redid the confirmation tests (test 40) and produced higher lead recoveries with less zinc contamination.

SGS Lakefield Research Phase 5 – Testwork on the Prairie Creek Ore and Mine Water September 2008

The principal objective of the Locked Cycle Test Program was to produce a quantity of flotation products for environmental and marketing purposes.

Summary of Results:

- It was determined that the use of mine water had no significant effect on sulphide lead / zinc flotation.
- The use of mine water had a negative effect on oxide lead flotation. Adjustment in the reagent additions was required to restore floatability.
- The run-of-mine composite contained an appreciable amount of secondary copper minerals, which resulted in the activation of zinc minerals and a selectivity problem during lead sulphide flotation. Adjustment in the reagent additions was required to improve slime depression and restore floatability.
- The results obtained in the heavy liquid separation were better than those obtained in the Phase 4 testwork with respect to waste rejection.

13.2.2 Comminution

The original Kilborn feasibility study indicated a design Bond work index of 10.3 kWh/tonne on unpreconcentrated ore. A series of ball mill grindability tests by SGS Lakefield were carried out on the preconcentrated master composite ore and on the master composite as is. The results obtained showed that the ore was medium hard having a Bond work index of 8.5 to 11.1 10.3 kWh/tonne (average 9.7 kWh/tonne).

13.2.3 Dense Media Testing

Heavy liquid separation, or Dense Media Separation (DMS), was originally contemplated and tested by Hazen Research Inc. in December 1997. Recovery and separation indicators from this simulated test and further work were completed by Confidential Metallurgical

Services in 2005 with testwork completed by SGS Lakefield. More detailed follow-up DMS testwork has been completed by SGS Lakefield in their Phase 1 & 2 Reports.

Subsequent HLS work was performed by SGS Lakefield on the Master composite and the two sub-composites. The results showed between 19.0% and 26.5% of the total feed was rejected as waste with low loss of metal value.

Subsequent pilot DMS testing was performed by SGS at their Vancouver facilities (SGS Vancouver) on a composite. The DMS production achieved a weight reduction of 28.7% and achieved recovery of 97% or greater for all value metals except for Au, which was at 85%.

13.2.4 Flotation Testing

Most of the flotation development work was conducted by SGS Lakefield on the Master Comp heavy liquid product + fines sample. Confirmation tests were performed on the Master Comp plus dilution (i.e. ore 'as is') and the individual composites (high and low oxidation).

Confirmation tests were also conducted on two sub-composites (i.e. sulphide Pb/Zn flotation). Although reasonably good sulphide lead and zinc recoveries were obtained, the selectivity between lead and zinc was not as good as that obtained on the Master composite due to sample contamination with heavy liquid.

To date approximately 133 individual flotation tests have been completed, of which 20 were locked cycle tests.

Based on the metallurgical testwork conducted to date, the anticipated metallurgical performance is presented in Table 13.1.

Table 13.1 Anticipated Metallurgical Performance

	Vein Ore	Stratabound	Weighted Average Recoveries
DMS Plant Losses			
Mass	27.0%	13.0%	
Silver	2.8%	1.0%	
Lead total	1.7%	1.0%	
Lead oxide	3.1%	1.0	
Zn Total	2.1%	1.0%	
Concentrate Grade			
Lead sulphide %Pb	74.0%	59.8%	
Lead oxide %Pb	55.6%	55.0%	
Zinc sulphide %Zn	61.4%	53.9%	

	Vein Ore	Stratabound	Weighted Average Recoveries
Recoveries			
Silver to PbS	64.3%	61.7%	
Silver to PbOx	14.5%	5.0%	
Silver to ZnS	15.9%	28.0%	
Lead sulphide	94.0%	90.0%	
Lead oxide	75.8%	65.0%	
Zinc sulphide	93.1%	90.7%	
Weighted Average Recoveries			
Zinc	%		75.0%
Lead	%		88.0%
Silver	%		92.0%

13.2.5 Dewatering Testing

The Phase 3 study by SGS Lakefield tested the settling characteristics of the flotation tails and the settling and filtration characteristics of the four concentrates. The results obtained showed the following:

- The tailings from both composites (Levels 930 and 870) contained an appreciable amount of clay-like slimes. Therefore, the settling rates were relatively low. In addition, the thickener overflow contained an appreciable amount of suspended solids.
- There was a strong indication that the tailings water cannot be recycled to the operating plant without pre-treatment. This issue was considered when the decision was made to partition the Water holding pond into two cells to eliminate tailings water recycle.
- Reasonable settling was achieved with the addition of flocculent at a neutral pH, controlled with acid (pH 6.8 to 7.0).
- All four concentrates settled well with the addition of small quantities of flocculant.
- The filter cake moisture for the sulphide lead and zinc concentrates was between 9% and 10%.

13.3 Backfill Testing

Golder Paste Technology Ltd. (Golder Paste Tec) was engaged by CZN to perform a pre-feasibility study on flotation tailings and DMS reject for the purpose of backfill disposal.

Three different backfill materials were tested:

- 100 wt% flotation tailings
- Blend 1, 80 wt% tailings, 20 wt% DMS float
- Blend 2, 50 wt% tailings, 50 wt5 DMS float

The testing required to develop a conceptual design and cost estimate were as follows:

- pH determination
  - The pH for the tailings was in the range of 9.4 and the DMD float was approximately neutral at 7.2.
- Mineralogical composition
  - The dominant gangue materials in both were dolomite and quartz followed by smaller amount of mica.
- Particle size distribution
  - The particle size distribution of the tailings sample was 80 wt % passing 69 microns whereas the DMS float was 80% passing 9.4 mm which is probably finer than the case in a full scale plant.
- Specific gravity
  - The specific gravity results of 3.1 for tailings and 2.8 for the DMS float are normal.
- Slump vs. Solids content
  - Testing indicated that the flotation tailings are moderately sensitive to water addition and this favours a batch paste system over a continuous paste system.
- Yield stress, water bleed and viscosity
  - The results of yield stress, water bleed and viscosity testing gives an indication of how long paste can be left idle in a pipeline without settling. The results indicated that paste can be left idle in the pipe a minimum of 30 minutes without major segregation and after this time it would be easy to restart flow.
  - A follow up paste testing program performed by Mine Paste Engineering (MPE) is currently under way to test various combinations of binder, flotation tailings and DMS float material. The intent of the program is to more accurately define backfill costs.

14 MINERAL RESOURCE ESTIMATES

14.1 Introduction

The existing Mineral Resource estimate discussed in the 2007 and 2011 Technical Reports, was reviewed and restated by Ms D Nussipakynova, P.Geo of AMC, who takes responsibility for the estimate. This estimate, which is based on the same data as that of 2007, now supersedes the 2007 and 2011 Mineral Resource estimates. Elements of the review included a review of the dominating, estimation, classification and reporting, and these are discussed below. There are two models developed: one for the MQV and Stockwork zones, and a separate one for the Stratabound mineralization. The reader is also referred to the earlier reports as the block estimates have not changed and only the classification and reporting has been altered. All the modelling by MineFill was carried out in Surpac and the review by AMC was carried out in Datamine software and the current models are named prairiemqv2_amc_cl and prairesms_amc_cl.

The summary results of the estimate for the three zones combined, at a cut off of 8% Zn equivalent (Zn Eq[1]) are shown in Table 14.1 below. The Prairie Creek deposit contains a significant amount of Inferred resources which, upon further delineation, has the potential to extend the life of the mine.

Table 14.1 Mineral Resources at 31 May 2012

Classification	Tonnes (M)	Zn (%)	Pb (%)	Ag g/t	Cu (%)
Measured	1.700	12.1	9.7	155	0.28
Indicated	3.731	10.2	10.5	162	0.32
Measured + Indicated	5.431	10.8	10.2	160	0.31
Inferred	6.239	14.5	11.5	229	0.57

Notes:

1. Mineral Resources are stated as of 31 May 2012.
2. Mineral Resources include Mineral Reserves.
3. Stated at a cut-off grade of 8% Zn-Eq based prices of $1.30/lb for both zinc and lead, and $35/oz for silver.
4. Average processing recovery factors of 78% for Zn, 89% for Pb and 93% for Ag.
5. Average payables of 85% for Zn, 95% for Pb and 81% for Ag.
6. $ Exchange rate = 1 CD/USD.

14.2 Data Used

The resource estimates relied on all the underground channel samples, in addition to surface drillcore and underground drillcore data collected by the Company since 1992. Channel samples were modelled as drill holes. The 1992 to 1998 assay database was

[1] See Section 15.3 for the Zn Eq formula

verified by MRDI, and MineFill verified the 2001 to July 2007 assay database. Only those assays that were deemed verified were used for purposes of resource estimation. No data collected since 2007 was used, as the recent drilling has been outside the extents of the model.

The data used for the estimates is shown in Table 14.2.

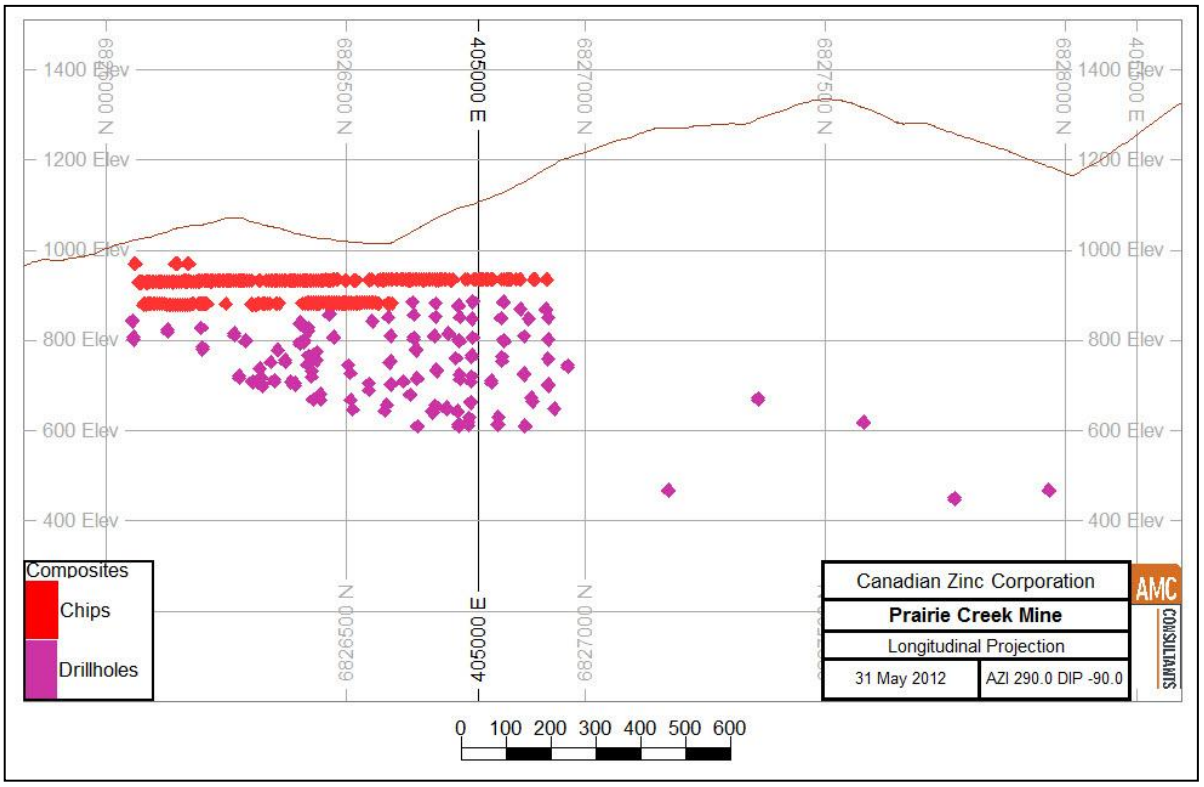
Table 14.2 Data Used in May 2012 Resource Estimate

Area	Drillholes		Channels	
	Holes	M	Samples	Composites
Surface	173	46,671	-	-
U/G	51	10,694	943	331
Total	224	57,365	943	331

The channel samples are predominantly from the Main Quartz Vein (MQV) and were taken from the three underground levels: 23 samples from 970L, 670 from 930L, and 250 samples from 870L.

The MQV data plotted on a long projection is shown in Figure 14.1. This demonstrates that the estimate in the upper levels is wholly supported by channel sampling and the deeper mineralization is supported by diamond drilling. The samples used for the Stockwork estimation are both core and channel samples and the Stratabound mineralization is estimated from diamond drill core only.

Figure 14.1 Distribution of Data Types in MQV



14.2.1 Bulk Density

The collection of bulk density measurements is described in Section 10.6. The bulk density measurements are interpolated into the blocks in the MineFill models using the assigned sample values based on the analysis described earlier. The statistics for the SG composites are presented in Table 14.3, which does not include stockwork material due to the limited amount of readily available, material-relevant SG data and a lack of an applicable regression function. An average SG of 3.31 was instead applied throughout the stockwork block model, which is a reasonable average for MQV material.

Table 14.3 Statistics on Bulk Density Values

Variable	MQV	Stratabound
Number of samples	454	199
Minimum value	2.63	2.84
Maximum value	4.87	4.45
Mean	3.26	3.61
Variance	0.12	0.19
Standard Deviation	0.35	0.44
Coefficient of Variation	0.11	0.12

14.3 Domain Modelling

14.3.1 Geology Model

The major lithological contacts were modelled by CZN geologists, in addition to the major faults.

14.3.2 Mineralized Domains

CZN geologists outlined three main types of Main Zone mineralization: MQV, stockwork, (STK), and stratabound (SMS). These solids were verified at the time by MineFill against the database and found that the solids created were representative of the mineralized zones.

In AMC's review of the shapes provided, it was found that there were some discrepancies between the data and the wireframes supplied. It would appear that drilling carried out at the time of the estimate was included, but the wireframes were not adjusted to the new intersections. However, the composites were flagged such that they were included in the estimate. This is not material, and should be cleaned up in the next modelling exercise. It was also seen that the MQV wireframe stopped at the southern drilled section and was not continued some distance beyond the holes. Again, this is not material and should be altered to fit the interpretation including some reasonable extrapolation for the next estimate.

In Figure 14.2 the three zones are shown looking Northwest, with the MQV shown in red, the STK in blue and the SMS in green. Figure 14.3 shows all three mineralized solids with the zones in plan view with the same colour scheme.

Figure 14.2 Long View of Mineralized Domains Looking Northwest

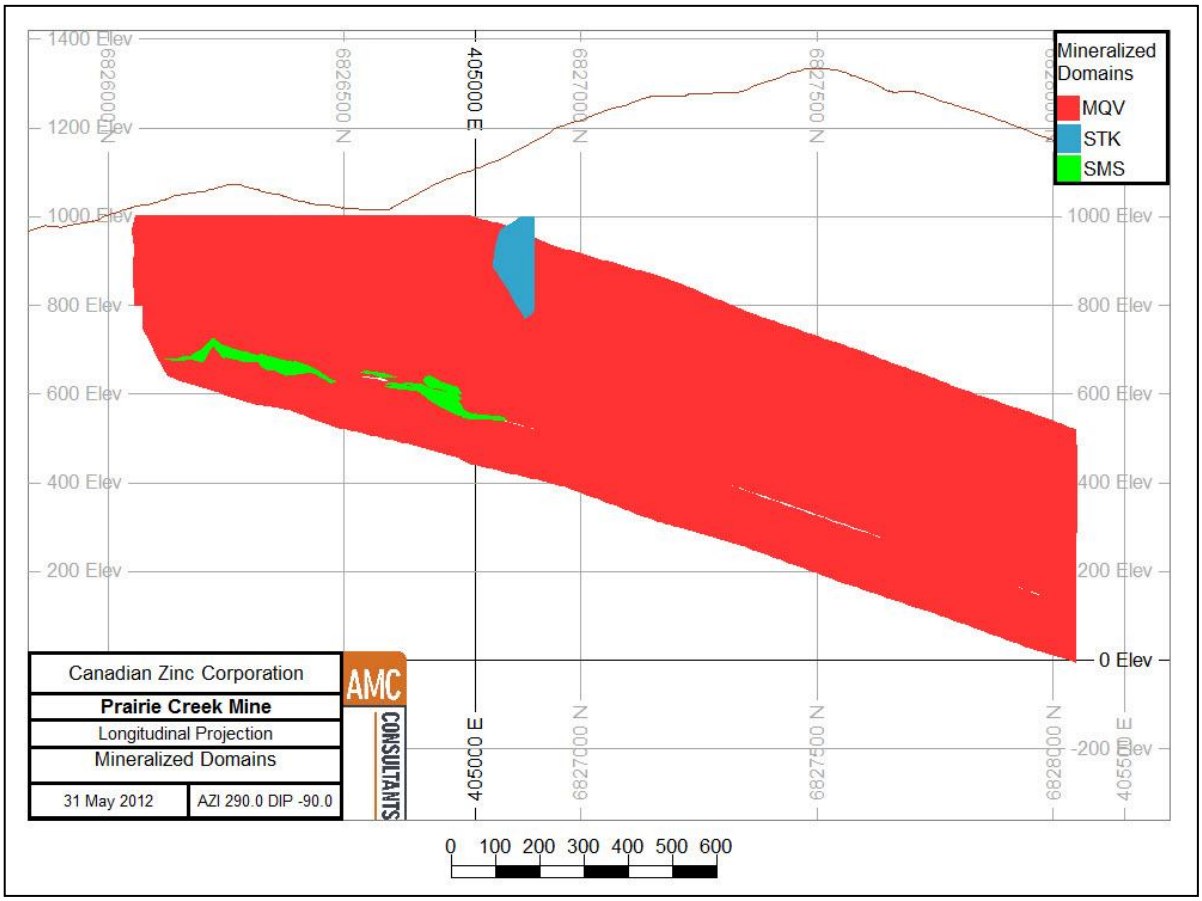
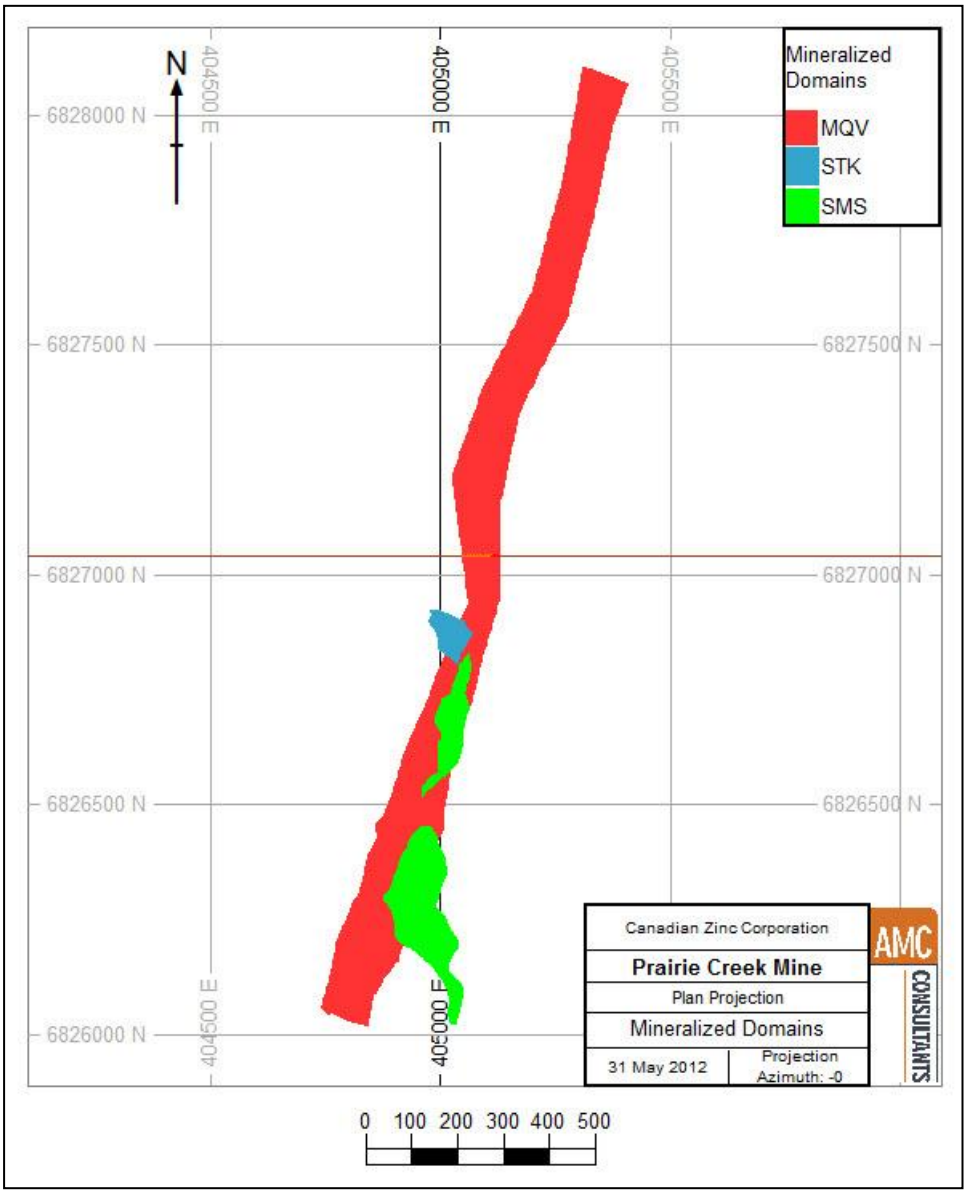


Figure 14.3 Mineralized Domains Plan View



14.4 Statistics and Compositing

14.4.1 Raw Statistics

AMC was provided both the raw and composite data for both the chip and drill core samples for the MQV. This was reviewed by AMC and the results are tabulated in Table 14.4 below.

Table 14.4 Statistics for MQV Raw and Composite Data

Chip Samples	Raw Data				Composite Data			
Field	**Zn %**	**Pb %**	**Ag g/t**	**Cu %**	**Zn %**	**Pb %**	**Ag g/t**	**Cu %**
N Samples	954	959	959	795	331	331	331	331
Minimum	0	0	0	0	0	0.12	0	0
Maximum	57.02	53.72	2220	9.33	41.78	35.24	1073	2.51
Mean	12.71	10.97	202	0.47	12.64	11.00	204	0.38
Core Samples	**Raw Data**				**Composite Data**			
Field	**Zn %**	**Pb %**	**Ag g/t**	**Cu %**	**Zn %**	**Pb %**	**Ag g/t**	**Cu %**
N Samples	2236	2236	2236	2236	117	117	117	117
Minimum	0	0	0	0	0.03	0.04	0.69	0.001
Maximum	64.12	69.88	1739	6.13	34.04	38.64	563.99	1.45
Mean	8.6	11.39	174	0.36	9.1	11.61	182.18	0.38

The comparison of the mean grades of the all elements in raw data versus composites shows a very slight difference. However it has been noted that the mean grades for zinc in chip samples are higher than the mean grades in drillholes. The reason for this discrepancy is unknown. Contributing factors may be different sampling methods, slight vertical zonation in the vein and / or preferential oxidation of zinc sulphides.

14.4.2 Composite Statistics

To facilitate grade estimation and statistical analysis, drillhole samples were composited for each Main Zone mineralization type: MQV, STK and SMS. Separate analyses were carried out for silver, copper, lead, zinc and specific gravity. All composites were confined to the respective mineralized zones. Those for the MQV were composited over the entire vein intercepts, and therefore of unequal length. Composites created for the STK and SMS zones were of a nominal two metre length. Table 14.5 shows the composite statistics for all samples for each zone.

Table 14.5 Composite Statistics by Metal for All Zones

	Zinc			Lead		
	MQV	**STK**	**SMS**	**MQV**	**STK**	**SMS**
Upper Cap	N/A	N/A	N/A	N/A	N/A	N/A
Number of samples	454	66	199	454	66	199
Minimum value	0.03	0	0.022	0.04	0	0.016
Maximum value	41.78	25.47	34.88	43.01	14.76	42.36
50.0 Percentile (median)	9.68	2.377	9.5	10.426	0.935	3.948
Mean	**11.58**	**5.46**	**9.91**	**11.78**	**2.43**	**5.67**
Variance	75.79	42.4	53.55	59.61	11.8	34.9
Standard Deviation	8.71	6.51	7.3118	7.721	3.435	5.907
Coefficient of Variation	0.75	1.193	0.738	0.656	1.415	1.0442
Skewness	1.09	1.1414	0.501	0.899	1.91	2.445
Kurtosis	3.94	4.114	2.49	3.745	6.013	12.67

	Silver			Copper		
	MQV	**STK**	**SMS**	**MQV**	**STK**	**SMS**
Upper Cap	N/A	N/A	N/A	2.6	2.6	N/A
Number of samples	454	46	199	392	66	199
Minimum value	0.01	5.151	0	0.001	0	0
Maximum value	1,703.00	1741	410.1	2.51	1.196	0.31
50.0 Percentile (median)	179.70	62.15	39.55	0.366	0.033	0.01
Mean	**201.90**	**153.80**	**57.43**	**0.43**	**0.11**	**0.02**
Variance	20,123	68294	3887.07	0.124	0.037	0
Standard Deviation	141.90	261.3	62.35	0.352	0.193	0.04
Coefficient of Variation	0.70	1.699	1.09	0.813	1.74	1.67
Skewness	1.52	4.862	2.53	1.599	3.327	5.07
Kurtosis	7.85	29.67	11.88	7.04	16.89	33.3

14.4.3 Grade Capping

The MineFill model employed capping for copper only, and a level of 2.6% Cu was chosen (Table 14.5). AMC carried out an analysis using probability plots on the raw data and agrees with this figure for copper. AMC would suggest capping limits of 850 g/t Ag, in addition to 50% for both zinc and lead.

14.5 Block Model

Two block models were created by MineFill in SURPAC, one which encompassed the MQV and STK, and a second that encompassed the SMS solid.

The block model parameters are summarized in Table 14.6.

Table 14.6 Summary of Block Model Parameters

Type	MQV + STK			SMS		
	y	x	z	y	x	z
Minimum co-ordinates	6,825,984	404,681	-50	6,825,984	404,682	500
Maximum co-ordinates	6,828,336	405,125	1,102	6,826,912	405,210	800
User block size	16	4	16	16	16	4
Min. block size	2	0.5	2	2	2	0.5
Rotation	16.6°	0°	18°	16.6°	0°	18°
Total Number of Blocks	1,610,336			128,738		

Due to the similarity of their trends, the two block models use the same orientation. For the MQV model, the block size was reduced to four m in the X direction to better represent the width of the vein. For the SMS block model, a block size of four m was used in the Z direction to reflect the flat lying nature of the SMS mineralized zone. Note rotation has been carried out in two orientations. AMC would recommend a maximum of one rotation orientation.

14.6 Variography and Grade Estimation

Variography was carried out on the data by MineFill and is discussed in the 2007 and 2011 Technical Reports.

Block assay values were computed by the inverse distance to the second power (ID2). Three passes were performed for silver, copper, lead, and zinc:

- The first pass utilized an octant search with a minimum of four samples to a maximum of 24 samples, the range found through variogram modeling and no more than three empty adjacent octants.
- The second pass utilized an ellipsoidal search with a minimum of two samples, a maximum of 24 samples and a range of 300 m.
- A third pass utilized an ellipsoidal search with a minimum of two samples, a maximum of 24 samples and a range of 500 m (to establish the amount and position of any additional resources that were not captured by the of the 300 metre search radius).

Estimations were performed inside each mineralized solid; the MQV plus STK block model took precedence over the SMS model where the solids overlapped.

The SG block model values for MQV and SMS were also estimated from measured and computed values, also using inverse distance ID^2. The parameters used for purposes of estimation are those of lead because consideration of the available SG dataset showed that lead assay grades have the greatest influence on SG values.

AMC made one change to the estimation, as in the MQV there were some non-estimated blocks within the wireframe at the lower levels. While this part of the solid did not have any data, it is part of the interpreted vein system and was allocated grades which were the average of the blocks in the portion of the vein above. This part of the model is an Inferred Mineral Resource, and it was considered geologically justified to assign grades to this volume. The effect of this can be seen on the Inferred line for MQV in Table 14.10.

14.6.1 Resource Classification

The MQV was initially reviewed and, due to the spotty nature of the classification, it was decided to reclassify by altering the boundaries slightly, and removing outliers. This gave a more realistic classification and this is shown in a schematic long projection in Figure 14.4.

A reclassification of the SMS zone was also carried out, which removed some outliers and consolidated the classification where reasonable. A plan projection of the SMS zone is shown in Figure 14.5.

