January 29, 2001

CANADIAN ZINC CORPORATION

PRAIRIE CREEK MINE SCOPING STUDY JANUARY 2001

CAPITAL COST – CDN$40.5 MILLION

PRODUCES - 95 MILLION POUNDS SALEABLE ZINC per annum, plus by product copper & lead concentrates containing extensive silver

BREAKEVEN CASH COST – **US 34.5 CENTS** per pound saleable zinc

PROJECT IRR – 45.6% BEFORE TAX AND FINANCING

PROJECT NPV – CDN$97.2 MILLION (@ 10% DCF)

Mine Life Current Resources – 18 Years

Located in Northwest Territories – Northern Canada
Wholly owned by **Canadian Zinc Corporation – CZN – TSE**.
Subject to 2% royalty capped at CDN$8.2 million
Originally constructed 1982, but never operated – CDN$100 million mine, mill and infrastructure in place
Resource – 11.8 million tonnes grading 12.5% Zn, 10.1% Pb, 0.4% Cu, 161 gpt Ag on 2.5 Km of a 14 Km mineralised trend. Potential tonnage – open ended.
Economic Cooperation Agreement in place with Nahanni Butte Dene of Deh Cho First Nations

Re-development Proposal –

- 1,500 tonne per day mine and mill
- Mechanise underground, Alimak stoping, short hole stoping, cut and fill
- Incorporate new gravity front end to mill
- Paste backfill addition to back of mill
- All weather road access and ferry required
- Re-permit operation, earlier permits lapsed.

Construction time 3-6 months

In Production, subject to permitting, 2003

CANADIAN ZINC
CORPORATION

PRAIRIE CREEK MINE

NORTHWEST TERRITORIES

SCOPING STUDY

Vancouver, British Columbia
January 29, 2001

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EXECUTIVE SUMMARY AND CONCLUSIONS

The Prairie Creek Mine, 100% owned and operated by Canadian Zinc Corporation (CZN-TSE), is situated in the southern Mackenzie Mountains of the Northwest Territories. The site contains over $100 million of minesite infrastructure in today's dollars, including a 1,000-ton per day mine and mill constructed in 1981 but never operated. Canadian Zinc Corporation (formerly San Andreas Resources Corporation) acquired the property in 1991 and has since been successfully expanding the mineral resource to its present level by surface diamond drilling exploration.

The Prairie Creek Deposit contains a significant mineral resource of over 11.8 million tonnes grading 12.5% zinc (Zn), 10.1% lead (Pb), 161 g/tonne silver (Ag) and 0.4% copper (Cu). Three distinct styles of base metal mineralisation have so far been discovered on the property: Mississippi Valley-type cavity style (MVT), Vein and Stratabound. MVT Zn-Pb mineralisation is exposed in the northern portion of the property within a marginal carbonate sequence. The latter two styles of mineralisation occur within the Prairie Creek embayment feature in a Siluro-Ordovician sedimentary sequence. The majority of the current mineral resource reports to Vein Pb-Zn-Ag-Cu and occurs in a crosscutting steeply east dipping fault with a northerly strike. The remainder of the mineral resource occurs as Stratabound massive sulphides, which were recently discovered proximal to the Vein mineralisation. The close proximity of the two styles of deposit may indicate a somewhat similar genetic origin.

The principal Vein structure cuts through Ordovician age dolostones and graphitic shales of the Whittaker and Road River Formations. The distinctly different style of mineralisation, termed Stratabound, was encountered in 1992 during deeper drilling near the Vein. The Stratabound mineralisation presently consists of a resource of 1.4 million tonnes grading 10.3% zinc, 5.0% lead and 53 g/tonne silver. This style of deposit is found, so far, wholly within the Upper Whittaker Formation dolostones. Drill holes such as PC-92-15, have cut significant composite intercepts such as 4.4% lead and 9.3% zinc over 28 meters of core. Stratabound sphalerite-galena-pyrite sulphides occur predominately in a subunit of the Whittaker termed the Mottled Horizon located approximately 200 meters below the present mill level (870 m) underground workings.

Since 1991, CZN has completed 40,000 meters of surface diamond drilling and an underground sampling program, which has greatly expanded the inventory of known resources on the property to the current 11.8 million tonnes. The discovery in 1992 of Stratabound type mineralisation in the main zone opened up multiple exploration targets for the discovery of further Stratabound deposits. Potential for significant increases in Stratabound mineralisation exist throughout the property. Drilling on Zone 3 has so far been restricted by topography limiting the areas of potential mineralisation that can be drilled. Such Stratabound mineralisation has the potential to significantly increase the tonnage fed to the mill, due to its much higher tonnes per vertical meter of depth.

The Vein-type deposit remains open ended to the north and south of the current resource, which is defined over only 2.5 kilometers of the 16 kilometer prospective corridor. With many Vein occurrences exposed throughout its length, the prospects for additional Vein and Stratabound material are excellent over the remaining 14 kilometers of subsurface. Consequently, there remains excellent potential for discovering additional massive sulphides in the vicinity of the Prairie Creek Mine, which will further add to the already substantial mineral resource base.

The mine and mill were originally developed by Cadillac Explorations Ltd. ("Cadillac") on a 2 million ton reserve, aimed at the production of a copper/lead/silver concentrate, as part of the attempt by the Hunt brothers of Texas to control the silver market. In 1982, Cadillac went into receivership, leaving a $100 million infrastructure in today's dollars, including 5 kilometers of underground development on three levels, a 200-man camp and a 1,000 ton per day mill, which has never been used for production. In August 1991, San Andreas Resources Corporation (subsequently re-named Canadian Zinc Corporation in 1999) entered into an agreement to purchase the mine and worked to extend the known resource.

The Prairie Creek Mine is located on land claimed by the Nahanni Butte Dene Band (the “Nahanni”) of the Deh Cho First Nations (“DCFN”) as their traditional territory. The DCFN are engaged in ongoing negotiations with the Government of Canada and the Government of the Northwest Territories in what is referred to as the Deh Cho Process. The negotiations are currently at the Interim Measures and Agreement In Principle stage. The outcome of the negotiations is expected to be a Final Agreement that will provide, amongst other things, for the implementation of a Deh Cho form of government to oversee the delivery of programs and services to residents within the DCFN territory. It is expected that the negotiations will take some five to seven years to complete.

In 1996, the Company and the Nahanni successfully negotiated and executed the Prairie Creek Development Cooperation Agreement (the “Agreement”). The overall intent of the Agreement was to establish and maintain a positive and cooperative working relationship between the Company and Nahanni in respect of the further development and operation of the mine, while at the same time supporting an economically viable and environmentally sound operation and maximizing economic opportunity and benefits to Nahanni and other Deh Cho First Nations. This Agreement foresaw the many benefits which could accrue to the Nahanni and the DCFN in conjunction with development of the road and mine, and made provision for maximizing opportunities to realize these benefits. To this end, the Agreement provides employment and contracting opportunities as well as equity participation for the Nahanni and the DCFN. The reaching of this Agreement was endorsed by Band Council Resolution and supported as well by the DCFN by Tribal Council Resolution.

In the Agreement, Nahanni proclaimed its support for the mine and the establishment of the access road in recognition of the significant benefits to Nahanni and the DCFN communities as a whole, and undertook to assist the Company in procuring permits, approvals and licences necessary to bring the mine into production, as well as grants, guarantees or other financial assistance from Government towards the establishment of the access road.

CZN believes that the all weather access road component of the mine re-development proposal provides an excellent opportunity to realize many of the anticipated benefits of the Agreement and maximizes opportunities for the DCFN at the very outset of the mine re-development, through operations and into the future. CZN has presented a series of options to the Nahanni and the DCFN for consideration for proceeding with the access road development with respect to the goals of achieving maximum benefit to the DCFN, while at the same time meeting CZN's requirements for achieving economic viability through minimizing costs to the Prairie Creek operation.

Opportunities will be created for employment, contracting and joint ventures beyond direct employment at the mine including such things as the concentrate haul, road construction and maintenance, ferry operation, expediting and freight forwarding, personnel transport and fuel supply. CZN is committed to continuing to work closely with the Nahanni and the DCFN to fulfill the provisions of the Agreement and ensure that First Nations communities in the area have ongoing input into the re-development plans for the mine.

In early 2000, a new management team took over at CZN with a view to placing the mine into production. A review of the data available on the Prairie Creek Project, plus a preliminary review of the economics of the operation suggested that a number of key components warranted re-examination by today's standards, in order to optimize operational efficiencies and therefore project economics. These included:

- Increased throughput, to maximize economies of scale,
- Increased use of mechanization both in the mine and the mill, to reduce the manpower costs of the operation,
- Reduced use of power per tonne, since the operation will be diesel power operated,
- The provision of an all weather road access to the site to ease shipping and inventory constraints and reduce shipping and road costs to the smelter,
- Reduction of the volume of tailings to the tailings pond, and required size of tailings storage facilities,
- Reduction of mercury in the concentrate to reduce smelter penalties, by increasing the volume of Stratabound mineralisation treated to reduce smelter penalties, and
- Other improvements to project economics.

The new team identified a number of key elements for profitable operation of the property including the raising of throughput to the mill to at least 1,500 tonnes per day, through the addition of a gravity pre-concentrator on the front of the mill and the inclusion of a paste backfill plant on the end of the mill to place the majority of the tailings underground. The addition of the pre-concentrator also allows the operation to handle more waste dilution and thus to mechanise underground, reducing mining costs and personnel, further aiding the project economics.

The Company recognized that these aims could be addressed in a number of different ways and in order to evaluate these various opportunities, engaged in this Scoping Study as an exercise to determine the size and extent of the improvements to project economics that could be achieved. Project staff and consultants have examined a number of different routes to reach the above targets and have identified a preferred scheme of arrangement for the mine and mill operation. The basic parameters identified include:

- The use of gravity separation to increase the throughput of the mill to at least 1,500 tonnes per day, without increasing the primary grinding mill size,
- The separation of the different flotation streams early in the mill to segregate and separate different reagent regimes and reduce reagent build up,
- The reduction in the volume of water to be treated before disposal, by the reduction in the volume of material to the flotation circuit,
- The use of paste or thickened backfill to reduce the tailings dam requirements of the operation,
- The increased use of mechanized mining, aided by the gravity plant to handle increased mining dilution,
- The increased use of automation and mechanization to reduce on site manpower,
- The construction of an all weather road, possibly in conjunction with the Nahanni Butte Dene Band, as a Deh Cho First Nations Initiative, to reduce working capital and operating costs,
- The installation of more efficient diesel generation to decrease fuel costs per tonne, and a consideration of supplemental hydro power, and
- The sequencing of mining to increase the tonnages of Stratabound mineralisation fed to the mill, thus reducing smelter penalties.

These parameters have been combined in a revised mine and mill layout and operating regime, which takes advantage of the above possibilities and potentially increases the economic returns to the shareholder. The Project team, aided by outside consultants has taken this layout and costed the operation of the mine based on a number of assumptions as to operating parameters to generate capital and operating costs and likely financial returns from the Project

Summary of Financial Performance

Because of the many variables in the financing and operation of the mine at Prairie Creek, it is difficult to accurately predict the final outcome of the operation. However, the base case modeled in this Scoping Study indicates a **break-even cash cost** of production **of US$34.5 cents per pound of saleable zinc** after by-product credits, but **before financing and taxation**. The operation will take advantage of the existing mine and mill infrastructure put in place by the Hunt Brothers in 1982, at a cost of CDN$67 million, but never operated. The replacement cost of this mine and mill can be estimated at $100 million in today's dollars.

Capital costs for the new operation are indicated to range from **CDN$40.5 million** to CDN$21.7 million depending on the final configuration of the operation and the construction of an all weather, or winter only, access road. A number of additional upside scenarios exist for the operation and these will be examined further during the feasibility study process.

The base case financial model indicates that the operation at a capital cost of CDN$40.5 million would have a **pre-tax and financing IRR of 45.6%** and an **NPV at a 10% discount rate of CDN$97.2 million dollars** over the first ten years of a minimum 18 year mine life. The study used long term metal prices of $0.90 per lb – Cu, $0.50 per lb – Zn, $0.25 per lb – Pb and $5.50 per ounce Ag. The Canadian dollar was kept constant at US$0.66. On this basis and with the mine capable of producing just over 95 million pounds of payable zinc per year; for every cent the Zn price is over the break-even production cost of US$ 34.5 cents per pound, pre tax and financing cash flow increases by around US$0.64 million per annum.

A number of significant upside opportunities have been identified which will further enhance project returns. These opportunities require additional work as part of the ongoing feasibility study, however for the purposes of this study they are summarised below:

Table 1 – Scoping Study Variables

Item	**Capital Cost in CDN$**	**Potential for Success**	**Change in Production cost per lb of Zinc in US cents**
Basic operation at 550,000 diluted tonnes per year	$40 million inc. all weather road and contingency	High, subject to Feasibility	34.5 cents per lb break even cash cost including smelter participation all royalties etc.
Operate mine and mill over winter road	Decrease in Cappex by $18 million,	Requires large increase in working Capital	Increase in cash cost of metal of around 1-2 cents
Operate mine for 8 months a year over winter road	Decrease of Cappex by at least $18 million	Requires much smaller increase in working capital	Increase in cash cost of metal of around 1-2 cents
Increase Grade from mine	Nil, high grade from existing development	High, Much higher grades available	Reduces cash cost by 3.6 cents for 10% increase
Install crusher and belt below Stratabound	$3-5 million	High, but adds to capital cost	Reduces cash cost by 1.4 cents US per lb Zinc
Increase rejection in gravity plant and increase throughput to 615,000 tonnes per year	No cost	High, as above	Reduce cash cost by 1.4 cents per lb Zinc
On site treatment of Copper Conc. For Ag and Hg	Around $10 million	Unknown, requires further testwork	2 – 4 cent reduction in cash cost dependent on products made
Purchase of Titan royalty	Say $1-2 million	High	0.75 cent reduction in cash cost.
Increase percentage of Vein to mill	Nil.	Simple for first two years then more difficult	Minor increase in Vein %age reduces cash cost by 1 cent

Decrease cost of Vein mining	Nil if by contractor	Good, but depends on orebody configuration	Un-quantifiable
Install hydro generation	Approximately $6 million.	Depends on suitable water source	0.95 to 1 cent per lb reduction in cash cost of Zinc

If a low cost entry-level mine is required, the possibility of starting the mine over a winter road with seasonal working should not be ignored. In this scenario, a smaller mine could be put into production for around $20 million, within 3 months of establishing the winter road.

All cost figures should be considered order of magnitude estimates (+25% -15% order of accuracy) and will require verification by more detailed study and pilot scale operation in order to convert this Scoping Study into a bankable Feasibility Study. **Please note that all references to dollars are to Canadian dollars unless otherwise specified.**

It should be noted that the Economic Assessment in this Scoping Study is preliminary and partially based on Resources that are considered too speculative to be categorized as Reserves in accordance with National Instrument 43-101. In addition, the Scoping Study is preliminary in nature and despite the existing underground development in the ore body and the on-site mill; the assumptions made within the Scoping Study and its subsequent results may not be attained. It is for this reason that the Company has put forward a development program designed to lead to a full bankable feasibility study and obtaining of the applicable permits, prior to final development taking place.

It is the Company's intention to carry out work in 2001 to increase the confidence levels in the above data through a program of Pilot Plant operation on site, geological and geotechnical examination and delineation drilling. In order to continue with the development of this project and to provide certainty to the future funding partners, or banks and other stakeholders in the project, it is also necessary to complete a bankable Feasibility Study to confirm the final mine design, metallurgical and financial performance, etc. and allow the construction of an operation capable of generating a reasonable return to shareholders and to support significant levels of debt funding.

The Study was examined by Micon International Ltd of Toronto, who confirm that all of the elements necessary for a scoping study have been incorporated by the Company and that the assumptions made within are considered reasonable for a study of this nature.

The Company is now working to examine the various alternatives outlined within the Scoping study and to convert this into a bankable feasibility study. As part of this program, it is the Company's intention to carry out a Pilot Plant program within the existing essentially complete 1000-ton per day mill, to carry out additional delineation drilling and to drive a decline to provide a drilling platform and to access ore below the current 5 Km of underground workings. The Company also continues to work on the re-permitting of the operation, which had a full permit in 1982, but never operated.

Based on the above figures, it appears that a mine at Prairie Creek could have a long and profitable lifespan, with current resources giving a mine life in excess of 18 years. When combined with the exploration potential for the area, this could lead to the development of a significant mining camp and a major profit center for CZN, the Nahanni Butte Dene Band, Northwest Territories and the Deh Cho First Nations.