The results of this reclassification can be seen in the Table 14.10, which shows a comparison between the 2007 / 2011 and May 2012 estimates.

Figure 14.4 Long Projection Looking West with MQV Classification

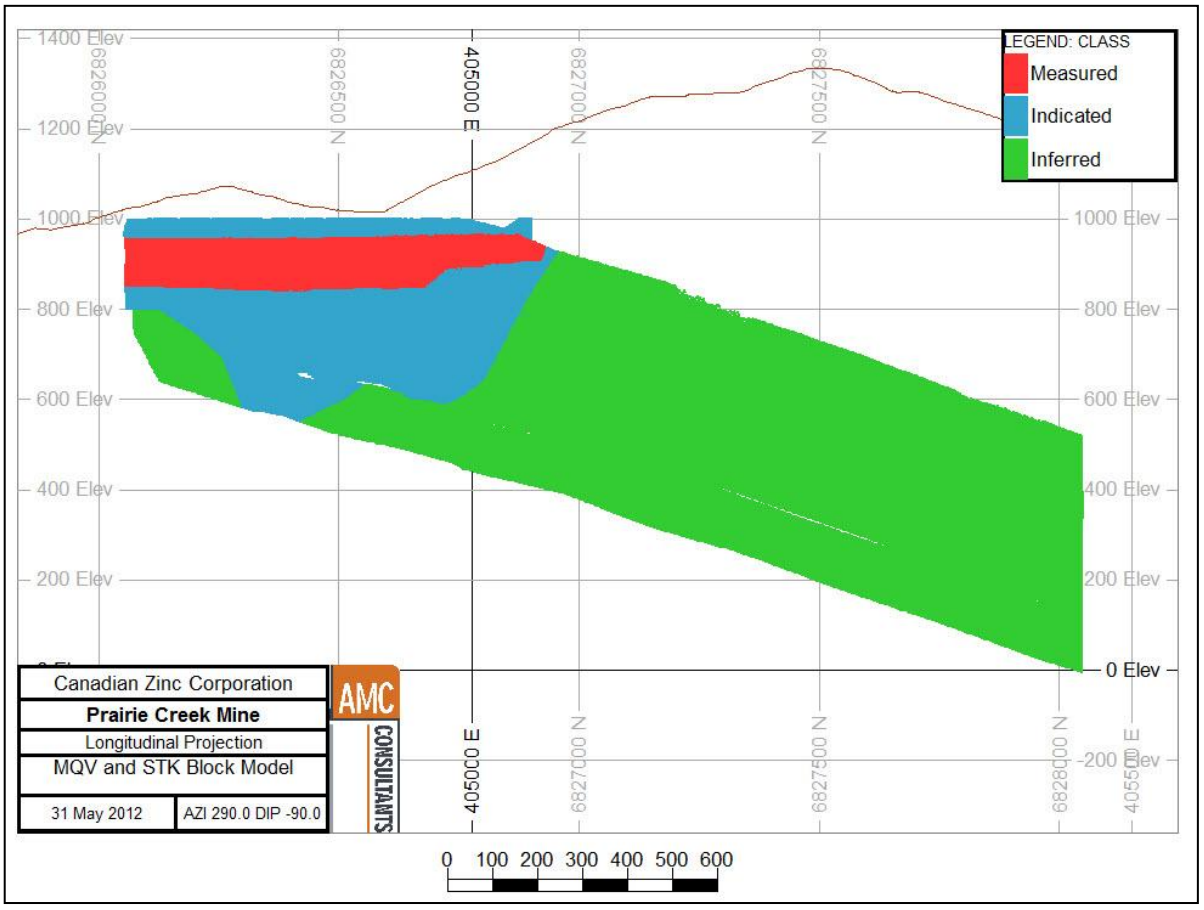
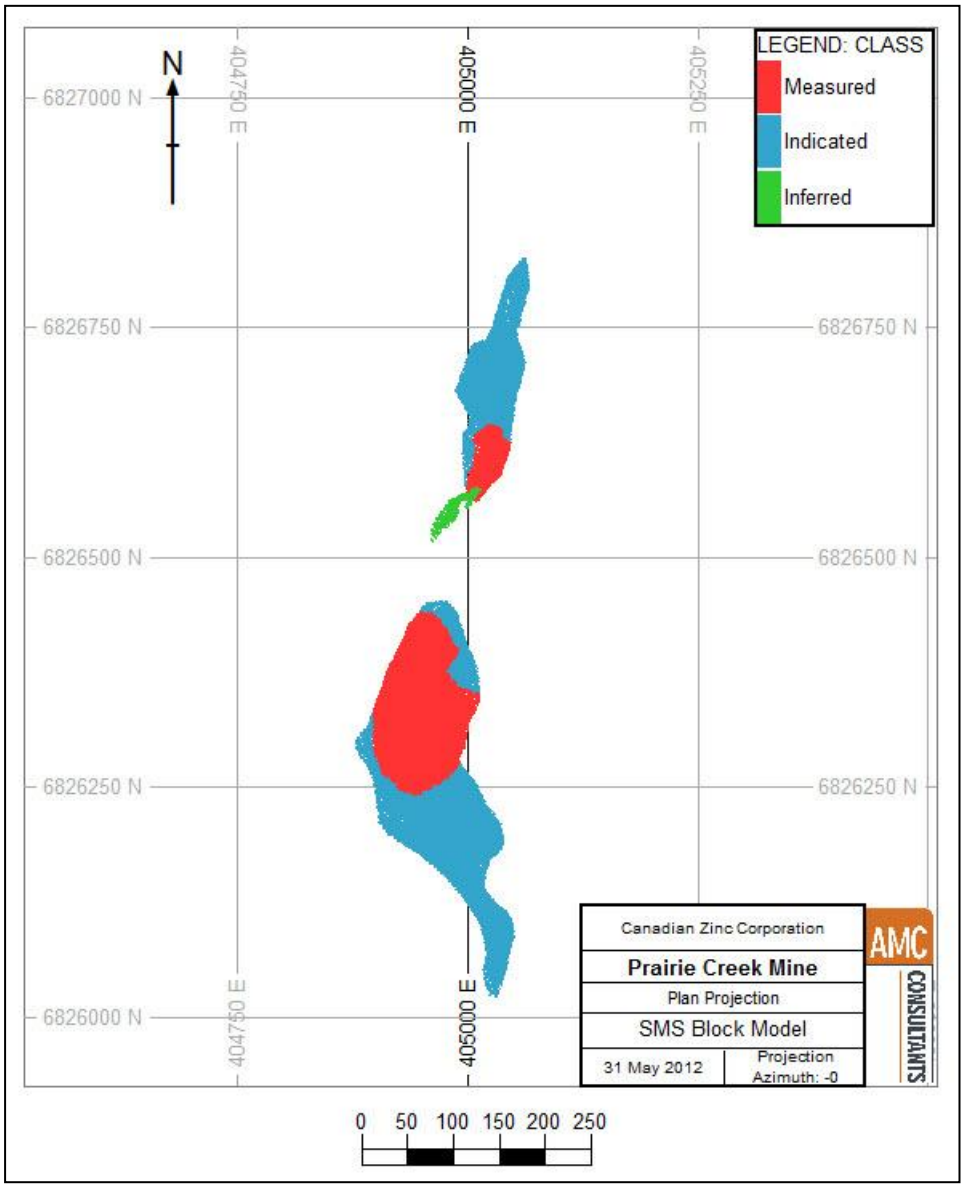


Figure 14.5 Plan Projection with SMS Classification



14.6.2 Block Model Validation

Visual checks were carried out by AMC to ensure that the grades respected the raw data and lay within the constraining wireframes.

The comparisons of the composite grades to the block model grades by zone are summarized from the 2007 Technical Report and are shown in Table 14.7.

Table 14.7 Comparison of Composites to Outputs

	Composites			Block Models		
Zinc	**MQV**	**STK**	**SMS**	**MQV**	**STK**	**SMS**
Number of samples	454	66	199	617	88	171
Minimum value	0.03	0.00	0.02	0.18	1.43	1.03
Maximum value	41.78	25.47	34.88	28.41	17.23	19.50
Mean	11.58	5.46	9.91	10.63	5.09	10.64
Lead						
Number of samples	454	66	199	617	88	171
Minimum value	0.04	0.00	0.02	0.58	0.42	0.73
Maximum value	43.01	14.76	42.36	28.79	0.79	18.51
Mean	11.78	2.43	5.67	13.68	2.31	6.41
Silver	**MQV**	**STK**	**SMS**	**MQV**	**STK**	**SMS**
Number of samples	454	46	199	617	88	171
Minimum value	0	5	0	15	12	8
Maximum value	1703	1741	410	422	151	178
Mean	202	154	57	228	43	69
Copper						
Number of samples	392	66	199	617	88	-
Minimum value	0.00	0.00	0.00	0.01	0.02	-
Maximum value	2.51	1.20	0.31	1.23	0.27	-
Mean	0.43	0.11	0.02	0.49	0.10	-

Note the composites are not declustered.

14.7 Mineral Resource Estimate

The results from the AMC review and reclassification are shown in the tables below. Table 14.8 shows a summary of the Mineral Resource at a cut off of 8.0% Zn equivalent (Zn Eq) for all zones. This cut off is based on costs, recoveries and payable product used in the Mineral Reserves and uses an elevated zinc price of $1.30/lb. All these values are shown as notes in Table 14.8. With the deposit being relatively high value moving the cut off grade makes little difference at these levels, other than for the lower grade Stockwork Zone.

Table 14.8 Mineral Resources at 31 May 2012

Zone	Classification	Tonnes (M)	Zn (%)	Pb (%)	Ag g/t	Cu (%)
Main Quartz Vein	Measured	1.055	13.2	11.5	209	0.45
	Indicated	2.680	10.5	12.7	200	0.43
	Measured + Indicated	3.736	11.3	12.4	202	0.43
	Inferred	6.236	14.5	11.5	229	0.57
Stockwork	Indicated	0.410	7.7	3.7	69	0.15
Stratabound	Measured	0.640	10.5	6.8	67	0.00
	Indicated	0.641	10.6	5.4	63	0.00
	Measured + Indicated	1.281	10.5	6.1	65	0.00
	Inferred	0.003	12.4	5.1	46	0.00
Combined	Measured	1.700	12.1	9.7	155	0.28
	Indicated	3.731	10.2	10.5	162	0.32
	Measured + Indicated	5.431	10.8	10.2	160	0.31
	Inferred	6.239	14.5	11.5	229	0.57

Notes:

1. Mineral Resources are stated as of 31 May 2012.
2. Mineral Resources include Mineral Reserves.
3. Stated at a cut-off grade of 8% Zn-Eq based prices of $1.30/lb for both zinc and lead, and $35/oz for silver.
4. Average processing recovery factors of 78% for Zn, 89% for Pb and 93% for Ag.
5. Average payables of 85% for Zn, 95% for Pb and 81% for Ag.
6. $ Exchange rate = 1 CD/USD.

AMC is not aware of any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other similar factors which could materially affect the stated mineral resource estimates.

In Table 14.9, the totals are shown at a range of cut-offs with the preferred resource estimate emboldened. The same notes apply as above.

Table 14.9 Mineral Resource Estimates, Range of Cut-off Grades

Cut off Grade Zn Eq %	Tonnes (M)	Zn (%)	Pb (%)	Ag g/t	Cu (%)
MQV Measured and Indicated Resources					
0	3.820	11.0	12.1	199	0.43
4	3.778	11.2	12.3	201	0.43
5	3.765	11.2	12.3	201	0.43
6	3.758	11.2	12.3	202	0.43
7	3.746	11.2	12.3	202	0.43
8	**3.736**	**11.3**	**12.4**	**202**	**0.44**
9	3.730	11.3	12.4	203	0.44
10	3.725	11.3	12.4	203	0.44
SMS Measured and Indicated Resources					
0	1.293	10.5	6.1	65	-
4	1.288	10.5	6.1	65	-
5	1.288	10.5	6.1	65	-
6	1.287	10.5	6.1	65	-
7	1.287	10.5	6.1	65	-
8	**1.281**	**10.6**	**6.1**	**65**	**-**
9	1.254	10.7	6.2	66	-
10	1.228	10.8	6.3	67	-
STK Indicated Resources					
0	0.686	5.8	2.7	50	0.11
4	0.638	6.1	2.8	53	0.12
5	0.554	6.7	3.1	58	0.13
6	0.508	7.0	3.3	62	0.14
7	0.452	7.4	3.5	66	0.14
8	**0.410**	**7.7**	**3.7**	**69**	**0.15**
9	0.375	8.0	3.9	73	0.16
10	0.321	8.5	4.1	77	0.16

14.8 Comparison with 2007 Resource Estimate

Table 14.10 shows a comparison of the May 2012 AMC Mineral Resource figures with those reported in the 2007 and 2011 Technical Reports.

Table 14.10 Comparison of May 2012 and 2007 / 2011 Resource Estimates

	May 2012				2007 / 2011			
Class	**Tonnes (M)**	**Zn (%)**	**Pb (%)**	**Ag g/t**	**Tonnes (M)**	**Zn (%)**	**Pb (%)**	**Ag g/t**
MQV								
Measured	1.055	13.2	11.5	209	0.939	13.1	11.6	212
Indicated	2.680	10.5	12.7	200	2.945	11.2	12.7	212
Measured + Indicated	3.736	11.3	12.4	202	3.883	11.6	12.4	212
Inferred	6.236	14.5	11.5	229	5.516	13.6	11.5	216
STK								
Indicated	0.410	7.7	3.7	69	0.682	5.9	2.7	50
SMS								
Measured	0.640	10.5	6.8	67	0.611	10.9	6.7	68
Indicated	0.641	10.6	5.4	63	0.663	10.2	5.5	62
Measured + Indicated	1.281	10.5	6.1	65	1.275	10.5	6.1	65
Inferred	0.003	12.4	5.1	46	0.020	10.5	5.7	56
Total								
Measured	1.700	12.1	9.7	155	1.550	12.2	9.7	155
Indicated	3.731	10.2	10.5	162	4.290	10.2	10.0	163
Measured + Indicated	5.431	10.8	10.2	160	5.840	10.7	9.9	161
Inferred	6.239	14.5	11.5	229	5.420	13.5	11.4	215

Notes:
1. AMC figures at an 8%Zn Eq cut off
2. MineFill classification reported at zero cut off as total volume within the zone wireframes
3. Copper differences are minor so only main contributing metals shown above

The main reasons for the differences between the 2012 and 2007 / 2011 statements are:

- Review of the classification which consolidated the categories and also discarded isolated areas based on poor support.
- Inclusion of a portion of the MQV which has poor data support but strong geological correlation as Inferred Mineral Resource.
- Application of an 8% Zn Eq cut off grade in May 2012.

Comments and Recommendations

As discussed above, in AMC's review of the model highlighted discrepancies between the data and the wireframes supplied. It would appear that drilling carried out at the time of the estimate was included, but the wireframes were not adjusted to the new intersections. However the composites were flagged such that they were included in the estimate. This is not material; however, AMC recommends this be cleaned up in the next modelling exercise.

AMC also noted that the MQV wireframe stopped at the southern drilled section and was not continued some distance beyond the holes. Again this is not material but AMC recommends that wireframes should be altered to fit the interpretation including some reasonable extrapolation for the next estimate.

AMC recommends employing capping on raw data has follows 2.6% Cu, 50% Pb, 50% Zn and 850 g/t Ag. The MineFill model employed capping for copper of 2.6%.

Due to the close spacing of the chip samples on the MQV mineralization, AMC recommends that chip sample composites are created by equal length and the data is declustered.

Rotation of the block model has been carried out in two orientations. AMC recommends a maximum of one rotation orientation or preferably an unrotated model.

As discussed in Chapter 13, three penalty elements, antimony, arsenic and mercury, report to the final concentrates. AMC recommends that these elements be modelled.

Secondary oxidation is locally developed in the MVQ mineralization with variable depths and severity as discussed in Section 7.3.1. AMC recommends calculating the percentage oxide component in the lead and zinc and modelling this information.

The inclusive cost for all modelling is $50,000.

15 MINERAL RESERVE ESTIMATES

15.1 Introduction

To convert Mineral Resources to Mineral Reserves, mineable stope designs were designed within the deposit at sufficient widths to create stopes that can be readily accessed and practically mined (for the given equipment) that incorporate a mining recovery and generate a dilution factor. Only Measured and Indicated Mineral Resources have been converted to reserves, any Inferred material within the designed stopes is treated as waste and allocated a zero grade.

15.2 Methodology

Given the size of the single boom jumbo and 2 cu. yard scoops to be used at Prairie Creek, a minimum mining width of 2.5 m and an average lift height of 5 m was used during stope design. Mining dilution averaged 22% for the MQV reserve and 10% for the SMS reserve, highlighting the more massive, albeit lower metal grade, nature of the SMS resource. Dilution was reported directly out of the mine design using the Gemcom MineSched program, however areas reporting under 10% dilution, such as some areas within SMS, were bumped up to report a minimum of 10% dilution. An average density of 2.7 t/m^3 has been assumed, based on previous lab specific gravity measurements, for waste material in dilution, added with zero grade.

The MQV block model was designed with minimum block dimensions within the mineralized area of the model of 0.5 m width, 2 m height and 2 m depth. The SMS block model was designed with minimum block dimensions of 2 m width, 2 m depth and 0.5 m height.

15.3 Mining Cut-off Grades

The cut-off grade was calculated from the parameters shown in Table 15.1.

Table 15.1 Costs and Recoveries Used to Establish Cut-off Grade

Item	Value
Operating costs (variable costs)	$162/t
Zn recovery	75%
Pb recovery	88%
Ag recovery	92%
Zn payable	85%
Pb payable	95%
Ag payable	81%

The equation used to calculate the mining cut-off grade is as follows:

ZnEq = Grade of Zn in % + Grade of Pb in %*(Price of Pb/Price of Zn)*(Pb recovery*Pb payable)/(Zn recovery*Zn payable) + Grade of Ag in gpt*(Price of Ag/grams per t.oz/lbs per t/Price of Zn)*(Ag recovery*Ag payable)/(Zn recovery*Zn payable)*100

Mining Cut-off grade in % ZnEq = Variable cost/Value of 1% ZnEq

Variable cost = cost per tonne mined for mining, processing and transportation

Within the cut-off grade equation above, zinc and lead grades are added as whole numbers (i.e. 9% zinc would be added into the equation as “9”, not “0.09”).

For additional clarity, the equation is illustrated in Figure 15.1 in a tabular format.

Table 15.2 Calculation Formulas for Zinc Equivalency

Realisable Value ($/T)	=	Grade x price	x	Recovery & payables normalization
	=	Zn kg/t x Zn $/kg	x	1
	+	Pb kg/t x Pb $/kg	x	(Pb recovery x Pb payable) (Zn recovery x Zn payable)
	+	Ag g/t x Ag $/g	x	(Ag recovery x Ag payable) (Zn recovery x Zn payable)

Where: kg/t (Zn or Pb) = 10 x % (Pb or Zn)
$/kg (Zn or Pb) = 2.20462 x $/lb (Zn or Pb)
$/g (Ag) = $/oz (Ag) / 31.10348

$$\%Zn_{eq} = \frac{\text{Realisable Value (\$/T)}}{\text{\$/kg (Zn) x 10}}$$

Mining cut-off grade = 10% Zn equivalent.

A milling cut-off grade was calculated in addition to a mining cut-off grade. This was calculated in the same way that the mining cut-off grade was calculated although it removes mining costs from variable operating costs. A variable cost of $95/t was used in calculating the milling cut-off grade. Mining costs are removed because the milling cut-off grade makes the assumption that the block of material being evaluated will be mined regardless of its value. However it will be sent to the waste rock pile or the mill depending on if its average grade is above that of the milling cut-off grade. This cut-off grade is used for marginal material that needs to be mined in order to access more profitable material ahead of the working face.

The milling cut-off grade = 6% Zn equivalent

Due to the high grade and extensive persistence of the MQV material, the cut-off grade had minimal effect upon reserve tonnages and grades.

15.4 Mining Recovery

Due to the high grade and continuous nature of the MQV resource, 97% of the resource was estimated to be recoverable. To account for sill pillars, a combination of smaller lift heights and tighter filling is planned at the top of each stope.

The SMS resource will be mined via room-and-pillar mining methods. It is assumed that 15% of the mineable area will be left as pillars during this operation. This will be further quantified by a geotechnical program at Prairie Creek this summer. Areas within thinner sections of the SMS resource were considered prohibitively dilutive and not included within the mineral reserve. Overall mining recovery of the SMS resource is estimated to be 57%.

15.5 Mineral Reserves

Mineral Reserves Statement

The Prairie Creek Mineral Reserves are estimated in Table 15.2 below.

Table 15.3 Mineral Reserves at May 2012

Zone	Class	Tonnes (M)	Zn (%)	Pb (%)	Ag (g/t)
Main Quartz Vein	Proven	1.278	10.8	9.4	172
	Probable	3.140	8.7	10.5	165
	Proven and Probable	4.418	9.3	10.2	167
Stratabound	Probable	0.803	9.5	5.7	62
Total Mineral Reserves		5.222	9.4	9.5	151

1. Cut-off grade of 10% ZnEq based upon total variable operating cost of $162/t including mining, processing and transportation. Metal prices assumed are Zn = $1.10/lb, Pb = $1.10/lb and Ag = $28/t.oz.
2. Average processing recovery factors of 75% for Zn, 88% for Pb and 92% for Ag.
3. Average payables of 85% for Zn, 95% for Pb and 81% Ag.
4. Exchange rate = 1 CAD/USD

Recoverable reserves tonnages within the MQV are higher than the corresponding resource tonnages. This is due to dilution waste that is included within the diluted reserves. Grades of the reserves drop accordingly with respect to the addition of waste at zero grade.

15.6 Comparison of Measured and Indicated Resources to Proven and Probable Reserves

Table 15.3 compares Measured and Indicated Resources to Proven and Probable Reserves.

Table 15.4 Measured and Indicated Resources versus Proven and Probable Reserves

Item	Resource (Measured & Indicated)	Reserve (Proven & Probable)	% change
MQV tonnes	3,736,000	4,418,000	+18%
SMS tonnes	1,281,000	803,000	-37%
Zn grade	10.8%	9.4%	-15%
Pb grade	10.2%	9.5%	-7%
Ag grade	160 g/t	151 g/t	-6%
In-situ lbs Zn	1,293 M lbs	1,078 M lbs	-20%
In-situ lbs Pb	1,221 M lbs	1,093 M lbs	-12%
In-situ oz Ag	27.9 M oz	25.4 M oz	-10%

Reserve tonnes within the vein are 18% higher than Resource tonnes and Reserve grades of Zn, Pb and Ag are 6% - 15% lower than Resource grades, largely due to mining dilution.

16 MINING METHODS

16.1 Summary

The Prairie Creek mine (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 (970 mL, 930 mL, 880 mL) have already been established at the Mine and these are targeted to be mined early in the life of the proposed operation. Within these three levels a number of shrinkage stopes were being developed and generated a stock pile of 50,000 tonnes of ore stored near the mill. The MQV deposit continues at depth and additional mining levels will be established on the structure. As mining progresses to depth, mining feed generated from the MQV will be supplemented by the deeper stratabound (SMS) deposits. Both deposits will also be accessed by a single ramp development. Initial mining will be focused in the upper parts of the mine including the existing three levels of workings; lower levels will be developed to depth through a trackless ramp access over the first five years of operation.

Mining will be conducted at the Mine primarily using the mechanized cut-and-fill method on the narrow vein structure, with some potential use of the room-and-pillar mining method on the SMS material. An average mining rate of 1,350 tonnes per day of diluted ore is targeted. During full production, approximately 500,000 tonnes of ore per year will be mined over an 11 year life of mine with a total mining reserve of 5.2 million diluted tonnes at 9.4% Zn, 9.5% Pb and 151 g/t Ag.

Vein material is the main focus of mine production and will be extracted throughout the life of the Mine. The MQV has been drill-defined over a 2,000+ m long semi-continuous structure with a vertical extent exceeding 400 m. The vein structure is exposed in over 800 m of backs within the existing three levels of underground development. Vein material consists of high grades of lead, zinc and silver and small amounts of copper with the SMS containing similar zinc values but significantly lower values of lead and silver. The SMS also contains significant amounts of iron in the form of pyrite and is only drill defined at this stage.

The SMS mineralization occurs approximately 200 m below the valley floor elevation and requires significant underground development in order to access it. This underground development will serve dual purposes: accessing the SMS deposit for mining and further accessing MQV material, which is known to occur at depth. Commencement of mining of the SMS ore is presently scheduled to start during the tenth year of the mine life.

Mining of both ore types will be mechanized using electric-hydraulic drilling, electric-pneumatic rock-bolting machines, diesel-powered scoop-trams and diesel-powered haul trucks. Access to the mine will be primarily through the existing 870 portal (site elevation which accesses the 880 Level). This portal will be slashed out to 4.5 m high by 4.5 m wide to accommodate for larger trackless equipment. Underground development, existing from the 1980's will be fully utilized to help minimize the amount of pre-development required to

achieve mine operation. The existing three levels of the mine, known as the upper mine, will account for the majority of mill feed in the initial years of operations. As development progresses, ore will be accessed from the lower mine and will sustain the necessary feed required to keep the mill running at full capacity.

The present mine plan is based entirely on a detailed Measured and Indicated Resource category of approximately 5.4 million undiluted tonnes of MQV and SMS material as discussed in Chapter 14.

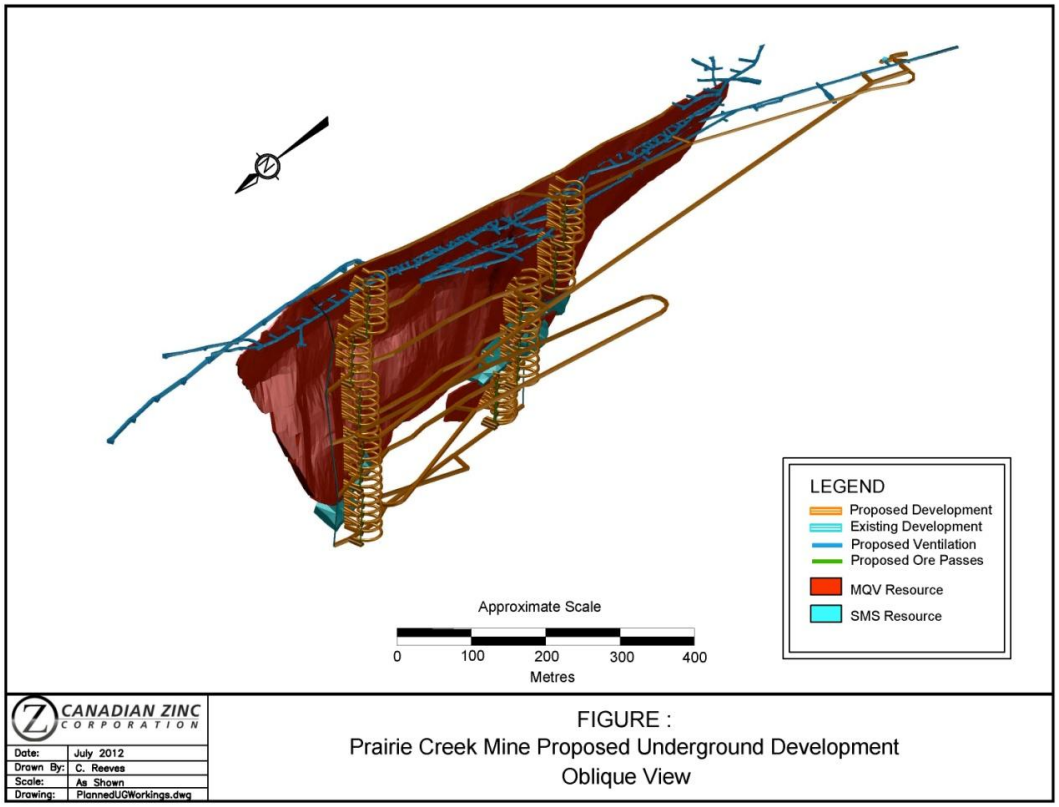
Ground conditions in existing development underground are good and the existing workings have stood unsupported for close to 30 years with minimal bolting.

In 2007 and 2008, Canadian Zinc contracted Procon Mining and Tunneling to drive a 600 m long 3 m by 3 m exploration decline hanging-wall tunnel at the north end of the then existing 880 level workings. Standard bolt patterns were put in place, and apart from the soft rock within the crosscut of the vein caused by the high grades, ground conditions were reasonable. As was expected, the tunnel was quite wet and required moderate pumping. However, the workings were drying out at by the end of the program, indicating the drawdown of the groundwater table had been achieved. It was also noted that the MQV structure was a significant aquifer for groundwater.

It should be noted that previously the 880 portal and 880 level have been referred to as the 870 portal and the 870 level. The 880 and 870 naming convention refers to the same area and should be considered synonymous.

Figure 16.1 shows an orthogonal view of the planned underground workings.

Figure 16.1 Orthogonal View of Planned Underground Workings



16.2 Mining Method

The Prairie Creek Mine plan will primarily use mechanized overhand cut-and-fill mining methods on the MQV structure. The stopes that were developed, and in some instances mined, for the shrinking mining method, will be converted to the cut-and-fill method. Certain wider and shallower dipping areas of stratabound mineralization will be mined using room-and-pillar mining methods.

Ore will be delivered by 20 tonne haulage trucks via the main access ramp to the 880 level dump pocket. The broken ore will then be transported by conveyor from the pocket to the existing main conveyor feeding the primary crusher in the Mill on surface. A dense paste tailings mixture will be used as backfill for the mining operation.

16.2.1 Mechanized Cut and Fill Mining

Cut-and-fill mining is commonly used to mine steeply dipping narrow vein deposits. It is extremely selective and has high mining recovery. Also, utilizing this mining method tends to minimize dilution, resulting in the need for less waste handling. Cut-and-fill mining is

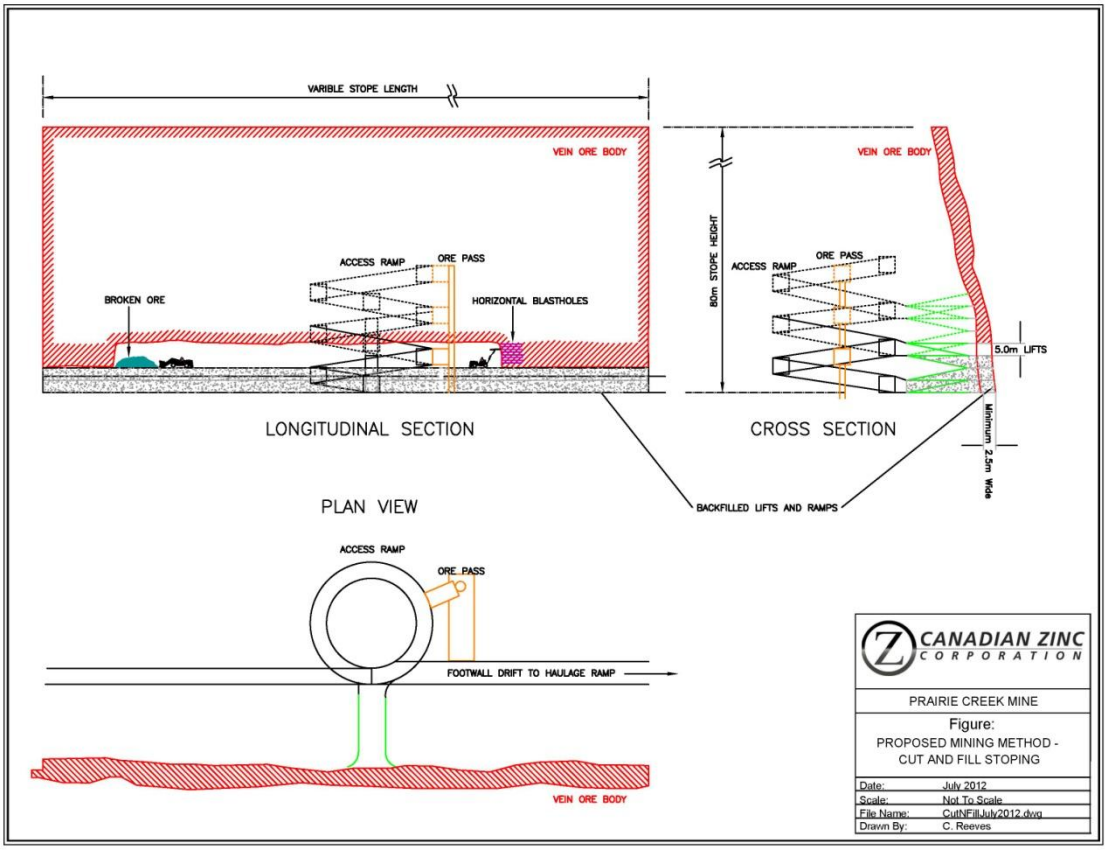
conducive to the tailings waste management plan that uses the paste tailings as backfill which acts as the running surface for every subsequent lift. A drawback of this method is the extensive development required, low production rate and high cost per tonne to mine the ore. In this case, however, the advantages outweigh the disadvantages.

At Prairie Creek, all mining of the MQV will be conducted with cut-and-fill mining or some variation thereof. The mining levels of 880 and above have already been established but need to be expanded upon. Mining below the 880 level will require 4 new main levels, each at 80 m intervals. Mining will take place within an initial cycle of drilling, blasting, roof support, mucking and eventually followed by backfilling that will typically last several months. Access will be gained to the base of the stoping section by ramp and a cross-cut drift driven from each of the main mining levels to the hanging wall of the ore body. Drifting is then continued along the full length of the stope, in the sill of the stope, in the MQV, with slashing to the full width of the stoping section. In the event that the unsupported width is less than the stoping section then a drift and fill mining method can be employed, to ensure the maximum amount of ore is removed. The vast majority of the mineralized vein at Prairie Creek is contained within a very narrow width allowing for ore within a given stope to be removed in a single pass.

After the first 5 m lift has been completed in the cut-and-fill stope, preparations for the second cut will be made. The access ramp roof will be slashed and the waste material from the roof is back slashed into the stope to form a bulkhead to aid filling. Prior to mining the second lift, at least partial filling of the mined portion will be necessary, as drilling for the second lift will be drilling horizontal holes (breasting). This new lift (5 m) will then be mined from the roof of the existing stope, following the discussed sequence of roof control, mucking and then filling. The ramp access is once again slashed to provide access to the next level and so on until the upper level of the stoping panel is reached.

Cut-and-fill mining at Prairie Creek will use a range of mechanized equipment, including one boom jumbo drills, 4-7 tonne scoop trams, 20 tonne articulated haul trucks and mechanized scaling, bolting and support equipment. It is yet to be determined if it may be more cost efficient to purchase or lease to purchase the equipment. A schematic of the cut and fill method is shown in Figure 16.2.

Figure 16.2 Cut and Fill Mining Method



16.2.2 Room-and-Pillar Mining

The SMS mineralization at the Mine represents a different style of resource to that of the vein deposit because of the apparent flatter attitude and wider ore intersections. Due to the greater amount of tonnes per vertical metre, the SMS mineralization is capable of sustaining higher production rates from mechanized room-and-pillar mining.

Mining recovery using room-and-pillar mining methods is lower than that of cut-and-fill because some ore is left as pillars to support the roof of the excavation. Productivity should be higher in this area, as equipment space will be less constrained.

16.3 Geotechnical

Limited geotechnical work within the mine has been conducted to date. There currently exists three levels of underground workings in the upper mine. These levels were driven in the early 1980s and have held up well. During the 2007 underground program, bolts were installed as a precautionary measure only in sections totally barren of support. These levels give good confidence that the current mine plan and ground support regime will be appropriate in the upper mine and give reasonable assurance regarding the lower mine. A geotechnical program has been planned for the summer of 2012 to confirm the geotechnical characteristics of the lower mine. These studies may result in reducing the planned 5 meter lift and/or altering the current ground support regime.

16.4 Hydrogeological

The mineralized MQV structure constitutes the main conduit for water to access the mine. Significant quantities of water run through the vein as evidenced by high flows from holes drilled from the new underground decline, increasing with depth. Testing has been done regarding permeability, recharge and flow rates to assess the implications water will have upon mine pumping, discharge and safety.

Sump pumps will be located on each level of underground workings near the access to the main ramp. Each level will have a slight gradient towards the sump area that will have a small pond that will act as a decantation pond. The majority of the water within this pond will be pumped up the haulage ramp to the water storage pond. Some of the water will be skimmed from the top of the pond to be used for drilling. Flow rates increase with depth, requiring significantly larger sumps as the mine progresses deeper.

The Mine will be a wet mine and managing groundwater will be a significant aspect of successful underground mining. Presently, natural groundwater drains out of the 870 mL portal during the summer season at an average of 20 L/s. Noted increases with local rainfall has indicated some contribution from surface runoff. At the bottom of the mine, when in full production the groundwater flow is estimated to produce up to 100 L/s of water. Dewatering of the vein structure, which tends to act as a natural aquifer for groundwater, will be necessary in order for mining to proceed on a consistent advancement. Groundwater within the vein will be tapped into, prior to mine development, via boreholes from the furthest extent of various access drifts in order to pre-drain water upstream of the vein. This water will be pumped from a sump near the boreholes to the larger sumps near the access to the main haulage ramp on each level. From there it will be pumped to surface to be stored within the water storage pond.

All water discharged from the mine will either be sent to the mill as feed water or be pumped into the currently existing tailings pond facility which will be revised and converted into a Water Storage Pond modified with liners and berms prior to commencement of mining. From there, water will be sent to the proposed new water treatment plant. The mine water will undergo separate water treatment (simple lime treatment) than the mill process

water but the bulk of the volume of water treatment will be mine water. This pond has been designed to hold all mine and mill water prior to treatment and discharge into Prairie Creek. A lift will be added to the existing pond facility to provide more storage capacity and contingencies to almost eliminate discharge during the winter when the flow in Prairie Creek is low and relatively un-dilutive compared to spring and summer.

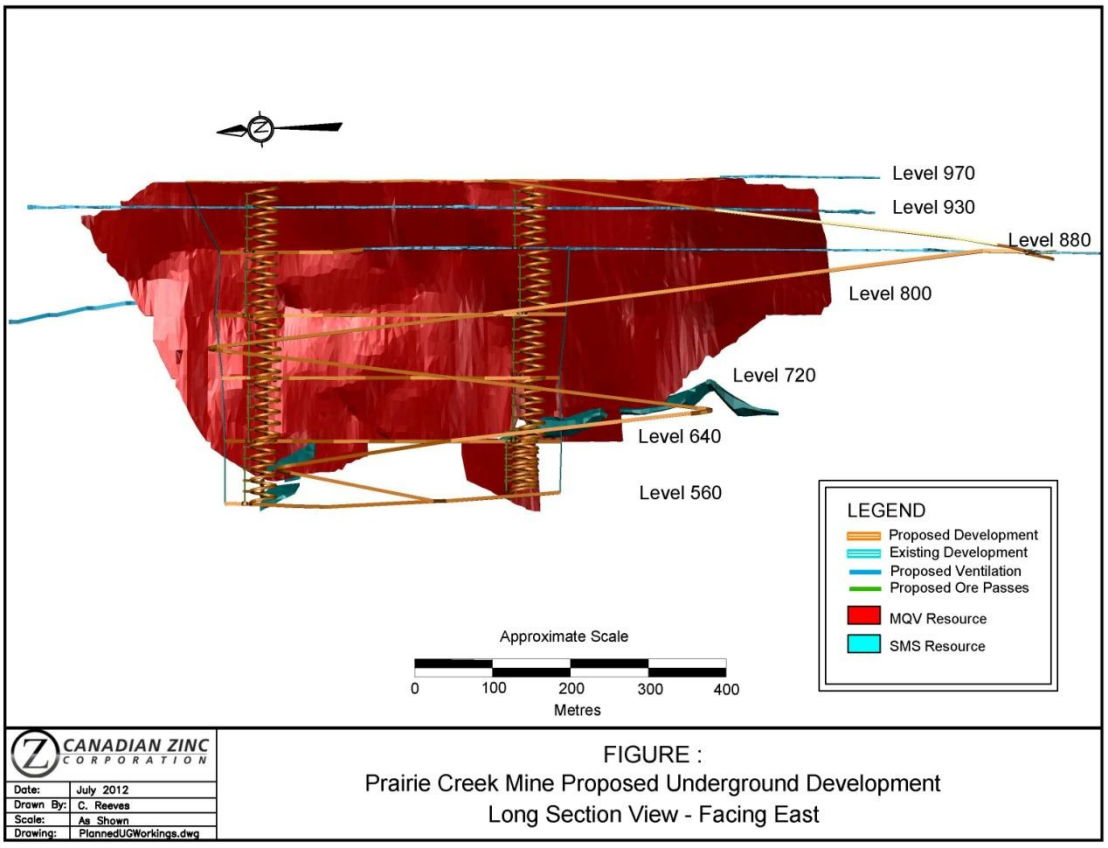
16.5 Mine Design

Mine design was conducted using a combination of Gemcom Gems and Surpac products. Table 16.1 lists the design parameters upon which the underground mine design was based. Figure 16.3 shows a long section of the planned underground workings.

Table 16.1 Underground Mine Parameters

Parameter	Design Criteria
Minimum mining width	2.5 m
Lift height	Up to 5 m
Vertical distance between sublevels	80 m
Vent raise dimensions	2 m x 2 m
Orepass dimensions	2 m x 2 m
Main haulage drive and sublevel drifts	4.5 m wide x 4.5 m high (max 15% grade)
Ore access ramps and cross-cuts	3.2 m wide x 3.2 m high (max 15% grade)

Figure 16.3 Planned Underground Workings in Long Section

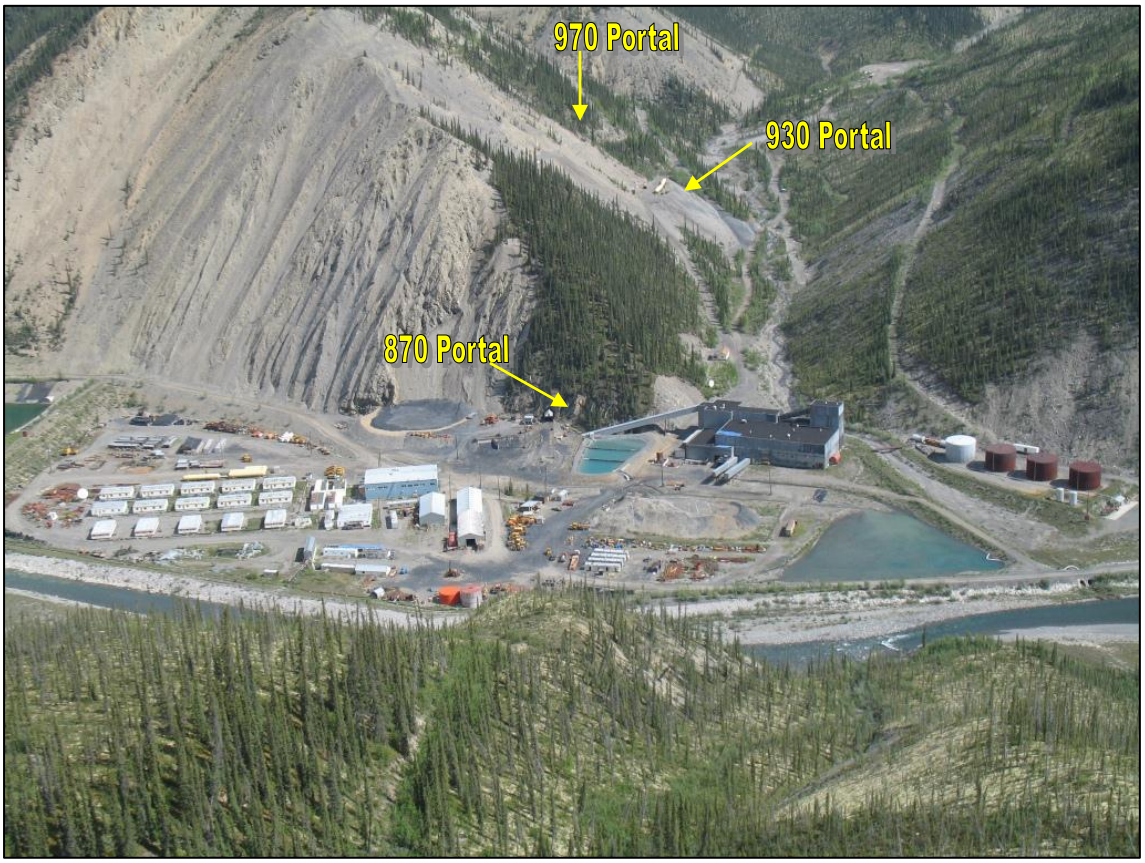


16.6 Development and Production

16.6.1 Existing Development

Main Zone underground development was carried out between the 1970s and the early 1980s, initially for purposes of exploration and later for purposes of facilitating production at the planned mining rate of 1,000 tons per day. In 2006/2007 the Company completed some additional Main Zone development driving a new decline, to facilitate underground resource definition drilling. This work included the installation of a new ventilation system and electrical sub-stations, a track upgrade and general rehabilitation. A new water treatment facility was installed and a new mine water polishing pond was constructed near the 870 mL portal. There currently exist three levels of underground workings as shown in Figure 16.4 below.

Figure 16.4 Locations of the Three Existing Underground Portals at Prairie Creek



970 Metre Level

This level contains 220 m of footwall drift with six crosscuts at 30 metre intervals. Access to the level is limited because it has not been fully rehabilitated. The 970 level is not connected to either the 930 level or the 880 level.

930 Metre Level

930 level consists of 940 m of trackless footwall haulage drift. There are 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. Approximately 50,000 tons of vein material was mined from 930 mL in 1981/82; this material is currently stockpiled next to the processing plant.

A small amount of underground drilling was completed on 930 mL in the early 1980s. At the time of Mine closure in 1982, preparations were underway to create a small backfill plant on 930 mL, but this work remains uncompleted. Limited workshop storage facilities were

completed and a mine air heater was installed, which was indirectly fired by propane. Limited services were available and concrete pads for substations were installed.

In preparation for the 2006/07 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 horse power ventilation fan was installed at the portal. The fan forces air down a manway to 880 mL where a new decline has been excavated by the Company.

880 Metre Level

The portal for the 880 mL (known as the 870 portal) is adjacent to the mill feed conveyor; the 880 mL itself contains approximately 610 m of tracked footwall haulage drift, 380 m of vein drifting and approximately 150 m of other development.

In the early 1980s, a number of shrinkage stopes were developed on 880 mL, as well as two manways (including a service raise) and two orepasses that connect 880 mL to 930 mL. A small ramp was developed at the end of the haulage drift, which provides access to a 50 ton capacity ore bin for loading rail cars. A sump system was also developed, which was excavated to act as a settlement pond and water treatment site (the latter was in part installed to improve process water quality that the results of metallurgical testing showed to influence metal recoveries to concentrate). Services (air, water and power) were brought into the Mine, from the processing plant, on 880 mL, from where they were distributed to 930 mL, via the developed raises.

The Company completed rehabilitation of 880 mL in 2004 and 2005, including timber stripping, rock bolting, installation of electrical facilities, and completion of a refuge station and track extension. The 880 Level portal was used extensively as the main underground access for the 2005/2006/2007 underground exploration programs.

Decline Development

A new crosscut (880-07) was developed in 2006, at the end of the existing 880 mL workings, about 1,000 m from the 870 mL portal. Its purpose was to enable internal access to the planned collar position of a new decline from which underground drilling was to take place in 2006 and 2007, with the objective of firming-up the 1998 Main Zone mineral resource.

Procon Mining and Tunnelling Limited of Burnaby, B.C. (Procon), was contracted by the Company to undertake the development work.

During 2006, the 3.2 metre by 3.2 metre decline was driven at minus 15 percent grade for 70 m into the hangingwall and then was turned north 90° to parallel the strike of the vein. It was driven a further 280 m from the turning point, along which distance six drilling stations were established at 50 metre centers (from section line 50,600N to 50,850N, inclusive).

This Phase I development was completed in December 2006; the stop-point was immediately beyond the 50,900N section line.

Phase II of the planned development comprised an additional 200 m of decline development that established five more drilling stations at 50 metre centers (from section line 50,900N to 51,100N, inclusive). Phase II drilling from the new stations was only partially completed and it was planned to continue the program in the future.

Zone 7 and 8 Development

Zones 7 and 8 were explored by underground drifting during 1969: 280 m of drift were developed on Zone 7 and 240 m of drift were developed on Zone 8. Both drifts are about 330 m below the surface vein showings. The drift portals have since been filled in and will need to be reopened to allow the underground workings to be examined.

A sump system currently exists on the 880 level, excavated to provide for solids settling and water treatment. Presently, all mine water reports to the final sump prior to flowing out of the portal.

16.6.2 Pre-Production and Mine Development

Prior to production, new mining equipment and supplies will be brought to site via a winter road. This will include scoop-trams, haul trucks, electric-hydraulic jumbos/drills, explosives, rock-bolting jumbos, as well as miscellaneous hand-held drills, electric ventilating fans and pumps.

Underground development prior to production will comprise of tunnel enlargement and extension at various points of the existing levels. Stope access development and installation of service utilities will also be conducted during pre-production. The existing 880 level will be slashed out to 4.5 m x 4.5 m and tracks will be removed.

Access to the mine will continue to be primarily via the existing 870 portal to access the 880 level, which will be slashed out to allow for the entrance of larger equipment into the mine.

An inclined ramp and declined ramp will be collared underground near this portal along with an ore transfer facility (dump pocket) and shop facility also established underground. The 880 level will provide facilities for heating the ventilation intake flow, and will enable equipment to be transported in and out of the mine. Development muck haulage and underground water discharge to surface will be from this level. The production haulage ramp, required to transport ore from below the 880 level, will be driven with dimensions of 4.5 m by 4.5 m, in order to accommodate 20 tonne haul trucks and various service vehicles, will be driven at grades from -12% to -15%. The main haulage ramp will access the current known resource by means of a series of sublevel haulage drifts (80m level spacings), ore access ramps, and crosscuts which will allow for overhand cut-and-fill to occur within the stopes. The main haulage ramp will be extended to the 880 level where loaded trucks will dump into the dump pocket.

The dump pocket will be established underground approximately 50 m from the 880 portal. A large vibrating screen (grizzly) with a rock breaker will limit all coarse materials reaching the ore bin. The ore bin will have a capacity for approximately 1,000 t run-of-mine (ROM) ore and will feed a conveyor that will in turn feed a small stockpile within a sprung structure outside of the mine with additional 1,000 t ROM ore storage. A conveyor from this stockpile currently exists and will continue to feed the primary crusher.

Intake fans will be installed near the 870 portal to draw fresh air into the mine. During the winter, the cold air will be heated by a diesel fuelled boiler located close to the entrance. The 930 level presently has surface access and will provide a second means of egress should emergency evacuation be required. The 930 portal will be the location of the primary exhaust ventilation fans which draw the used air from the mine.

During pre-production, orepasses will be developed so that all ore from the upper mine will be dropped down through orepasses into drawpoints (re-mucks) on the 880 level from which scoops will load haul trucks. As the mine moves into production, various sublevels, accesses and other orepasses will be established.

The pre-production phase of the Project is expected to take approximately 12 months and be performed by a contractor. All remaining major development within the mine will be finished by the end of year 5 (after 4 years of production) and will be conducted by the owner.

16.6.3 Production

Rates

Production rates at Prairie Creek will be approximately 225 tonnes per day per production team (drill and scoop). There will be a day shift and night shift and three production teams resulting in an average production of 1,350 tonnes per day, although this can fluctuate.

Transportation of Ore and Waste

Haul trucks of 20 tonne capacity will be used within the mine to transport waste and ore from the development headings and drawpoints, respectively. Scoop-trams will dump ore from stopes into orepasses located close to the stope access ramps. These orepasses will feed into drawpoints and the trucks will ultimately be loaded from drawpoints by another scoop. Loaded trucks will drive up the production ramp to dump ore at the dump pocket located on the 880 level near the 870 portal.

In order to achieve 100% disposal of tailings underground, no development rock will be left underground. It will be hauled up the ramp and out of the mine and placed in the surface Waste Rock Pile. Table 16.2 shows estimated volumes of waste components stored in the waste rock pile. Waste will be temporarily dumped to the west of the 870 portal from where it will be loaded by a surface loader onto a surface haul truck and hauled to the waste

dump. Only dense media separation float material waste rock will be placed back in the mine, as it will be used as aggregate for paste backfill running surfaces.

Table 16.2 Estimated Waste Quantities in the Waste Rock Pile

Item	Expected Volume (m^3)
Development waste rock	300,000
DMS waste rock	220,000
Inert scrap material	40,000
Total waste rock pile	**560,000**

While all tailings will be placed underground as paste backfill, there will be some need to temporarily store tailings during times when there are limited stopes available for backfill. In this situation, tailings will be dry-stacked and placed in the temporary tailings storage pad to the west of the mill. When backfill space becomes available in the mine, these tailings will be placed back into the paste plant to meet blend specifications, prior to being sent underground.

Ore Stockpiles

Currently on site there exist close to 50,000 tonnes of oxidized ore stockpiled in the yard. Further assessment of this stockpile is warranted before any revenue can be allocated towards it since it has been broken and exposed on surface for over 30 years. The 970 development ore drive will be the only primary development in ore throughout the mine. This development will supplement the existing 50,000 tonnes of ore with another 5,000 tonnes of ore that will be used during mill commissioning and start-up. This ore will be stored on temporary storage pads east and west of the 870 portal. The pads will consist of a lined area with a sand-under-gravel surface for liner protection. Pad drainage will report to a sump, and will be routed to the pump box for mine water. These stockpiles will be reduced during the mill start-up period.

After the mill start-up period, a smaller 1,000 tonne underground ore storage site, in combination with an enclosed 1,000 tonne surface ore storage site on surface will be used as ROM stockpiles. In addition to mill storage, these will be drawpoints and orepasses that will be appropriate to provide approximately one half week of mill feed. This storage will sustain the mill during times when mining operations are temporarily reduced or suspended, possibly due to mechanical issues. The permanent surface ore storage site will retain pad design and runoff management from the larger temporary site.

Production Sequence

The mine will operate for 360 days a year on a two shift per day basis. Personnel will work ten hours per day, with two hours between shifts to allow for blasting smoke to clear.

The stoping or production sequence will start with mining within the upper levels of the MQV mineralization on the 930 and 880 levels. Shortly thereafter, mining of the 800 level will begin with subsequent mining of the lower levels in time. The focus will be to mine as much of the upper levels early on to defer development expenses incurred from accessing the lower levels.

16.6.4 Production Schedule

Table 16.3 shows the production schedule for the mine.

Table 16.3 Production Schedule

ITEM	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	TOTAL
Ore Mined, mt	4,510	208,380	502,135	495,442	498,025	499,707	502,662	499,556	488,619	507,991	503,495	511,160	5,221,683
Development waste rock mined, mt	55,598	57,187	110,819	109,737	113,414	80,694	9,864	12,521	31,973	18,101	543	-	600,449
Mine Feed													
% Zn as Sulfide	4.07%	6.90%	6.79%	7.13%	8.44%	8.33%	7.48%	7.16%	6.88%	7.64%	7.81%	9.45%	7.68%
% Pb as Sulfide	2.94%	4.92%	5.15%	6.33%	8.26%	7.32%	7.68%	9.09%	10.79%	10.51%	4.95%	4.55%	7.35%
% Pb as Oxide	2.94%	3.70%	3.79%	3.52%	2.57%	2.23%	1.72%	1.47%	0.84%	0.83%	1.79%	2.12%	2.15%
% Zn as Oxide	2.71%	3.72%	3.81%	3.06%	2.01%	1.97%	1.35%	0.83%	0.26%	0.27%	1.16%	1.30%	1.69%
Ag (g/t)	125.54	169.21	172.59	175.82	164.25	157.91	157.49	159.99	168.26	166.38	82.34	99.95	151.04
% Cu	0.29%	0.38%	0.39%	0.38%	0.34%	0.35%	0.35%	0.34%	0.34%	0.35%	0.08%	0.11%	0.31%

Note: grades relate only to ore mined. Development waste has zero grade.

16.7 Mobile Equipment

The following mobile new / used equipment will be purchased and / or leased at some point over the life of the mine for the underground operation. Most equipment will be required at the start of the operation; however a small number of pieces will be sourced in the 2nd year of operation. No contracts have been entered into with the manufacturers specified in Table 16.4.