Figure 1 – Effect of Changed Economic Variables on Project Return

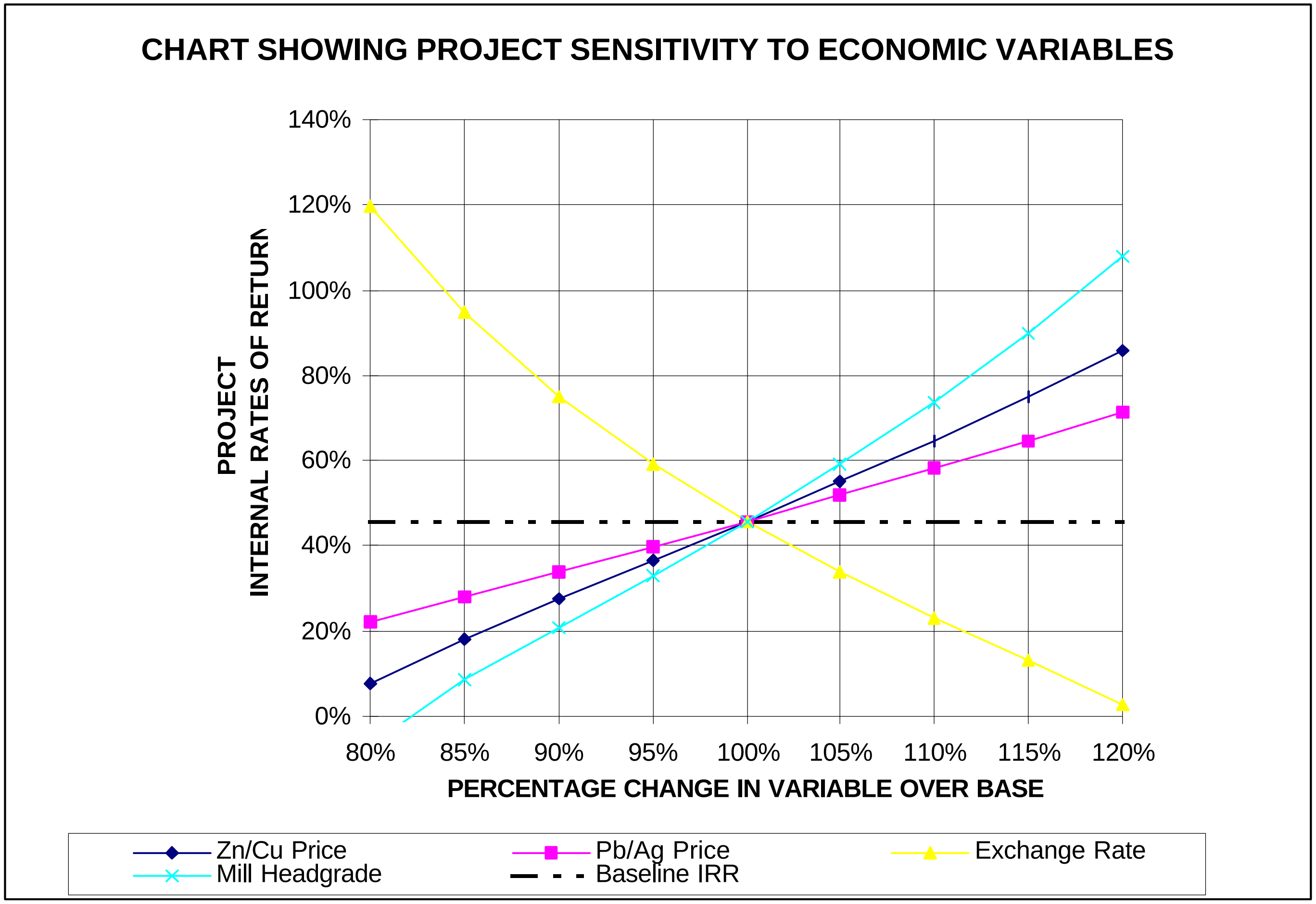
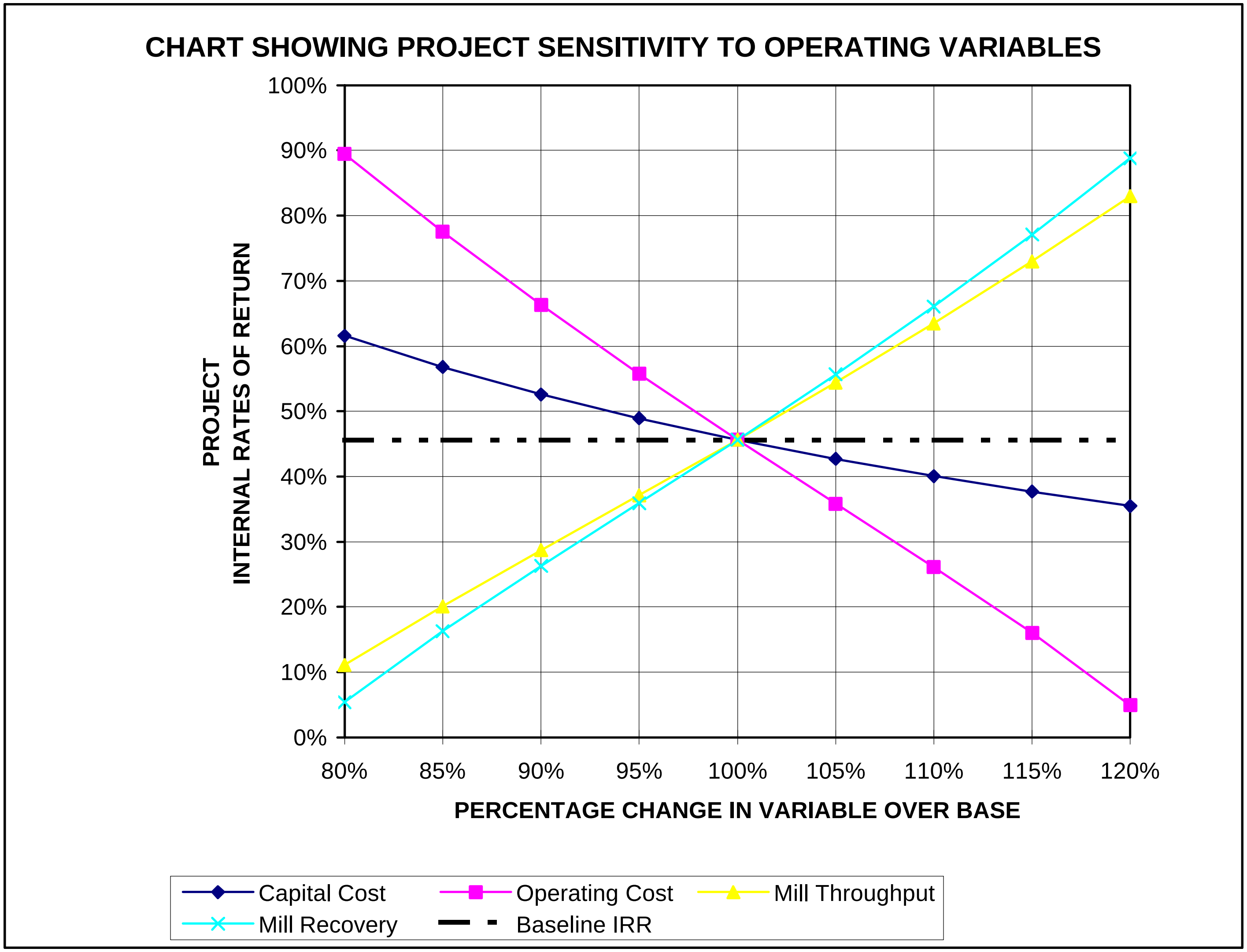


Figure 2 – Effect of Changed Operating Variables on Project Return



January 29, 2001

Letter from Micon International

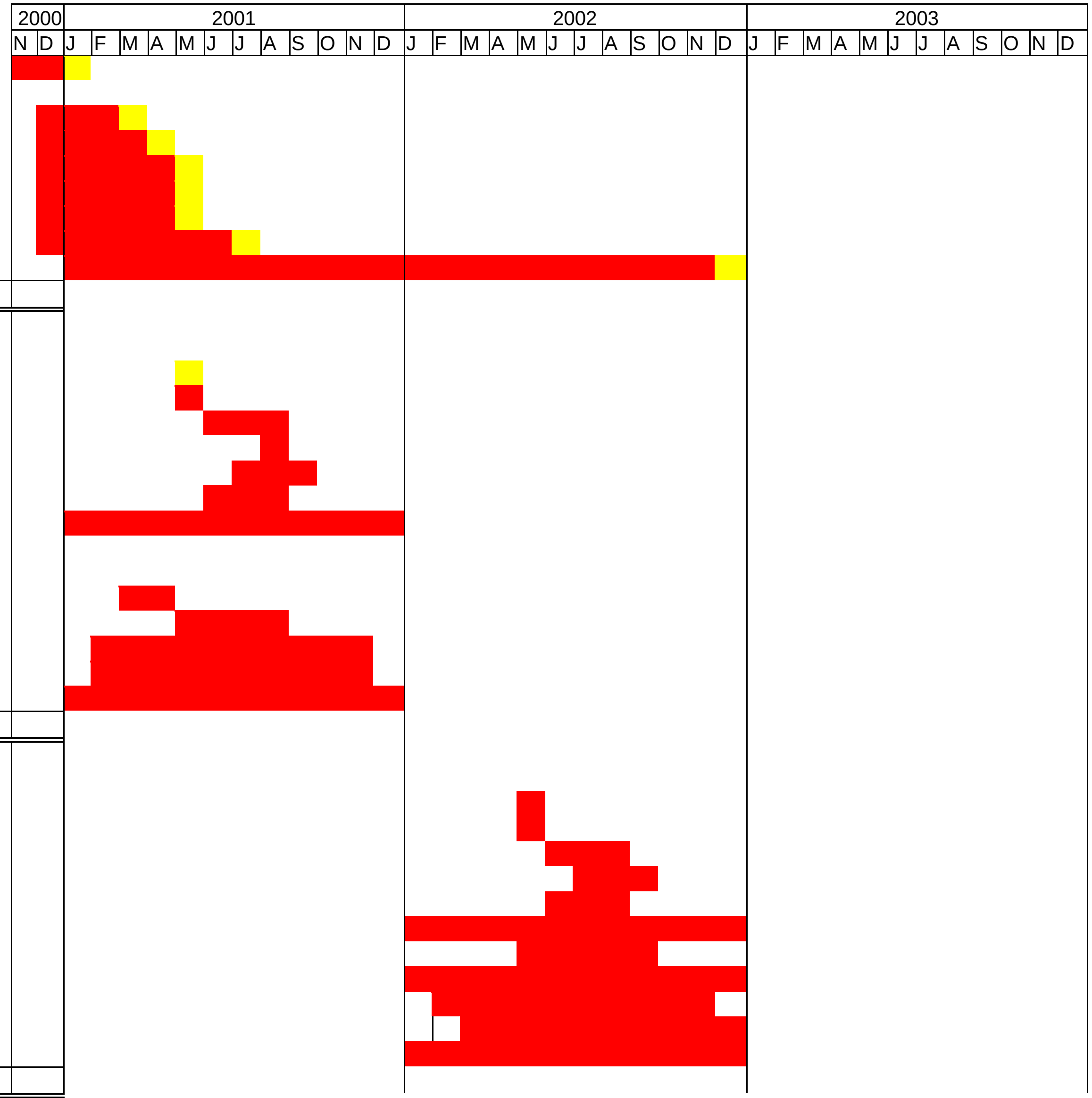
Please refer to Micon Letter.pdf

Filed with this report

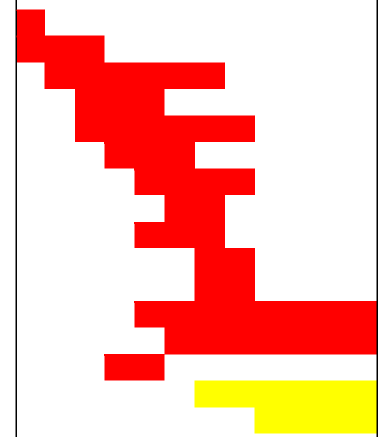
Figure 3 – Prairie Creek Project Timetable

Prairie Creek Project Timetable

Item	Cost ($1000)	Duration Months
Scoping Study	$100	3
Permits		
Permit for 2000 drill programme	$2	4
Permit for reclamation of Cat Camp	$2	5
Permit for Pilot Plant Operation	$10	6
Permit for Decline Development	$10	6
Permit for Surface delineation Drilling	$10	6
Permit for Exploration of rest of site	$15	8
Application for water and operations permit for Mine		24
Total Permits in 2000	**$49**	
2001 Programme		
Opening of Camp for 2001	$10	1
Preparation of equipment for Decline Drivage	$100	1
Driving 600 m of decline	$1,100	3
Cross cut 70 m to vein	$120	1
Underground delineation drilling	$500	3
Surface Delineation Drilling + 2000 programme	$500	3
Environmental Studies	$1,000	12
Geotechnical Studies	$250	4
Other permitting studies & consultation	$750	12
Pilot Plant fabrication and commissioning	$250	2
Pilot Plant Operation	$300	4
Feasibility Studies	$300	10
Permitting	$75	12
Head Office and administration	$500	12
Total 2001 Programme	**$5,755**	
2002 Programme		
Opening of Camp for 2002	$10	1
Preparation of equipment for Decline Drivage	$100	1
Deepen decline by driving 300 m	$500	3
Underground delineation drilling	$200	3
Surface Drilling programme	$200	3
Environmental Studies	$200	12
Geotechnical Studies	$150	4
Other permitting studies & consultation	$250	12
Financing Studies	$200	10
Permitting	$50	12
Head Office and administration	$500	12
Total 2002 Programme	**$2,360**	



2003 Programme		
Receive Permit to construct ice road		
Construct & operate Ice Road	$500	3
Construct Permanent road and airstrip	$17,000	6
Construct barge landings and import barge	$1,200	3
Commence UG Development	$5,000	6
Import new milling gear, plus construction material	$5,000	3
Rehabilitate Tailings pond etc	$3,000	4
Extend mill building	$1,000	2
Complete rehabilitation of mill	$5,000	3
Construct concentrate stores	$1,000	2
Construct load out at railhead	$500	2
Commence underground mining		8
Drive decline towards Stratabound		7
Upgrade Camp facilities	$750	2
Commence mill production		
Ship first concentrate		
Permitting	$50	12
Head Office and administration	$500	12
Total 2003 Programme	**$40,500**	



Various Minesite Photos

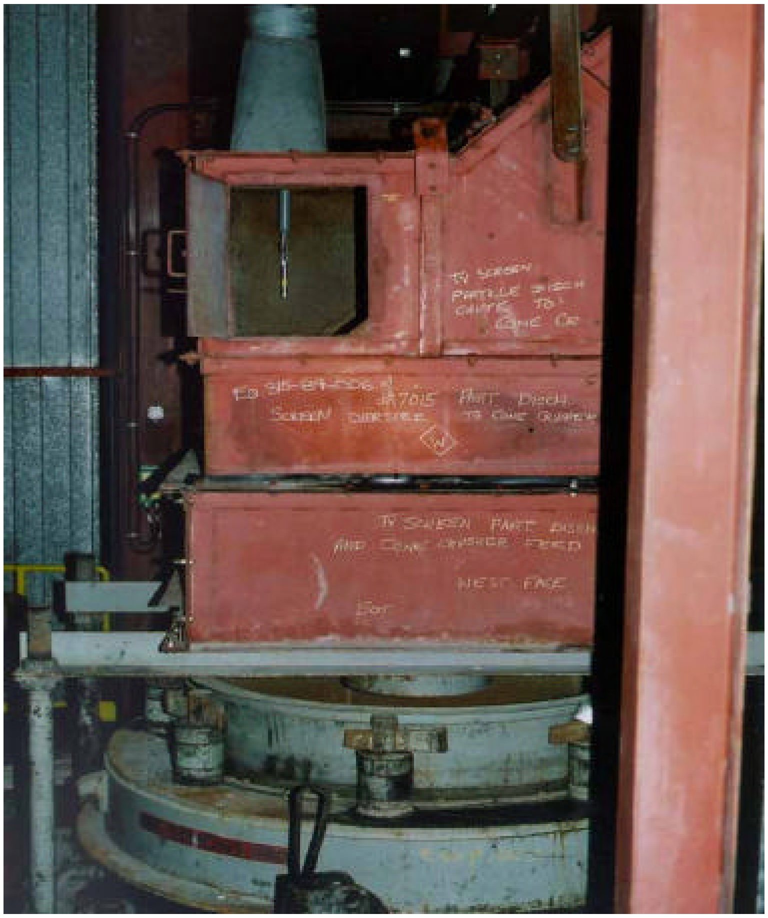
Assay Office – N.E. Side of Mill

Coarse Ore Conveyor Gallery – View Towards Shops and Camp

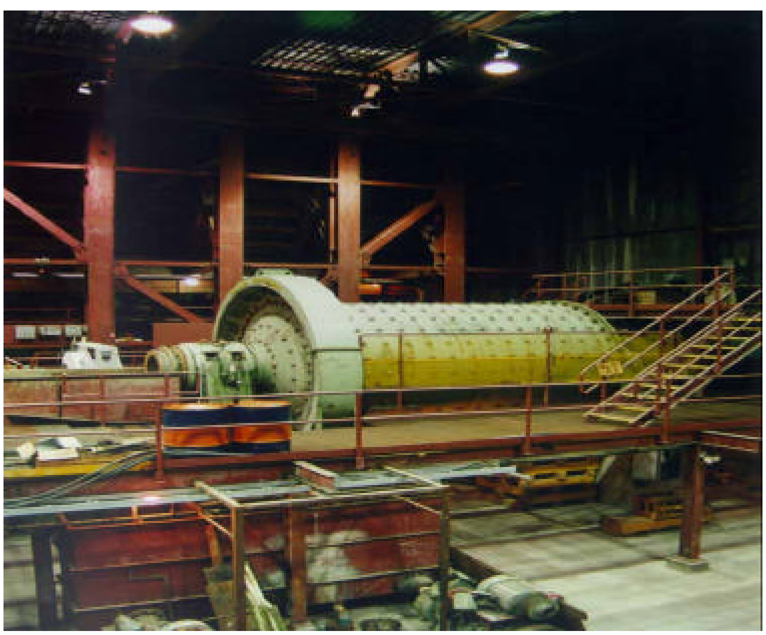
Mill Building from South East

Mill Building from South

Metal Detector & Conveyor (Crushing Plant)



5 ½ Foot Short Head Crusher



700 HP Main Mill

Mill Control Room

1150 HP Cooper Bessiner Unit (1 of 4)

Interior of Main Mill Showing Reline

Larox Pressure Units (Mill Basement)

Geho Pump (Mill Basement)

Miners quarters

President & CEO Malcolm Swallow at the stratabound discovery hole

Company staff speaking with Alan Taylor, V.P. Exploration at #2 portal

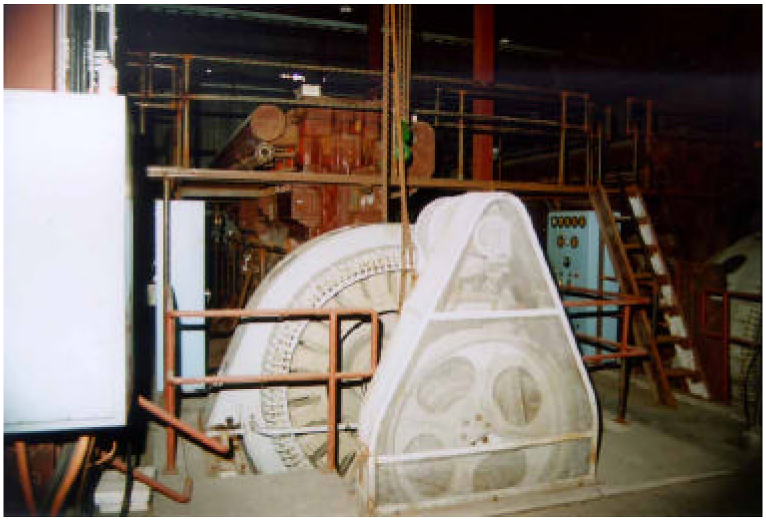
Old Winter Access Road

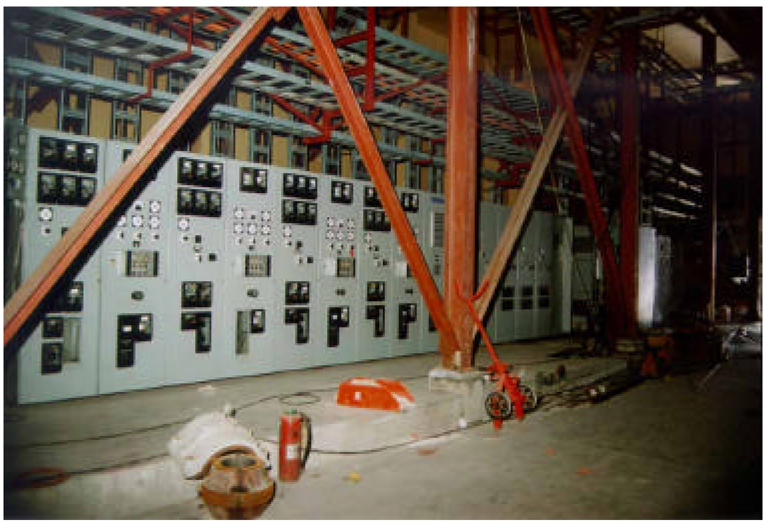
Lower Portion of Access Road Near Mine


Mill Spares in Basement


Mill Spares in Basement


Mill Spares in Basement


Power House – 1150 kw Cooper Bessemer Diesel Generator


Power house – Motor Control Center


Drafting Area in Office Building


Plant Site – Crane and Mine Timber Supply


Plant Site - Mill Supplies and assay Office Equipment, Camp in Background


Typical Conditions Underground, 930m Level

Typical Conditions Underground, 930m Level

Timber Service raise 930m-870m

Typical High Grade Intercept

35’ Thickener

’35 Thickener with Tailings Cyclones in Fore Ground

Ball Mill, View from Mill Control Room

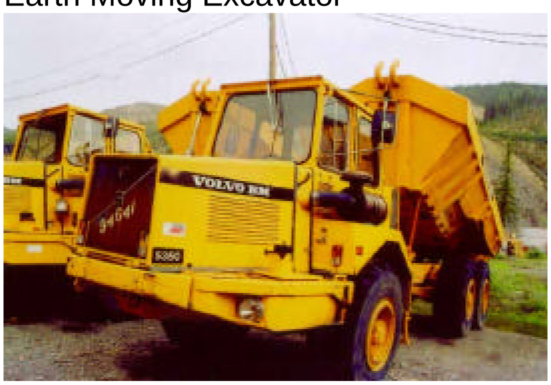
75 HP Regrind Mill

Prairie Creek Airstrip

Mill Yard with Mobile Equipment & Office Building

Underground Scoop Tram

Earth Moving Excavator



Articulated Ore Trucks

Earth-moving Equipment

Portable Cedar Rapids Crusher - Located Beside Airport Runway

Portable Sand Wash Plant – Located Beside Airport Runway

Fuel Tank Farm

PRAIRIE CREEK PROJECT SCOPING STUDY

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1. INTRODUCTION

1.1 ACKNOWLEDGEMENTS

Throughout its recent history, a number of partial studies have been carried out on the Prairie Creek Project aimed at capitalizing on the $100 million legacy of plant and equipment left behind by the Hunt brothers. These studies and the work carried out by the new management since June of 2000 have resulted in a significant data base of information, which requires the preparation of a scoping study to more accurately define the potential of this spectacular deposit and to focus the work required to bring the Project from vision to bankable reality.

For this reason the Company has prepared this Scoping Study to best examine the likely size and style of future mining operations at Prairie Creek. This Scoping study will form the basis for future development of the deposit and allow more accurate focusing of work on the Project, to bring the concepts developed in the Scoping Study to the practical and financial reality of a bankable feasibility report over the next year. The Scoping study will identify areas that require additional work to provide certainty to the owners, future investors and regulatory authorities, that a profitable mine can be operated on the site in a practical and environmentally responsible manner.

The current organization of the Company is based around a small core of highly experienced mining industry professionals, supported by independent consultants and contractors as required. As a consequence a significant number of people and Companies have been involved in preparation of this Scoping Study, concentrating on their own areas of expertise. In order to produce a sensible, readable and realistic report, these separate elements were drawn together in-house and copies of the background reports are appended or available for inspection as appropriate.

On the completion of the report, Micon International Limited, mineral industry consultants of Toronto, were invited to carry out an independent examination of the complete study to confirm veracity and technical correctness of the Scoping Study's results.

Canadian Zinc Corporation would like to take this opportunity to thank all the people and companies involved, in advance, for their efforts which were above and beyond the call of duty and to acknowledge the efforts of the following individuals and companies in the preparation of this report:

- Malcolm J.A. Swallow, B.Sc. (Hons) Min Eng., C.Eng, FIMM, President & CEO Canadian Zinc Corporation, Mining Engineer;
- Alan B. Taylor, M.Sc., B.Sc. Geology, P.Geo., Vice President – Exploration, Canadian Zinc Corporation, Geologist;
- J. Peter Campbell, B.Sc. Biology, Vice President – Project Affairs, Canadian Zinc Corporation, Biologist;

- Paul Sterling, B.ApSc. Chemical, P.Eng., Vice President, Moonray Corporation, Mineral Technologist;
- Gary W. Hawthorn, B.Sc., P. Eng., Principal, Westcoast Mineral Testing Inc., Mineral Processing Engineer;
- Gordon Parkin, B.Sc. Min Tech., Director, North East Minerals Ltd., Mineral Technologist;
- Ernst Binder, P.Eng., Project Manager, Procon Mining and Tunnelling Ltd., Senior Mining Engineer;
- Mark Tindall, B.Sc., P.Geo., Principal, Tindall Geoservices Inc., Consulting Geologist;
- Alastair Findlay, M.Sc., P.Geo., Consulting Geologist;
- Rick Wilson, Consultant, Project Manager, UEE - Unit Electrical Engineering Ltd.;
- K. Wayne Savigny, Ph.D., P.Eng., P.Geo., Vice President, BGC Engineering Inc., Geotechnical Engineer/Engineering Geologist;
- Mitchell Zulinick, General Manager Interior Operations, Arrow Transport Systems, Transportation pricing and alternatives; and
- Harry Burgess, B.Sc. (Hons) Min Eng., P.Eng., Vice President, Micon International Ltd., Mining Engineer.

In addition the Company would like to thank the innumerable unnamed people who have helped in every way to prepare this report and to acknowledge their efforts on the Company's behalf. We appreciate all that you have done for us so far and look forward to imposing on you in the not too distant future as the Project progresses.

Finally, the Company must acknowledge the heroic efforts of Rhonda Schultz and Kathleen Hancock in the typing, correlating and ordering of this report, without whose discipline and determination we would still be arguing about the titles, spelling and format.

Malcolm Swallow
Canadian Zinc Corporation
Vancouver, January 2001

1.2 PROJECT BACKGROUND

Prairie Creek Mine, located in the Northwest Territories of Canada, contains a significant Zn-Pb-Ag-Cu in situ resource, which has been the focus of exploration since the early 1900's. In addition to this mineral resource, over $100 million of mine related infrastructure has been placed on site along with significant exploration and research resulting in the establishment of a large database of information.

Although fully permitted in 1982 (since lapsed,) the mine has never achieved actual production. The primary objective of CZN, which owns 100% of the Prairie Creek Mine, is to place the mine into production in the near future.

1.3 LOCATION AND ACCESS

The Prairie Creek Mine is located in the South Nahanni Mining District of the Northwest Territories near the Yukon border, at 61°33' N latitude and 124°48' W longitude. The nearest communities include Nahanni Butte 90 kilometers southeast, Fort Liard 170 kilometers south, Fort Simpson 185 kilometers east, and Yellowknife 550 kilometers east, all in the Northwest Territories. The town of Fort Nelson, British Columbia, is located 340 kilometers south and is the nearest point of access to B.C. Rail.

The minesite is located on the east side of Prairie Creek, approximately 48 kilometers upstream from its confluence with the South Nahanni River. The Nahanni National Park Reserve boundary lies 32 kilometers downstream of the mine.

Year round access is presently by charter aircraft, generally from Fort Nelson or Fort Simpson, which are serviced by scheduled commercial airlines. The mine is accessed by a 1,000 meter gravel airstrip that is located on the flood plain of Prairie Creek, approximately 1 kilometer north of the site. Previous access in the 1980's was via a 170 kilometer long winter road from the Blackstone crossing on the Liard Highway. This road was utilized to transport the bulk of the building materials and equipment into the site and is still in reasonable condition and capable of early re-activation.

1.4 CLIMATE

The minesite is at an elevation of 850 meters above sea level and is situated in topography characterized by low mountains and narrow valleys with an average relief of 300 meters. Short summers and long winters are typical of the area's sub-arctic climate, where the mean annual temperature is -5°C. Annual precipitation is approximately 40 cm, most of which falls as rain. The minesite area is located within the Alpine Forest-Tundra section of the Boreal Forest characterized by stunted fir and limited undergrowth.

Figure 4 - Property Location Map

Over Size photo.

If you would like a copy, please contact the Company at 1-866-688-2001 toll free or 604-688-2001.

2. CLAIMS AND TENURE

CZN owns 100% of the Prairie Creek Mine and Property. A 2% net smelter royalty is held by Titan Pacific Resources Limited which is presently held as a 40% salvage right to the plant and equipment until payout which is capped at $8.2 million. Upon full payment the royalty will be terminated. The main land holding comprises eight mining leases and two surface leases. The Company also holds seven additional mineral claims. Details of the Project tenements are shown in the Table below and on Figure 5; geology and claims.

Table 2 – Prairie Creek Property Claims Listing

Property Type	Claim #	Lease/Claim Name	Area Ha	Area Acres
Mineral Claims				
Claim	F22751	SAN 4	1,003.30	2,479.20
Claim	F22752	SAN 5	1,003.30	2,479.20
Claim	F22753	SAN 6	701.40	1,733.20
Claim	F67134	GATE 1	731.59	1,807.75
Claim	F67135	GATE 2	1,003.30	2,479.20
Claim	F67136	GATE 3	1,003.30	2,479.20
Claim	F67137	GATE 4	1,003.30	2,479.20
Claims total			*6,449.49*	*15,936.95*
Surface Leases (held in Over holding Tenancy)				
Surface Lease	95F/10-5-3	Minesite	113.60	280.74
Surface Lease	95F/10-7-2	Airstrip	18.20	45.07
Surface Lease total			*131.80*	*325.81*
Mining Leases				
Mining Lease	ML 2854	Zone 8-12	743.00	1,835.99
Mining Lease	ML 2931	Zone 4-7	909.00	2,246.18
Mining Lease	ML 2932	Zone 3	871.00	2,152.28
Mining Lease	ML 2933	Rico West	172.00	425.02
Mining Lease	ML 3313	Samantha	420.05	1,037.96
Mining Lease	ML 3314	West Joe	195.86	483.99
Mining Lease	ML 3315	Miterk	43.70	107.98
Mining Lease	ML 3338	Rico	186.16	460.00
Mining Leases total			*3,804.35*	*9,401.02*
Grand Total			***10,253.84 Ha***	***25,337.97 Acres***

3. FIRST NATIONS PARTICIPATION

The Prairie Creek Mine is located on land claimed by the Nahanni Butte Dene Band (the "Nahanni") of the Deh Cho First Nations ("DCFN") as their traditional territory. The DCFN are engaged in ongoing negotiations with the Government of Canada and the Government of the Northwest Territories in what is referred to as the Deh Cho Process. The negotiations are currently at the Interim Measures and Agreement In Principle stage. The outcome of the negotiations is expected to be a Final Agreement that will provide, amongst other things, for the implementation of a Deh Cho form of government to oversee the delivery of programs and services to residents within the DCFN territory. It is expected that the negotiations will take some five to seven years to complete.

3.1 PRAIRIE CREEK DEVELOPMENT COOPERATION AGREEMENT

In 1996, the Company and the Nahanni successfully negotiated and executed the Prairie Creek Development Cooperation Agreement (the "Agreement"). The overall intent of the Agreement was to establish and maintain a positive and cooperative working relationship between the Company and Nahanni in respect of the further development and operation of the mine, while at the same time supporting an economically viable and environmentally sound operation and maximizing economic opportunity and benefits to Nahanni and other Deh Cho First Nations. This Agreement foresaw the many benefits which could accrue to the Nahanni and the DCFN in conjunction with development of the road and mine, and made provision for maximizing opportunities to realize these benefits. To this end, the Agreement provides employment and contracting opportunities as well as equity participation for the Nahanni and the DCFN. The reaching of this Agreement was endorsed by Band Council Resolution and supported as well by the DCFN by Tribal Council Resolution.

In the Agreement, Nahanni proclaimed its support for the mine and the establishment of the access road in recognition of the significant benefits to Nahanni and the DCFN communities as a whole, and undertook to assist the Company in procuring permits, approvals and licences necessary to bring the mine into production, as well as grants, guarantees or other financial assistance from Government towards the establishment of the access road.

Some specific considerations as set out in the Agreement pertaining to road development (the "Access") and opportunities relating thereto are as follows:

- Nahanni recognizes the all weather road and ferry are required and beneficial to both parties, and fully supports establishment of the Access
- Nahanni will grant an easement for the Access
- Nahanni will provide all necessary assistance in obtaining and maintaining permits to establish the Access
- Nahanni will use its best efforts to procure maximum financial assistance for the Access

- Nahanni shall enjoy preferential access to economic opportunities including open book negotiated contracts
- CZN shall have a minimum target of 20% employees from DCFN communities
- CZN shall require non-First Nation contractors to have a target of not less than 20% employees from DCFN communities
- Nahanni will receive a 5% equity interest of profits before taxation, but after recovery of prior capital and development costs
- Nahanni will be granted an option to purchase either a 10% or 15% working interest in the Project for $6 or $9 million, inflation adjusted on completion of a Feasibility Study, but before construction
- Nahanni will grant a 5% equity interest to CZN in all tourism ventures which make use of the airstrip or access road
- Nahanni will deliver to CZN 50% of any cost savings arising as a consequence of their participation

3.2 DCFN JOINT VENTURE OPPORTUNITIES

CZN believes that the all weather access road component of the mine re-development proposal provides an excellent opportunity to realize many of the anticipated benefits of the Agreement and maximizes opportunities for the DCFN at the very outset of the mine re-development, through operations and into the future.

While all weather mine access road construction could be pursued by CZN in the traditional manner; that is applying for a land use permit and upon issuance contracting out construction to build the road, CZN believes there is an opportunity to approach road development either jointly with or solely as a **Deh Cho Initiative**. From CZN's perspective, a direct DCFN involvement in any road development may provide access to funding not readily available to the Company, thereby improving economic projections by reducing the capital costs associated with bringing the mine into production. From the DCFN's perspective, development of a DCFN all weather road would create opportunities for capacity building and introduce the First Nations to significant long-term employment and economic benefits. As well, other opportunities, beyond those immediately available through development and operation of the mine, are evident in the form of tourism and other resource ventures.

Through construction and operation of the road as a Deh Cho Initiative, the DCFN would maintain a direct (joint or independent) interest in the road from the beginning and over the longer term, and would also benefit from other opportunities associated with the presence of the road (i.e. tourism, oil and gas). As a Deh Cho Initiative, the DCFN would control access onto the road and into their traditional territory by virtue of the ferry crossing, thereby eliminating one of the common concerns with respect to road development. Such an arrangement would require an agreement between CZN and the DCFN which would set respective responsibilities, assure availability of the road for use, determine appropriate user fees, etc.

CZN has presented a series of options to the Nahanni and the DCFN for consideration for proceeding with the access road development with respect to the goals of achieving maximum benefit to the DCFN, while at the same time meeting CZN's requirements for achieving economic viability through minimizing costs to the Prairie Creek operation. These options are:

1. **CZN road**
   - CZN would include the all weather road in its development proposal, acquire the necessary permits, construct the road and purchase the ferry at its own cost
   - CZN would maintain the road and operate the ferry at its own cost
   - Business and employment opportunities in respect of the construction and maintenance of the road, and operation of the ferry would fall to the DCFN as per the Agreement

2. **CZN - DCFN Joint Venture Road**
   - DCFN would pursue and acquire maximum possible funding through Indian and Northern Affairs Canada, the Government of the Northwest Territories, Treaty negotiations (Interim Measures Agreement) or other available funding mechanisms for road and ferry
   - Funds so acquired would be applied towards Nahanni's (or DCFN's) purchase of equity participation in the operation as provided for under the Agreement
   - Development of the road would be pursued by the Joint Venture; costs would be shared on a pro rata basis
   - DCFN would maintain the road and operate the ferry through a development corporation or JV
   - Operating/maintenance costs would be shared by the JV on pro rata basis

3. **DCFN Road**
   - Road and ferry to be taken on as a Deh Cho Initiative in its entirety
   - DCFN would pursue and acquire maximum possible funding
   - CZN would fund shortfall
   - DCFN to submit development proposal; obtain permits; construct, operate and maintain road
   - CZN to pay a user fee to cover operating costs and appropriate level of profit; user fee to be reduced at outset subject to payback of CZN funds injected; user fees to be charged to any other road users on a pro-rata basis
   - A Road User Agreement setting out respective responsibilities would have to be negotiated and agreed upon up front

Opportunities will be created for employment, contracting and joint ventures beyond direct employment at the mine including such things as the concentrate haul, road construction and maintenance, ferry operation, expediting and freight forwarding, personnel transport and fuel supply.