Table 16.4 Equipment Required for Mining

Type of equipment	Number Required	Manufacturer / Model
Emulsion loader	2	HandiLoader 1000
Single-boom jumbo	4	Atlas Copco S1D boomer
3.5 – 4t scooptram (LHD)	3	Sandvik Toro151 LH203
7t scooptram (LHD)	1	Sandvik T6 LH307
20t Haul trucks	3	Sandvik TH320
4.4 m3 transmixers (paste)	2	Normet Utimec MF 500 Transmixer
Grader	1	Miller M86
Lube/fuel truck	1	Getman A64
Personnel carrier	1	Getman A64
Pick-up trucks	3	Toyota Tundra
Scissor lift	1	Miller scissor lift
Bolter	2	Atlas Copco Boltec 235
Scaler	1	Brokk 330D
Flatbed	1	

16.8 Personnel

Manpower is scheduled for a rotation of three weeks in and three weeks out during operations. Most positions in operations require a day and night shift, while technical positions typically only require a day shift. Table 16.5 lists the people that will be employed by Canadian Zinc during full operations. This list does not include contractors. A total of 145 people will be employed within the mining technical, production and maintenance departments. Some small redundancy was built into the personnel requirements to account for absenteeism.

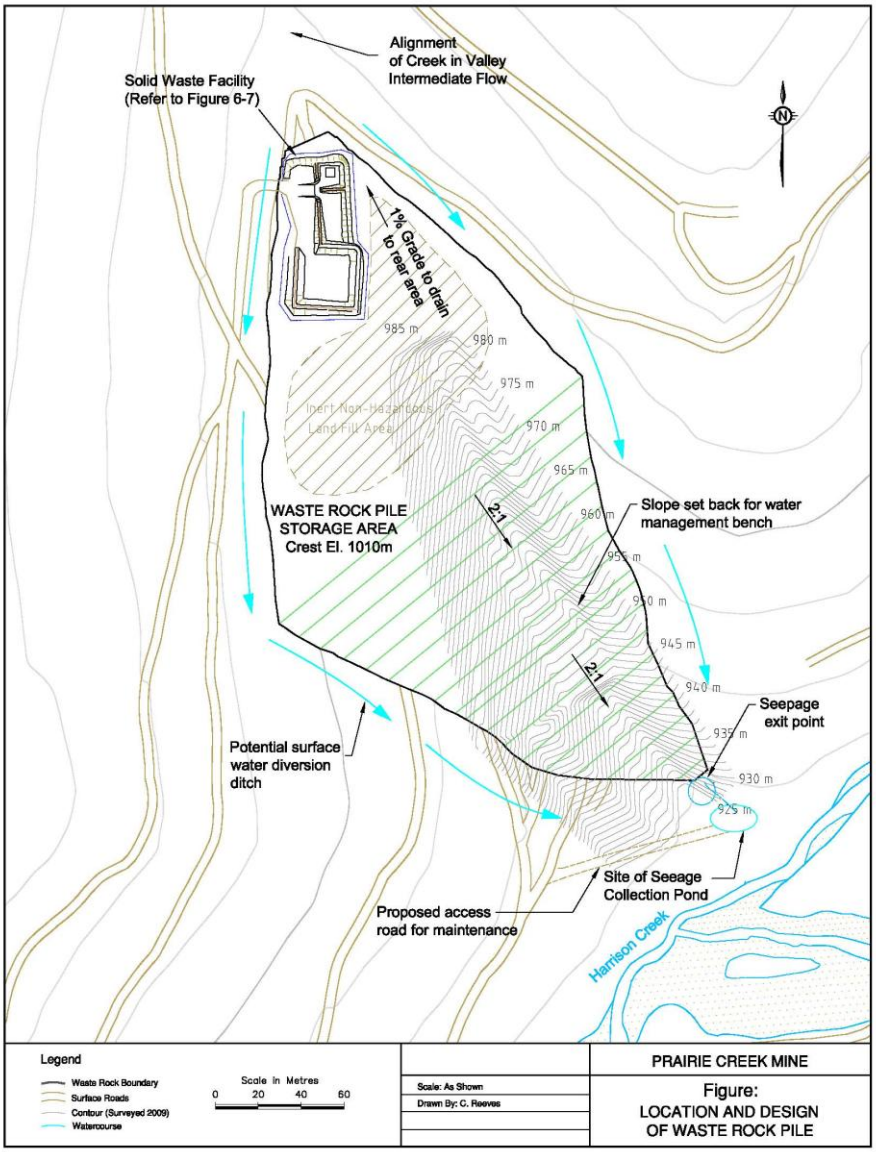
Table 16.5 Required Personnel

Technical	
Position	**Total Employed**
Mine superintendent / Chief mine engineer	2
Long range planner	1
Mine production engineer	2
Chief/Assistant Chief mine geologist	2
Geologist	2
Surveyor	2
Surveyor helper	2
Grade control samplers	2
Total	17
Production	
Mine foreman	**2**
Mine supervisor	6
Jumbo operator	18
Scoop operator	18
Haul truck operator	12
Blaster	8
Blaster helper	8
Bolter / scaler equipment operator	8
Labourer / Service crew	12
Paste truck driver	8
Paste placement and stope set-up	8
Total	108
Maintenance	
Foreman	**2**
Heavy duty mechanic	6
Welder	4
Lube/service	2
Electrician	2
Labourer	4
Total	20

16.9 Waste Rock

Waste rock from underground development operations will be dumped directly west of the 870 portal onto a staging area. From this staging area, waste will be transported to the newly created Waste Rock Storage facility located approximately 800 m upstream of Harrison Creek from the mill concentrator (Figure 16.5). A portion of the dense media separation (DMS) float reject rock will be transported to the waste rock storage facility as well. The entire operation is expected to produce 600,000 tonnes of development waste rock and 1,297,000 tonnes of DMS float waste rock. A significant portion of the DMS waste rock will be used as an aggregate to augment increased strength to the paste. It is estimated that approximately 520,000 m^3 of development and DMS waste will be stored in the waste rock pile at closure.

Figure 16.4 Waste Rock Pile



Although the waste rock is considered Non-Acid Generating (NAG) due to its high content of carbonate material, diversion ditches will be constructed upstream and around the waste rock pile to prevent leaching from surface runoff through the waste rock pile. For any water that does get to the Waste Rock Pile, via rain or groundwater, there will be a collection ditch downstream preventing any leachate from entering the nearby Harrison Creek. All water from the collection ditch will be piped to the Water Storage Pond for treatment prior to discharge.

16.10 Paste Backfill

Paste Description

Paste backfill will be the only material used as backfill within the mine. The existing mill concentrator will be upgraded with a newly installed paste backfill plant that will return 100% of the flotation tailings produced at Prairie Creek back underground as a thick paste backfill. The newly installed DMS plant, at the front end of the mill, will remove a significant amount of waste rock from being processed, resulting in the production of less tailings. This is a high grade base metal ore deposit; consequently a high percentage of the mill feed will become contained metal in concentrates, thereby further reducing the amount of tailings produced. All the aforementioned factors contribute to allowing full disposal of the tailings produced at the Prairie Creek Mine back underground as paste backfill, thus negating the need for a permanent surface tailings facility.

The paste plant will be located northwest and directly adjacent to the mill. 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. Most of the paste will be deposited as one of two types: 10” slump, 1% binder with only tailings or 6” slump, 8% binder, 50% tailings, 50% DMS float waste rock. The binder likely to be used will be Ordinary Portland Cement (OPC), which is very expensive to ship to the mine site. In order to reduce significant freight costs of transport testing will be done on OPC blended with fly ash and OPC blended with ground iron blast furnace slag to determine if there is any benefit from using a different binder.

Within each lift during the mining operation, both types of paste will be deposited. Each lift is planned to be 5 m high. The top 1.5 m of the paste backfill in the lift is considered to provide the running surface. As cement is expensive, the bottom 3.5 m of each lift will be composed of the 10” slump tailings only paste, while the running surface will be composed of the tailings, DMS mixture. DMS float waste is used as an aggregate to construct a surface strong enough to support the various operating equipment running along its surface during production.

A minimum curing time of seven days will be required to ensure sufficient strength capabilities of the running surface. Within the current plan, paste is left for a minimum of 12–15 days for curing. 450 m3 of paste will be poured on a typical day to ensure mining operations move forward and there is not an excess build-up of tailings on surface.

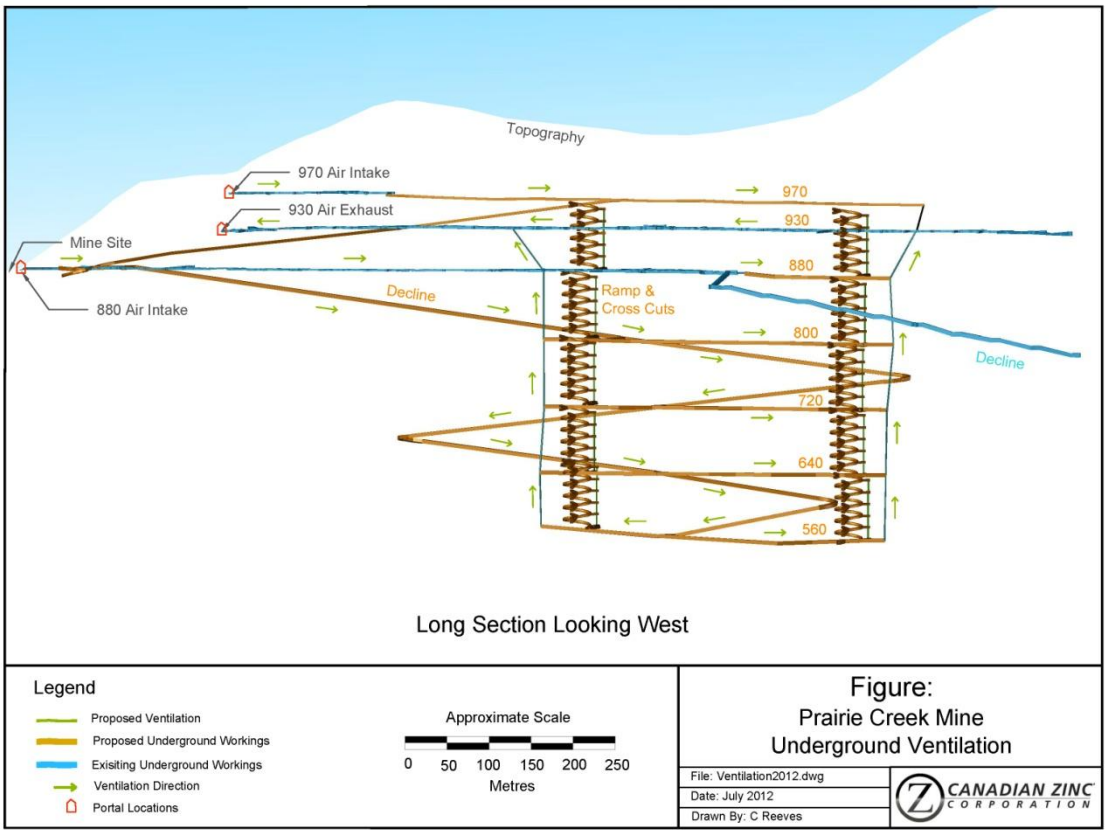
Paste Distribution Centre

Due to the low slump of a large portion of the paste, it cannot be pumped a long distance laterally and up into the mine. Therefore, paste will be transported via transmixers to a newly established 970 level underground distribution centre, where it will be piped downwards to the rest of the mine. The paste distribution centre will be located at close to the highest elevation within the mine. However, due to the density of the paste, gravity on its own will not be sufficient to distribute paste to all of the stopes within the mine. Positive displacement pumps will be used to move paste laterally through the line to its intended receivers.

16.11 Ventilation and Heating

Ventilation is required to remove diesel exhaust and blasting fumes, and to provide fresh and warm air to all areas of the mine where personnel will be working. The ventilation circuit will be under negative pressure with the primary exhaust fans located at the 930 m portal. When the mine is fully developed, 170,000 cfm of fresh air will be drawn through the 880 level and will move air down to the lower levels of the mine (Figure 16.6). The intake air fans will be located at the 870 portal, directing air through the heat source generated by the boiler.

Figure 16.5 Schematic of Underground Ventilation



The Mine air will be heated when necessary during October through April to a minimum temperature of +1˚C. This is to provide a comfortable working temperature and to prevent mine water from freezing within pipes near surface. The diesel fired boiler heater will have a capacity of 130,000 BTU/min and will be located in series with the primary forcing fans at the 870 portals. The heater will not usually be run at capacity since it will only be used to heat the underground as a supplement to the radiated glycol-based heat recovered from the new power generation units that will act as the primary method of underground heating.

There will be a number of ancillary booster fans throughout the mine that will be used to regulate flows to various areas of the mine. There will also be auxiliary fans used to ventilate active production headings.

16.12 Electrical

Power will be fed at 4.16kV through both the 870 and 930 portals. Both of these lines will connect to the mobile power stations located at the base of the active stope access ramps, where power will be transformed to 600V for operating equipment. Two lines provide a redundancy; if one line is damaged during operations, the second line will solely supply power until the first line has been repaired. Power underground is mainly required for ventilation, drilling, pumping, paste facilities and other uses such as lighting, the equipment maintenance facility and the ore dump.

16.13 Explosives

Emulsion-type explosives are proposed to be used as the primary underground explosive. This is in order to reduce and have better control of ammonia concentrations building up in associated mine water which could prove to be problematic to meeting prescribed effluent quality criteria. The use of emulsions does not guarantee the reduced ammonia and underground practices would still have to be closely monitored and supervised in order to reduce this potential problem.

An emulsion plant will be built on site to lessen the amount of explosives that will need to be brought and stored on site. The intent is to have a contractor build and operate the explosive plant for future acquisition by the Mine.

Significant amounts of ammonium nitrate, sodium nitrate, sensitizer and fuel oil will need to be brought to site and stored. These will be stored in a fabric building near the emulsion plant, in accordance with explosives regulations. Boosters will be periodically flown in to site and stored separately from explosives as per regulation. Explosive components will be stored on pads located southeast of the Mine on the adjacent floodplain. Detonators will be stored separately.

Emulsions are effective in wet environments and they have low ammonium nitrate leaching rates. Bulk emulsions planned for Prairie Creek leach at a rate 2,500 times less than that typical of ammonium nitrate-fuel oil.

The emulsion plant will be constructed and operated by a major explosives manufacturer. The plant will be rented to CZN on a lease-to-own contract over five years (subject to an agreement) and CZN would pay a fee for explosives materials and personnel from the explosives manufacturer.

16.14 Mine Services

Mine services/supplies to support mining operations comprise compressed air, communications, water supply, water discharge, the mine emergency warning system and refuge stations.

Compressed Air

Operations will not require excessive amounts of compressed air on a regular basis due to the reliance on electrical energy. The major use of compressed air will be for rock bolting purposes. Approximately 1,200 cfm of compressed air will be available to mine equipment at any given time. Two existing compressors on site will be used and will be powered electrically. Compressors will be located outside of the 880 portal. Compressors will pump air into the mine in 6” pipes. Receivers will be located periodically through the mine to keep up flow and pressure requirements. Pipes will be stepped down to 4” at sublevels and then 2” to feed production headings.

Water Supply and Discharge

Water for the mine will be required for drilling and blasting. Groundwater inflows underground will be collected in sumps where sediments will be allowed to settle. The top of these settling ponds will allow water to be skimmed off and used underground for drilling and blasting. Potable water will be brought into the mine from a fresh water line on the 880 level as required.

After primary settling in local underground sumps, mine water above the 880 level will flow by gravity to the 870 portal, and from there will be pumped to the Water Storage Pond. Water from the lower mine levels will be pumped up the main decline to the 870 portal. From the 870 portal, water will flow by gravity to the water storage pond. From there, it will undergo various treatments prior to discharge.

Underground Communications

The mine will be equipped with leaky feeder communication infrastructure to allow for radio communication underground, as well as to surface. This system will also have capabilities of sending video and data feed from specific areas of interest to surface.

Comments and Recommendations

- The mine geotechnical program as currently planned for the summer of 2012 should be carried out to confirm ground support requirements and stability control during operations.
- More detail needs to be incorporated into the dump pocket design for the run-of-mine ore. This acts as a main artery of the mine and it is crucial that its design is highly functional and capable of supporting the expected throughput.

- A mine dewatering plan should be constructed to ensure safety at the face during operations as there is potential for water pressure build-up during mining.
- Equipment selection should be further reviewed and refined to identify if there is any merit in allowing for variations in the size of the drills and scoops (smaller and larger) or if standardization of the equipment size (as is currently planned) optimizes efficiencies.
- Oxides were built into the mine plan with discrete defined boundaries at various elevations. While these values were based on assay data, they were not interpolated into the block model used for mine planning. Oxides should be interpolated into the model so that scheduling will represent a more accurate picture of the oxide component in mill feed allowing for high oxide pockets to be deferred to later in the mine life as much as possible (due to lower metallurgical recoveries of oxides).
- Deleterious elements such as mercury, antimony and arsenic were assumed constant around all areas of the mine. These elements should be interpolated into the block model. There have been some indications that mercury within the SMS resource is considerably lower than that within the vein. If this were modelled into the production schedule, it could have the effect of lowering the significant mercury penalties associated with the zinc concentrate, thereby improving the Net Smelter Return (NSR) of the zinc concentrate produced from the SMS resource.
- Further continuation of exploration drilling activities toward the north end of the mineral resource in order to upgrade inferred resources into measured and indicated categories that can then be incorporated in the mining reserve.

Opportunities with Respect to Mining

- It has been shown that all of the vein material can be mined via the cut-and-fill mining method. Cut-and-fill is considered to be a labour intensive, low productivity method that is used to increase selectivity and decrease dilution. Within the vein, there exists areas of wider mineralization that are steeply dipping. The characteristics of these areas suggest that some may be amenable to a form of longhole/sublevel stoping mining method. If these areas are capable of supporting this more bulk-type of mining method, it would have the effect of lowering operating costs, increasing mine productivity and could also allow for more tailings to be stored underground as there would be a lower amount of cement required during backfill.
- Currently 20 tonne haul trucks are the proposed trucks for the mine. Within the planned size of development, it may be possible to operate 26 – 30 tonne trucks, albeit in somewhat tighter conditions. This should be further discussed with equipment manufacturers to confirm this potential. If larger trucks can be employed without significantly altering development costs/plans, a trade-off should be looked at between capital and operating costs of the type of truck purchased.
- Mine design on this project is at a fairly detailed level; however dilution is relatively high for such a persistent deposit mined via cut-and-fill. Dilution accounts should be considered conservative and may be able to be lowered in the next stage of design.

17 RECOVERY METHODS

17.1 Process Design Criteria

The original processing plant was designed to produce three sulphide base metal concentrates; lead, zinc and copper. The copper concentrate was expected to contain in the order of 7,800 g/t of silver and was the primary target commodity at that time. This section summarizes the changes to the original design.

Based upon the previous testing described in Section 13 the anticipated metallurgical performance is provided in Table 13.1, which is reproduced below in Table 17.1.

Table 17.1 Anticipated Metallurgical Performance (repeat of Table 13.1)

	Vein Ore	Stratabound	Weighted Average Recoveries
DMS Plant Losses			
Mass	27.0%	13.0%	
Silver	2.8%	1.0%	
Lead total	1.7%	1.0%	
Lead oxide	3.1%	1.0	
Zn Total	2.1%	1.0%	
Concentrate Grade			
Lead sulphide %Pb	74.0%	59.8%	
Lead oxide %Pb	55.6%	55.0%	
Zinc sulphide %Zn	61.4%	53.9%	
Recoveries			
Silver to PbS	64.3%	61.7%	
Silver to PbOx	14.5%	5.0%	
Silver to ZnS	15.9%	28.0%	
Lead sulphide	94.0%	90.0%	
Lead oxide	75.8%	65.0%	
Zinc sulphide	93.1%	90.7%	
Weighted Average Recoveries			
Zinc	%		75.0%
Lead	%		88.0%
Silver	%		92.0%

Recent metallurgical testing and a regional concern about the use of cyanide has caused the process design criteria to be modified to produce a lead oxide concentrate and not produce a copper concentrate. Under these conditions the majority of the silver will now be contained in the lead sulphide concentrate.

To increase the feed grade to the flotation circuit and to reduce the grinding power per tonne of concentrate produced, a dense media separation (DMS) plant has been designed into the process after fine crushing and before the fine ore bin. It is rated at 85 tonnes per

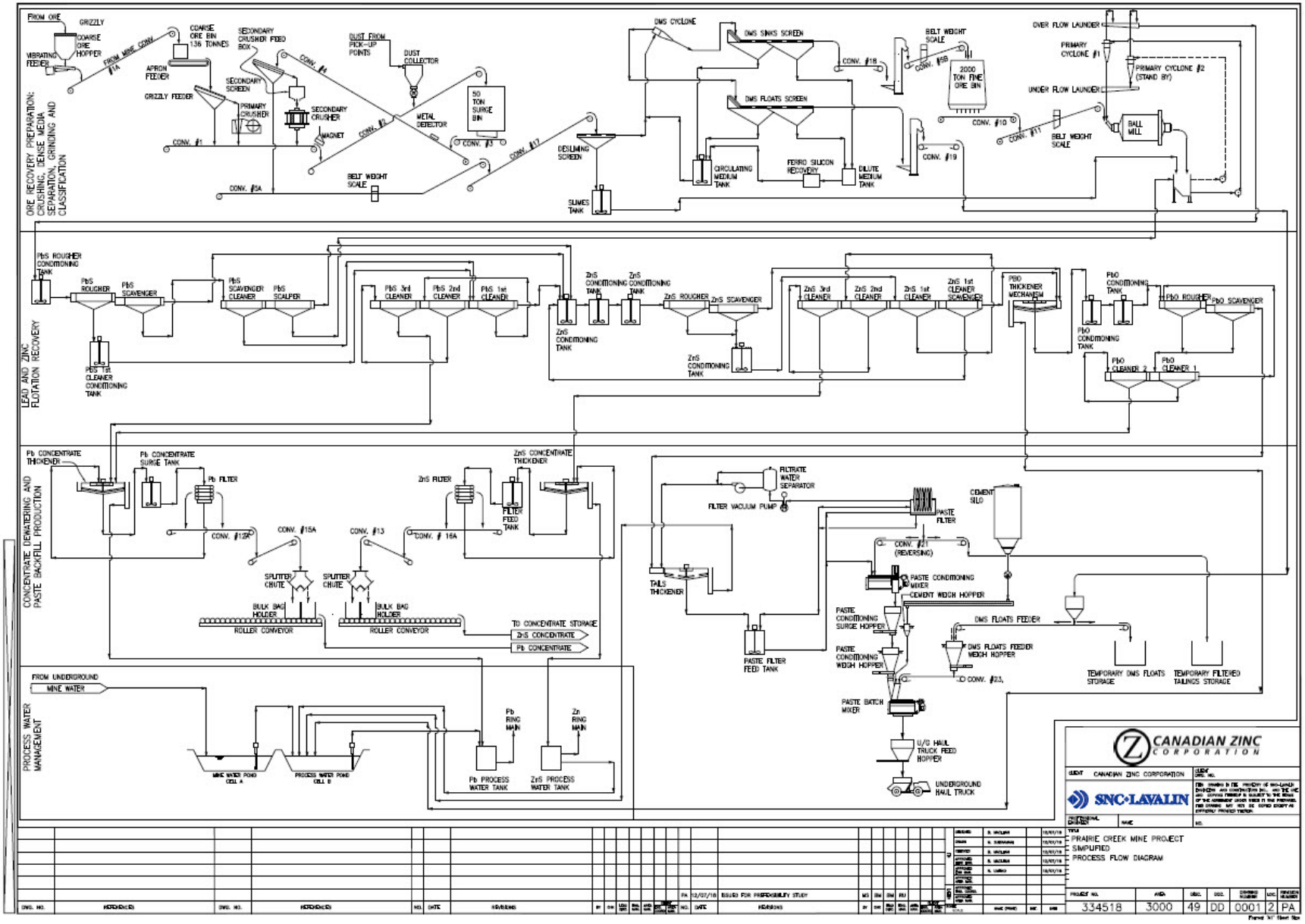
hour and designed to reject about 24% crushing plant feed. DRA Americas Inc., of Peterborough, Ontario, was engaged by CZN to prepare a pre-feasibility level process flowsheet and capital and operating cost estimate for the DMS plant and its recommendations formed part of the Pre-feasibility Study.

The original grinding circuit was designed to treat 910 tonnes per day of ore with a Work Index of 9.8 kWh per tonne. The Work Index of the DMS sink fraction averaged 9.5 kWh per tonne and the designed milling rate has increased a nominal amount to 920 tonnes per day. The original product size was 80% passing 92 microns that has now been lowered slightly to 80% passing 80 microns and the regrind mill has been eliminated from the circuit. At present the corrected ball mill power is 9.4 kWh per tonne compared with the installed primary grinding power of 11.9 kWh per tonne (ref: Cadillac Explorations Ltd. Prairie Creek Definitive Feasibility Study - Kilborn Engineering Ltd., September 29, 1980).

The original flotation circuit initially produced a bulk lead/copper sulphide concentrate followed by a zinc sulphide concentrate. A copper sulphide concentrate was then floated out of the bulk lead/copper concentrate. The current flotation circuit is comprised of a lead sulphide circuit followed by a zinc sulphide circuit and finally a lead oxide circuit. A schematic of the simplified process flow diagram is shown in Figure 17.1. This flow sheet is the recommended flotation scheme of SGS Lakefield in their Phase 4 report dated June 20, 2007. The zinc oxide circuit was eliminated from the circuit because the low grade of the concentrate is presently uneconomic.

The original Backfill plant concept was to deslime the flotation tailings by two-stage cycloning and then to pump the deslimed tailings to an underground storage facility. In order to store 100% of the flotation tailings underground along with a significant portion of the DMS reject, a Cemented Paste Backfill plant has been designed into the process. Golder Paste was engaged by CZN to prepare a pre-feasibility level process flowsheet and capital and operating cost estimate for the Cemented Paste Backfill plant. The design and costing was updated by MPE Engineering Ltd.

Figure 17.1 Simplified Process Flow Diagram



17.2 Crushing

The following subsections describe the major items of the crushing circuit.

17.2.1 Primary Crushing

Run of mine (ROM) ore will be delivered to an area adjacent to the existing ore dump pocket. The ore will be fed over a static grizzly and conveyed to a 40 tonne coarse ore bin. It will be withdrawn by an apron feeder that will discharge onto a grizzly feeder to screen the minus 100 mm material from the feed and discharge the oversize material into the primary jaw crusher. The crusher product and the minus 100 mm material will be combined on a conveyor located below the primary crusher. The primary jaw crusher has a feed opening of 610 x 914 mm and with a closed side setting of 75 mm will produce an 80% minus 100 mm product at a capacity of 190 tonnes per hour.

17.2.2 Screening

The combined grizzly undersize and primary crusher product will be conveyed to a 40 tonne surge bin. The surge bin will feed the secondary vibrating screen. The oversize from the vibrating screen will feed the secondary cone crusher. The undersize from the screen will be conveyed to the DMS plant.

The 1,524 x 4,267 mm double deck-vibrating screen will be fitted with a 25 mm top deck and a 12 mm bottom deck.

17.2.3 Secondary Crushing

Oversize material will pass to the 1,676 mm (5.5 feet) short head secondary cone crusher. The throughput for the secondary crusher is 178 tonnes per hour. The discharge from the secondary crusher will be conveyed to the 40 tonne surge bin which is re-cycled to the secondary vibrating screen. The screen undersize will be conveyed to the DMS plant.

17.3 Dense Media Separation

The crushed ore from the crushing plant will be fed to a feed preparation screen located in the DMS plant. These two plants operate 16 hours per day. The screen will remove the minus 14 mesh (1.2 mm) fines from the DMS feed. The screen oversize will feed the dense media cyclone feed pump box where a media of water and ferrosilicon will provide the slurry to enable a cyclone cut at a specific gravity of 2.8. A single cyclone will operate with a capacity of 80 tonnes per hour. The cyclone overflow (DMS reject) will be fed to a wash screen to recover ferrosilicon media before being conveyed to the temporary coarse rejects storage area. The cyclone underflow (sinks) will also be fed to a wash screen to recover the ferrosilicon media. The DMS coarse reject will contain the lower specific gravity gangue material consisting mainly of dolomite and quartz. The washed sinks will be dewatered with a dewatering screen and conveyed to the fine ore bin. The fine screen undersize will be pumped to an agitated holding tank. The holding tank discharges by a pump to the ball mill cyclone feed pump box at a constant rate over the 24-hour mill operating day.

Recovery of ferrosilicon from the wash screens will be accomplished by magnetic drum separators. The DMS plant will include mixing and storage equipment for ferrosilicon make up to the DMS circuit.

The washed DMS float fraction is transferred by conveyors directly to the Backfill Plant.