Other opportunities under discussion include a joint venture wilderness tourism operation, in the area of the Ram Plateau, utilizing the new access road and proposed Sundog airstrip.

CZN is committed to continuing to work closely with the Nahanni and the DCFN to fulfill the provisions of the Agreement and ensure that First Nations communities in the area have ongoing input into the re-development plans for the mine.

4. GEOLOGY AND MINERALISATION

4.1 REGIONAL

The Prairie Creek Mine is located in the southern portion of the Mackenzie Mountains physiographic subdivision within the Northern Cordillera Geosyncline. The Southern Mackenzie Mountains are underlain by Lower Paleozoic carbonates of the Mackenzie shelf, and associated basinal limestones, dolostones and shales.

Structurally the prevalent orientation of faulting and folding is north-south. Faults and fold axial planes dip both east and west. A number of north trending thrust faults cut through the region. The east dipping Tundra Thrust Fault (located on the Prairie Creek Property) and, 30 kilometers to the west, the west dipping Arnica Thrust Fault define the present margins of the Prairie Creek paleobasin, which accumulated a thick Devonian sequence of sediments (including the Cadillac and Funeral Formations). The two principal styles of base metal sulphide mineralisation occur within the Prairie Creek basinal feature in the Ordovician-Silurian Whittaker and Road River Formations.

4.2 PROPERTY GEOLOGY

The northern part of the property is underlain by a marginal carbonate sequence of the Root River, Camsell and Sombre Formations. This sequence is bounded to the west by the east dipping Tundra Thrust, which forms the eastern boundary of the Prairie Creek basinal sequence.

In the southern part of the property the mine is geologically situated on the eastern margin of the Prairie Creek Embayment (Morrow D.W. and Cook D.G., 1987). This ancient basinal feature is composed primarily of a conformable sequence including the Lower Ordovician Whittaker Formation dolostones, Silurian Road River Formation shales, and Cadillac Formation thinly bedded limy shales. Lower to Middle Devonian Arnica and Funeral Formation dolostones and limestone overlie this assemblage on the northern part of the property.

In the southern part of the property faulting and folding axis trend generally north-south, resulting in windows of older Road River shales cored by the Whittaker Formation dolostones being exposed along the core of the main Prairie Creek anticline. The Prairie Creek anticline is structurally bounded to the east by the PC Fault and to the west by the Gate Fault.

4.2.1 Property Base Metal Mineralisation

There are three main styles of base metal mineralisation so far located on the property: Vein sulphides, Stratabound sulphides (both of which occur in the southern part of the property) and Cavity infill sulphides (MVT) which are found in the marginal carbonates in the northern sector of the property. Exploration at Prairie Creek has revealed many base metal mineral showings along the entire 27 kilometer length of the property. Historical exploration of the property has led to the referencing of these surface mineral

Figure 5 - Geology, Showings & Land Holdings

Over Size photo.

If you would like a copy, please contact the Company at 1-866-688-2001 toll free or 604-688-2001.

Figure 6 – Main Zone Plan View

Over Size photo.

If you would like a copy, please contact the Company at
1-866-688-2001 toll free or 604-688-2001.

Figure 7 – Main Zone Longitudinal Section

Over Size photo.

If you would like a copy, please contact the Company at
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Figure 8 – Cross Section 50,325 North

Over Size photo.

If you would like a copy, please contact the Company at 1-866-688-2001 toll free or 604-688-2001.

Figure 9 – Cross Section of SD1 Massive Sulphide Zone

Over Size photo.

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showings by name (refer to Figure 5). In the northern section of the property MVT type showings occur and are referred to, from north to south over a distance of 10 kilometers, as the Samantha, Joe, Horse, Zulu, Zebra and Road Showings. In the southern part of the property, where Vein occurrences were exposed at surface, the mineral showings were referred to as sequentially numbered Zones. Some of these mineral Zones, as a result of recent exploration by drilling, are now known to also contain Stratabound mineralisation in subsurface. The subsurface area above the underground workings is referred to as Zone 3. Originally Zones 1 and 2 occurred adjacent to Zone 3, however, as a result of continuing exploration, Zone 1 and 2 are now incorporated and considered part of Zone 3. A further expression of Vein mineralisation, known as the Rico Showing, is located 4 kilometers to the north of Zone 3. Zone 3 contains the present estimated mineral resource which includes both Vein and Stratabound mineralisation. Extending south from the minesite (or Zone 3) are a series of other Vein exposures referred to as Zones 4 through 12, extending over a distance of 10 kilometers.

4.2.1.1 Vein Sulphides

Quartz Vein Sulphide mineralisation occurs in a north-south trending 16 kilometer long corridor in the southern portion of the property (referred to as Zones 1 through 12). The bulk of the mineral resources outlined to date on the property are established on only one of these Vein occurrences, namely Zone 3 (which includes Zone 1 & 2).

The vein in Zone 3 strikes approximately north and dips steeply to the east (variable from -40°E to -90° and averages -65°E). Most of the surface mineralised zones at Prairie Creek occur within Road River Formation shales. These showings generally occur close to the axial plane of a tight north-south doubly plunging anticline. Mineralisation comprises galena, sphalerite, lesser pyrite, tennantite-tetrahedrite as massive to disseminated in a quartz-carbonate-dolomite matrix. Silver is present in equal amounts both in galena and tennantite-tetrahedrite. Vein widths are variable (from <0.1m to >5m) but overall averages indicate a horizontal thickness of approximately 2.7 meters. The most extensive known Vein occurrence is in Zone 3 where underground development has proven 940 meters of strike length and diamond drilling has indicated a continuance of the vein for a further 1.2 kilometers. The vein remains open to the north and is expected to continue at depth for a further 4 kilometers. Evidence of the vein continuing is the Vein occurrence at the Rico Showing on surface 4 kilometers north on strike of the vein. In Zone 3, the vein appears to be a tensional fault feature co-planar to a tight N-S trending fold axis.

At the end of the 930 meter level workings, the main vein dissipates into the mid Road River shales. Rock competency appears to be a controlling feature on formation of the vein hence in the upper shales of the Road River and Cadillac Formations the vein is not well structured. Drilling at depth has indicated a continuance of the vein however little information is available below the 600 meter elevation mark.

Also of note towards the end of the 930 meter level workings (crosscut 30) are a series of narrow (average 0.5 meter wide) massive sphalerite-tennantite veins striking at approximately 40° to the main Vein trend. This zone is referred to as Vein Stockwork and carries a calculated mineral resource based on underground sampling and limited diamond drilling. It is proposed that the Stockwork system formed as tensional openings formed by primary movement along the Vein fault structure.

4.2.1.2 Stratabound Sulphides

Stratabound mineralisation was discovered in 1992 while drilling to extend Vein resources at depth. So far indications of Stratabound mineralisation have been found by drilling along the trend of the Prairie Creek Vein System over a strike length of more than 3 kilometers. This type of deposit has so far been located by drill holes in Zones 3, 4, 5 and 6 (Findlay, October 2000). Stratabound massive sulphides occur largely within a mottled dolostone unit of the Whittaker Formation close to both the Vein system and the axis of the Prairie Creek anticline.

Stratabound sulphide mineralisation has now been identified in three stratigraphic horizons of the Upper Whittaker Formation. Mineralisation consists of sphalerite-pyrite-galena, totally replacing the host dolostone with little apparent alteration. Apparent thicknesses of the Stratabound zone of up to 28 meters have been drill intercepted. Stratabound mineralisation is generally fine grained, banded to semi massive, consisting of massive fine grained sphalerite, coarse grained galena and disseminated to massive pyrite. This type of sulphide mineralisation appears to be genetically related to the Vein mineralisation, however it is different in its mineralogy and structural setting.

There are presently no underground workings that intercept Stratabound material. The main drill defined Stratabound deposit occurs 200 meters below the 870 meter level at the minesite.

4.2.1.3 Cavity Style Sulphides

Cavity-style infilling sulphide mineralisation is located at the Zebra showing which is one of the southern showings in a belt that extends for 10 kilometers to the north and includes the Zulu, Joe and Samantha showings. Mineralisation is comprised of colliform rims of sphalerite, brassy pyrite and minor galena with or without later dolomite infilling. Mineralisation is hosted within the Root River Formation.

Mineralisation occurs discontinuously at approximately the same stratigraphic horizon along this NNW trend. This sulphide appears to be classic Mississippi Valley Type mineralisation occurring in open cavity type settings.

This style of mineralisation is similar to some of the deposits mined at Pine Point, Northwest Territories. Since these showings occur in a more remote location of the property and are somewhat lower grade they have not been the focus of any major exploration.

Plate 1 – Overview of Zone 3

Over Size photo.

If you would like a copy, please contact the Company at
1-866-688-2001 toll free or 604-688-2001.

Plate 2 – Minesite Overview

And

Plate 3 – Underground Vein Exposure

Over Size photo.

If you would like a copy, please contact the Company at 1-866-688-2001 toll free or 604-688-2001.

Plate 4 – High Grade Vein

And

Plate 5 – Vein Drill Core

Over Size photo.

If you would like a copy, please contact the Company at 1-866-688-2001 toll free or 604-688-2001.

Plate 6 – Stockwork Zone

And

Plate 7 – Stratabound Drill Core

Over Size photo.

If you would like a copy, please contact the Company at 1-866-688-2001 toll free or 604-688-2001.

5. HISTORY

Mineralisation was discovered at Prairie Creek in 1928 and limited work was conducted on the property thru 1966. Cadillac Explorations Ltd. then acquired the property and, from 1966 to 1969, discovered and explored twelve mineralised Vein zones along a linear northerly trend now known as Zones 1 through 12 (refer to Figure 5).

In 1970, the property was optioned to Penarroya Canada Ltd. Underground development of Zone 3 was carried out, as well as bulk sampling and preliminary metallurgical test work. Some surface drilling was carried out on Zones 6, 7 and 9.

In 1975, Noranda Exploration Company optioned the southern portion of the property, drilled eight holes and subsequently dropped the option in the same year. Between 1970 and 1980, extensive underground development of Zone 3 took place. In 1980, a group led by Nelson Bunker Hunt and William Hunt of Texas agreed to finance the mine into production based on a study completed by Kilborn Engineering (B.C.) Ltd. Financing negotiations were completed with Procan Exploration Company Ltd., a private company owned by the Hunt brothers. In 1980, Cadillac Explorations Ltd. acquired the 1,200 tons per day Churchill Copper concentrator that was dismantled and shipped to the Prairie Creek location.

By May 1982, the surface facilities were 90-95% complete, and mine preparation work to produce an initial 500 tons per day had been finalized and completed. At that time the silver price collapsed and Cadillac Explorations Ltd. was forced into bankruptcy, after a total of $64 million had been expended on the Project. The mine, together with all other assets of Procan, was tied up in litigation until 1990.

In 1991, Conwest Exploration Company Limited acquired the property and granted an option to the Company, then known as San Andreas Resources Corporation, to acquire a 60% interest in the property. Since 1991, the Company completed four consecutive years of surface diamond drill exploration campaigns, greatly expanding the pre-existing resource and in 1993 exercised their option to acquire 100% of the mine.

In 1992, the discovery of Stratabound mineralisation in the main zone opened up multiple exploration targets. In 1993/94 the Company, in addition to diamond drilling, also completed additional environmental assessment research and further metallurgical studies. The 1995 drill program explored for extension of the vein to the north on 200 meter step-out sections all of which successfully intersected the Vein structure. This demonstrated the continuance and strength of the Vein that now had a drill defined strike length of 2.1 kilometers and remains open at depth and along strike.

In 1996, the Company successfully completed the negotiation of The Prairie Creek Development Co-operation Agreement with Nahanni Butte Dene Band of the Deh Cho First Nations. During 1997/98 the Company completed a compilation of information which included infill underground sampling, metallurgy studies, file organization, updates of sections and plans, and an independent resource evaluation.

In 1999, San Andreas Resources Corporation's name was changed to Canadian Zinc Corporation. The Company completed a grassroots exploration program on areas that are underlain by similar type geology and structural settings as found in the main zone. The Company staked four claim blocks (the Gate Claims) and carried out some prospecting and geochemical sampling that resulted in discovery of a new Vein showing and indicated strong geochemical zinc anomalies over parts of the Whittaker Formation.

A change of management and board members took place at the Company's May 2000 Annual General Meeting. CZN subsequently undertook a regional reconnaissance helicopter exploration program to assess regional metal potential. At the same time further studies were entered into to support the present Scoping Study.

6. PRIOR EXPLORATION

6.1 GENERAL EXPLORATION

Prospecting and sampling has been carried out on the Prairie Creek Property for over 50 years. An extensive network of cat roads were established throughout the southern part of the property. The roads allowed backhoe and caterpillar trenches to be excavated on the various showings. There is extensive trenching completed by blasting on Zones 3, 7 and 8. The property has also been soil sampled in part, and geophysical surveys were carried out including IP, UTEM and VLF. Aerial surveys including orthophotography, aerial photography and detailed land surveys have been completed.

There have been at least three regional reconnaissance exploration programs, the most recent of which was carried out by CZN in the year 2000. Numerous reports exist on both the property and region and certainly much work remains unreported.

6.2 DIAMOND DRILLING

To the end of 1970 a total of over 10,000 meters of core drilling (the majority from surface) and 3,800 meters of underground development exploration had been completed.

Table 3 – Diamond Drilling Completed at Prairie Creek

By zone and Year (in meters)

Zone	Prior to 1970	1970	1971-80*	1992	1993	1994	1995	Totals
1/2/3/4	3,748	2,902	4,000	6,535	7,028	8,731	9,718	**38,914**
5	316		3,000	0	362	97	1,062	**4,521**
6		615	1,500	0	650	2,051	0	**4,816**
11	104	103		0	33	0	0	**136**
7 & 8	796	2,160		0	279	0	0	**2,439**
Rico				0	0	372	193	**565**
Zebra				0	0	2,025	703	**2,728**
Totals	**4,964**	**5,780**	**8,500**	**6,535**	**8,352**	**13,276**	**11,676**	**54,119**

*estimated

It is estimated that approximately 120 diamond drill holes were completed at Prairie Creek Mine prior to CZN's involvement. Since 1991 CZN has completed 129 diamond drill holes, core from which is presently stored in racks at the minesite.

6.3 UNDERGROUND DEVELOPMENT

6.3.1 Zone 3

Zone 3 contains the most extensive underground workings which have been developed since the 1970's. Zone 3 is presently accessed by three levels of underground workings with the lowest level, the 870 meter level, being on valley floor at the minesite. The Company has not extended any of the underground workings during the course of its exploration. Approximately 40,000 tonnes of Vein material was mined from the 930 meter level in 1981 and is presently stockpiled beside the mill.

6.3.1.1 970 meter (3,170 ft) Level

This level contains 220 meters of footwall drift with six cross-cuts at 30 meter intervals. Access to this level is limited at this time due to caving at the portal entrance. The current condition of the workings is unknown.

6.3.1.2 930 meter (3,050 ft) Level

The 930 meter level consists of 940 meters of footwall haul drift with 32 cross-cuts at 10 meter centers. There are 630 meters of Vein drifting and 480 meters of other development. This level has a number of shrinkage stopes developed with active drawpoints and is presently prepared to operate at 500 tons per day in a trackless operation. A small amount of underground drilling was completed in this level. Preparations were underway to create a small backfill plant on this level however this remains uncompleted. Conditions on the level are excellent with little or no evidence of falls of ground or deterioration of the workings. Support on the level is minimal and none is currently required.

6.3.1.3 870 meter (2,850 ft) Level

This level contains 610 meters of footwall haulage drift (tracked), 380 meters of Vein drifting and 150 meters of other development. The portal is located at mill level. This level has a number of shrinkage stopes developed and is tracked out to the mill. There are two manways (including a service raise) and two ore passes connecting this level to the 930 meter level. Also a small ramp was developed at the end of the level, which provides access to a 50 ton ore bin to load rail cars. Also on this level is a sump system which was excavated to act as a settlement pond and water treatment site. At the present time there is limited access into this level due to development of an ice plug near the portal entrance. Conditions on the level are excellent with little or no evidence of falls of ground or deterioration of the workings. Support on the level is minimal and none is currently required.

6.3.2 Zones 7 and 8

Zones 7 and 8 were explored by underground drifting during 1969: 280 meters on Zone 7 and 240 meters on Zone 8. Both drifts are about 330 meters below the surface Vein mineralisation at these showings. The portals to these workings have since filled in and need to be reopened to allow further examination.

7. RESERVE AND RESOURCE ESTIMATION

Over the years there has been a number of mineral reserve and resource calculations and estimates completed on the Prairie Creek deposits (refer to Table 4). While the applied economic and cutoff parameters have varied with the time, overall the reports generally reflect a consistently high-grade lead-zinc-silver-copper Vein deposit. In September of 1980 Kilborn Engineering completed a Feasibility Study for the Prairie Creek Deposit which calculated a Vein reserve of 1,629,000 tons grading 11.16% lead, 12.17% zinc, 5.54 oz/Ton silver and 0.43% copper. It was on this basis that the main financing was raised in order to place the Prairie Creek Mine into production. The Kilborn ore reserve estimate was based on three levels of underground workings and approximately 43 other data points, all within close vicinity of the underground workings of Zone 3. The Kilborn Vein ore reserve included a 15% dilution factor at zero grade. Procan Exploration Company in 1983 completed additional underground sampling from the three levels and incorporated this into a subsequent reserve calculation. Furthermore a 1993 reserve calculation (Cominco) included additional surface diamond drilling completed by the Company on the Vein at depth below the workings and the initial drill intercepts of the Stratabound sulphide material.

Previous reserve estimates have included mineral reserve blocks in other zones such as 7 and 8, however these have not been included in this Scoping Study since the primary focus of this report is exclusively on Zone 3 (which includes 1 & 2).

Since Stratabound sulphide material was not discovered until 1992, none of the mineral estimates prior to this time consider it as a resource. Both the Vein and Stratabound deposits remain open along strike.

Step-out drill holes at depth on the main zone were completed by the Company in 1993/4 exploration campaigns incorporated into an in-situ mineral resource calculation in 1995 (Simons). Overall the Vein resource was projected at depth and along approximately 1.2 kilometers of strike length beneath the workings based on some 40 recent diamond drill holes and previous data. Additional Stratabound sulphide intercepts were also included into this mineral resource estimation.

As a Qualified Person representing Canadian Zinc Corporation, I, Alan B. Taylor, do so state that in my opinion the Historical Reserve Estimates presented in Table 4 are estimated on somewhat different categories than that adopted in the CIM Standards on Mineral Resources and Reserves – Definitions and Guidelines as incorporated in National Instrument 43-101 due to the date in which they were completed. However, it is apparent that reasonable economic parameters were applied to estimate mineral reserves which include a completed feasibility study by Kilborn Engineering in 1980. The pertinent reports are referenced in the Bibliography Section 24 of this report and these reports are available for review at the Company's' office upon request.

Table 4 – Historical Ore Reserves and Resources: Prairie Creek Mine

No 3 Zone

Behre Dolbear & Company 1970 for Pennarroya Canada: Reserves

	Category	Tons	Pb %	Zn %	Ag Oz/Ton	Cu%
Vein		***1,372,000***	***11.75***	***14.23***	***6.07***	***0.40***

James & Buffam 1972: Reserves

	Category	Tonnes	Pb %	Zn %	Ag Oz/Ton	Cu%
	Proven	**534,000**	**9.71**	**12.04**	**5.19**	**0.41**
	Drill Indicated	**471,000**	**12.04**	**12.38**	**5.43**	**0.41**
Vein	***Total***	***1,005,000***	***10.80***	***12.20***	***5.44***	***0.41***

Kilborn Definitive Feasibility Study September 29, 1980: Reserves

	Category	Tons	Pb %	Zn %	Ag Oz/Ton	Cu%
	Proven	740,000	10.96	12.88	5.60	0.46
	Probable	377,000	11.08	12.65	5.34	0.37
	Possible	512,000	11.50	10.80	5.59	0.44
Vein	***Total***	***1,629,000***	***11.16***	***12.17***	***5.54***	***0.43***

Procan Exploration 1983: Reserves

	Category	Tonnes	Pb %	Zn %	Ag gm/t
Vein		***1,069,057***	***13.01***	***13.34***	***220.40***

Macleenan for San Andreas 1992, in CESL 1993: Reserves

	Category	Tons	Pb %	Zn %	Ag Oz/Ton
Vein	Proven	374,928	12.66	14.02	6.36
Vein	Probable	957,991	16.68	17.08	7.97
Vein	Possible	838,634	16.01	16.58	7.89
Stratabound	Probable	335,268	8.54	16.60	2.49
Stratabound	Possible	355,844	7.62	16.16	2.32

Simons Mining Group, January 1995: In situ Geological Resource

	Category	Tonnes	Pb %	Zn %	Ag gm/t	Cu%
Vein		4,019,280	14.29	12.02	218.23	0.42
Vein Stockwork		719,990	8.20	17.44	122.35	0.27
Stratabound		1,144,530	7.08	12.95	73.13	0.02
Zone 3 Total		***5,883,800***	***12.14***	***12.86***	***178.27***	***0.33***

8. CURRENT RESOURCES

In 1995, the Company completed a series of 200 meter step-out diamond drill holes to test for extension of the Vein to the north of the previous resource. This program was successful at intercepting the Vein structure in every hole very close to the anticipated target and increased the confidence in the structure continuing even further towards the north. This 1995 drilling effectively doubled the drill tested strike length of the Vein resource to over 2.3 kilometers from the previous resource estimate. The prospective Stratabound host horizon was not tested in these holes and remains open. In order to take into account the effect on resources of the step-out drilling completed by the Company an independent resource estimate was completed.

Table 5 shows the "estimated resource" contained in Zone 3 as calculated by MRDI Canada, a wholly owned subsidiary of AMEC E&C Services Limited. (January 1998) in the course of an independent review of the Project. A total geological resource in Zone 3 was estimated at 11,846,000 tonnes grading 10.1% lead, 12.5% zinc, 161 g/tonne silver and 0.4% copper. The information used in the resource estimates was derived from the Company's diamond drill hole data and from a number of the more clearly defined drill logs from previous operators. The database for compilation incorporated 1,529 sample assays from the Vein (both underground channels and diamond drilling), 39 samples from the stockwork (both underground channels and diamond drilling) and 282 sample assays (drilling only). The silver grades were cut to 600 g/tonne. Specific gravity laboratory measurements were provided for 231 Vein samples and 22 Stratabound samples. MRDI completed regression analysis to determine an appropriate function to calculate specific gravity for the remainder of the samples. The geological resource was classified into measured, indicated and inferred resources, based upon level of confidence according to the Australasian Code for Reporting Identified Mineral Resources and Ore Reserves, using drilling grid spacing and continuity of mineralisation as determined through the geostatistical review of the data. MRDI staff visited the property site and agreed that the data and interpreted model represents the Prairie Creek Deposit.

The complete methodology used by MRDI in estimating the Prairie Creek mineral resource may be found in Appendix I of this report.

As a Qualified Person representing Canadian Zinc Corporation, I, Alan B. Taylor, do so state that in my opinion the categories of measured, indicated and inferred mineral resources in The Australasian Code for Reporting of Mineral Resources and Ore Reserves used by MRDI are substantially equivalent to the categories of measured, indicated and inferred mineral resources in the CIM Standards on Mineral Resources and Reserves – Definitions and Guidelines as incorporated in National Instrument 43-101.

The "1998 mineral resource estimation" reflects the impact of step-out exploratory drilling at depth and along strike of the previously known mineral reserve subsequently completed by CZN since acquiring the property. While tonnage has increased in direct proportion to area of influence, the basic Vein mineralogy and structure remains the same. It is also of interest to note that the MRDI "Geological" Resource grade in the Vein of 12.5% zinc, 11.4% lead, 178 g/tonne silver and 0.4% copper compares very closely with Kilborn's 1980 grade estimate of the Vein which was 12.17% zinc, 11.16% lead, 190 g/tonne silver and 0.4% copper. The "MRDI resource" grades include all intercepts in a specific area, and has had no blocks removed by a cut off grade.

It is fully understood that the 1998 mineral resource is strictly an in-situ mineral resource estimation and further delineation drilling and underground drifting is required in order to raise the confidence level of the resources. This is the objective of the proposed 2001 exploration, definition and delineation program which would also include underground driving, drifting and drilling both on the Vein structure and areas of Stratabound mineralisation.

Table 5 – Current Mineral Resource Table

MRDI Canada January 1998

	Category	Tonnes	Pb %	Zn %	Ag gm/t	Cu%
Vein	**Measured**	542,000	13.0	12.5	197.0	0.4
Vein	**Indicated**	1,434,000	12.8	11.2	190.0	0.4
Subtotal		**1,976,000**	**12.9**	**11.6**	**191.9**	**0.4**
Vein	**Inferred**	**7,412,000**	**11.0**	**12.7**	**174.0**	**0.4**
Stockwork	**Measured**	79,000	15.0	31.1	294.0	0.7
Stockwork	**Indicated**	228,000	5.6	14.5	134.0	0.4
Subtotal		**307,000**	**8.0**	**18.8**	**175.2**	**0.5**
Stockwork	**Inferred**	742,000	5.0	14.6	145.0	0.4
Stratabound	**Measured**	500,000	5.4	10.5	51.0	0.0
Stratabound	**Indicated**	785,000	5.1	10.6	59.0	0.0
Subtotal		**1,285,000**	**5.2**	**10.6**	**55.9**	**0.0**
Stratabound	**Inferred**	**124,000**	**2.7**	**7.9**	**26.0**	**0.0**

9. EXPLORATION POTENTIAL

The deposit at Prairie Creek has enormous potential for extension to the existing 12 million tonne resource. The current resource is defined over only 3 kilometers of a 14-16 kilometer mineralised strike length. The discovery in 1994 of Stratabound mineralisation over approximately 1.5 kilometers of the existing defined resource, is an indicator of the potential for further Stratabound mineralisation over the remaining 12–14 kilometers of the identified strike length.

The Project is therefore faced with two distinctly different geological opportunities. First, further exploration will increase the level of confidence in the current resources allowing for the opportunity of their conversion to reserves; and secondly to further increase the resource on the Prairie Creek Project by exploration drilling outside Zone 3 over the remainder of the 14 kilometer strike length. Both offer excellent opportunities for expansion of the mineral resource.

9.1 PRE-PRODUCTION DEFINITION DRILLING, DECLINE AND UNDERGROUND DRILLING

A significant amount of pre-production definition drilling will be required to raise the current inferred resource into a measured and indicated category. Much of this drilling can be carried out from surface, using the mine's two existing Super 38 drill rigs to drill the area between the mill building and the proposed portal entrance some 500 meters to the north. The use of the Super 38s will reduce drilling cost due to ease of access and size of machinery available. This drilling will be carried out during 2001, from existing roads and drill pads in the area. The objective of this drilling is to both increase confidence in the Vein resource below the 870 meter level and to outline additional Stratabound mineralisation.

An underground decline consisting of approximately 600 meters of drivage and 70 meters of crosscuts is proposed for 2001. The objective of this development is to establish underground drill stations from which to further delineate both the Vein and Stratabound mineral deposits in order to increase the drill intersect database, hence increasing the confidence level of the resource towards a reserve.

It is proposed to initially drill from underground on 50 meter stations further defining the Vein below the existing 870 meter level where mining would initially commence. The portal to this –15% decline would be located approximately 300 meters northeast of the existing 930 meter level portal. A crosscut at the end of this decline would access fresh Vein material at approximately the 825 meter level. This decline may or may not be utilized in the eventual development of the mine but will serve as means to further explore the deposit.

9.2 PROJECTED DECLINE DEVELOPMENT AND ADDITIONAL DRILLING COSTS

Program	Estimated Cost
Decline	$1,500,000
Underground Definition Drilling	500,000
Surface Diamond Drilling	500,000
Property Exploration	500,000
Total Estimated 2001 Expenditure	**$3,000,000**

Development costs for a 600 meter, 3 meter x 2.3 meter decline at –15% was calculated by Procon Mining and Tunnelling Ltd. (2000). This underground program would utilize existing equipment available on site and only bring in such additional equipment necessary to efficiently complete the job. Additional underground diamond drilling delineating both the Vein and the Stratabound potential would cost in the order of $500,000. Further definition drilling from surface would take place near the new decline portal area working towards the minesite and would utilize the Company's two Longyear 38 diamond drills that it presently has on site.

A mill Pilot Plant would be operating during the season in order to further refine the metallurgical process sheet. The Pilot Plant would operate within the existing mill structure at a rate of approximately 1 tonne of ore per hour during the summer and fall season.

9.3 VEIN POTENTIAL

Four kilometers to the north of the last drill hole on the Vein (PC-95-125) is the Rico Showing. This consists of high grade Pb-Zn-Ag-Cu Vein sulphide occurring within the Arnica Formation in the same structural setting as the Vein, near the axis of the Prairie Creek Anticline. This sulphide material occurs as Vein sulphide, which has moved vertically within the stratigraphy from the minesite host rock units, which are at depth under the Rico Showing. Geological mapping and geochemical sampling also indicate the continuance of the Vein along strike of the drill-defined resource (Jenks 1986).

This northerly continuance coupled with the 12 separate Zone Vein occurrences over 10 kilometers to the south of the mine indicate further additional potential to significantly extend the present Vein resource.

9.4 STRATABOUND POTENTIAL

The potential for discovering additional Stratabound massive sulphides in the immediate vicinity of the Prairie Creek Mine is excellent (Findlay 2000). The Stratabound horizon in the area of the proposed decline contains the potential to host significant undiscovered sulphide bodies in addition to the presently known deposits. This is due to the general lack of any detailed drilling in the prospective horizons.

The principal exploration target is the Whittaker Formation in the immediate vicinity of the Prairie Creek Vein system, which extends for 10 kilometers to the south. To the north the host stratigraphy plunges for 4 kilometers at 15°, hence becomes somewhat deeper, and remains totally unexplored.

Stratabound mineralisation was previously intercepted in Zone 6 which is 3 kilometers south of the minesite. This type of sulphide has also been located through drilling in Zones 4 and 5.

9.5 SURROUNDING GROUND

The Gate claims are underlain by similar geology and structure with that of Zone 3 and a grassroots exploration program in 1999 discovered a Pb-Zn-Ag-Cu vein in an identical type setting with that of the main zone. Also of significance was the discovery of a significant zinc anomaly in soils, overlying the Upper Whittaker Formation, which has characteristics similar to that of Stratabound mineralisation. This area has not been drill tested and should be the focus of a future exploration program.

Similar geologic occurrences adjacent to the Gate claims should be tested for further mineralisation. A helicopter reconnaissance sampling program (Taylor and Findlay 2000) indicated anomalous zinc values associated with regionally occurring black shales along with a number of other specific anomalies. The lack of a strong metal signature around known mineral showings indicates there still remains a large area surrounding the known Prairie Creek deposits that need to be further evaluated in detail for additional mineral potential.

10. DESIGN CRITERIA

10.1 GENERAL

10.1.1 Location

Country	Canada
Province	Northwest Territories
Nearest Town	Nahanni Butte/Fort Simpson, Northwest Territories
Nearest Shipping Point	Fort Nelson, British Columbia
Nearest Major Center	Yellowknife, Northwest Territories

10.1.2 Elevation

Mine	860 meters
Processing Facility	860 meters

10.1.3 Climate

Annual Precipitation	400 mm
Minimum Temp	-25^{o}C
Maximum Temp	+13^{o}C
Mean Temperature	-5^{o}C

10.1.4 Communications

External	Satellite Phone system
Internal	Two-way radio

10.1.5 Utilities

Electric Power	Diesel fired generators
3.8 MW loaded / 2.7MW operating	2400 volt, 60 Hz
Potable Water	From Fresh Water Wells

10.2 MINING

10.2.1 Design Basis

Total production, tonnes/year	550,000
Ore production, tonnes/day	1,500
Dilution tonnes per day	225
Operation, days/year	340
Operation, days/week	7
Operation, shifts/day	2
Operation, hours/shift	10-12 hours depending on regulations

10.2.2 Diluted Ore Resources (MRDI 1998)

Vein, ore tonnes		9.3 million
grade,	Pb %	11.4
	Zn %	12.5
	Cu %	0.4
	Ag G/T	178.0

Stratabound, ore tonnes		1.40 million
grade,	Pb %	5.00
	Zn %	10.30
	Cu %	0.02
	Ag G/T	53.00

Total, ore tonnes		11.8 million
grade,	Pb %	10.1
	Zn %	12.5
	Cu %	0.4
	Ag G/T	160.0

In addition, 15% dilution at 30% of in situ grade has been added to these figures.

10.2.3 In-Situ Density

Vein (t/m^3)	3.28
Stratabound (t/m^3)	3.75
Waste (t/m^3)	2.67

10.3 CRUSHING

10.3.1 Crushing Design Basis

Production Schedule	540,000 tonnes/year
	340 days/year
	1 shifts/day
	12 hours/shift
Availability	75%
Design Feed Rate, t/day	1,750
t/hr	190

10.3.2 General

Feed Method	Conveyor from Dump Chute
Feed Size	80% passing 150 mm
Feed Size - top size	200 mm
Moisture, %, nominal	4.0

Specific Gravity of Product, t/m^3	3.4

10.3.3 Primary Crusher

Feeder Type	Grizzly Feeder
Opening, mm	100
Crusher Type	Jaw (609.6mm x 914.4mm)
Installed Power, kW	37
Feed size passing, mm	200
Closed Side Setting, mm	75

10.3.4 Secondary Crusher

Screen Type	Double Deck
Top Deck, mm	25
Bottom Deck, mm	10
Installed Power, kW	11.5
Crusher Type	Short-head Cone (5 ½feet)
Closed Side Setting, mm	10
Installed Power, kW	150

10.3.5 Fine Ore Bin

Bin Size, tonnes (live)	1,800

10.4 MILLING

10.4.1 Milling Design Basis

Production Schedule		540,000 tonnes/year
		340 days/year
		2 shifts/day
		12 hours/shift
Availability		93%
Design Feed Rate,	t/day	1,750
	t/h	80

10.4.2 Gravity Concentration

10.4.2.1 General

Method	Spiral concentration
Design Feed Rate, t/day	1,750
To Flotation, t/day	1,020
To Backfill Plant, t/day	730

10.4.2.2 Sizing Screen

Screen Type	Double Deck
Top Deck, mm	1.0 (16 mesh)
Bottom Deck, mm	0.5 (32 Mesh)
Installed Power, kW	11

10.4.2.3 Fines Separation

Feed Method	Sump Box and pump
Feed Size	Minus 0.5mm
Specific Gravity	3.0
Separation Method	Cyclone
Undersize,	To Falcon Concentrator
Oversize,	To Rougher Spiral

10.4.2.4 Rougher Separation

Feed Method	Cyclone Oversize
Feed Size	Minus 0.5mm
Specific Gravity	4.0
Separation Method	Spiral
Concentrate,	To Cleaner Spiral
Tails,	To Scavenger cyclone, fines to Falcon and Oversize to Backfill Plant

10.4.2.5 First Cleaner Separation

Feed Method	Rougher Spiral Concentrate
Feed Size	Minus 0.5mm
Specific Gravity	4.5
Separation Method	Spiral
Concentrate,	To Lead Flotation Circuit
Tails,	To Zinc Flotation Circuit

10.4.2.6 Middlings Separation

Feed Method	Sump Box and pump
Feed Size	0.5mm< x < 1.0mm
Specific Gravity	3.4
Separation Method	Spirals
Concentrate,	To Primary Ball Mill
Tails,	To Backfill Plant

10.4.2.7 Zinc Flotation Feed Separation

Feed Method	From lead flotation tails, First Cleaner Spiral Tail, Falcon Middlings
Feed Size	Minus 0.5mm
Specific Gravity	4.5
Separation Method	Spiral
Concentrate,	To Copper/Lead/Zinc Flotation Circuit
Tails,	To Zinc Flotation Circuit

10.4.3 Grinding

Type	Single stage ball milling
Work Index, kWh/T	8
Ball Mill	(3,048 mm dia. X 4267mm length)
Design Feed Rate, t/h	90
Feed size, mm	15
Product Size, mm	50% passing 0.75
Installed Power, kW	522