17.4 Grinding

The 3,048 mm diameter x 4,267 mm long overflow ball mill is fed from the 1,800 tonne capacity fine ore bin by a conveyor. The discharge from the ball mill will be pumped to the primary cyclones. Cyclone underflow will return to the ball mill for additional grinding and the cyclone overflow at 80% passing 80 microns will feed the conditioning tank at the front end of the lead sulphide flotation circuit.

17.5 Flotation

The ground, upgraded ore is treated by flotation in the existing mechanical flotation cells to sequentially recover lead sulphide, zinc sulphide, and lead oxide concentrates. All of the existing flotation cells will be used and modifications to the piping system will be required. Additional flotation cell will be required to produce the lead oxide concentrate.

17.5.1 Lead Sulphide Flotation

The ground ore is treated in a lead sulphide flotation circuit consisting of lead sulphide rougher flotation. The rougher concentrate is treated in three stages of cleaner flotation to produce a final lead sulphide concentrate. Lead sulphide rougher and first cleaner tailings are combined and form the feed to the zinc sulphide flotation circuit.

17.5.2 Zinc Sulphide Flotation

The zinc sulphide feed is conditioned in three stages and subjected to rougher flotation. The zinc sulphide rougher concentrate is further conditioned and cleaned in three stages to produce a final zinc sulphide concentrate. The first cleaner stage includes a first cleaner scavenger in which the concentrate is recycled to the feed to the zinc rougher conditioning stage. The first cleaner scavenger tailings join the zinc rougher tailings as feed to the lead oxide circuit.

For stratabound ore, which contains little oxide or carbonate minerals, and which has significant pyrite, the zinc sulphide rougher tailings and first cleaner scavenger tailings are combined and discarded as final flotation tailings.

17.5.3 Lead Oxide Flotation

The thickened feed to the lead oxide flotation circuit is conditioned with reagents in two stages and subjected to rougher and scavenger flotation. The concentrate is treated with two stages of cleaning produce the final lead oxide concentrate.

17.5.4 Reagents

The reagents used in the test work conducted to develop the current Prairie Creek flotation circuit flowsheet are shown in Table 17.2

Table 17.2 Flotation Reagent Suite

Lead Sulphide Circuit	
Na_2CO_3 (soda ash)	pH modifier
P82 (mixture of $ZnSO_4$, $Na_2S_2O_3$ and $Na_2S_2O_5$)	zinc depressant
AQ4 (mixture of Na_2SiO_3, Accumer 9000 and $NaPO_4$)	slime dispersant
SIBX (sodium isobutyl xanthate)	collector
DF067 (Dynafloat 067)	frother
MIBC (methyl isobutyl carbinol)	frother
Zinc Sulphide Circuit	
Na_2CO_3 (soda ash)	pH modifier
$CuSO_4$ (copper sulphate)	activator
SIBX (sodium isobutyl xanthate)	collector
3894 (Cytec 3894)	promoter
AQ4 (mixture of Na_2SiO_3, Accumer 9000 and $NaPO_4$)	slime dispersant
Lead Oxide Circuit	
Na_2S (sodium sulphide)	lead sulphidizing agent
DV177 (acrylic acid)	depressant
SIL N (sodium Silicate)	depressant
SIBX (sodium isobutyl xanthate)	collector
DF067 (Dynafloat 067)	frother
AQ4 (mixture of Na_2SiO_3, Accumer 9000 and $NaPO_4$)	slime dispersant

17.6 Thickening and Filtration

Lead Concentrates

The lead concentrates will be thickened in the existing conventional lead concentrate thickener. Both concentrates are much coarser than originally designed (80% passing 80 microns vs. 80% passing 35 microns) which is expected to materially increase the thickener capacity. The underflow from the thickener will feed the lead concentrate stock tank. Thickened concentrate will be fed from the stock tank to either of the two existing Larox pressure filters. Both filters are model PF19 and can and will be expanded to PF25 sizing with the addition of more plates to the frame's capacity. The lead concentrate filter cake will discharge onto a conveyor leading to a concentrate bagging plant.

It is currently planned to market the lead sulphide and lead oxide concentrates as a blended product. If the ongoing concentrate marketing investigation concludes a marked improvement in the overall economics by placing lead sulphide and lead oxide concentrates with different smelters, then an additional small lead oxide concentrate thickener and stock tank will be required.

Zinc Concentrate

The zinc concentrate will be thickened in the existing conventional zinc concentrate thickener. This concentrate is much coarser than originally designed (80% passing 80 microns vs. 80% passing 35 microns) and this is expected to materially increase the thickener capacity. The underflow from the thickener will feed the zinc concentrate stock tank. As with the lead concentrates, the thickened zinc concentrate will be fed from the stock tank to either of the two expanded Larox pressure filters. The zinc concentrate filter cake will also discharge onto a conveyor leading to the concentrate bagging plant.

17.7 Concentrate Bagging

The two filter discharge conveyors discharge into a splitter chute that can fill either of two bulk bags, which are supported on a bag filling frame on a roller conveyor. After a bag is filled the splitter directs feed into the empty bag. The full bag is lifted off the frame by an overhead crane fitted with a scale. The bags are then moved to the storage shed using a front end loader fitted with a special bag lifting hook.

17.8 Concentrate Storage

In light of current global environmental issues relating to contamination of metals along transport corridors of mineral concentrates the Company has decided to place the lead and zinc concentrates into bulk bags. These bags are capable of containing up to 4 tonnes of concentrate material and have been used at the Newmont's Eskay Creek mining facility which has recently closed. SNC-Lavalin and CZN personnel visited the rail loading facilities while it was still in operation at the Kitwanga railhead in northern British Columbia, to investigate and adapt where applicable the system that they developed over the life of the mine.

Figures 17.2 shows a bulk bag with scale and Figure 17.3 shows a bulk bag being loaded.

Figure 17.2 Concentrate Bulk Bag

*Note bag hook filled with scale

Figure 17.3 966 Loader Fitted with Bag Lifting Extension

Concentrate Storage – Mine Site

Concentrates will be stored in a bagged concentrate storage facility. The concentrate bags are capable of being stacked at least 4 high. The facility will be located to the southwest of the Mill and will be sized to store 70,000 tonnes of concentrates.

17.9 Backfill Plant

The final flotation tailings and the DMS coarse rejects streams will be combined to produce a mix for use as backfill in the underground workings. The flotation tailings will be a slurry

stream with a particle size of approximately 80% passing 80 microns while the coarse rejects will be 1 to 12 mm in size. The relative mass splits of each of these streams is variable and dependent on the grade, degree of oxidation, and amount of internal and external dilution included in the mining sequence. On average, twice as much flotation tails by weight will be produced compared to the DMS coarse rejects

The final flotation tailings will be fed to a 7 m diameter thickener. The thickener underflow will be fed to a tailings filter feed tank. The thickened tailings will be filtered in a tailings filter, yielding a filter cake with 15–20% moisture.

A conditioning mixer will mix the filtered flotation tailings with thickened flotation tailings to obtain 80% solids content. The conditioning mixer discharges via an overflow into a batch surge hopper.

Part of the DMS coarse rejects will be transported directly to the backfill mix area by a conveyor that will discharge into a storage bin. The bin has been sized to provide adequate surge capacity in the event of variations in coarse rejects to flotation tailings mass, and for down-time/maintenance schedules. The coarse rejects will be free draining with a moisture content of about 6%. The tails and DMS rejects will be batched together with binder (1.5-3% cement and/or fly ash) in a batch mixer. The batch will be consolidated in the batch mixer while adjusting with water to give the desired consistency. The blended mix will be discharged into a hopper and then to a truck for transport to the underground workings.

In the event that the underground workings are temporarily unavailable for backfill, the filtered tailings DMS coarse rejects will be directed to a 500 tonne capacity temporary storage bins. In the event these become full the material will be moved by truck to open storage areas for future reclaim.

17.10 Energy, Water and Process Materials

New diesel engine power plant will include five 1.5 MW, 4.16 kV units and power distribution switchgear to ensure adequate plant power availability with some standby capacity.

The estimated average electrical running load with the mine and all surface plants operating is slightly over 5 MW. For 12 hours each day when the crushing and DMS plants are not operating the average demand is expected to drop below 4 MW.

The process plant requires approximately 75 m^3/h of process water, all of which is supplied by the underground mine.

The existing 400,000 m^3 water storage pond will be divided into two cells by placing an intermediate divider berm. All mine water, having different water chemistry, will be pumped into Cell A. Mine water will either pumped to the Mill as makeup process water or to the Water Treatment Plant for treatment and discharge to Prairie Creek via the Catchment Pond.

At full mine production of approximately 500,000 tonnes of ore per year the tonnage of the principal process materials, other than the previously described energy and water requirements, are shown in Table 17.3.

Table 17.3 Process Materials

Process Materials	Tonnes per year
Crushing	
Jaw Crusher Liners	15
Cone Crusher Liners	30
Grinding	
Ball Mill Liners	13
Grinding Balls	194
Flotation Reagents	
Flocculant	1
DF067	12
SIBX	46
MIBC	1
Soda Ash	592
P82	53
AQ4	204
Copper sulphate	533
3894	6
DV177	4
SIL N	141
Sodium sulphide	243
Paste Plant	
Binder (Portland Cement)	5000
DMS Plant	
Ferro-silicon	176

18 PROJECT INFRASTRUCTURE

18.1 Camp and Surface Facilities

Much of the existing infrastructure can be used in its current state, with relatively minor refurbishment. The administration and mine ‘dry’ building, maintenance shops, tank farm and sewage treatment plant need only minor upgrades. Site water management structures are also in place, including the catchment pond for all run-off prior to site discharge, and the recently constructed polishing pond for the polishing of treated water. Other infrastructure, the major ones being the power plant and the incinerator, are now obsolete and will be replaced with more efficient equipment.

The kitchen and some of the accommodation trailers have deteriorated beyond repair and will be replaced. These will be demolished and removed to the WRP. Fifty of the trailers are currently useable, and these will continue to be used for the construction period and for overflow purposes during operations. A new two-storey accommodation block is planned. The block will contain a kitchen and recreation facilities, and will be linked to the administration building by an utilidor. The new block will be the same height as the administration building, and will occupy a much smaller footprint than the existing trailers.

More details for each infrastructure component are provided below.

18.1.1 Dormitories and Catering

The existing accommodations comprise modular dormitories. Some units will require replacement, while approximately 50 will be retained for construction and overflow during operations.

A new 110-man two-storey camp has been budgeted to replace the deteriorated components of the existing camp. The new camp will have glycol heating and will be tied into the existing water, sewer and electrical infrastructure.

18.1.2 Air Strip

No physical changes will be required for continued use of the existing airstrip for crew changes and bringing in supplies, however, some instrumentation and beacons may be installed to assist in aircraft approach.

18.1.3 Fuel Consumption and Storage

The existing four tanks in the tank farm have a combined capacity of approximately 6.8 million litres. The annual diesel fuel requirement is estimated to be approximately 11 million litres at the peak mining rate of 500,000 tpy which does not occur until the second operating year. The breakdown between mining equipment, surface, plant equipment, and diesel power generation is given in Table 18.1.

Table 18.1 Annual Diesel Consumption Breakdown

Item	Litres
Mill and Surface Equipment	900,000
Mine Equipment	700,000
Supplemental Air Heating	650,000
Power Plant	8,700,000
Total	10,950,000

Fuel is re-supplied to the site annually between mid-January and the end of March. On this basis fuel storage capacity will need to be increased to 9.5 million litres by increasing the height of the existing tanks by approximately 5 m.

Figure 18.1 The Diesel Tank Farm at Prairie Creek

An external and internal inspection of the fuel tanks by an independent tank engineer, Roosdahl Engineering Enterprises, has indicated that the tanks require a nominal amount of upgrading so that they can be utilized in future operations.

Figure 18.2 Sewage Treatment Plant

18.1.4 Sewage Treatment

The existing Sewage Treatment Plant (STP) is a secondary level extended aeration treatment plant.

Sewage treatment 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 disinfected 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, and any excess will be dried and then taken to the incinerator for combustion.

Sewage will be transported within each building and pumped to the STP from strategically located lift stations through force mains in the utilidor. For any sewage generation in outlying areas, the sewage will be collected in local holding tanks and removed via a sewage collection tanker truck for treatment at the STP.

The treatment of the raw sewage is based on a biological oxygen demand (BOD_5) of 220 to 300 mg/l. The flow rate per capita per day of 400 litres is estimated to have a loading of 220 to 300 mg/l of total suspended solids (TSS). The design parameters for treated effluent quality are BOD_5 : <20 mg/l, and TSS: <20mg/l.

18.1.5 Garbage Incineration

CZN will incinerate food waste and other acceptable combustible wastes including sewage treatment plant sludge. The recommended incinerator is a double-chamber design and is considered best suited for this application.

The incinerator will be located adjacent or close to the kitchen to easily accept food waste. Other combustible waste will be transported by truck. Once combustion of waste is complete, the incinerated ash will be hauled by covered truck for burial in the landfill.

18.2 Electrical System

The electrical power system provides power to the mill, surface infrastructure, underground mine and camp. The power plant switchgear delivers power to the mine underground and to a number of plant unit substations for further distribution to the miscellaneous process loads. It generally covers the following:

- New diesel engine power plant with five (5) diesel engine generators and power distribution switchgear.
- Upgrade of the existing installations; lighting, grounding, cabling, cable trays, CCTV, fire protection and heat tracing.
- Upgrade of the underground installations.
- New dense media separation (DMS) plant.
- New paste plant.
- New water treatment plant.
- Integration of the existing 700 kW 600 V diesel engine generator to serve as a standby unit to feed the accommodation complex.

Site inspections were conducted in 2008 and 2011. The existing electrical equipment will be used to the greatest extent as possible. An estimate of all of the electrical equipment that requires replacement has been prepared which includes a number of items that are most probably still useable but that will require testing to confirm.

18.2.1 Power Supply and Heat Recovery

New diesel engine power plant will include five (5) 1.5 MW, 4.16 kV units and power distribution switchgear to ensure adequate plant power availability with some standby capacity. The engines will likely be delivered fully factory installed and tested and packaged in individual trailers. The trailers may be installed outside the existing plant structures. The existing Cooper Bessemer diesel engines will be removed from the power plant and this area will be used to house the water treatment plant.

Each engine will be furnished with waste heat recovery equipment, drawing the heat from all the heat sources. The heat will be transferred to the plant by a mixture of glycol media. Additional heat boilers will be required to supplement the heat during the winter months and to supply the heat for the ventilation air for the underground.

The engines will be self-starting, battery operated, requiring no black start input. During the cold months, the engines will require some pre-warming of the jacket water.

The existing 2.4 kV switchgear will be replaced. New 4.16 kV switchgear compatible with the new generating plant will be installed. The new switchgear will allow automatic loading and unloading of the diesel engine generators to match the plant load, allowing for some spinning reserve to ensure safe plant motor starting and shut-downs.

The estimated average electrical running load with the mine and all surface plants operating is slightly over 5 MW. For 12 hours each day when the crushing and DMS plants are not operating the average demand is expected to drop below 4 MW. The peak load occurs during the winter periods. One generator set will be available as standby. There is also approximately 1 MW of construction power diesel generators available, operating at 600 V to supplement any emergency power requirements. The fuel consumption is expected to be around 11,000,000 litres / year.

18.2.2 Electric Heat Tracing

Heat tracing will be used for the pipe containing process water taken from the water storage pond. The pipes will be partly buried and partly laid on sleepers to be covered with snow during the winter months. Fire suppression systems will be designed with dry mains and will not need heat tracing. The pipes will have to be purged by air occasionally to remove the moisture from the pipe and hydrants.

Insulated plastic pipes with slots for heat tracing cables are available at site and will be utilized. Plastic pipes need to be heated by using lower temperature rated heat tracing cables. Thus plastic pipes need more cables and controllers and energy for the same heating effect as the carbon steel pipes.

The ambient thermostats will be turned on at +5°C for the outdoor pipes. The buried pipes will carry their switching thermostats on the pipes. The cables will be sized to operate for temperatures of –45°C in the open and –7°C for buried pipes, with a safety margin of 20%.

Each buried pipe will be provided with 2 x 100% rated heat tracing cables. Due to the length of pipes (600 m), series resistance heat tracing cables are preferred over the self regulating types. Connection to the heat tracing cables will be by Teck90 cables at the junction boxes, located along the pipes.

Power supply sources at 208V are from off-site facilities in proximity of the power points from dedicated heat tracing transformers and panels. Power distribution supplying the heat tracing will be at the top of the list of loads to be maintained during plant outages.

18.2.3 Closed Circuit TV

Plant monitoring is provided at the specific locations to observe the flow and handling of ore and materials in the process building and the ore conveying systems.

The monitoring is by the field TV cameras feeding into the video management equipment and video monitors located in the central control room.

18.3 Plant Control

The plant site control system; DCS (Distributed Control System) or PLC (Rockwell Control Logic), will be decided upon following the full review of process requirements. Local control rooms are provided to facilitate operations of the facility process areas. These areas include separate, but integrated control systems for the powerhouse and the process plant and facilities.

18.4 Fire Detection and Suppression Systems

The fire detection and suppression system is partly installed. The hardware already installed will be reused as much as suitable to be able to operate with the new modern fire control panel. The existing main control panel Edwards Model 6500 is located in the office building. This panel will be retained to serve the offices, and will be integrated with the 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 consistent with the equipment presently in use at site, conforming to National Fire Protection Association requirements.

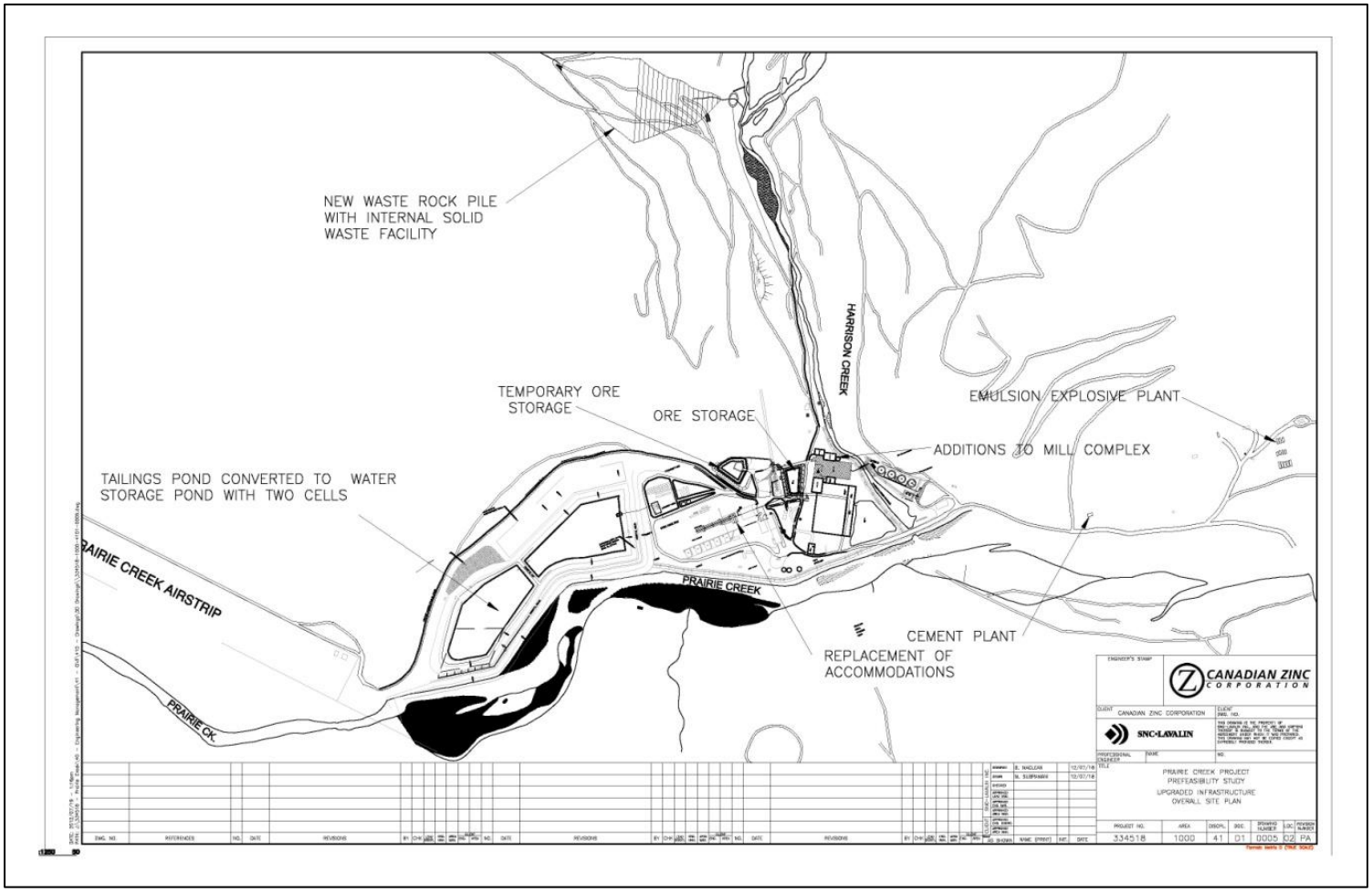
The water for the fire suppression will be taken from the water ring main placed around the plant facilities. The reference drawing is attached. The water main will be laid generally within the buildings and Arctic corridors. In areas between the buildings, the water main will be buried. The water suppression system will operate as a dry system, whereby the mains will be filled with water on the demand of the fire detection system.

The type of fire detection and suppression will depend on the use of the facility, as follows:

- The offices are equipped with fire detection and sprinklers.
- Camp trailers will be purchased with installed detection and suppression devices. These shall be connected to the water main and the fire control panel.
- The assay lab will be equipped with installed detection and suppression devices. These shall 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.
- The process building generally does not have much of the flammable materials. Local manual stations shall be placed at various locations. The suppression will be accomplished by water pipe hose stands located within the building.
- The power plant will include a high fog system over the engines and the pipe hose stands within the building.
- CO_2 hand held extinguishers will be located throughout the buildings.

Figure 18.3 shows the proposed infrastructure upgrades.

Figure 18.3 Proposed Infrastructure Upgrades



18.5 Winter Road

The existing access road from Lindberg Landing to Prairie Creek Mine was constructed in 1980 and operated for two winter seasons carrying over 800 loads into the developing Mine site. The road alignment that is presently be evaluated in detail reroutes sections of the alignment to higher less environmentally sensitive ground.

Figure 18.4 shows the entire haul route from the Mine site to Fort Nelson.

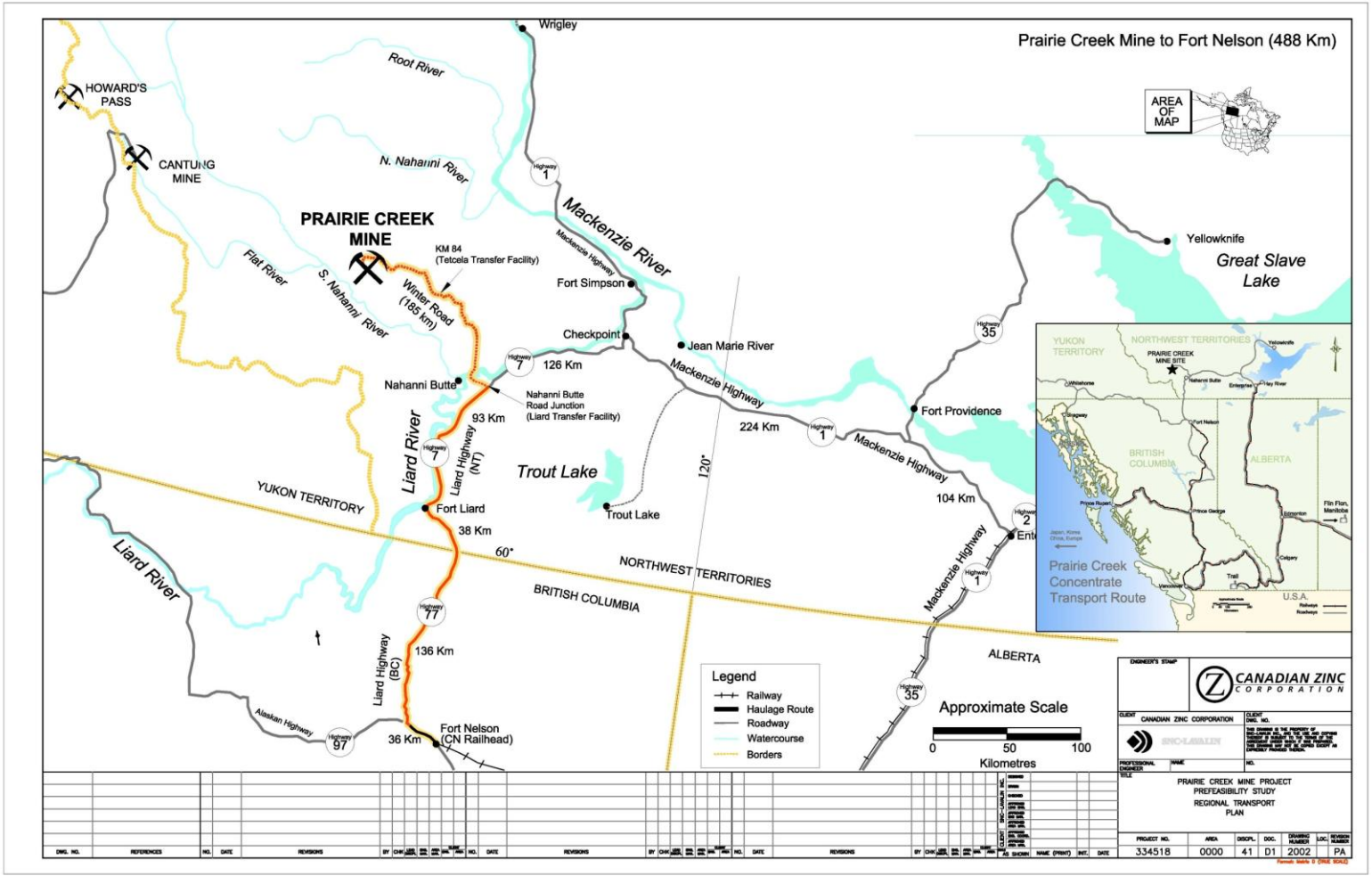
18.5.1 General Operation Plan

The winter road operation plan is designed to transport up to 120,000 tonnes per year of lead and zinc concentrates from the mine site to the rail head at Fort Nelson, and to re-supply the mine with approximately 15,000 tonnes of operating supplies and 12 million litres of diesel fuel.

The plan is based on the following:

- An ice bridge over the Liard River for 40 tonne gross vehicle weight (GVW) traffic is available on average from mid-January to the end of March.
- Winter road construction can commence from the minesite about 1 November of each year and on average will be completed to the Tetcela Transfer Facility (TTF) at Km 84.5 by 1 December, winter road construction will continue toward the ice bridge which will be available by early to mid January.
- The winter road will be operated for minimum 40 tonne GVW traffic from the mine site to the TTF from 1 December until mid-January, at which time the total length of the winter road will on average be available.
- A temporary, unheated storage structure at the TTF will be constructed to store concentrates in bags up to 50,000 tonnes starting on 1 December and ending on approximately 1 March of each year. It is proposed that this would be a prefabricated fabric type structure.
- A staging area referred to as the Liard Transfer Facility (LTF) will be constructed for temporary fuel storage, tractor/trailer transfers, additional concentrate storage facilities, and reloading of the winter road haul trucks with incoming supplies (back haul) at the end of the winter road and on the Liard highway site of the ice bridge.
- A need to minimize the size of the contracted haulage fleet due to competition from others requiring these services at the same time as the winter road operating season.
- A captive fleet of haulage trucks will be operated by CZN from the mine. The trailers will be capable of hauling bagged concentrate on the outbound leg of the haul and diesel fuel and operating supplies on the inbound leg.

Figure 18.4 Overall Site Area Map



Winter road construction from the Liard Highway to the mine site beginning after an ice bridge has been installed reduces the time available for winter road use. The current plan is to construct the winter road from the mine site out to the highway using CZN's own road construction fleet with completion timed to coincide with the opening of the ice bridge at the Liard highway. This plan also takes advantage of colder temperatures earlier at the higher elevations traversed by the western sections of the road close to the mine.