10.4.4 Flotation

Production Schedule	340 days per year 2 shifts per day 12 hours per shift
Operating Availability	+95%
Design Feed Rate, tph	30

10.4.4.1 Flotation Reagents - Vein Ore (grams per tonne of Ore)

Lime	none
Cytec 3418A	30
Sodium Isopropyl Xanthate	60
Copper sulphate	600
Sodium Sulphide	50
Glycol Frother	40

10.4.4.2 Flotation Reagents - Stratabound (grams per tonne of Ore)

Lime	3,000
Cytec 3418A	60
Sodium Isopropyl Xanthate	none
Copper sulphate	550
Sodium Sulphide	50
Glycol Frother	40

10.4.4.3 Lead Concentrate

Recovery Method	Gravity Rougher followed by regrind and two stages of flotation cleaning
Regrind Work Index, kWh/T	8
Regrind Ball Mill	(1829mm dia. x 1829mm length)
Design Feed Rate, t/h	14
Feed size, mm	P80=80 mics
Product Size, mm	80% passing 0.03
Installed Power, kW	100

Combined Production (2/3 Vein Ore and 1/3 Stratabound Ore)

Production, t/d	165
Production, tpy	56,100

10.4.4.4 Zinc Concentrate

Recovery Method	Gravity Rougher followed by regrind and two stages of flotation cleaning
Regrind Work Index, kWh/T	9
Regrind Ball Mill	(1524mm dia. X 1829mm length)
Design Feed Rate, t/h	20
Feed size	P80=80 mics
Product Size, mm	80% passing 0.05
" "	P80=50 mics
Installed Power, kW	75

Combined Production (2/3 Vein Ore and 1/3 Stratabound Ore)

Production, t/d	250
Production, tpy	85,000

10.4.4.5 Copper Concentrate (Vein Ore only)

Recovery Method	Regrind followed by bulk rougher flotation and two stages of flotation cleaning
Regrind Work Index, kWh/T	9
Regrind Ball Mill	(610mm dia. X 914mm length)
Design Feed Rate, t/h	5
Feed size	P80=80 mics
Product Size, mm	80% passing 0.05
" "	P80=50 mics
Installed Power, kW	37
Production, tpd	6
Production, tpy	2,000

10.4.5 Concentrate Dewatering and Storage

10.4.5.1 Lead Concentrate

Recovery Method	High Capacity thickener followed by pressure filtration
Production, t/d (max / design)	165
Thickener	
Type	High Capacity
Diameter, m	4.6
Specific Gravity, t/m^3	6.0
Filter area, m^2/tonne/hour	5.0
Storage	
Type	Conveyor feed to covered structure
Size, m x m	25.0 x 40.0
Capacity, tonnes	9,200 (8 weeks)

10.4.5.2 Zinc Concentrate

Recovery Method	High Capacity thickener followed by pressure filtration
Production, t/d	250
Thickener	
Type	High Capacity
Diameter, m	4.6
Specific Gravity, t/m^3	4.0
Filter area, m^2/tonne/hour	5.0
Storage	
Type	Conveyor feed to covered structure
Size, m x m	35.0 x 50.0
Capacity, tonnes	14,000 (8 weeks)

10.4.5.3 Copper Concentrate

Recovery Method	Thickener followed by vacuum filtration
Production, t/d	6
Thickener	
Type	Standard
Diameter, m	2
Specific Gravity, t/m^3	4.0
Filter area, m^2/tonne/hour	1.2
Storage	
Type	Bagged in supersacs
Capacity, tonnes	outside / unlimited

10.4.6 Tailings and Paste Backfill

Source of feed	Flotation tailing + Gravity Tailing
Method	Existing Thickeners followed by belt filtration, Pug mill mixer with cement addition and Geho pump distribution
Design Feed Rate, t/h	60
Thickener	2 – 10m diameter existing
Belt Filter	2 - 18.5 m^2 filter area
Vacuum Pump	
Installed Power, kW	37
Pug Mill	
Installed Power, kW	112
Geho Pump	
% Solids	85
Installed Power, kW	112

10.4.7 Concentrate Shipping and Handling

Concentrate will be shipped to Fort Nelson to a load-out. The concentrate will be railed from Fort Nelson to the nearest port or to Canadian smelters.

10.4.8 Reagents Other Than for Flotation

Flocculent g/tonne ore (con thickening)	< 5
Cement, kg/tonne (backfill as required)	40.0

11. MINING

The ore bodies at Prairie Creek are characterised by their high grade and strong specific gravity differences between ore (bedded or Vein type massive sulphides) with specific gravities from 3.28 to 3.75 and waste, with specific gravity around 2.67. Two very distinct types of mineralisation will be exploited in the early years of the operation, Vein style and Stratabound ore. Mining methods will be mechanized, using a gravity circuit in the mill to reject unwanted and extraneous dilution, caused by mechanization underground and areas of narrow but high grade ore.

Access will be by ramp below the level of the mill, with the possible installation of conveyor haulage to bring ore from the deeper Stratabound reserves to the surface. Use will be made of the existing underground development left over from the eighties, possibly by slashing out undersize development to accommodate larger and more efficient trackless equipment.

Every effort will be made to mechanise below ground, to reduce manpower and to reduce power costs by the use of conveyor haulage where applicable. The mill will treat the different ore types on a campaign basis. This will introduce some complications in the separate handling of Stratabound and Vein style ore. Support underground will be by paste or cemented backfill from the mill via a new paste fill plant. This will have the advantages of reduced water handling, since the backfill does not need to decant and the reduction of tailings storage facilities required on surface.

Mining at Prairie Creek has the advantage of starting off with a significant developed ore reserve above the valley floor, which can be recovered while other mining areas are developed, thus lowering early development costs.

Maximum advantage will be taken of existing development in the mine, however initial production tonnages from each level and ore type will be a limiting factor in any production plan. Fortunately conditions underground are excellent, with the existing workings have stood unsupported and undamaged for over 20 years. Conditions on the site are such that the mine air will require heating in winter, and pumping can be expected once workings drop below the valley floor since the mineralisation is in limestones.

11.1 EXISTING DEVELOPMENT

Zone 3, adjacent to the mill, contains the most extensive underground workings which have been developed since the 1970's. Zone 3 is presently accessed by three levels, with the lowest level, the 870 meter level, being on valley floor at the minesite. The Company has not extended any of the underground workings during the course of its more recent exploration. Approximately 40,000 tonnes of Vein material was mined from the 930 meter level in 1981 and is presently stockpiled beside the mill.

11.1.1 970 meter (3,170 ft) Level

This level contains 220 meters of footwall drift with six cross-cuts at 30 meter intervals. Access to this level is limited at this time due to caving at the portal entrance. The current condition of the workings is unknown.

11.1.2 930 meter (3,050 ft) Level

The 930 meter level consists of 940 meters of footwall haul drift with 32 cross-cuts at 10 meter centers. There are 630 meters of vein drifting and 480 meters of other development. This level has a number of shrinkage stopes developed with active drawpoints and is presently prepared to operate at 500 tons per day in a trackless operation. A small amount of underground drilling was completed in this level. Preparations were underway to create a small backfill plant on this level, however this remains uncompleted. Limited workshop facilities were completed as was a rock doctor's shop and other storage facilities. A mine air heater was installed on the level, indirectly fired by propane. Limited services are available and concrete pads for substations have been installed. Conditions on the level are excellent with little or no evidence of falls of ground or deterioration of the workings. Support on the level is minimal and none is currently required.

11.1.3 870 meter (2,850 ft) Level

This level contains 610 meters of footwall haulage drift (tracked), 380 meters of vein drifting and 150 meters of other development. The portal is located at mill level. This level has a number of shrinkage stopes developed and is tracked out to the mill. There are two manways (including a service raise) and two ore passes connecting this level to the 930 meter level. Also, a small ramp was developed at the end of the level, which provides access to a 50 ton ore bin to load rail cars. Also on this level is a sump system, which was excavated to act as a settlement pond and water treatment site. Services (air and water, plus power) were brought from the mill to the mine via this level, then up the raises to the level above. At the present time there is limited access due to development of an ice plug near the portal entrance. Conditions on the level are excellent with little or no evidence of falls of ground or deterioration of the workings. Support on the level is minimal and none is currently required.

11.2 REVISED ACCESS PROPOSAL

Future access will be by ramp from surface using a raise as a second means of egress and to provide through ventilation. The production ramp, which will be driven at –15°, will be 4.6 meters high by 5.2 meters wide to accommodate 25 tonne trucks and a conveyor belt if required. Where possible the ramp will take advantage of existing development and the possibility of enlarging the proposed exploration decline will not be ignored. The ramp will be placed at the center of gravity of the current known resources and will provide access to the orebody by means of a series of crosscuts at appropriate level intervals depending on the mining method selected.

Since the ramp will be driven zigzag fashion to stay in the center of the deposit, ventilation raises and ore passes if required will be driven from turn to turn, using longhole equipment or an Alimak raise climber. Subsequently, the ramp will approach the Stratabound mineralisation, passing below it to provide sumps and ore passes for this significant area of mineralisation.

11.3 PROPOSED MINING METHOD

Mining methods at Prairie Creek will be selected dependent upon orebody geometry and hanging and footwall conditions. Because of the introduction of a gravity plant in the mill, dilution, while not desirable, can be traded off against increased mining efficiency. As a consequence, mining methods will be selected to gain the maximum possible increases in production efficiency since the two most expensive aspects of any operation in the North are manpower and fuel costs.

Methods that will be considered will include:

11.3.1 Typical Mining Methods – Vein Mineralisation

Economic mineralisation at Prairie Creek is dominated by Vein mineralisation in Zone 3. By far the largest majority of ore is Vein type, constituting approximately 9.3 out of the 11.8 million tonnes of the MRDI resource. Vein mineralisation is typically from 1.5 to 8 meters thick and dips from 40° to 90°. Continuity within the receptive horizon is excellent, with an average thickness over the whole resource of 2.7 meters. Specific gravity of in situ Vein mineralisation averages 3.28 t/m^3. On this basis, each square meter of Vein mined will produce 8.6 tonnes of ore with a global grade of 11.4% lead, 12.5% zinc, 0.4% copper and 178.0 g/t silver.

The major problem with the Vein style mineralisation is the relatively low tonnes per vertical meter, typical of this type of deposit. Assuming a strike extent over the receptive horizon of 650 meters, then this equates to approximately 5,800 tonnes per vertical meter. A typical 30 meter stope in this material would need to advance approximately 1.8 meters per day to produce 500 tonnes.

11.3.1.1 Shrinkage Stoping

The traditional mining method in this type of material has been shrinkage stoping and indeed this is the method that was originally chosen for the operation in 1982. A number of shrinkage stopes are fully developed on the bottom two levels of the existing mine (930 meter and 870 meter levels). Assuming a typical shrink stope in this mine is 30 meters long and stretches level to level (60 meters), then the tonnage contained would be approximately 15,940 tonnes. Because dilution is generally low with shrinkage stoping this would be close to the total tonnes extracted or about 16,000 tonnes.

Typically, a shrinkage stope will produce approximately 1/3 of its possible daily broken tonnage as swell pulled from the stope during development and will only allow complete

Figure 10 – Conceptual Mine Plan (Ramp Haulage)

Over Size photo.

If you would like a copy, please contact the Company at
1-866-688-2001 toll free or 604-688-2001.

Figure 11 – Conceptual Mine Plan (Conveyor Haulage)

Over Size photo.

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extraction on completion of mining. As a consequence and assuming a lift of 1.2 meters daily, then the total broken ore production from a developing stope would be in the region of 100 tonnes (320 tonnes broken times 30%). When being pulled, the tonnage available from a shrinkage stope can be as high as the transport and mucking system can stand. In this case production levels of 500-700 tonnes per day from an individual level would be quite possible with trackless equipment.

Shrinkage stoping requires significant development on the production level, followed by the driving of a series of drawpoints and a service raise to the top of the stope in the pillar connecting with the level above. This raise can be half timbered to provide access to the stope throughout its length and a mill holed second raise can be brought up with the muck, using steel half sections available on site. All raises are driven by hand, using timber platforms and a tugger to raise equipment and supplies into the stope. Subsequently an undercut drift is mined above the production level and stoping is commenced by slashing down the back using the raise at one end, or the mill hole raise as a free face.

Drilling is by stoper or jackleg using hand held equipment, and the process is labour intensive and expensive. After each lift is broken, the muck pile must be leveled and the swell pulled to allow the miners working room. Because the miners are working in the whole stope, the back must be intensively inspected and supported as necessary to provide a safe working environment for the men.

On completion of mining, the stope is pulled empty and then bulkheaded to take backfill or paste fill. Because of the number of drawpoints for a typical 30 meter stope (five) plus the vein drive, this requires seven bulkheads before filling can commence. As can be seen, shrinkage is thus labour intensive and costly and because of the production profile, requires at least four stopes in all stages of mining at any one time (development, breaking, pulling, filling). Between adjacent stopes, a pillar is left, which is subsequently extracted by shrinkage against the cemented fill of the two adjoining stopes.

Typical productivity in a shrinkage stope of this type is low, with mining costs in the $50-$60 per tonne to the mill. The only real advantage of shrinkage stoping is its ability to accurately follow the hanging and footwall of the orebody and thus reduce dilution to the mill, and control the hanging and footwall, particularly in cases where bolting and meshing of the contacts is necessary to prevent subsequent failure. This is not currently the case at Prairie Creek, where existing shrink stopes in the early phases of production (breaking) have shown no indication of hanging and footwall failure or need for local support. Similarly, with the introduction of a gravity circuit in the mill, the need to control or reduce dilution to maximise ore grade to the mill is no longer as important as tonnes per man shift and cost of ore from the mine. As a consequence, the mine will avoid as far as possible the use of shrinkage stoping.

Figure 12 – Proposed Mining Methods - Shrinkage Stoping

Over Size photo.

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11.3.1.2 Alimak Stoping

The introduction of the Alimak mechanical raise climber revolutionized the driving of raises. This air, diesel, or electrically driven platform climbs up a monorail bolted to the back of the raise. Services are carried to the face using pipes integrated into the monorail or by hose. On long raises an Alicab or separate service platform is provided to travel from the working platform to the ground.

Using the Alimak, much longer raises can be achieved faster and easier than by conventional timbered or blind raising. Longer rounds can be drilled, since more time is available to the miners at the face and larger raises can be easily driven off the Alimak platform. Raises are typically driven at least 100 meters and in exceptional circumstances can be extended to 300 meters plus. Because of the relatively low cost of an Alimak raise climber ($150,000 each) this method for mid range blind raising has become an industry standard against more capital intensive raise boring or cage raising methods.

From the Alimak raising method, Alimak stoping was developed to mechanise shrinkage style ore bodies. Typically, the orebody is blocked out with levels up to 100 meters apart vertically. Alimak stopes are set out on 30 meter centers, with only a single central drawpoint, plus a vein and a footwall drive completing the level development required. Adjacent stopes are mined in sequence so that when one stope is mining, the adjacent stopes on either side are only in development.

From the central drawpoint, an Alimak raise is driven to the level above over the full width of the orebody. The hangingwall of the raise is bolted and secured to provide safe cover throughout the operation of the stope. Subsequently the Alimak platform is equipped with a pair of longhole machines pointing in opposite directions mounted on bar sliders. These machines can easily drill off the 10-12 meters from the Alimak raise to the designated edge of the stope along strike. The provision of bar mounts allows the machine to move across the orebody to accurately position the drill holes to break the ore.

The use of longhole machines in this manner allows the operation to take advantage of a number of increases in efficiency. The machines drill a larger 2.5 inch hole at a much higher rate than the traditional handheld machine. One operator can operate two machines within the same stope. Less holes are required to break the ore. The addition of an Alicab in each stope means that drilling can continue without having to bring the platform down until the stope is completed, blasting then being carried out off the platform after the longhole machines have been removed.

Typically, once an Alimak stope has been developed, it can produce at very high rates, only restricted by the ability of the mucking system to handle the ore. Remote control scooptrams, operating from the drawpoints within the adjacent stopes, can pull the stope empty from under the unsecured back via the vein drive and muck to trucks on the footwall drive. Because the Alimak is used to provide access and a drilling and

Figure 13 – Proposed Mining Methods – Alimak Stoping

Over Size photo.

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blasting platform, a stope can be pulled empty after every blast. The use of remote control scoops means that men do not have to work under an unprotected back.

Drawbacks of Alimak stoping are increased dilution due to breakage of the hangingwall and footwall by errant or undirected drilling, and increased use of capital due to the need to have at least two stopes (and consequently Alimak climbers and Alicabs) in operation for each production stope (one producing and one developing; the Alimak machines are effectively captive for each stope until completed). Typically, Alimak stoping forms a checker board pattern, with alternate stopes being mined and filled with cemented fill. Subsequent pillars are mined against two adjoining filled stopes and blasted right to the cemented fill. This pillar stope is then filled with un-cemented fill to reduce cement consumption.

Sill pillars can be left in ore, below main levels, or these can be avoided by the use of cemented fill in the bottom of each stope to form a sill pillar. Typical costs for Alimak stoping in this type of circumstance depend on the thickness of the ore, but range from \$20 to \$30 per tonne to the mill, plus the costs of development.

The use of Alimak raising will be considered for mining both the existing developed areas on the 870 and 930 meter levels and new areas of the mine. Alimak stopes could be established from the 870 meter level by the driving of an Alimak garage in the hangingwall of the existing shrinkage drawpoints on the 870 meter level. The central raise of the shrinkage stopes would then be equipped with the Alimak rail and driven as a conventional Alimak stope up to the 930 meter level and beyond as required. Stoping could be carried out from the 870 meter level all the way to the 970 meter level if mineralisation permits. Because of the methods high productive capacity and the existing development on the 870 meter level, this would be the preferred approach for mining from 870 meters up.

11.3.1.3 Shorthole Open Stoping

As with Alimak stoping, technology in the form of the remote control scooptram brought about a sea change in production methods for narrow ore bodies dipping at over 30°. Mining costs underground are composed of labour costs and the costs of breaking and transporting the ore. Shorthole open stoping takes advantage of the ability of the remote controlled scooptram to work under an unsupported back and remove broken ore from the open stope.