The construction and operations sequence is as follows:

1. The winter road is completed from the mine site to the TTF by CZN by 1 December.
2. Concentrate haulage to the TTF commences on 1 December. The size of the operating fleet is 21 trucks when concentrate output reaches the 120,000 tonnes/year maximum. Each truck transports 30 tonnes of bagged concentrate.
3. Between 1 December and 15 January, the section of the winter road between the TTF and the ice bridge is constructed.
4. Before the ice bridge opens on 15 January:
    a. Up to 70,000 tonnes of concentrate is in storage at the mine site.
    b. Up to 50,000 tonnes of concentrate is in storage at the TTF.
    c. A contractor haulage fleet of 21 trucks is marshaled at the staging area south of the river along with the first tranche of operating supplies and fuel.
5. When the ice bridge opens:
    a. The contractor fleet initially moves all of the concentrate from the TTF to the Liard Transfer Facility (LTF) staging area where the trailers are doubled up and hauled as B-Trains on the existing highway to Fort Nelson.
    b. After all of the concentrate has been removed from the TTF, the contractor fleet commences haulage from the mine site to the LTF area.
    c. At the same time as the above, the CZN haulage fleet is hauling concentrate from the mine to the LTF area, and returning with 15000 tonnes of operating supplies and fuel.

The selection of operating periods for the ice bridge and the different sections of the road are based on historical data of ice bridge operations at Fort Simpson and Nahanni Butte, winter road operating experience in the north, and local knowledge regarding both.

18.5.2 Tetcela Transfer Facility (TTF)

The proposed location of the TTF is at Km 84.4, adjacent to the existing winter road alignment. A geotechnical evaluation will be required to precisely locate the final location.

The facility will consist of one or more prefabricated framed structures with stressed membrane (fabric) covers.

The structures will not be heated, whereby maintaining the frozen state of the concentrate bags. The structure(s) will have the capacity to store 50,000 tonnes of concentrates.

Site infrastructure will include a mobile camp with potable water and sewage treatment supplied by the facilities at the mine site. Mobile, diesel powered lighting plants will provide both outdoor and storage shed lighting. A small fuel tank and the generator will be located within a bermed and lined area. Communications will be supported by satellite phone to monitor road traffic and keep in contact with the Mine site and elsewhere.

At the end of each winter operating season, the doors of the structures will be closed to prevent the ingress of rainfall and entry of wildlife. The mobile camp will be relocated to the mine site.

A front-end loader fitted with a bag-loading boom with an operator and helper will load or unload at the estimated two trucks per hour transfer rate.

18.5.2 Liard Transfer Facility (LTF)

A staging area is required at or near the junction of the Winter Road and the Liard Highway to temporarily accumulate and transfer operating supplies, fuel and concentrate to and from trucks heading in either direction. Furthermore there is a concern that CN may experience railcar delays at Fort Nelson so this staging area may be required for up to 100,000 tonnes of bagged concentrate storage.

The LTF will consist of the following components:

- A 400,000-litre, diked temporary fuel storage tank which would be capable of storing 1.5 days of fuel transfer capacity.
- A fuel transfer module capable of simultaneously unloading a fuel B-train into the temporary fuel storage tank, filling a 20,000-litre fuel transfer tank and re-fuelling the associated tractor.
- An operating supplies lay down area for incoming operating supplies.
- A 100,000 tonne bagged storage shed facility.
- An operations camp with the associated infrastructure either at the facility or in the vicinity.

Facility operating personnel during the peak season are expected to be as follows:

Table 18.2 LTF Operating Personnel

Personnel	No.
Foremen	1
Operators	4
Labourers	2
Total	7

18.5.3 Fort Nelson Load-Out Facility

The Fort Nelson Load-Out Facility (FNLO) is required to offload and store the incoming bagged concentrate from the mine and the LTF, and to load railcars. The design is based upon the Kitwanga load out used by the Eskay Creek mine over the 14 years that this mine

operated. The storage capacity of the FNLO has not been established and is dependent on rail car availability from CN and concentrate delivery rates to the smelters. The design includes four 250 m long sidings, which are sufficiently long to store 30 empty, and 30 full rail cars. The rail car scale is located inside the building. The bags of concentrate are unloaded and loaded using a Cat 966 front-end loader fitted with a bag-loading boom. This size of loader is also capable of moving loaded railcars down the siding.

18.5.4 Equipment Fleet – Winter Road

The operating fleet as mentioned above will consist of 11 trucks in the low hauling years and increased to 21 trucks when the concentrate produced is at a maximum. Along with the main haul feet, one additional truck will be on standby and another truck in maintenance. This will prevent delays and ensure the concentrate is hauled within the time frame of the winter road.

18.6 Water Treatment Plant

Excess water in the water storage pond will be treated in a water treatment plant (WTP). The WTP is comprised of two separated processes. These are:

- Mine water treatment
- Mill process water treatment

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

- Mine water treated with hydrated lime to a pH>9 was sufficient to meet the effluent quality.
- Mill process water required 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.

Lime Treatment

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 flocculant addition and suspended solid removal. Space has been reserved 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.

Sulphide / Ferric Treatment

The Mill process water does not meet the Metal Mining Effluent Regulations requirements with lime treatment alone so it requires a different treatment strategy. The process water will first be sent to an inline mixer where the addition of 1N sulphuric acid will reduce the pH to 4.5. Sodium sulphide will be added at this time and allowed to react for 30 minutes. Following the sodium sulphide addition, hydrated lime is used to increase the pH to 9.0 in another incline mixer. Ferric sulphate is then added to precipitate out Zn and As, and reacted for 60 minutes. To facilitate metal reduction, air is added in the last tank. The

treated water will then be sent to the same clarifier as the lime treatment flow where flocculent is added to settle the precipitates.

18.7 Mobile Equipment

CZN has an extensive fleet of mobile equipment onsite a portion of which would be capable of supporting the onsite requirements. The mobile equipment fleet required for the operation and maintenance of all the surface facilities areas (including the roadways, airstrip) is listed in Table 18.4.

Table 18.4 Mobile Equipment

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

19 MARKET STUDIES AND CONTRACTS

19.1 Commodity Markets

19.1.1 Metal Prices

A number of sources were reviewed to determine a market consensus for the price of lead, zinc and silver including Thompson Reuters and Bloomberg surveys and some private forecasts. Prices used in the financial analysis are shown in Table 19.1.

Table 19.1 Prices of Zinc, Lead and Silver used in Financial Analysis

Metal	2014	2015	Long Term
Zinc ($/lb)	$1.20	$1.20	$1.00
Lead ($/lb)	$1.20	$1.20	$1.00
Silver ($/oz)	$28.00	$28.00	$26.00

C$:US$ exchange rate: C$1:US$
Figures are estimated using Q2 2012 dollars

19.1.2 Price of Zinc

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. Over the period 2014 – 2015 the global economy is forecast to stabilise and gradually return to normal and as a consequence zinc consumption is forecast to increase. Galvanizing of steel continues to be the number one consumer of zinc. At the same time a number of existing large mines are scheduled to close due to depletion of reserves. Significant closings are the recent closure of the Lennard Shelf mine and the expected 2013, 2014 closings of Brunswick, Lisheen and Perseverance mines. The latter three mines will lead to an annual loss of 515,000 tonnes of contained zinc. 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 deficits, with higher prices anticipated before the market returns to near balance with longer term prices higher than levels experienced in 2011 and 2012 which resulted from the global economic recession overshadowed by the euro zone crisis.

The current zinc price of 85 cents per pound is considered by most in the industry to be unsustainably low because the average direct cost to produce zinc metal is high and approaching this value. The low price results mainly from depressed demand in Europe because of the recession there, which has consequentially led to the building of London Metal Exchange (LME) stocks to 940,000 tonnes. As a result, some zinc mines are closing. With an annual growth in demand expected to average 2–4% over the next several years, higher zinc prices are expected.

19.1.3 Price of Lead

The demand for lead metal continues to be moderately strong despite the recession in Europe. The main driving force behind demand is from China, which increased demand in 2011 by 420 tonnes, or 9.9%. Chinese demand is mainly for batteries, including small lead-acid batteries designed for providing power for pedal bikes. These pedal bicycles have become a recent and popular fashion and are becoming popular in India as well.

For a number of years China has been the world's largest producer of lead, and until recently, was also the largest exporter. However, increasing domestic demand for lead in batteries has caused exporting to decrease and an early resumption is not expected.

These trends should help the lead price to move higher. The international Lead and Zinc Study group estimate that there was a surplus of 156 tonnes or 1.5% in 2011, but a deficit seems likely as time proceeds.

19.1.4 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 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. For these reasons demand has risen, despite the substantial losses of its former major markets.

Another recent and major factor in the silver price through the past two years has been speculation. During 2010 and 2011, hoarding appears to have accounted for more than 10% of total demand. Trends in silver often follow trends in gold, which has been the target of heavy speculation related to inflation and the European central bank problems.

19.2 Smelting Terms

19.2.1 Zinc Smelting Terms

For the past 50 years there has been an understanding between the sellers and buyers of zinc concentrate that the smelters should receive their treatment charge per dry metric tonne (dmt) at 38% of the value of the payable zinc generally contained in a dmt. For the year 2012, because of the growing shortage of zinc concentrates the tradition has been broken and treatment charges have decreased significantly for contracts and even more so for spot prices. The general expectation is that a return to normal conditions should be expected but the break is so severe that for the next several years the treatment charge may remain below the 38% guideline.

At this time, there is some downward pressure on treatment charge as the new smelters in China and India struggle to buy tonnages of concentrate that they need. For this reason the treatment charge may shrink below the long-term average of 38% and then recover back to this level in the near future.

CZN should receive payment for 85% of the contained zinc within the concentrate at LME prices. It is estimated that after deducting 3.5 oz per dmt of silver, CZN will be paid 70% of the remainder silver at LME prices.

Mercury content within the concentrate is high, resulting in estimated significant penalty charges at the smelter. Mercury penalty deductions have been applied in the cash flow sheet based on an escalating scale for each 10 ppm that the mercury content exceeds 100 ppm deduct $0.30/dmt up to 500 ppm. For any excess over 500 ppm deduct $0.50 per 10 ppm excess.

19.2.2 Lead Smelting Terms

The terms for lead concentrate have not gone through the dramatic reduction for year 2012 that has been seen for zinc concentrate. A modest escalator on the treatment charge is now imposed at high lead prices and the penalty schedule for undesirable impurities has been stiffened. Refining charges for silver have increased over the past years.

CZN should receive payment for 95% of the contained lead within the concentrate at LME prices for refined lead, with a minimum deduction of three units. CZN should receive payment for 95% of the silver content at LME prices, after deducting 50 g per dmt of silver, less a refining charge of $1/oz.

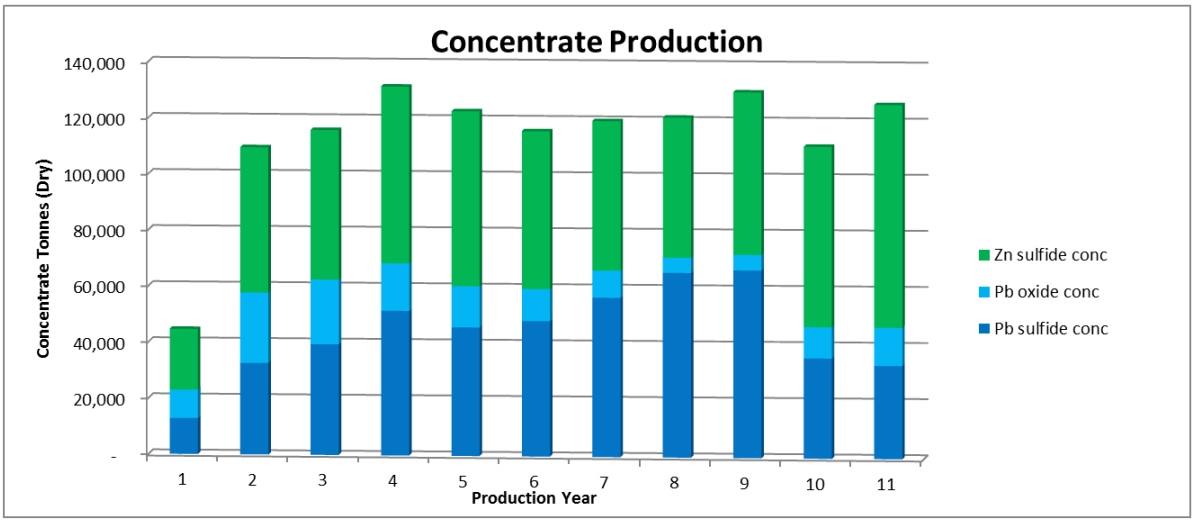
From a sum of these payments deduct a treatment charge of $120 per dmt. This treatment charge is subject to a $0.05 escalation for each $1.00 by which the LME price exceeds $2400 U.S. per tonne. There is expected to be no downscale for prices below $2400 U.S. per tonne.

Penalties for arsenic, antimony, bismuth and mercury have also been applied and deducted from the lead concentrate revenue.

19.2.3 Marketing of Prairie Creek Concentrates

The projected concentrate production from an operating Prairie Creek Mine is shown in Figure 19.1.

Figure 19.1 Projected Concentrate Production from an Operating Prairie Creek Mine



The Prairie Creek mine contains high grades of lead, zinc and silver and in turn produce high grade concentrates of lead and zinc. Table 19.2 and 19.3 shows the results of a locked cycle test for the MQV and SMS mineralization.

The concentrates produced from the Prairie Creek Deposit are known to be of high grade but intermediate quality, containing concentrations of penalty elements such as mercury,

antimony and arsenic. Appropriate smelter penalties have been applied to the Prairie Creek concentrates in the cash flow model generated from the Preliminary Feasibility Study.

The presence of the significant amounts of deleterious elements is expected to present some challenges in marketing these concentrates but the Company has strong indications from outside parties of their interest in purchasing these concentrate products. Now that the preliminary economics are established and the permitting is well advanced the Company can enter into serious negotiations with interested parties in securing some form of off-take agreement for the Prairie Creek concentrates.

Table 19.2 Concentrate Specifications from Bulk Underground Composite for MQV

Element	Lead Sulphide	Lead Oxide	Zinc Sulphide
Pb %	71.5	56.5	4.02
Zn %	3.93	6.09	60.1
Cu %	1.96	0.42	0.31
Fe %	0.14	1.06	0.36
Co %	<0.02	<0.02	<0.02
As %	0.35	0.27	0.09
Sb %	1.09	0.67	0.12
Sn %	<0.002	<0.002	<0.002
S %	12.9	1.27	30.0
C (t) %	0.17	4.09	0.43
Ge g/t	<4	<4	6
Se g/t	10	23	10
F %	<0.01	<0.01	<0.01
Ti %	0.004	0.009	0.004
Ca %	0.04	0.43	0.11
Mg %	0.024	0.21	0.061
Mn %	0.002	0.007	0.007
Al2O3 %	0.08	0.24	0.1
SiO2 %	0.48	6.07	1.36
Bi %	<0.002	<0.002	<0.002
Cd %	0.034	0.047	0.36
Hg g/t	562	936	2730
Au g/t	0.03	0.05	0.08
Ag g/t	1034	438	190
Cl g/t	90	52	57
In %	<0.002	<0.002	<0.002
Ga %	<0.004	<0.004	<0.004

Source: SGS Phase 4 report, June 2007

Table 19.3 Concentrate Specifications Generated from SMS Material from Drillcore Composite

Element	Zinc	Lead	Source
Lead	0.8%	59.8%	SGS, 2005
Zinc	53.9%	5.4%	SGS, 2005
Copper	0.14%	0.034%	G&T, 1994
Iron	4.2%	15.5%	G&T, 1994
Cobalt	<0.001%	<0.001%	G&T, 1994
Arsenic	<0.01%	0.03%	G&T, 1994
Antimony	0.01%	0.054%	G&T, 1994
Manganese	0.01%	<0.01%	G&T, 1994
Cadmium	0.19%	0.01%	G&T, 1994
Mercury (ppm)	580	40	G&T, 1994
Gold (oz/ton)	0.006	0.005	G&T, 1994
Silver (oz/ton)	5.8	12.3	G&T, 1994
Nickel	0.002%	0.001%	G&T, 1994

Specifications for the zinc and lead concentrates produced from the SMS resource are based upon work done by SGS in 2005 and G&T Metallurgical in 1994. The main economic minerals, lead and zinc, were tested in the Phase 1 and 2 report from SGS in April 2005. All other elements tested are from assays of concentrate produced by G&T Metallurgical in 1994. Due to the age of the G&T work, these values were not used in the Net Smelter Return (NSR) calculations of the financial model. The NSR calculations use the conservative assumption that SMS concentrate will have the same concentrations of deleterious elements as that of the MQV and was penalized accordingly.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

The text below was written by Richard Pope of Dillon Consulting Limited (Dillon) after reviewing the PFS information and having discussions with CZN. To the extent permitted under NI 43-101, the Qualified Persons disclaim responsibility for this section of the report.

20.1 Project Overview

The Property contains a base metal deposit containing zinc, lead and silver. As currently planned, the proposed development involves:

- Construction of an underground lead zinc mine producing up to a maximum of 1,200 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 additional water storage pond.
- Re-clearing of the existing winter road from the mine site to the Liard Highway and re-aligning portions of the winter road route.

The Project is located in the southern Mackenzie Mountains in the south-west corner of the Northwest Territories within the traditional territories of the Nah?a Dehe Dene Band (NDDB) and the Liidlii Kue First Nation (LKFN). The nearest community is Nahanni Butte located approximately 90 kilometres to the southeast of the Project site. There is no permanent road access to the Project Site other than the existing winter road. Other communities within 200 kilometres of the site include Fort Simpson, Fort Liard, and Wrigley.

20.2 Overview of Existing Information

A range of studies, surveys and other existing information were reviewed as part of this assessment. Available information evaluated included: environmental baseline studies; regulatory reports/documents; socio-economic data; and summaries of existing agreements with communities and stakeholders, see Table 20.1. Much of this information has been compiled by CZN or has been collected as part of baseline and environmental assessment activities. Some of the baseline information collected at the site dates back to the 1970's.

Table 20.1 Summary of Documents and Information Sources Reviewed

Document Title	Author	Year
Joint Press Release - Canadian Zinc and Nahanni Butte Dene Band Sign Impact and Benefit Agreement for the Prairie Creek Project	Canadian Zinc Corporation and Nahanni Butte Dene Band	2011
Joint Press Release - Canadian Zinc and Liidlii Kue First Nation Sign Impact and Benefits Agreement for the Prairie Creek Project	Canadian Zinc Corporation and Liidlii Kue Dene Band	2011
Press Release - Canadian Zinc and Government of the Northwest Territories Sign Socio Economic Agreement for the Prairie Creek Project	Canadian Zinc Corporation	2011
Prairie Creek Mine Project – Socio Economic Agreement	Canadian Zinc Corporation	2011
Letter – Environment Canada's Final Submission for the Prairie Creek Mine Project & Comments on the Recent Possible Project Modifications (September 13, 2011)	Environment Canada	2011
Prairie Creek Environmental Summary	D. Harpley (Canadian Zinc Corporation)	2012
Aboriginal Affairs and Northern Development Canada (AANDC) Closing Arguments – Canadian Zinc Corporation Proposed Prairie Creek Mine – EA0809-002 (September 13, 2011)	AANDC	2011
Canadian Zinc Corporation Regulatory Applications for Prairie Creek Operations	Canadian Zinc Corporation	2008
Press Release – Minister Confirms Environmental Impact Review Not Required for Prairie Creek Mine (June 14, 2012)	Canadian Zinc Corporation	2012
Media Release - The Review Board approves Canadian Zinc Corp proposed Prairie Creek Mine (December 8, 2011)	Mackenzie Valley Review Board	2011
Joint Press Release – Canadian Zinc and Parks Canada Sign Renewed Memorandum of Understanding (March 5, 2012)	Canadian Zinc Corporation and Parks Canada	2012
Canadian Zinc Website (accessed June 2012) – various pages related to the Prairie Creek Project (History, Community, Permitting)	Canadian Zinc Corporation	2012
Technical Report on the Prairie Creek Mine, Northwest Territories Canada	Stone, D.M.R. and S.J. Godden (Minefill Services Inc)	2007
Press Release – Additional Permits Approved for Prairie Creek Mine (May 15, 2012)	Canadian Zinc Corporation	2012
Report of Environmental Assessment and Reasons for Decision EA0809-002: Canadian Zinc Corporation Prairie Creek Mine (December 2011)	Mackenzie Valley Review Board	2011
Prairie Creek Mine Developer's Assessment Report. Environmental Assessment of Prairie Creek Mine EA 0809-002 (March 2010)	Canadian Zinc Corporation	2010
Terms of Reference for the Environmental Assessment of Canadian Zinc Corporation's Prairie Creek Mine - EA 0809-002 (June 2009)	Mackenzie Valley Review Board	2009
Development of Site-Specific Water Quality Guidelines for Prairie Creek, NWT (March 2010)	Dubé, M. and A. Harwood Saskatchewan Research Council	2010
Prairie Creek Mine Consolidated Description Report (February 2012)	Canadian Zinc Corporation	2012
Directive and Work Plan – Prairie Creek Mine (May 11, 2012)	Mackenzie Valley Land and Water Board	2012

20.2.1 Environmental Setting and Potential Environmental Concerns

The following sections provide an overview of environmental aspects related to the Prairie Creek Project.

20.2.1.1 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 Project 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.
- Vein and stratabound 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.
- 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).
- 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.

20.2.1.2 Water Quality

The Project is located in an environmentally sensitive watershed of the South Nahanni River. As a result, particular attention has been paid by the Company to 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 as part of CZN's Surveillance Network Program (SNP).

Studies referenced in the Project Developer's Assessment Report (DAR) 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. Based on the findings of this study 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 abnormally low flow periods (*e.g.*, winter months). As a result, CZN proposed to monitor flows in the creek, and if flows are found to be lower than normal, discharge will be temporarily reduced (*i.e.*, water would be held in the onsite pond).

Following mine closure, it is expected that there will be no drainage from mine portals as the mine and access tunnels will be completely filled. 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.

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 potentially occurred before 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 cadmium and mercury are not stable in the natural environment and will be attenuated by various natural reactions. As such, CZN concluded that it is likely that no additional impacts on water quality will occur after mine closure when compared to pre-mine conditions.

20.2.1.3 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 mitigative 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.

20.2.1.4 Aquatic Environment

The Project is located on the eastern side of, and adjacent to, Prairie Creek, about 43 kilometres upstream from the creek's confluence with the South Nahanni River.

Aquatic Flora and Fauna

Bull trout and mountain whitefish are commonly found in Prairie Creek near the mine; however, no evidence of spawning has been found downstream of the Project site. Based on the water quality predictions (including toxicity testing), it is expected that mine operations will not impact the aquatic environment.

20.2.1.5 Protected Areas

The Project is located close to (but outside of) the Nahanni National Park Reserve. In 2009, the Nahanni National Park Reserve (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.

20.2.1.6 Cumulative Effects

Given the remote location of the Project, it is expected that there will be very little additional activity in the future which could contribute to cumulative effects. If the Mackenzie Gas Pipeline construction occurs during the life of the Project, there is the potential for a significant regional disruption, but it is unlikely to significantly affect the mine because the pipeline will require short-term skilled labour. 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. Project development includes detailed plans and programs that are expected to prevent or mitigate potential impacts. Key items of proposed environmental management plans 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 in a paste backfill mix. Approximately half of the DMS reject rock from the mill will also be placed underground in the mix. The remaining DMS rock together with waste rock from mine development will be placed in an engineered Waste Rock Pile (WRP). This approach has several advantages: 1) following mine closure there will be no mine waste on the Prairie Creek floodplain; and 2) the underground workings will be backfilled.

Seepage from the WRP will be collected at the toe in a lined seepage collection pond. The pond will be connected to the site water management system, either by pipeline to the mill or 870 Portal, or by borehole to the underground mine workings.

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 the 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

Two main sources of water to be managed during mine operations are:

1) Drainage from the Mine
2) Process water from the Mill

Both water sources will contain metals in varying amounts; although, the process water is expected to contain a greater variety of metals plus residues from flotation chemicals.

A large pond was originally built on site with dykes and a clay lining 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 will also be placed in the process effluent cell, to be reclaimed at a later date. Temporary storage is required as the underground stopes will not be available for paste backfill until approximately 5 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. Most of the 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).

The treatment and release rates of process effluent will vary depending on existing flows in Prairie Creek in order to minimize receiving water metal concentrations. A detailed water balance has been prepared for the Project site to help guide how water will be stored, managed and treated seasonally, and to predict the concentrations of metals in receiving waters.

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. The potential for spills or leaks along the access road will be minimized by controlling road use and using industry-standard containers for transport and storage. Winter conditions will assist in the containment of any spills until a response team can complete a clean-up. The bags of concentrate being transported will be frozen, but road bed samples will be collected along the route to make sure material is not being lost.

20.4 Permitting

20.4.1 Overview of the Regulatory Process

As the Project Site is located within the Mackenzie Valley, all permitting activity relating to land and water use at the site are subject to the *Mackenzie Valley Resource Management Act* (the 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 Licenses for projects outside settled land claim areas in the Mackenzie Valley.

Applications for a LUP or a Water License 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 Environmental Assessment (EA) or can proceed directly to regulatory review for the issuance of a LUP and/or Water License.

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.

20.4.2 Permits and Licenses

A summary of key events and milestones shown in Table 20.2 includes those for site permitting and licensing activities.

Table 20.2 Summary of Project Events and Permitting Milestones

Event / Milestone	Date	Comments
Project placed on care and maintenance	1982	The Prairie Creek Mine was largely constructed and permitted with an existing 1,000 tonne per day mill and related infrastructure
Original Water License, Land Use Permit and Surface Leases	1982	Water License (Class A) N3L3-0932 (DIAND); Land Use Permit N80F249 (for the winter road); Surface Leases (for the Mine Site and airstrip); the Water License and Land Use Permit have subsequently lapsed
Property acquisition	1993	San Andreas Resources (renamed Canadian Zinc in 1999) acquires 100% interest of Project
Exploration activities	1993 - 2007	Ongoing resource delineation including major drilling programs
Prairie Creek Development and Cooperation Agreement	1996	Agreement between CZN and NDDB
Rehabilitation of underground workings	1997	-
Scoping Study completed	January 2001	CZN completes Scoping Study to outline and guide re-development of mine.
Application for exploration and access permits	2001	CZN applied for two surface exploration drilling permits, an underground exploration permit and a permit for use of part of the road from the property
Surface exploration drilling permits issued	2001	Issued following completion of EA
Land Use Permit and Water License issued	September 2003	Underground exploration and pilot plant (was never commissioned); Water License MV2001L2-003 and LUP MV2001C0023
Development of underground decline	2006/2007	Part of ongoing exploration activities noted previously
Land Use Permit issued	11 May 2006	LUP MV2004C0030 allows for surface exploration and diamond drilling activities
Land Use Permit issued	11 April 2007	LUP MV2003F0028 allows CZN to reopen the access road from the Liard Highway at Lindberg Landing to the Mine
43-101 Technical Report Prepared	October 2007	MineFill Services Inc.
Project Open Houses	November 2007	Before CZN submitted applications for operating permits, community open house meetings were held in various Dehcho communities
Applications to MVLWB	May and June of 2008	Applications triggered a preliminary screening of the Project: -MV2008L2-0002: Type A Water License, Prairie Creek Mine -MV2008D0014: Type A Land Use Permit, Prairie Creek Mine -MV2008T0012: Type A Land Use Permit, Liard Transfer Facility -MV2008T0013: Type A Land Use Permit, Tetcela Transfer Facility

Event / Milestone	Date	Comments
Referral to MVRB	11 August 2008	Proposed applications noted above referred to MVRB for Environmental Assessment
Memorandum of Understanding	21 October 2008	MOU between CZN and LKFN; Declaration of intention to work together
Memorandum of Understanding	4 November 2008	MOU between CZN and NDDB; Declaration of intention to work together
Terms of Reference (TOR) issued for EA and Developers Assessment Report (DAR)	June 2009	MVRB issues TOR for the EA and production of DAR. Use of the access road was scoped into the EA, but the MVRB indicated it would consider changes to the existing alignment should they be proposed
Expansion of the Nahanni National Park Reserve (NNPR)	June 2009	The NNPR officially expanded. The Mine area is now surrounded by the expanded park but is not part of the park, and approximately half of the length of CZN's access road crosses the park
Parks Canada Issues Water Licence and Land Use Permit to CZN relating to the access road	June 2009	Permit Number 2009-W02 and Permit Number 2009-L02 valid until April 10, 2012
Developers Assessment Report (DAR) submitted to MVEIRB	March 2010	DAR submitted by CZN initiating the EA review phase. DAR concludes that the impacts of the development and operation of the Project can be effectively managed and minimized by implementing a mitigation program. The assessment report indicates that the Project will not have any significant impacts on fish or aquatic life and will not have any significant adverse effect on the ecological integrity of the NNPR
Impact Benefit Agreement	January 2011	CZN and NDDB
Impact Benefit Agreement	June 2011	CZN and LKFN
Public Hearings	22 – 24 June 2011	Nahanni Butte and Fort Simpson
Socio-Economic Agreement for Prairie Creek Mine	22 August 2011	The Agreement with the GNWT 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
Closing Arguments from federal regulators, including Aboriginal Affairs and Northern Development Canada	13 September 2011	Closing Arguments – Canadian Zinc Proposed Prairie Creek Mine – EA0809-002
Approval to proceed from the MVRB	8 December 2011	Project Approval to proceed to permitting, including long list of commitments as part of Scope of Development
Prairie Creek Mine Consolidated Project	15 February 2012	Report submitted to the MVLWB and Parks Canada in support of applications: MV2008L2-

Event / Milestone	Date	Comments
Description (CPD) submitted		002; MV2008 D0014; Tetcela Transfer Facility; MV2008T0012; and Access Road LUPs/WLs
Application for new LUP and amendment to existing Class B Water License	20 February 2012	New LUP for a second decline and request for amendment to Water License MV2001L2-0003
Renewed MOU between CZN and Parks Canada	5 March 2012	CZN and Parks Canada renew their MOU regarding the operation and development of the Prairie Creek Mine and the management of Nahanni National Park Reserve
New LUP issued and Water License amended	10 May 2012	LUP MV2012C0008 and Water License amendment issued
Directive and Workplan received from the MVLWB	11 May 2012	Document defines the information requirements and process schedule leading to operating permits for the Mine
Additional permits approved for Project	15 May 2012	New mine decline development permit issued; Road Land Use Permit extended
Decision form AANDC	8 June 2012	Minister of AANDC advises the MVRB that they will not order an environmental impact review of the Project, nor will they refer the project to the Minister of Environment for a *Canadian Environmental Assessment Act* joint panel review
Parks Canada Issues Notice of Two year extensions to Water License and Land Use Permit for the access road.	5 July 2012	Permit Number 2009-W02 and Permit Number 2009-L02 extended until April 10, 2014

The key milestone for the Project was the MVRB approval of the Project proposal for operations EA on December 8, 2011. The MVRB concluded, pursuant to paragraph 128(1)(a) of the *Mackenzie Valley Resource Management Act*, 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, see Table 20.3 for summary.