Levels are driven on 20 meter intervals in ore using two boom electro hydraulic jumbos. In the case of Prairie Creek, each level would be driven in ore to the extremity of the orebody, with a parallel footwall drive, if required, to provide access. On reaching the mineralised extremity of the ore, or in the case where a footwall drive is employed at convenient intervals along strike, a two boom rig carrying a longhole machine and a basket on the other arm is introduced to the stope for production drilling.

This rig drills parallel longholes from level to level (approximately 15-18 meters long) to allow the ore to be broken to a longhole drilled slot raise at the end of the designated stoping block. Drilling is advanced ahead of production blasting until the stope can be blasted without damage to adjacent development and machinery. Production holes are then blasted into the open stope, and muck is removed using remote control scoops as per Alimak stoping. (The overall pattern of shorthole stoping can be thought of as Alimak stoping turned through 90° into the horizontal.)

Production from the stope continues both horizontally and vertically until stope limits are met. This can be much larger than other methods, since the determining factor becomes available mining levels and the ability of the in situ rock to remain in place unsupported. Typically, a footwall drive is included with this mining method to allow the development of adjacent stopes. However if the central ramp access is placed correctly and sufficient development is in place ahead of subsequent mining, then it is possible to advance development to the limits of the orebody in vein only, and subsequently retreat from the fringes of the ore zone to a central ramp.

If the mining sequence is correct, then drawpoints need only be provided on every third level to allow pulling of the ore to completion. Subsequent to mining, cemented fill is introduced to the fringes of the stope and the stope is then filled with paste or cemented fill as appropriate. As with Alimak stoping, mining and filling take place on a checker board pattern to ensure security of the hanging and footwall.

Advantages of shorthole stoping include simplicity of equipment since only scooptrams, trucks, two boom jumbos and two boom production rigs are used, and the ability to share equipment between production areas since all equipment is mounted on rubber tires. Typical costs for this method are, again, in the $20 to $30 per tonne delivered to the mill, plus development costs.

Shorthole open stoping would be employed in areas at the mine where Alimak stoping was difficult or where a shortage of Alimak equipment meant more efficient utilization of existing two boom production rigs. Advantages with shorthole open stoping are that development is carried out on the level using electro-hydraulic equipment pulling longer rounds. Disadvantages are the need for a footwall drive to allow machinery to move around producing stopes.

11.3.2 Typical Mining Methods – Stratabound Mineralisation

Stratabound mineralisation at Prairie Creek currently represents a lower proportion of the total resource: 1.4 million tonnes grading 5.0% lead, 10.30% zinc and 0.02% copper with 53.0 g/t silver. However, it represents a significant upside to the deposit for a number of reasons. These include much flatter lying and thicker ore intersections (up to 28 meters thick) and significantly lower levels of penalty elements (principally mercury, arsenic and antimony) in the ore. Metallurgical recoveries of this type of ore are higher to a slightly lower grade concentrate. Because of the higher tonnes per vertical meter, the Stratabound ore is capable of sustaining higher production rates with suitable mechanised mining. The Stratabound ore zones are open in almost all directions, and it is expected that significant increases in Stratabound tonnage can be achieved through future drilling.

Figure 14 – Proposed Mining Methods – Shorthole Open Stoping

Over Size photo.

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Because of the variable thicknesses and geometry of the Stratabound mineralisation, a number of different mining methods may be appropriate for this orebody. Delineation drilling to be carried out over the next year will give a much clearer picture of the likely mining method, plus the mining conditions likely to be encountered, however for the purposes of this scoping study, it has been assumed that a modified cut and fill or open stoping method will be employed.

11.3.2.1 Cut and Fill Mining

Cut and fill mining has been used extensively as a low cost mining method in flatter lying ore bodies. Typically, access is gained to the base of the stoping section by ramp, and a drift is driven to the hangingwall of the orebody. This drift is then slashed out to the full width of the stoping section, or to the maximum unsupported width. In the event that the unsupported width is less than the stoping section, then like the other mining methods, separate stopes and pillars can be developed for subsequent mining after filling, or pillars can be left in the ore (room and pillar mining). Typical room and pillar cut and fill mining has produced extraction ratios in excess of 87% (Black Angel – Greenland) and produces high efficiency, low cost ore.

After the first lift has been completed in the cut and fill stope, the access ramp roof is slashed and the waste material from the roof is back lashed into the stope to form a barricade to aid filling. Paste fill is subsequently introduced into the stope and capped off with cemented fill to provide a running surface for the production equipment. Another lift is then slashed from the roof of the existing stope and the filling sequence is repeated. The ramp access is once again slashed to provide access to the next level and so on until the hangingwall of the stoping panel is reached.

Cut and fill mining uses all mechanised equipment, including two boom and three boom jumbo drills, large scooptrams and trucks and mechanised scaling and support machines. Because of these efficiencies of scale and the non captive nature of the equipment, the method is flexible and cheap, with typical mining costs per tonne being in the $15 to $25 range plus development costs. An additional cost from this orebody would be the cost of hauling the ore to the mill, which would be significantly reduced by the use of conveyor haulage.

11.3.2.2 Longhole Open Stoping Mining

Open stoping mining is a generic term for mining methods that leave the stoped out void open, prior to subsequent filling of the stope. Longhole stoping, as its name suggests, includes the use of longholes to increase the efficiency of the mining method. Typically for this style of mineralisation, access would be gained to both the top and the bottom of the orebody. Drawpoints would be developed in the bottom of the block to extract the ore, and a slot raise would be developed between the top and the bottom of the stoping block to provide a free face for blasting.

Development on the top of the stoping block can include the slashing out of the whole of the top of the stope to the hangingwall (Topslice and Bench) or the driving of a number

Figure 15 – Proposed Mining Methods – Cut and Fill Stoping

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of drilling drifts over the top of the orebody to allow drill coverage of the whole stope while maintaining the integrity of the hangingwall. Then, 2.5 inch longholes are drilled from both the bottom and the top drifts to allow the ore to be blasted to the slot raise. Alternatively, 4-6 inch blastholes can be drilled as down holes from the top of the stope to the bottom of the stope (Sudbury/Inco method) and the stope effectively quarried from the top level.

Broken ore is extracted from the bottom of the stope and the stope is finally cleaned out using remote control scoops. On completion of mining, the stope is barricaded and filled with either paste or cemented fill, depending on whether it is a primary stope or a pillar. Subsequently adjacent pillars are blasted to the cemented fill and filled with paste.

Longhole stoping is highly efficient and cheap being the corollary of quarrying underground. The method depends on sufficiently large orebody geometry and a competent hangingwall, although the method has been used successfully where extensive support of the hangingwall has been necessary (Rubiales – Spain). Costs can be low due to the high level of mechanisation, dipping as far as $10 per tonne broken, plus the cost of development plus haulage of the ore to the mill.

11.4 PASTE BACKFILL

Paste backfill has become the preferred method of fill for underground operations for a number of reasons. These include increased use of mill tailings underground, reduced or eliminated decant requirements when compared with conventional fill, decreased cure time, increased fill strength with reduced cement consumption, improved fill pumping characteristics, etc. While not all mill tailings are suitable for paste backfill, the tailings from Prairie Creek when combined with the rejects from the gravity pre-concentrator should be suitable for placement underground using thickened or paste backfill.

The tailings from the flotation circuit will be thickened using flocculent in one of the two 35 foot diameter thickeners in the mill and mixed with rejects cycloned from the tailings stream of the gravity plant. Thickened tailings will then be pumped to belt filters that will reduce the moisture content of the tailings stream to approximately 12-18% moisture from where the thickened tailings will be fed to a pug mixer.

In the pug mixer, the tailings will be mixed with 3-5% cement or fly ash as required and the resultant mixture pumped to the mine using the existing Geho pump or a substitute positive displacement pump. Mix strength and loop pumping tests will be carried out using the Pilot Plant tailings to set final mixes and pumpability parameters, however, it is expected that the vast majority of the tailings can be placed underground in this manner.

Costs of placement of fill will vary, however the major cost of paste fill is in the cement addition, which has a cost of approximately $250/tonne delivered to Fort Nelson. Onward transport to the mine is approximate $15 per tonne. Thus for a cemented fill

Figure 16 – Proposed Mining Methods – Longhole Stoping

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containing 3% cement, the cost for the cement alone is $7.95 per tonne placed. The use of un-cemented fill is therefore generally maximised where no support is required or where the fill can be contained. Lower cost alternatives to act as pozilans, such as pulverized fly ash have been used successfully to reduce the cost of cemented fill. However, the likely cost of fill at this operation will be in the $10 per tonne for cemented fill and $2.50 to $3.00 per tonne for un-cemented fill including pumping and underground costs of the fill operation. For the purposes of the Scoping Study exercise it has been assumed that 60% of the fill placed underground will be cemented.

11.5 MINING COST ESTIMATES

The cost of mining underground is made up of a number of elements. These can be considered under the following headings:

- Access, including horizontal development, ramps and second means of egress
- Development to ore
- Development in ore
- Breaking of ore
- Haulage of ore to the mill
- Handling of waste
- Backfilling
- Services, including heating of air, ventilation, pumping, compressed air, water, etc.
- Supervision
- Exploration and delineation drilling

For the purposes of the Scoping Study, while a number of possible mining methods have been outlined, it is difficult to precisely define the methods to be used within this level of study detail. As a consequence, a preliminary development plan has been laid out and costed by Procon Mining and Tunnelling Ltd. of Burnaby. (2001) They have assumed that Alimak or shrinkage stoping will be used above the existing 870 m. level, to take advantage of the existing work and that short hole open stoping will be employed in the vein below this level. Cut and fill mining has been selected for the Stratabound for the purposes of this study, and all haulage is by truck via a ramp.

Procon assumed that only 90% of all ore is recoverable, (98% in shrinkage stopes) and calculated a per tonne cost of development for each mining method, over the first 4.9 million tonnes of the resource. This per tonne cost was split by stoping method and applied to each production tonne to generate an annual development cost. Similarly Procon developed a mining cost on a mining method basis and this was applied to generate annual stoping costs. The mine supplies fuel and power to the contractor at zero cost and all camp costs are presumed to be to the expense of CZN.

A number of more sweeping assumptions have been made regarding mining costs for breaking of ore and operation of the mine.
These are summarized as follows:

1. Mining costs are based on contractor supplied equipment and labour rates on a target cost contract.
2. All capital equipment will be supplied and maintained by the contractor.
3. Geological supervision and mine planning will be provided by the mine within the management overhead.
4. Services including power will be provided by the mine to the contractor.
5. Waste is presumed hauled to surface by Procon, although most likely it will be backlashed underground to reduce haulage costs.
6. Delineation drilling and exploration is provided by the mine at a fixed cost per tonne of $1.50.
7. Backfilling will be available as required and will be assumed to be 60% cemented and 40% un-cemented. Only 90% of all tailings are available for fill. Cost for cemented fill will be approximately $10 per tonne and for un-cemented fill will be $2.50, including labour, piping, etc.
8. Camp costs for all personnel, plus the costs of airfares, etc. have been included in CZN's labour sheet (Table 8, section 19.1).
9. No additional freight costs have been added to the mining cost. These are handled in the transport section of the model.
10. Mining costs for ore based on Procon's work, will be assumed to be $30 per tonne of Vein material mined using mixed, Alimak/Shrinkage stoping, $20 per tonne for short hole open stoping and $22 per tonne of Stratabound, all delivered to the mill, excluding access development costs and fuel costs, added separately.
11. Dilution is presumed to be 15% in Vein and 15% in Stratabound mineralisation. In all cases, dilution is presumed to carry 30% of the in situ grade of the ore.
12. Grades for material mined from the Vein and Stratabound areas of the mine are presumed to carry the grades of the overall resource. No attempt has been made to schedule the higher-grade material early in the development, with the exception of feeding only Vein ore during the development phase of the Stratabound ore (Year 1 and the much of Year 2).
13. It is assumed that sufficient ore will be developed in the early part of the mine life to allow the feeding of Vein material only to the mill (from three levels) until such time as additional Stratabound resources are developed.

These assumptions and the estimate from Procon have been combined into a financial model for the operation of the mine. This is presented in the economic analysis section of this report. The effects of varying the cost of mining through selection of differing mining methods can also be varied to confirm the effect of different mining methods on the overall profitability of the operation.

12. METALLURGY

12.1 INTRODUCTION

Numerous test work-studies have been carried out on the Prairie Creek ore since 1980. These studies were reviewed by Gary Hawthorn of Westcoast Mineral Testing Inc. as outlined below. The study of the various reports and proposed flow sheets produced to date indicate a changing picture regarding the intentions and priorities of the production regime.

The following sections discuss previous test work performed and current test work philosophy.

12.2 MINERALOGY

12.2.1 Vein Ore

The mineral composition of the Vein material consists of low iron sphalerite, galena and tetrahedrite-tennantite hosted in carbonate and lesser quartz. This material contains silver in solid solution with the tetrahedrite-tennantite and to a lesser extent with the galena. The material is somewhat oxidized (perhaps 15-20% of the lead and 10% of the zinc) to cerussite (lead carbonate) and smithsonite (zinc carbonate), but with only minor oxidation of the tetrahedrite-tennantite.

12.2.2 Stratabound

The Stratabound material contains pyrite, sphalerite and galena, totaling 60-85% sulphides. This zone contains essentially no copper. The silver is contained in solid solution with both the galena and sphalerite.

Both ore types contain mercury in solid solution (much higher in Vein-type ore) with both the sphalerite and tetrahedrite-tennantite, and as such follow the respective minerals into the various flotation concentrates.

12.3 METALLURGICAL TEST WORK – PREVIOUS WORKERS

12.3.1 Laboratory Review

G. Hawthorn of Westcoast Mineral Testing Inc. commented as follows:

Although the testing reports indicate that a primary grind to P80=50 microns is required, the data suggests otherwise, and P80=100 microns may be entirely satisfactory (G&T Metallurgical – 1994; See KM370 pg.15, KM440-13, and LR2252-13). KM370 indicates that grind is not a factor in lead roughing, and in zinc roughing it does not affect recovery, although the RC grade is higher with a finer primary grind. Useful grinding could be accomplished more cost effectively with regrinding. Note that operating plants benefit from differential grinding of the high specific gravity minerals, so they invariably operate at coarser grinds and higher throughput rates than is indicated from laboratory batch grinding tests.

Spreadsheets and graphs have been created from several of the reports (Lakefield Research – 1994: LR2252, G&T Metallurgical – 1994: KM370/ 462 (Stratabound), and KM440/454/462/474 (Vein)) so that this reviewer can better understand the lead rougher flotation variables and their effect upon process metallurgy. They suggest that no matter what lead rougher conditions are used with respect to grind and depressants, 20-30% of the zinc floats with the lead RC, the majority of which is returned to the zinc circuit via the lead cleaner tailing No. 1 (CT1). In all probability regrinding of the lead RC is beneficial. Unfortunately, several "depressants" recommendations were made based upon RC grade, while ignoring the much more important metal deportment.

The data indicated the need for regrinding of the zinc and probably the lead or copper/lead rougher concentrates. Although regrinding almost certainly has a very important role, the optimum location for regrinding was not established.

Virtually none of the testing included oxide metal analyses (the exception being KM497 using a very high-grade sample of about 3X the expected feed grade) and as such did not provide any useful data regarding the distribution of the sulphide minerals. The oxidation of lead and zinc is quite variable, but based upon head sample analyses, is generally about 20% and 10% respectively in the Vein material. The Stratabound material is not oxidized.

Oxide lead flotation was evaluated in a single test (LR2252-6), producing a significant incremental oxide lead recovery into a low-grade oxide RC.

The role of "depressants" was not well established. On reviewing the data, it became apparent that there was very little difference in the test results with and without the use of depressants. These include NaCN, lime, $ZnSO_4$, SO_2, and MBS (sodium metabisulphite). As a result of this, it was incorrectly concluded that depressants are very important to depressing the "naturally floatable" sphalerite. Lakefield (LR2252) concluded "The most effective zinc depressant was $ZnSO_4$". G&T in KM034 (Vein ore) states "Depression of pre-activated zinc by the extensive use of powerful depressants such as sodium cyanide appears to be mandatory". Later (KM440) they replaced cyanide with MBS.

G&T supported a lime-only approach to zinc depression for the Stratabound material in KM462, after earlier (KM370) stating "the lime-cyanide scheme using 500 g/tonne of each reagent offered the most promising flotation conditions".

G&T's endorsement for 5 kg/tonne of MBS for the Vein material is probably both not necessary and a very expensive option given that the cost of MBS, as shown in the Simon's 1995 study, is $0.75/kg.

Lakefield (LR2252) noted that "The slime gangue minerals (dolomite) and graphitic material complicated flotation", and in fact expended many tests in the evaluation of gangue "depressants". Later G&T included a prefloat stage in part of the series KM370 & 424 (Stratabound) without comment, and discontinued it without comment. The addition of a gravity circuit as currently proposed, would bypass this slimes question.

The most compelling issue is the presence of mercury in solid solution with tetrahedrite-tennantite and sphalerite that will limit the marketing options for both the copper and zinc concentrates. No amount of conventional mineral processing will eliminate this problem. This Scoping Study is based upon the production and sale of concentrates that will attract significant penalties but will be saleable.

12.3.2 Concentrate Quality

Several undesirable metals follow the various metals into concentrates because of the mineral composition of the Prairie Creek Deposit.

Antimony and arsenic are present in the copper minerals tetrahedrite and tennantite, so that any reduction in the content of either of these will also reduce both the copper and silver content.

Mercury (Hg) is present in the lattice structure of both tetrahedrite-tennantite and sphalerite, and within the processing plant nothing can be done to reject this undesirable metal without also rejecting the desired metals. The only options that exists to deal with mercury is to:

- Accept a substantial smelter penalty (the preferred choice), or
- Process the concentrates (one or both of the copper and zinc concentrates) through hydrometallurgical or pyrometallurgical circuits for which there has been very little industrial precedent, but which have been intensively investigated in recent years, particularly as a result of the development of more ion specific resins analogous to the well known copper SX-EW process.