Table 20.3 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, 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 fulfill its commitments, and as a result did not provide legally enforceable measures. Two Board members had dissenting opinions to the final decision that the Project was not likely to have any significant impacts on the environment but recommended that the Project be allowed to proceed subject to conditions. Some groups (Dehcho First Nations, Canadian Parks and Wilderness Society, Mining Watch) encouraged the Minister to send back the EA to the MVRB for further review. However, in a Decision dated June 8, 2012, the Minister of AANDC, on behalf of the responsible Ministers with jurisdiction, including the Minister of the Environment, the Minister of Fisheries and Oceans, the Minister of Environment and Natural Resources, the Minister of Transport Canada and the Minister of Environment and Natural Resources of Government of the Northwest Territories, advised the Review Board of the Decision that the Ministers will not order an environmental impact review of the proposed development of the Prairie Creek Mine, nor will they refer the proposal to the Minister of the Environment for a *Canadian Environmental Assessment Act* joint panel review.

Following the December 2011 EA decision, the project was referred to the MVLWB for the processing of permits required to operate the mine (a range of separate permits required).

In support of operational permit applications, including Water Licenses and LUPs for a revised access road, a Consolidated Description Report (CDR) was submitted to MVLWB and Parks Canada on 15 February 2012. The Water Licenses for the road are required for proposed permanent bridges and the use of water for seasonal road base construction. A

LUP from Parks Canada is required for the mid-point transfer facility because the facility would be within the expanded NNPR and not in MVLWB jurisdiction. On 11 May 2012, CZN received a Directive and Work Plan from the MVLWB which defines information requirements and a process schedule leading to operating permits for the mine. It was requested that the information be submitted to the MVLWB by 17 August 2012. The information request covers a range of topics including general information, financial, construction, waste, water, aquatic effects monitoring, and management plans.

Current Permits and Licenses

CZN currently has several permits and licenses that are active and in-place including three LUPs and two Water Licenses (see Table 20.4 for summary) issued by the MVLWB under the *Mackenzie Valley Resource Management Act*, and an LUP and Water License from Parks Canada for the portion of the existing road that crosses the NNPR.

Water License (MV2001L2-0003) and the current LUP (MV2012C0008) allow CZN to continue with underground exploration while awaiting operational permits. LUP MV2004C0030 is valid until May 2013 and provides for surface exploration and diamond drilling at sites throughout the Prairie Creek Property.

LUP MV2003F0028 (valid until 2014) allows CZN to reopen the access road from the Liard Highway at Lindberg Landing to the mine. While CZN does not currently intend to use the LUP to re-open the whole road, the permit does allow repairs and maintenance to be conducted on the all season portion of the alignment. CZN also obtained a companion Water License MV2007L8-0026 for this work in May 2008 (valid for 5 years). With the expansion of the NNPR, approximately half of the access road now crosses the park. Subsequent to park expansion, Parks Canada issued permits for the existing road comparable to those issued by the MVLWB. Parks Canada is also in the process of extending the road LUP for 2 years. In order to conduct road repairs, in addition to a Water License, CZN also required an authorization from the Fisheries and Oceans Canada (DFO) which was issued in July 2008.

Table 20.4 Summary of Current Permits and Licenses for Crown Land

Permit	Date of Issuance (duration)	Description
Water License (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 (first decline), and subsequently to continue with this exploration (second decline) while awaiting operational permits -Included application for pilot plant operation (plant was never commissioned)
LUP MV2001C0023	10 September 2003 (extended for 2 years, but has now lapsed)	
LUP MV2012C0008	Issued 10 May 2012 – expiring on 9 May 2017	-New LUP for a second decline -Included request for amendment to Water License MV2001L2-0003 – see above)

Permit	Date of Issuance (duration)	Description
LUP MV2004C0030	11 May 2006 (for 5 years) Extended for 2 years (valid to 10 May 2013)	-Allows for surface exploration and diamond drilling activities
LUP MV2003F0028	11 April 2007 (initially valid for 5 years but has been extended another 2 years – now valid to April 2014)	-Allows CZN to reopen the existing access road from the Liard Highway at Lindberg Landing to the Mine
Water License MV2007L8-0026	May 2008 (valid for 5 years)	-Water License for the existing road allowing repairs of erosion and the placement of material below the ordinary high water mark of the creek

20.5 Social and Community Aspects

A detailed socio-economic assessment was completed 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 confirmed economic prospects. The majority of the economic and social impacts will be generated through the participation of local labour and business in the area, including the communities of Nahanni Butte, Fort Simpson and Fort Liard. Participation will come in the form of direct employment, direct supply of goods and services, and spin-off activities. There will be a period of adjustment as people and communities integrate into the wage economy. For those living in the Project area, an operating Prairie Creek Mine is expected to offer an opportunity for a generation of employment, and will result in a population that is better educated, better trained and has access to future opportunities.

In August 2011, CZN negotiated a Socio-Economic Agreement with the 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.

One of the conclusions of the MVRB from their review of the Project EA in December 2011 was that “the Prairie Creek Mine has broad support from First Nations and communities within the Dehcho Region”. The MVRB also concluded that “In the Review Board’s view, socio-economic impacts and benefits are appropriately addressed through the Socio-Economic Agreement between Canadian Zinc Corp. and the Government of Northwest Territories”.

20.5.1 Aboriginal Consultation

Prior to CZN’s involvement with the Prairie Creek Mine, the local affected communities had a somewhat limited understanding of the Project’s existing infrastructure. CZN (then known as San Andreas Resources) first began to engage with the involved parties in 1992, which led to the first cooperative working agreement. The Prairie Creek Development and Cooperation Agreement was signed between the NDDB and San Andreas in 1996.

In 2001 Consultation activities were re-started in earnest with the NDDB and the LKFN. Senior staff from CZN met frequently with local Aboriginal groups, culminating in Memoranda of Understanding (MOU) being developed in 2010 leading to the signing of Impact Benefit Agreements (IBAs) with each of the NDDB and the LKFN in 2011.

The IBAs provide for an ongoing working relationship between CZN and the First Nations that respects the goals and aspirations of each party and enable community members to participate in the opportunities and benefits offered by the Prairie Creek Project. The IBAs also confirm the support of each group for the Prairie Creek Project. In addition the Agreements are intended to ensure that CZN undertakes operations in an environmentally sound manner. CZN has targeted minimum employment levels of 60% northern residents and 25% Aboriginal First Nations and has undertaken to maximize business opportunities for regional First Nations communities and northern businesses. CZN continues to provide employment, training programs, and community assistance for local Aboriginal groups. Specific examples include the employment of an Information Officer, the establishment of an annual scholarship program, site tours, and various community activities.

Over the past 15 years, CZN has had expanded dialogue with local groups, including the Dehcho First Nation (DCFN), Acho Dene Koe Band (Fort Liard), Sambaa K'e First Nation (Trout Lake), Jean Marie River First Nation (Jean Marie River), and the Village of Fort Simpson.

Before CZN submitted applications for operating permits, community open house meetings were held in November 2007 in Dehcho communities. Open houses were held in Fort Simpson, Nahanni Butte, Fort Liard, Wrigley, Trout Lake, Hay River, Kakisa, and Fort Providence and provided details of the Project and plans for operations. As part of the Environmental Assessment process public hearings were also held in each of Nahanni Butte and Fort Simpson.

Access road operations are expected to increase traditional land use in the area since a re-aligned access road will afford easier access to hunting areas and trap lines. A cooperative effort to control road access will be adopted because un-authorized use poses risks to human safety and to wildlife from hunting pressures.

20.5.2 Agreements with Government Agencies

In addition to the Socio-Economic Agreement with the GNWT noted previously, in 2008, CZN signed a MOU with Parks Canada relating to the Expansion of the Nahanni National Park Reserve and the future operation of the Prairie Creek Mine. The MOU was recently renewed on 5 March 2012. The MOU states the following:

- Parks Canada and CZN 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 CZN to develop the Prairie Creek Mine and has granted Land Use Permit 2009 – L02 to provide road access through the Park to the Mine area.

- CZN 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 – Nah?a Dehé Consensus Team.

20.6 Mine Closure Requirements

The closure and reclamation of mine sites in the Northwest Territories is regulated by the MVLWB through the Water License and Land Use permitting process. The Prairie Creek Mine will be closed as per an approved closure plan for the Project which is expected to include the following:

- Covering of the WRP with a 1-2 m thick clay-rich soil layer to minimize infiltration.
- Dismantling of site buildings and infrastructure (if they have no future use) and the burial of refuse within the WRP.
- Emptying the WSP, removing sediment to storage underground or in the WRP.
- Complete filling of underground access tunnels to prevent entry and the discharge of drainage.
- Removal of the synthetic liner from the WSP and burial within the WRP (note that the clay lining of the pond may be used in reclamation).
- Breaching and grading the WSP dykes.

It is expected that the airstrip at the Project site will be left for emergency landings.

Mine closure costs and requirements will be the subject of discussion between CZN and relevant authorities in the near future; however, costs have been generally accounted for in the cash flow model.

20.7 Recommendations

It is recommend that consultation activities continue with aboriginal groups, government agencies (e.g., Parks Canada) and other interested stakeholders to maintain a positive working relationship.

It is also recommended that the Company implement environmental studies as required to address information requests from the MVLWB as per their Directive and Work Plan received on 11 May 2012. This document defines the information requirements and process schedule leading to operating permits for the Mine.

21 CAPITAL AND OPERATING COSTS

21.1 Sources of Estimated Costs

The current overall Pre-Feasibility Study Capital and Operating Cost Estimates for re-commissioning the Prairie Creek Mine and Concentrator were compiled by SNC-Lavalin from information for which it was responsible and from information provided by other QP's associated with this 43-101 report. SNC-Lavalin, in compiling the Capital and Operating Cost Estimates, relied on information from the following QPs associated with this QP report.

Table 21.1 Sources of Inputs to Capital and Operating Cost Estimates

Component of input to the Capital and Operating Cost Estimates	Source
Owner's Costs	CZN
Mine Workforce Cost Estimate	BH
Mill & Surface Workforce Cost Estimate	SNC
Mine Capital Cost Estimate	BH
Mine Operating Cost Estimate	BH
Mill & Surface Capital Cost Estimate	SNC
Mill & Surface Operating Cost Estimate	SNC
Winter Road Installation & Maintenance Cost Estimate	SNC
Power Cost Estimate	SNC
Reclamation Cost Estimate	SNC

CZN=Canadian Zinc Corporation, BH= Barrie Hancock & Assoc Inc., SNC== SNC-Lavalin Inc.

21.2 Capital Cost Estimate

The current overall Capital Cost Estimate for the Pre-Feasibility Study has an intended overall accuracy of +20/-20%. The estimate is broken down into pre-production capital, working 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 11.

Table 21.2 Summary of Capital Costs

Description	Total ($M)
Pre-Production Capital	192.87
Working Capital	41.15
Sustaining Capital	11.34
Total Capital Cost	235.36

21.2.1 Pre-Production Capital

The pre-production capital cost estimate is divided by Work Breakdown Structure (WBS) cost areas as provided below in Table 21.3.

Table 21.3 Pre-Production Capital Cost Summary

	Total Cost ($M)
Direct Costs	
Underground Mine Development	$21.57
Mine Equipment	$8.90
Mill Upgrade - Structural & Mechanical	$7.06
Mill Upgrade - Piping	$2.09
Mill Upgrade - Piping Heat Traced Lines	$1.87
Mill Upgrade - Piping Waste Heat	$1.33
Mill Upgrade - Electrical	$19.90
Mill Upgrade - Instrumentation	$0.79
Dense Media Separation Plant (DMS)	$6.69
Mill Rock Storage Are DMS Floats	$0.13
Concentrate Storage Building	$3.99
Bagging Plant	$0.25
Tailings Paste Plant	$10.46
Water holding pond	$2.61
New Camp	$7.71
Waste Rock Storage Area	$0.27
Water Treatment Plant	$1.72
Winter Road Upgrade	$4.35
Liard Laydown Area Hwy 7	$9.20
Tectcela Storage Area	$3.89
Fort Nelson Storage Area	$1.83
Fuel Tank Modification	$0.78
Owner's Costs	
Owner's Cost	$12.76
Direct Costs	
EPCM Cost	$8.64
Construction Indirects	$14.44
First Fills	$0.52
Freight	$3.93
Spares	$0.48
Vendors Rep	$1.62
Contingency	
Contingency	$33.07
SubTotal Preproduction Capital	**$192.87**

21.2.1.1 Direct Costs

Mine Pre-Development and Mining

Underground pre-development at the Prairie Creek mine will be conducted by a mining contractor. Pre-development will consist of just under one year of mining, primarily focused on preparing the development in the upper mine to sustain full production. This work will begin in late 2013 and end near the end of 2014. Pre-production will be minimized wherever possible to defer expenses. CZN will purchase all of its own mining equipment to start production. CZN currently plans to take over development mining at the start of production, but may decide to retain the contractor in some capacity until all major development is completed.

The major items that will be mined by the pre-development contractor include:

- Slashing out existing 880 level to 4.5 m x 4.5 m.
- Extension of the 880 level.
- An internal ramp (3.2 m x 3.2 m) to the 970 level.
- 200 m of development on the 970 level (to allow for construction of the paste distribution centre).
- Two internal ore access ramps (3.2 m x 3.2 m) for the upper mine.
- Various ore passes and ventilation raises within the upper mine.

Pre-development costs have been estimated from a number of quotes from capable mining contractors, including Procon Mining and Tunnelling, Dumas, DMC Mining Services and Cementation Inc. Quotes were obtained from a variety of mobile equipment manufacturers and some costs were based upon engineering judgment and experience. Overall, pre-development and equipment capital costs, including contingency are estimated at $36.6M in 2013 and 2014, collectively.

Costs for mine equipment used in this estimate were sourced from a variety of new and used equipment suppliers including: Atlas Copco, Sandvik, McDowell Brothers Industries, Orica Mining Services, Miller Technology, Normet and SwedeVent.

Sustaining capital for mining, including the purchase and rebuild of mobile equipment, is estimated at $4.2M over the remainder of the mine life after year 1. A 20% contingency was applied to all capital expenditures.

Process Plant, Onsite and Offsite Infrastructure

All costing for equipment and buildings relating to upgrades of the process plant, onsite and offsite infrastructure for equipment was based on quotes received from equipment vendors in 2011 with a 3% escalation applied (based on Chemical Engineering Plant Index as of February 2012).

Civil, concrete, structural steel, piping, electrical and instrumentation bulk quantities were developed by SNC-Lavalin or a sub-consultant with pricing based on current in-house cost information.

The construction labour rate used in the estimate is based on a workweek of 84 hours, comprising of seven 12-hour shifts, Monday to Sunday, straight time for 40 hours and a 1.5 multiplier for overtime. A productivity factor of 1.4 is estimated to account for the working conditions, weather and long working hours. Current labour rates were based on 2012 wage rates provided by Alberta Construction Labour Relation Board.

21.2.2 Owner's Costs

Owner’s costs were provided by CZN and include a reclamation cost of $12.6M, with a 30% contingency to fund the cost of the Preliminary Closure and Reclamation Plan.

21.2.3 Indirect Costs

Engineering, Procurement & Construction Management (EPCM)

EPCM costs were based on a preliminary Manpower Front End Loading (MFL) prepared by SNC-Lavalin.

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.

Construction Indirects

Costs for temporary construction facilities and services were prepared by SNC-Lavalin.

Construction camp capacities have been developed using both direct and indirect man-hours. Quotations were used in pricing the 50-man rental construction camp at C$40,000 per month including estimated cost for shipping and installation.

Catering costs per person per day are based on $65/man-day seven days a week calculated from total on-site man-hours for direct and indirect labour.

Freight Cost, Spare Parts, Vendor Representatives

Freight cost for equipment was estimated at 10% of the cost of equipment. Freight cost for bulk materials was estimated at 5% of the cost of bulk materials.

Spares were estimated by SNC-Lavalin as 5% of mechanical equipment costs, excluding tanks.

Costs for vendor representatives were estimated by SNC-Lavalin as 1.5% of mechanical equipment costs, excluding tanks.

21.2.4 Contingency

Contingency was estimated by SNC-Lavalin in each area based on the level of engineering definition and method of pricing.

An overall average of 20% contingency has been applied to the total direct and indirect cost for the mine, mill, onsite and offsite infrastructure.

21.2.5 Sustaining Capital

Sustaining capital costs have been stated in constant second quarter 2012 Canadian dollars without any allowance for escalation or inflation. Sustaining capital costs have been developed as follows:

- The estimated sustaining capital for the mine and mine development was provided by CZN.
- Process and infrastructure sustaining capital was based on an estimated price of $1.00/tonne mined ore provided by SNC.
- A 20% contingency was applied to mine and mill sustaining costs.

21.2.6 Working Capital

Working capital included the full cost of operations for three months after commissioning plus the cost of additional operating supplies necessary to operate until the winter road is available to re-supply the mine.

It is assumed that the concentrate will be sold as produced or that an inventory financing arrangement from a smelter or third party will be arranged.

21.3 Operating Cost Estimate

21.3.1 Summary Total Operating Cost

The operating cost estimate was initially expressed in second quarter 2012 Canadian dollars and then developed to a pre-feasibility study estimate level of +20/-20% overall intended accuracy, over the life of the mine.

The annual tonnes and grade of mined ore is based on a detailed mine plan prepared by CZN. Operating costs are presented in each case for year 1 and year 5 to reflect the costs at the start-up mining rate of 213,000 tonnes/year and the estimated maximum rate of 500,000 tonnes/year. The estimated unit costs for each major operating cost are shown in Table 24-1. These values are the sum of both supplies cost and manpower costs during production years 1 and 5.

Table 21.4 Total Operating Cost (Labour and Supplies) Summary

Total Operating Cost	Year 1	Year 5	Year 1	Year 5
Tonnes/ year			212,890	499,707
	($/t)	($/t)	($M/year)	($M/year)
Processing	37.25	37.17	7.93	18.57
Mining	81.22	72.10	17.29	36.03
G&A	10.59	10.83	2.26	5.41
Site Surface	30.72	23.67	6.54	11.83
Transportation	51.31	60.30	10.92	30.13
Total	211.10	204.07	44.94	101.97

21.3.2 Annual Operating Manpower Costs

The manpower costs exclude 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 overall manpower costs for all of the major areas are summarized in Table 21-5.

Table 21.5 Total Manpower Cost Summary

Area Manpower	Year 1	Year 5	Year 1	Year 5
Tonnes/year			212,890	499,707
	($/t)	($/t)	($M/year)	($M/year)
Process	10.87	11.08	2.31	5.54
Mine	55.65	40.87	11.85	20.42
Site Services	6.25	4.64	1.33	2.32
G&A	5.25	3.67	1.12	1.83
Total	78.02	60.25	16.61	30.11

The manpower salary costs, provided by CZN, are based on all mill employees working a 12-hour shift, three weeks on and three weeks off. The majority of general maintenance costs are used in the mill, however for simplicity all general maintenance labour costs are under site services.

The manpower for mining operations is included in the supplies cost for mining. The breakdown for mining operating and labour costs were provided by CZN and developed on a cost per tonne of mined ore.

The manpower for site services includes all site and mill maintenance costs. Catering manpower costs are included in the site services operating costs.

The manpower for G&A includes all positions required for support from both the home office in Vancouver, BC and the Prairie Creek site.

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

21.3.3 Annual Supplies Costs

The overall supplies cost, including all of the major areas; mill, mine, G&A, site service and transportation, is summarized in Table 21.6.

Table 21.6 Total Supplies Cost Summary

Area Supplies Cost	Year 1	Year 5	Year 1	Year 5
Tonnes/year			212,890	499,707
	($/t)	($/t)	($M/year)	($M/year)
Processing	26.38	26.09	5.62	13.04
Mining	25.57	31.23	5.44	15.61
G&A	5.34	7.17	1.14	3.58
Site Surface	24.47	19.03	5.21	9.51
Transportation	51.31	60.30	10.92	30.13
Total	133.08	143.81	28.33	71.87

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

Mining costs are based on power, labour and other miscellaneous costs associated with mining activities.

Major categories of G&A costs have been estimated by CZN and include costs of office and general, professional fees and air flights.

Site Services include the operational costs of providing power to the mine, milling complex and the camp. Costs also include the day-to-day operation and maintenance of surface facilities, and infrastructure that support the mill and mine operations.

The estimated peak annual energy consumption at 1,200 tonnes per day is 4477 kW. This is based on the connected load from the equipment list with an allowance for heat tracing, building HVAC and lighting services. The power demand for the mills is estimated using the “Bond Work Index” for the ore being processed. A power cost of $0.31 kWh was calculated using a low sulphur diesel fuel delivered price of $1.25/litre provided by CZN.

The cost for transportation functions, winter road installation and maintenance is estimated to be $67/t of concentrate produced at 1,200 tonnes/day during peak production. The winter road construction will only be carried out for 50 days of the year while maintenance will continue through the following 75 days well the road is occupied for hauling concentrate.

22 ECONOMIC ANALYSIS

22.1 Sources of Estimated Costs

The financial model consists of multiple worksheets linked to the Pre-Tax and Pre-Finance Economic Model. SNC-Lavalin was responsible for compiling the Pre-Tax and Pre-Finance Economic Model from information for which it has been responsible for developing during the Pre-Feasibility Study and from information provided by other QP's associated with this Technical Report. SNC-Lavalin, in compiling this Pre-Tax and Pre-Finance Economic Model has relied on information from QPs who have provided cost estimates and pricing information.

Table 22.1 shows the sources for the financial model input.

Table 22.1 Financial Model Input Responsibility

Component of input to the Pre Tax and Pre Finance Economic Model	Source
Pre Tax and Pre Finance Economic Model	SNC
Sensitivity Chart @ 90%/100%/110%	SNC
Owner's Costs	CZN
Commodity Prices	BH
Mine Production Schedule	BH
Metallurgical Balance	SNC
Mine Workforce Cost Estimate	BH
Mill & Surface Workforce Cost Estimate	SNC
Mine Capital Cost Estimate	BH
Mine Operating Cost Estimate	BH
Mill & Surface Capital Cost Estimate	SNC
Mill & Surface Operating Cost Estimate	SNC
Winter Road Installation & Maintenance Cost Estimate	SNC
Power Cost Estimate	SNC
Net Smelter Return and Penalty Revenue Estimate	BH
Reclamation Cost Estimate	SNC

22.2 Assumptions

- The model is prepared on a pre-tax and pre-finance basis.
- The project is assumed to be financed at 100% equity and project operating cash flows throughout the project term.
- All costs included in this report and in the model are expressed in 2012 Q2 Canadian dollars without allowance for escalation or inflation, unless specified otherwise.
- The metals prices were estimated as shown in Table 13.2.

Table 22.2 Estimated Metal Prices for Zinc, Lead and Silver

	2014	2015	Long Term
Zinc ($/lb)	$1.20	$1.20	$1.00
Lead ($/lb)	$1.20	$1.20	$1.00
Silver ($/oz)	$28.00	$28.00	$26.00

- Within the financial model, revenue is recognized as concentrate is generated and does not account for any time delay between shipment and payment for concentrate. It is assumed that the concentrate will be sold as produced or that an inventory financing arrangement from a smelter or third party will be arranged.
- No salvage value for Prairie Creek Mine is included.

22.3 Summary of Outputs

The Pre-Tax and Pre-Finance Economic Model is attached as Appendix A. A summary of the output values is shown in Table 22.3

Table 22.3 Output Values of Financial Model

Capital	
Pre-production capital	$160 M
Pre-production contingency	$33M
Financial Analysis:	
Average annual EBITDA*	$66 M
Pre-tax NPV using a 8% discount rate	$253M
Pre-tax IRR	40.4%
Pre-tax payback period	3 years

* Average annual EBITDA does not include year 1 of production

22.4 Sensitivity Analysis

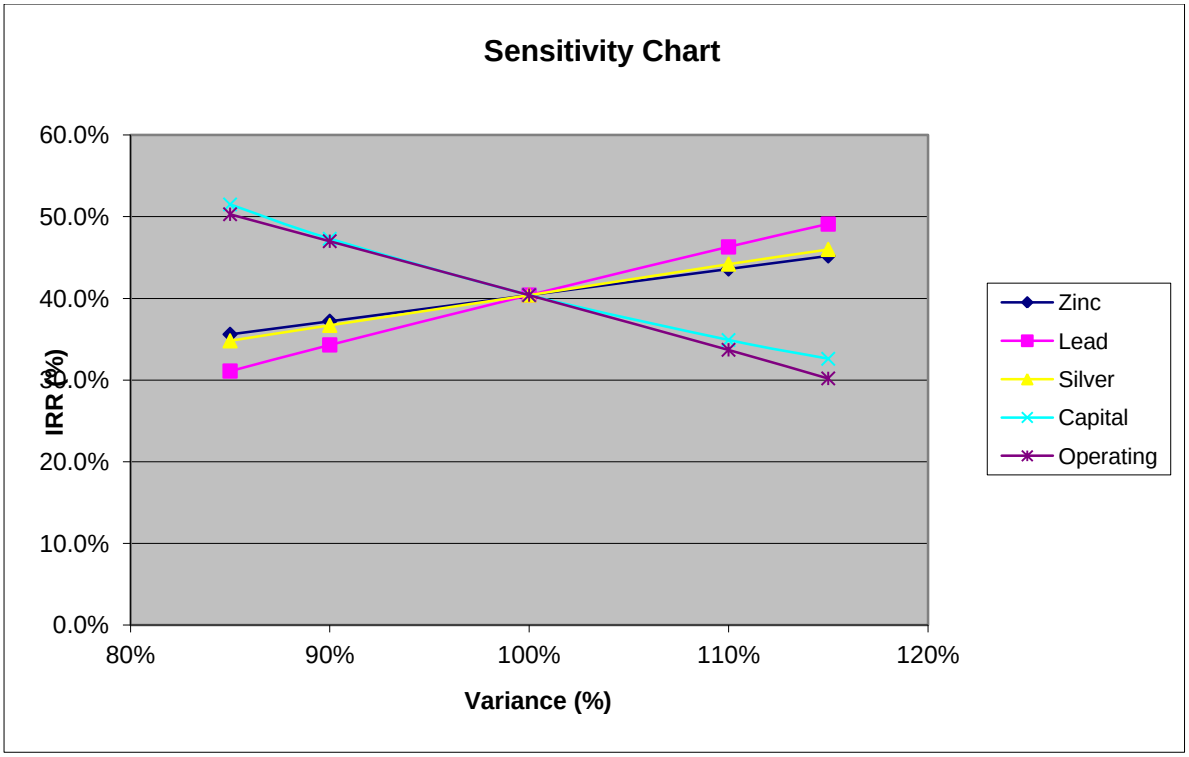
Table 22.4 shows the results, on a pre-tax basis, of a financial analysis with a +/- 10% sensitivity factor centering on the Base Case. Figure 22.1 is a spider diagram showing the sensitivities to lead, zinc, silver, operating costs and capital costs.

Table 22.4 Financial Analysis

	Low Case	Base Case	High Case
Metal Price Scenario	90%	100%	110%
Average Annual EBITDA* $M	$47	$66	$84
Pre-Tax NPV (undiscounted) $M	$303	$493	$683
Pre-Tax NPV @ 8% discount $M	$140	$253	$366
Pre-Tax IRR	27.4%	40.4%	52.8%
Pre-Tax Payback Period	3.8 yrs	3.0 yrs	2.5 yrs

* Average annual EBITDA does not include year 1 of production

Figure 22.1 Sensitivity Chart



The before-tax IRR of the Prairie Creek project is sensitive to zinc, lead and silver prices as well as capital and operating costs on a percentage basis.

22.5 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 around 15% of taxable income. The property will be subject to an 11.5% territorial income tax rate on taxable income. Northwest Territories royalties will typically be 13% 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 and certain other amounts exceeds the permitted deductions and allowances.

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.

The Prairie Creek Mine has Impact Benefit Agreements (IBAs) with the Nah?ą Dehé Dene Band and the Liidlii Kue First Nation. The terms of the IBAs are confidential; however, they have generally been accounted for in the cash flow model.

23 ADJACENT PROPERTIES

There are no adjacent properties. The Property is surrounded by, but is not included in, Nahanni National Park Reserve.

24 OTHER RELEVANT DATA AND INFORMATION

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

The South Nahanni River is highly valued as a wilderness recreation river and is used for canoeing trips during the summer months. These wilderness adventure tours are supported by a number of outfitting companies from as far away as Ontario.

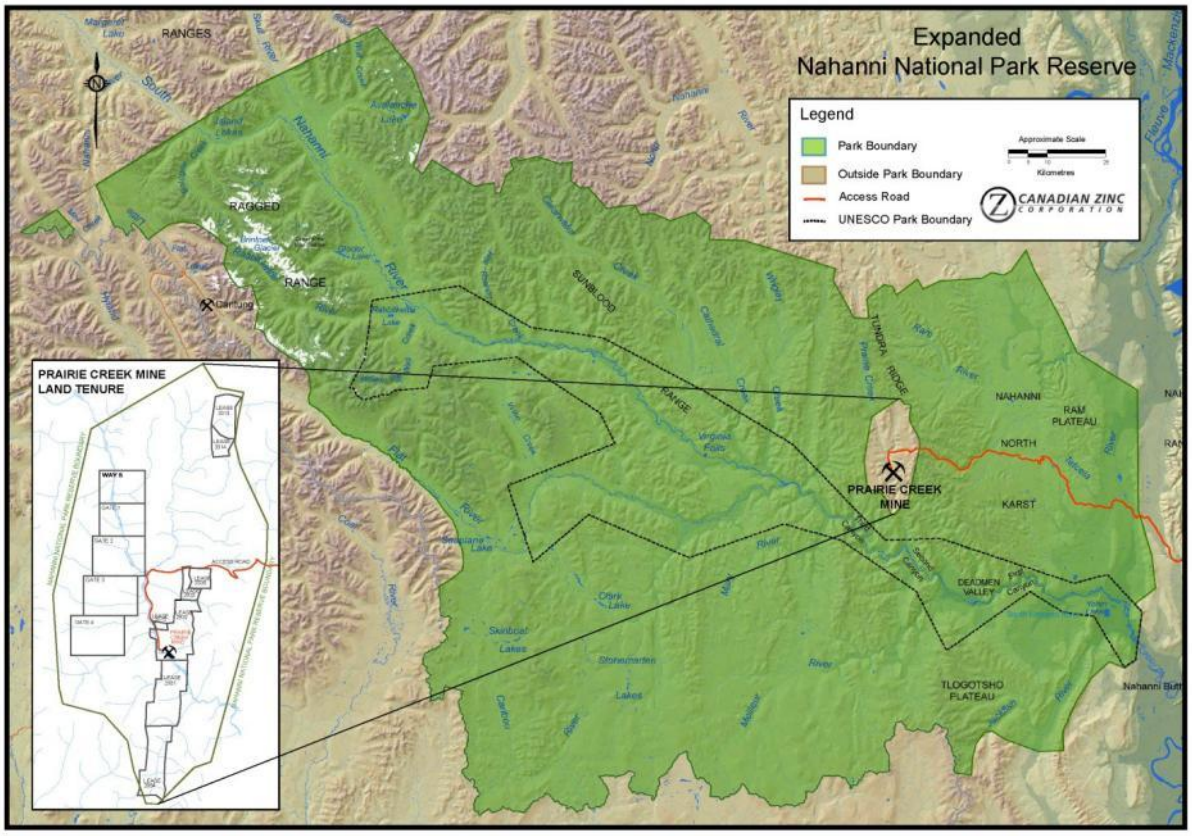
Parliament formally established NNPR in 1972, legally protecting it as Canada's 26th National Park under the Canada National Parks Act. It was established as a National Park Reserve in view of the fact that there were outstanding land claims in the area. It will only become a fully-fledged National Park once an agreement has been reached with the Dehcho First Nations.

Nahanni National Park Reserve is considered to be of global significance. In 1978, it was the first area added by UNESCO to its list of World Heritage Sites. There are only 13 sites in Canada designated as World Heritage Sites, eight of them being National Parks. Nahanni received this designation because of the geological processes and natural phenomena in the area. In UNESCO's view, Nahanni is special because it is an unexploited natural area. The presence in this area of three river canyons cutting at right angles to the mountain ranges, with walls of up to 1,000 m high, Virginia Falls which falls over 90 m, hot springs, sink holes and karst topography are considered a special combination.

In considering and approving the nomination of Nahanni National Park Reserve 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 Nahanni National Park Reserve 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 NNPR. 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 as shown in Figure 24.1.

Figure 24.1 Claims 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. The Government's decision on the expansion of Nahanni National Park 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. Theretofore, 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 Memorandum of Understanding (MOU) with regard to the expansion of the Nahanni National Park Reserve and the development of the Prairie Creek Mine, whereby:

- Parks Canada and Canadian Zinc agreed to work collaboratively, within their respective areas of responsibility, authority and jurisdiction, to achieve their respective goals of an expanded Nahanni National Park Reserve and an operating 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 Nahanni National Park Reserve 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 Nahanni National Park Reserve.

Parks Canada and Canadian Zinc (the Parties) 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 Dehcho First Nations, which will better identify, define and consider issues of common interest, including, among other things, access to and from the Prairie Creek Mine through the expanded NNPR and the park boundaries around the Prairie Creek Mine properties. The Parties have 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 MOU is an expression of the mutual intentions of the parties and is not legally binding or enforceable. The MOU does not create any new powers or duties, or alter or affect any rights, powers or duties established by law, including by the Parks Canada Agency Act and the Canada National Parks Act, or result in the Parties relinquishing any right, jurisdiction, power, privilege, prerogative or immunity.

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 constrain Parks Canada from doing so, subject only to the understanding that Parks Canada has agreed not to object to or oppose, in principle, the development of the Mine.

The MOU was valid for three years. On 5 March 2012, CZN and Parks Canada renewed their MOU regarding the operation and development of the Prairie Creek Mine and the management of Nahanni National Park Reserve.

The renewed MOU states the following:

- Parks Canada and CZN 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 CZN to develop the Prairie Creek Mine and has granted Land Use Permit 2009 – L02 to provide road access through the Park to the Mine area.
- CZN 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 – Nah?a Dehé Consensus Team.

25 INTERPRETATION AND CONCLUSIONS

The Prairie Creek Property contains a high-grade, silver-lead-zinc-copper vein that was explored since the early 1900s and then extensively explored and developed by Cadillac Explorations Limited (Cadillac) from 1966 to 1983. A mine was developed and the processing plant and surface infrastructure were built in the early 1980s, at a cost of C$64 million (1982 money). The operations were engineered and fully permitted to produce and process mineralized vein material at a rate of 1,000 tons per day. The 1982/83 fall in metal prices necessitated closure of the mine prior to production. This closure led to a change of ownership and eventually to CZN's involvement in 1992. Through a series of agreements between 1992 and 2004 the Company established an increasing interest in the property, plant and equipment resulting in a 100% interest in the Property by 2004.

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

AMC believes that the data collection and handling followed normal industry practice and the data is fit for purpose. However, while QA / QC samples were inserted and the results have been observed in the assay certificates, it is not clear if any analysis of the data was carried out. This has been remedied in the recent programs but these do not influence the current Mineral Resource.

Bulk density measurements have been obtained through direct measurement and calculations. While acceptable for the purposes of this report, AMC recommends that this area be reviewed prior to the next estimate, but does not consider this has a material impact on the resource.

AMC reviewed and restated the existing Mineral Resource estimate discussed in the 2007 and 2011 Technical Reports and took responsibility for it. Mineral Resources comprise 1.700 M tonnes averaging 12.1% Zn, 9.7% Pb, 155 g/t Ag and 0.28% Cu in the Measured category and 3.731 M tonnes averaging 10.2% Zn, 10.5% Pb, 162 g/t Ag and 0.32% Cu in the Indicated category. Inferred resources are 6.239 M tonnes averaging 14.5% Zn, 11.5% Pb, 229 g/t Ag and 0.57% Cu.

A portion of the Mineral Resources was converted to Mineral Reserves through application of suitable dilution factors in stoping blocks utilizing the cut-and-fill mining method for the Main Quartz Vein and room and pillar for SMS. A Mineral Reserve of 5.2 million tonnes, grading 9.4% Zn and 9.5% Pb, with 151 g/t Ag has been estimated.

Due to the high grade nature of the deposit, the majority of the vein resource will be mined, allowing for 97% of all of the Measured and Indicated Resources to be converted to Mineral Reserves. For the SMS mineralization the figure is a conversion of 57%.

The Preliminary Feasibility Study (PFS), which has been carried out to assess the viability of starting up the Prairie Creek Mine, has associated risks and opportunities. These are outlined in bullet point below.

Risks and Opportunities

Major Risks:

- Significant reduction in metal prices.
- Increase in fuel cost significantly in excess of offsetting increases in metal prices.
- A shortened winter road hauling season that could affect the ability to complete the annual concentrate removal and mine re-supply.
- Periods of abnormally cold weather over extended periods of time creating surface-related operating problems.
- Paste delivery sequencing problems creating surface storage issues inconsistent with operating permits.
- Water treatment plant disruption which may cause effluent quality outside compliance limits necessitating the temporary suspension of operations.

Major Opportunities:

- Continued road upgrades and bridge installations that would reduce winter road installation and maintenance costs and also decrease transport costs.
- Cycloning of the DMS feed screen undersize to upgrade feed to the grinding circuit.
- Copper / lead separation to produce a Cu/Ag concentrate that could be air-shipped all year around from the site.
- Use of a form of longhole / sublevel stoping rather than cut and fill in zones of wider mineralization which could reduce operating costs, increase mine productivity and allow for more tailings to be stored underground (less cement required during backfill).
- Use of higher capacity underground equipment to increase efficiency and productivity.
- Reduction in mine dilution in the next stage of design.

Other Relevant Data

- The Project is located close to, but outside of, the Nahanni National Park Reserve. In 2009, the Nahanni National Park Reserve 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.

26 RECOMMENDATIONS

Recommendations (with estimated cost where relevant)

Project optimization

There are a number of recommendations listed in the PFS including:

- Examine opportunities to improve efficiencies in transport, scheduling and logistics on the winter road.
- Consider financial alternatives to purchasing of significant equipment and other procurement.
- Review opportunities for early completion of construction, engineering and mine development programs to reduce start-up times required.
- Consider financial arrangements targeted to further reduce Working Capital needs.
- Undertake additional drilling programs, particularly towards the north end of the deposit, to increase the confidence level in the estimated resources and reserves and to identify additional resources. $2 million.
- Modify the mine plan to include increased resources and identify areas of the mine amenable to lower cost bulk mining methods. Optimization of mine schedule and equipment utilization should follow.
- Undertake further studies aimed at upgrading the zinc oxide concentrate to a commercial grade and producing a copper/silver concentrate to maximize potential future revenues. $100,000.

Data management and resource modelling

- Review bulk density measurement methods.
- Resolve slight discrepancies between drillhole data and wireframes and lack of extrapolation of the MQV wireframe beyond the southernmost drilled section.
- Restrict rotation of the block model to no more than orientation or use an unrotated model.
- Estimate antimony, arsenic and mercury in the next resource update, as these metals report to the final concentrates.
- Model and estimate the percentage oxide component in the MQV mineralization.
- Review the high grade capping policy.
- Composite chip samples to equal lengths and decluster the data.

The inclusive cost for all modelling is estimated at $50,000.

Mining

- Undertake currently planned geotechnical drilling program in the summer of 2012 to confirm ground support requirements and stability control during operations. $50,000.
- Incorporate more detail into the dump pocket design for run-of-mine ore. $10,000.
- Prepare a mine dewatering plan to ensure safety at the face during operations. $10,000.
- Review and refine equipment selection to identify if there is any merit in allowing for variations in the size of the drills and scoops (smaller and larger) or if standardization of the equipment size (as is currently planned) optimizes efficiencies.

Environment and social issues:

- Continue consultation activities with aboriginal groups, government agencies (e.g., Parks Canada) and other interested stakeholders to maintain positive working relationship.
- Implement environmental studies as required to address information requests from the MVLWB as per its Directive and Work Plan received on 11 May 2012.

27 REFERENCES

There is an exhaustive list of references in Stone D M R and Godden S J of MineFill Services, Technical Report of October 2007. Below is an abbreviated version of that list, as well as references cited in this report.

Acme Analytical Laboratories Limited. Methods and Specifications for Analytical Package Group 8 – Specific Gravity. E-mail Version 1.1. 22 February, 2007.

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G & T Metallurgical Services Ltd, 1994. 14.T.462 Metallurgical Testing 1994. Prepared for San Andreas Resources Ltd.

Geological Survey of Canada Open File 5344: Mineral and Energy Resource Assessment of the Greater Nahanni Ecosystem Under Consideration for the Expansion of the Nahanni National Park Reserve, Northwest Territories. December 2007.

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Morrow, D.W. and Cook, D.G. The Prairie Creek Embayment and Lower Paleozoic Strata of the Southern Mackenzie Mountains. Geological Survey of Canada, Memoir 412, Ministry of Supply and Services, Ottawa, Canada, 1987.

SNC-Lavalin Inc, Prairie Creek Project, Pre-Feasibility Study for Canadian Zinc Corporation, July 2012.

Smee & Associates Consulting Limited. Undated Certificates of Analysis, Canadian Zinc Standards 1, 2 and 3. Prepared by B.W. Smee, Ph.D., P.Geo. for Canadian Zinc Corporation.

SGS Lakefield Research Limited. Consultancy Report L.R. 10916-001, Report #1 Phase 1 & 2: An Investigation into the Recovery of Lead, Zinc and Silver from Prairie Creek Sulphide/Oxide Ore Samples. 01 April, 2005.

SGS Lakefield Research Limited. Consultancy Report L.R. 11098-001, Report #2 Phase 3: The Recovery of Lead, Zinc and Silver from Prairie Creek Sulphide/Oxide Ore Samples. 18 January, 2006.

SGS Lakefield Research Limited. Consultancy Report L.R. 11098-002, Report #3: Phase 4 – Development Testwork on The Recovery of Lead, Zinc and Silver from Prairie Creek Sulphide/Oxide Ore Samples. 20 June, 2007.

Stone D M R and Godden S J 2007. Technical Report on the Prairie Creek Mine, Northwest Territories 12 October, 2007. Prepared by MineFill Service Inc.

Taylor, A B, 2011. Technical Report, Prairie Creek Mine, Northwest Territories, Canada, 28 December 2011. Prepared by Canadian Zinc.

28 CERTIFICATES

J M Shannon

1. I, John Morton Shannon, P.Geo, do hereby certify that I am a Principal Geologist and Geology Manager for AMC Mining Consultants (Canada) Limited, Suite 1330, 200 Granville Street, Vancouver, British Columbia V6C 1S4.
2. The Technical Report to which this certificate applies is entitled *Prairie Creek Property, North West Territories, Canada, Technical Report for Canadian Zinc Corporation*, dated 15 June 2012 (the Technical Report).
3. I graduated with a B.A. Mod Nat. Sci. in Geology from Trinity College Dublin, Ireland in 1971. I am a registered member of the Association of Professional Engineers and Geoscientists of British Columbia, and the Association of Professional Geoscientists of Ontario, and a member of the Canadian Institute of Mining, Metallurgy and Petroleum. I have practiced my profession continuously since 1971, and have been involved in mineral exploration and mine geology for a total of 40 years since my graduation from university. This has involved working in Ireland, Zambia, Canada, and Papua New Guinea. My experience is principally in base metals and gold. I fulfill the requirements to be a “Qualified Person” for the purposes of NI 43-101.
4. I visited the Prairie Creek Property on 25 April 2012.
5. I am responsible for the preparation of all of Sections 1 – 12, part of 14, 18, and 23 – 26 of the Technical Report
6. I am independent of the issuer as described in section 1.5 of NI 43-101.
7. I have had no prior involvement with the property that is the subject of this Technical Report.
8. I have read NI 43-101 and certify that the parts of the Technical Report for which I am responsible have been prepared in compliance with the Instrument.
9. As at the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 9 day of August 2012

Original signed and sealed by

John Morton Shannon, P.Geo.

Dinara Nussipakynova

1. I, Dinara Nussipakynova, P.Geo., do hereby certify that I am a Senior Geologist for AMC Mining Consultants (Canada) Limited, Suite 1330, 200 Granville Street, Vancouver, British Columbia V6C 1S4.
2. The Technical Report to which this certificate applies is entitled *Prairie Creek Property, North West Territories, Canada, Technical Report for Canadian Zinc Corporation*, dated 15 June 2012 (the Technical Report).
3. I graduated with a B.Sc. and M.Sc. in Geology from Kazakh National Polytechnic University in 1987. I am a registered member of the Association of Professional Geoscientists of Ontario. I have practiced my profession continuously since 1987, and have been involved in mineral exploration and mine geology for a total of 25 years since my graduation from university. This has involved working in Kazakhstan, Russia and Canada. My experience is principally in database management, geological interpretation and resource estimation. 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.
5. I am responsible for the preparation of Section 14 of the Technical Report
6. I am independent of the issuer as described in section 1.5 of NI 43-101.
7. I have had no prior involvement with the property that is the subject of this Technical Report.
8. I have read NI 43-101 and certify that the parts of the Technical Report for which I am responsible have been prepared in compliance with the Instrument.
9. As at the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 9 day of August 2012

Original signed and sealed by

Dinara Nussipakynova, P.Geo.

J B Hancock

1. I, John Barrie Hancock, P.Eng. do hereby certify that I am a Consulting Mining Engineer for Barrie Hancock & Associates Inc., Suite 38, 20751 87th Avenue, Langley, British Columbia, V1M 2X3.
2. I graduated with a B.Sc. (Hons.) in Mining Engineering from Cardiff University College, University of Wales, in 1964.
3. I am a registered member of the Association of Professional Engineers and Geoscientists of British Columbia and a Life Member of the Canadian Institute of Mining, Metallurgy and Petroleum.
4. I have practiced my profession continuously since 1966, and have been involved in mining engineering including supervision of mining operations, exploration and evaluations for over 45 years, since my graduation from university. This has involved working in the U.K., Canada, and Zambia. I have also spent brief periods at properties and operations USA, Brazil, Chile, Peru and Mexico. While my experience is principally in base metals and gold, I also have experience in coal operations, uranium, potash and a variety of industrial minerals.
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 (NI 43-101) and I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.
6. I am responsible for the preparation of Sections 15, 16 and 19 of the Technical Report is entitled *Prairie Creek Property, North West Territories, Canada, Technical Report for Canadian Zinc Corporation*, dated 15 June 2012 (the Technical Report). I visited the Prairie Creek Property in July 2008.
7. I am independent of the issuer as described in section 1.5 of NI 43-101, but point out that Barrie Hancock & Associates hold 2,500 shares of Canadian Zinc Corporation.
8. I have carried out consulting services for Canadian Zinc Corporation since 2008, and thus have had prior involvement with the property that is the subject of the Technical report.
9. I have read NI 43-101 and Form 43-101F1 and I certify that the parts of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.
10. As at the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 9 day of August 2012

Original signed and sealed by

John Barrie Hancock, P.Eng.

B H Maclean

I. Byard Harold MacLean, P.Eng, do hereby certify that I am a consultant to SNC Lavalin Inc, Suite 1800 – 1075 West Georgia Street, Vancouver, British Columbia, Canada V6E 3C9.

2. I graduated with a B.A.Sc. in Metallurgical Engineering from the University of British Columbia in 1971 and M.A.Sc. in Environmental Engineering from the University of British Columbia in 1977.

3. I am a registered member of the Association of Professional Engineers and Geoscientists of British Columbia.

4. I have practiced my profession since 1971, and have been involved in mineral processing for a total of 30 years since my graduation from university. This has involved working mainly in Canada. My experience is principally in base metals and gold.

5. 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.

6. I am responsible for the preparation of all of Sections 13, 17, 18, 21 and 22 and part of Sections 20, 25 and 26 of the Technical Report titled *Prairie Creek Property, North West Territories, Canada, Technical Report for Canadian Zinc Corporation* dated 15 June 2012. I have visited the Prairie Creek Property on March 2008, June 2011 and July 2011.

7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.

8. I have had prior involvement, as an independent consultant, with the property that is the subject of the Technical Report.

9. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

10. As at the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report that I am responsible for, contains all scientific and technical information that is required to be disclosed to make the technical report clear and the findings well supported.

Dated this 9 day of August 2012

Original signed and sealed by

Byard Harold MacLean, M.A.Sc., P.Eng.

APPENDIX A

PRE-TAX AND PRE-FINANCE ECONOMIC MODEL

Table 28.1 Pre Tax and Pre-Finance Economic Model

ITEM	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	TOTAL
Mining													
Ore Mined, mt	4,510	208,380	502,135	495,442	498,025	499,707	502,662	499,556	488,619	507,991	503,495	511,160	5,221,683
Development waste rock mined, mt	55,598	57,187	110,819	109,737	113,414	80,694	9,864	12,521	31,973	18,101	543	-	600,449
	Apr-13	Apr-14	Apr-15	Apr-16	Apr-17	Apr-18	Apr-19	Apr-20	Apr-21	Apr-22	Apr-23	Apr-24	
Mine Feed													
% Zn as Sulfide	4.07%	6.90%	6.79%	7.13%	8.44%	8.33%	7.48%	7.16%	6.88%	7.64%	7.81%	9.45%	7.68%
% Pb as Sulfide	2.94%	4.92%	5.15%	6.33%	8.26%	7.32%	7.68%	9.09%	10.79%	10.51%	4.95%	4.55%	7.35%
% Pb as Oxide	2.94%	3.70%	3.79%	3.52%	2.57%	2.23%	1.72%	1.47%	0.84%	0.83%	1.79%	2.12%	2.15%
% Zn as Oxide	2.71%	3.72%	3.81%	3.06%	2.01%	1.97%	1.35%	0.83%	0.26%	0.27%	1.16%	1.30%	1.69%
Ag (g/t)	125.54	169.21	172.59	175.82	164.25	157.91	157.49	159.99	168.26	166.38	82.34	99.95	151.04
Net Smelter Return (US)													
Pb sulfide conc		$39,367,773	$98,822,529	$95,369,622	$113,791,007	$104,413,502	$107,912,645	$119,856,873	$135,897,581	$136,403,926	$58,438,953	$60,669,556	$1,070,943,967
Pb oxide conc		$12,694,604	$37,667,435	$35,536,341	$19,134,043	$17,093,064	$14,324,250	$12,875,562	$6,519,346	$6,695,041	$11,664,362	$14,598,100	$188,802,148
Zn sulfide conc		$21,076,957	$50,726,389	$42,908,060	$47,527,686	$46,827,314	$43,000,900	$41,247,255	$39,804,512	$44,501,193	$38,343,606	$48,032,450	$463,996,321
Total Net Smelter Return US$		$73,139,333	$187,216,353	$173,814,023	$180,452,735	$168,333,880	$165,237,795	$173,979,690	$182,221,440	$187,600,160	$108,446,921	$123,300,106	$1,723,742,435
Net Revenue (CANADIAN $)		$73,139,333	$187,216,353	$173,814,023	$180,452,735	$168,333,880	$165,237,795	$173,979,690	$182,221,440	$187,600,160	$108,446,921	$123,300,106	$1,723,742,435
Operating Cost													
Mining		$17,290,788	$33,480,963	$33,550,344	$34,593,360	$36,026,588	$31,691,899	$31,358,099	$31,590,658	$31,635,998	$30,947,639	$30,847,507	$343,013,845
Processing		$7,929,479	$18,638,838	$18,456,909	$18,527,118	$18,572,827	$18,653,177	$18,568,746	$18,271,427	$18,854,810	$19,607,616	$19,796,373	$195,877,320
Other (incl transportation)		$17,464,412	$38,653,214	$40,194,964	$44,043,843	$41,960,168	$40,234,680	$41,135,444	$41,458,347	$43,729,297	$39,146,946	$42,889,906	$430,911,220
G&A		$2,255,571	$5,413,369	$5,413,369	$5,413,369	$5,413,369	$5,413,369	$5,413,369	$5,413,369	$5,413,369	$5,413,369	$5,413,369	$56,389,263
SubTotal Operating:	$0	$44,940,250	$96,186,384	$97,615,586	$102,577,690	$101,972,952	$95,993,126	$96,475,658	$96,733,802	$99,633,474	$95,115,571	$98,947,155	$1,026,191,648
	Cost Y-1	Cost Y1											
Direct Costs	$22,315,298	$95,092,639											
Owner's Costs	$12,762,019	$0											
Direct Costs	$6,156,065	$23,467,659											
Contingency	$3,307,490	$29,767,414											
SubTotal Preproduction Capital	$44,540,874	$148,327,712											
Working Capital (includes 20% contingency)		$41,147,971											
SubTotal Sustaining Capital	$0	$250,056	$2,968,420	$948,536	$732,755	$1,222,623	$923,770	$920,042	$906,918	$930,164	$924,768	$613,392	$11,341,444
Total Capital Cost	$44,540,874	$189,725,739	$2,968,420	$948,536	$732,755	$1,222,623	$923,770	$920,042	$906,918	$930,164	$924,768	$613,392	$200,817,127

ITEM	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	TOTAL
Pre Tax Cash Flow													
Net Revenue	$0	$73,139,333	$187,216,353	$173,814,023	$180,452,735	$168,333,880	$165,237,795	$173,979,690	$182,221,440	$187,600,160	$108,446,921	$123,300,106	$1,723,742,435
Operating Cost (net of Working Capital)	$0	$3,792,279	$96,186,384	$97,615,586	$102,577,690	$101,972,952	$95,993,126	$96,475,658	$96,733,802	$99,633,474	$95,115,571	$98,947,155	$985,043,677
Capital Costs	$44,540,874	$189,725,739	$2,968,420	$948,536	$732,755	$1,222,623	$923,770	$920,042	$906,918	$930,164	$924,768	$613,392	$245,358,001
Pre-Tax Cash Flow	(44,540,874)	(120,378,684)	88,061,548	75,249,901	77,142,290	65,138,305	68,320,899	76,583,989	84,580,721	87,036,522	12,406,581	23,739,559	$493,340,758
Accumulated CashFlow	(44,540,874)	(164,919,558)	(76,858,009)	(1,608,109)	75,534,181	140,672,487	208,993,386	285,577,375	370,158,096	457,194,618	469,601,199	493,340,758	$493,340,758
Discounted Cash Flow	(44,540,874)	(111,461,745)	75,498,584	59,735,797	56,701,886	44,332,036	43,053,756	44,686,022	45,696,332	43,539,930	5,746,648	10,181,490	$273,169,862
Accumulated Discounted Cash Flow	(44,540,874)	(156,002,618)	(80,504,034)	(20,768,237)	35,933,649	80,265,685	123,319,441	168,005,463	213,701,795	257,241,725	262,988,372	273,169,862	$273,169,862