12.3.3 Discussion of Findings

Despite the above, a number of key issues have been highlighted by the Company's metallurgical team which could prove helpful in the redesign of the process layout. These include the following areas of concern:

- Liberation of lead,
- Adverse effect of recycled water on lead recovery,
- Migration of undesirables to copper,
- Ineffectiveness of certain depressants,
- General philosophy of doing everything by flotation, and
- No de-slime despite presence of dolomite and graphite in the ore.

A study carried out by Simons in January 1995 indicated from previous reports that a target grind of 80% passing 50 micron would be necessary to achieve adequate liberation of all metals (the original target was 80% passing 70 micron). The treatment of a combined ore feed of 577,500 tonnes was proposed yielding 63,178 tonnes of lead concentrates and 108,273 tonnes of zinc.

Average recoveries from a combination of Vein and Stratabound ore were calculated at 69% for lead and 84% for zinc. Product quality was indicated as 71.5% lead and 57% zinc respectively in the concentrates. In addition, copper concentrates could be produced from Vein ore at a 60% recovery and 23% copper. In order to achieve the above throughput by flotation only, it was proposed to install a new ball mill, at a cost of up to $1,000,000 plus installation. It was also proposed that flotation roughing capacity be increased by the addition of six 8 cubic meter cells. Both of the above items would require a significant building extension.

Although the Simons proposal appears to be a simple extrapolation of the earlier process philosophy some practical difficulties would have had to be addressed before the target throughput could be realised. The settling pond is of limited capacity and there are no tailings thickening facilities. The existing solids recovery system for backfill consists solely of hydro-cyclones. Even if these were extended to include small diameter high pressure de-slime units it is doubtful that they would be capable of removing material of less than 5 to 10 micron. Consequently, the overflow that would have to be discharged to the settling pond would transport a significant amount of solids over quite a short period of time, particularly if the grind of 80% passing 50 micron was achieved. It is estimated that assuming that 500,000 tonnes of ore were to be processed between 10,000 and 20,000 tonnes of slimes would report to the settling pond.

An additional problem would be that because of the relatively small water circuit, a complex build up of reagents could take place, which would result in a steady deterioration of recycle water quality, which would in turn affect process recovery and mineral selectivity. To compensate for this, a water treatment plant was proposed in order that the tailings water would be of a good enough quality either to be discharged to the river or be recycled to the plant.

12.3.4 Practical Alternatives

A report produced by Hazen Research in 1997 demonstrated that some initial beneficiation of the ore could be achieved by gravity techniques. Although the work was carried out on minus 12 mm material, extrapolated results at minus 1.18 mm indicated that over 50% of the feed could be discarded as a sand tailing with a total metal recovery of around 95%. If this result proved to be representative, current management felt that increasing production to the required level without making major changes to the existing plant would be a possibility.

The current review of metallurgical response of the ore based on all the previous test work has lead to the identification of a number of areas for improvement in metallurgy. This has included a revisit to the basic flotation response of the ore with a more comprehensive test work program, the inclusion of some form of pre-concentration to enhance grade fed to the ball mill and the addition of a paste or thickened backfill system to the mill for tailings disposal underground.

With these aims in mind, two programs have been set in progress: a full metallurgical test work program at Westcoast Mineral Testing Inc. to examine the flotation response of both Vein and Stratabound ore, and a gravity based separation program with Camborne School of Mines Associates (CSMA) and South West Metallurgical Services to identify methods to pre-concentrate the ore and possibly produce a gravity lead concentrate.

12.4 CURRENT TESTING PROGRAM

12.4.1 Gravity Test Work Program

A 100 kg sample of Vein ore was sent to CSMA consultants and South West Metallurgical Services in England so that the amenability of the ore for pre-treatment by gravity could be evaluated.

Preliminary results from the pre-concentration test work at CSMA indicate that the ore is amenable to gravity separation. Consequently, trials were carried out using a spiral test rig at South West Mineral Services with the following results:

CSMA Results

Size	**S.G.**	**Weight %**	**Lead**		**Zinc**	
			Assay	**Dist**	**Assay**	**Dist**
-1 mm + 0.075 mm	-2.9	68.1	1.02	7.5	1.46	11.2
	+2.9	31.9	26.88	92.5	26.64	88.8
	Feed	100	9.26	100	9.49	100

Total lead/zinc recovery: 91.1%

South West Met. Services Results

Size	**S.G.**	**Weight %**	**Lead**		**Zinc**	
			Assay	**Dist**	**Assay**	**Dist**
-1 mm + 0.125 mm	Lights	38.8	1.01	4	1.76	7.5
	Heavies	61.2	15.17	96	13.65	92.5
	Feed	100	9.68	100	9.04	100

Total lead/zinc recovery: 94.2%

12.4.2 Gravity Test Work Conclusions

Both the sink/float test at CSMA and the spiral work at South West indicated that gravity pre-treatment of the Prairie Creek Vein ore should be considered. Between 38% and

60% of feed crushed to 1 mm could be removed by gravity with a loss of only 5 to 10% of metals.

The lead and zinc head grades to the ball mill could be increased 50 to 100%. In addition the possibility of producing a limited amount of lead concentrate as a gravity product was also demonstrated.

With this in mind the processing and water problems already highlighted could be greatly reduced and possibly completely eliminated if a gravity pre-treatment system was installed before the milling circuit. There would also be a considerable saving in power consumption for grinding due to a reduction in volume to the mill and a subsequent reduction in work index due to the removal of silica during pre-treatment. Reagent consumption per tonne of product would also be significantly reduced. No additional milling capacity would be required to achieve the new target throughput.

12.4.3 Recommendations From Gravity Test Work

The gravity test work clearly indicates the significant benefits of pre-concentration by spirals. The spiral plant will reject up to 38.8% of the feed for a total loss of metal of around 5.6%. This performance can be further enhanced by test work on site to fine-tune the spirals. This should include the setting up of a Pilot Plant at the mine that would include both gravity pre-treatment and flotation equipment.

A program of laboratory scale flotation has been commenced on the concentrate samples produced from the gravity pilot study. No results are as yet available for the flotation of the increased head grade sample generated by the gravity test work, however, assuming the results are promising an extended pilot study will be considered for 2001.

12.4.4 Discussion of Flotation Test Work

Preliminary metallurgical test work performed by Gary Hawthorn of Westcoast Mineral Testing Inc. has produced the following comments from his Progress Report 1 on the flotation response of Prairie Creek ore.

12.4.4.1 Review of Current Testing Program

<u>Vein Zone</u>

CZN prepared three samples, as follows:

Composite No.	Pb Oxidation %	Zn Oxidation %	Description
V-1	56	45	"Vein rep" from wall chipping
V-2	31	28	"Vein low" from wall chipping
V-3	42	55	second "Vein rep" from wall chipping sent to England
V-4	10	3	"Vein low oxide"

A total of five flotation tests have been completed and an additional two are being assayed.

The V-1 sample was found to be highly oxidized only after the testing was started. This resulted in the collection of the V-4 composite from drill core, where oxidation is more apparent than on the exposed underground faces where additional oxidation may have occurred in the 20 years since the earlier development program. Because of this, the focus of the flotation testing was shifted to less oxidized material (V-4 composite) that was thought to be more representative of the zone. Later, after the optimum conditions have been established on the V-4 composite, offsetting testing will be undertaken on the other composites.

The first four tests were performed on the V-1 sample. The fifth test was performed on the V-4 composite. The last two tests were on spiral concentrate and tailing derived from the V-3 composite.

The significant conclusions from these tests are as follows:

1. The recovery of sulphide lead and zinc into rougher concentrates is not bad, at up to 93% of the lead and 91% of the zinc in test W-0004. However, the metal distribution is not desirable, with 30-40% of the sphalerite typically floating in the lead rougher concentrate, and 10% of the galena in the zinc rougher concentrate. This deportment is not caused by natural activation since the sphalerite is readily rejected into the cleaner tailing using "dilution" flotation.

2. The role of slimes losses was investigated with composites V-1, V-3 (no assay results yet) and V-4 (incomplete data). In the case of V-1, the only sample with complete data to date, elevated losses of both the oxide and sulphide lead and zinc in the minus 325 tailing fraction occurred (see W-00-04). The oxide losses were very elevated in the minus 325 fraction, the sulphide losses less so. This is consistent with the 1982 CSMRI fractional analyses.

   Economically, this may not be so important since the rougher stage recoveries of both lead and zinc were quite good. Nevertheless, this does bear investigation by the performance of flash flotation testing.

3. As indicated in 2., elevated assay grades are reported in the "slimes" fraction of the rougher tailing. Aside from these slime losses, the grade distribution in the coarser fractions indicated that there is no severe grind sensitivity at the rougher flotation stage. This suggests that a nominal primary grind of 50% minus 200 mesh, possibly with flash flotation, may be entirely adequate.

4. The role of regrinding of zinc concentrate grade was investigated in W-00-16 (V-4 composite). Although no regrinding was actually performed, a screen analysis of the zinc cleaner concentrate indicates an unusual pattern with the highest grade in the coarsest (plus 200 mesh) fraction, with lower grades at finer fractions, caused in part by elevated galena. Note that this was not a particular high-grade concentrate at 47% zinc, and the main diluent appears to be "gangue" given the only 80% content of combined galena and sphalerite.

   The pattern of the data suggests that there are two generations of sphalerite. This was essentially confirmed by CZN who indicated that there is a considerable amount of recrystallization that has resulted in both coarse clean sphalerite and heavy locking with galena.

   These results are not consistent with the CSMRI pilot plant concentrate data, in which the zinc concentrate grades were quite good at 56-60% zinc, did not significantly increase or decrease in the finer fractions, and the lead grade did not significantly increase in the finer fractions.

   This is an important characteristic of this ore that needs to be more thoroughly investigated.

5. The role of regrinding on lead concentrate grade was investigated in W-00-17, for which assay results have not yet been received. However, the CSMRI report indicates a similar pattern to the lead concentrate grade distribution that was reported for the zinc concentrate in test W-00-16.

6. The role of depressants during lead rougher flotation was investigated by using no depressant and comparing the results to the earlier G&T results in which $Na_2S_2O_5$ was used to "depress" sphalerite. The graphical data clearly indicates that in both cases the selectivity against zinc was poor but equally so. Deleting the G&T recommended 5 kg/tonne of $Na_2S_2O_5$, and using only a weak collector does not significantly alter the weight recovery versus metal recovery curves, but will significantly decrease the reagent costs.

7. Note that G&T used Minerec 2030 collector and this writer used Cytec 3418A. Either of these reagents performs well in lead flotation and provides good selectivity. The choice of 3418A by this writer was one of convenience and familiarity rather than any suggestion of superior performance. The less expensive SIX was used in the zinc roughers where selectivity is not an issue.

Note that in both tests W-00-11 (V-1 sample) and W-00-16 (V-4 sample), the vast majority of the zinc that floated in the lead rougher stage reported to the cleaner tailing using dilution flotation and no regrinding.

8. Bulk copper/lead rougher flotation was investigated in the two tests noted above, and in both cases the bulk concentrate was cleaned once, then subjected to separation. In W-00-11, lime at pH 11.4 (intended to depress only the galena) depressed both minerals. W-00-16 used both $Na_3S_2O_3$ and lime (pH 10.7) and was much more effective. The next test will investigate the use of lime only since in W-00-16 the two depressants may have been consuming each other.

9. The absence of pyrite in this ore type makes the flotation control much easier than it other wise would have been, in that the circuit can be operated at natural pH, with the probable exception of the copper/lead separation stage.

Stratabound Zone

CZN prepared a single composite from drill core. This was deemed to be satisfactory because of the absence of oxidation.

Eight tests were performed on this composite, mainly to investigate grind dependency.

The major comments on this testing include:

1. In grinds ranging from 45% to 99% passing 200 mesh, there were only modest differences in any of weight distribution, metal recoveries and distribution, and pyrite rejection. This strongly suggests that a primary grind any finer that 50-60% minus 200 mesh cannot be justified.

2. The data from two tests (W-00-02 and W-00-05) indicate that lead losses increase in the finer tailing fractions, as does zinc, although less emphatically.

3. In W-00-05, flash flotation was investigated, and was found not to be beneficial. In both cases, the overall rougher recoveries were satisfactory, so the slime losses were modest.

4. Test W-00-13 included cleaner flotation of three sequentially produced rougher concentrates on material that have been ground (not reground) to essentially all minus 200 mesh. The data indicated that the lead concentrate was diluted equally with sphalerite and pyrite. The zinc concentrate was diluted mainly with pyrite, suggesting an important role for lime in zinc cleaning both for the rejection of galena and pyrite. Note that optical microscopy indicated virtually complete liberation of the pyrite.

5. Reagent testing indicated that replacing the G&T proposed 2 kg/tonne of lime and SEX collector in lead roughing with Cytec 3418A produced similar weight recovery versus metal recovery curves, while significantly decreasing the reagent

costs. The use of lime will continue as a pyrite depressant in both lead cleaning and throughout the zinc circuit. Note that although 3418A is quite expensive, at $9.00/kg, the usage cost of about $0.09/tonne of ore is considerably less than the otherwise $0.30/tonne for lime plus about $0.03/tonne for PEX.

3418A was also used in the zinc roughing circuit to provide selectivity against pyrite.

12.4.4.2 Gravity Concentration

The gravity concentration studies conducted in 2000 have already been reported above. Comments below are therefore in the context of their influence upon future flotation testing.

The results of the preliminary gravity concentration tests, exclusively on Vein material, suggest that this is a very favourable and cost effective procedure to reject 40-60% of the original feed and only a very small proportion of the lead and zinc.

A second desirable feature of this procedure (one that is not so cost effectively achieved by flotation) is its ability to concentrate both oxide and sulphide lead and zinc into separate metal mineral products that will lend themselves to easier and less expensive downstream flotation.

This test work produced >20 kg of concentrate, that will provide the feed for future flotation testing of Vein ore.

12.4.4.3 Future Testing

<u>Locked Cycle Testing</u>

Both of these ore types are relatively complex from a processing perspective.

In the case of the Vein Zone, the batch bench scale testing indicates that 20-30% of the zinc reports to the lead cleaner tailing. All by itself, this indicates the need for locked cycle testing. Note that in the G&T locked testing (see KM-444-16), the zinc in the lead cleaner tailing floated quite well in the zinc roughing circuit, but about 50% of the lead and zinc losses were in the zinc retreatment tailing.

The Stratabound zone testing to date indicated elevated iron in the lead concentrate, suggesting the desire to eliminate this material, probably in the lead cleaner tailing. This also suggests the need for locked cycle testing.

<u>Vein Zone</u>

Only limited work, and that limited to cleaner flotation optimization, needs be done at the present time. As indicated above, that work will be undertaken on the retained sample of gravity concentrate.

A final stage of bench scale testing, including locked cycle testing, will be performed in conjunction with the proposed Pilot Plant gravity (spiral) concentration studies.

Stratabound Zone

Most of the important control parameters for this material have now been identified. However, because this material will not be subject to the proposed 2001 pilot studies, it is worthwhile to continue with the bench scale testing.

The program will almost certainly be limited to <4 tests to investigate the role of lime in both the lead and zinc cleaner circuits.

12.5 PREDICTIVE METALLURGICAL RESPONSE OF PRAIRIE CREEK ORES

In order to produce a financial and performance model for evaluation of the Project, it is necessary to prepare recovery estimates for metals to flotation concentrates. The predictive response of the mineralisation at Prairie Creek to conventional flotation has, in the past, been based on testwork by G&T, Lakefield and others in testwork carried out in periods up to 1994. This testwork was summarized by Simons in 1995 as follows:

12.5.1 Original Metallurgy – Vein Material

(From Simons Report 1995.)

Product	Tonnes / year	WT %	Ag g/t	Cu %	T Pb %	Ox Pb %	S Pb %	T Zn %	Ox Zn %	S Zn %
Cu conc	4,459	1.1	7500	23.0	10.0	0.2	8.8	12.0	0.1	11.9
Pb conc	50,591	12.0	650	0.8	75.0	0.5	74.5	5.0	0.0	5.0
Zn conc	80,843	19.2	70	0.1	4.0	0.0	4.0	57.0	0.5	56.5
Total Con	136,253	32.4								
Tailing	283,747	67.6		0.07	5.04	4.0	1.1	2.5	1.8	0.7
Feed	420,000	100.0	207	0.4	13.4	3.0	10.4	13.4	1.3	12.1

DISTRIBUTION

Product		WT %	Ag	Cu	T Pb	Ox Pb	S Pb	T Zn	Ox Zn	S Zn
Cu conc			38.6	60.0	0.8	0.1	2.3	1.0	0.1	1.0
Pb conc			37.9	23.7	67.5	2.0	86.5	4.5	0.0	5.0
Zn conc			6.5	4.7	5.8	0.0	0.7	82.0	7.4	90.0
Total Con			83.0	88.4	74.5	2.1	89.5	87.5	7.5	96.1
Tailing			17.0	11.6	25.5	97.9	10.5	12.5	92.5	3.9
Feed			100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0

Oxidation levels for these samples were high at 22% oxidised for the lead and 10% oxidised for the zinc in the sample. Generally, oxide levels in the deposit decrease with depth and are as low as 3% oxidised from drill holes below the water table. As can be seen, total metal recoveries to individual concentrates were poor (total lead to lead recovery 67.5% and total zinc to zinc 82.0%). This was largely an effect of cross contamination and poor recovery of oxide lead and zinc. However interestingly enough, recoveries of sulphide lead to the lead concentrate (86.5% recovered) and sulphide zinc to the zinc concentrate (90.0%) were reasonable and indicate better performance can be achieved in low oxide environments, or if the oxide minerals can be adequately sulphidised within the process. Mercury and other impurity levels were higher in the Vein mineralisation, since these impurities appear inextricably locked into the mineral matrix.

General levels of impurities in the Vein concentrates were as follows:

Product	Fe %	As %	Sb %	Hg ppm
Copper Concentrate	0.7	5.0	12.0	3,200
Lead Concentrate	1.2	0.2	0.5	250
Zinc Concentrate	0.8	0.2	0.5	2,000

As discussed in the later concentrate quality section, the high impurity levels in the copper concentrate are offset by the exceptionally high silver assays, resulting in a low but positive NSR and a way to reduce the penalty elements reporting to other concentrates. The lead concentrate is clean and of high value because of the high silver count and the zinc concentrate will attract a mercury penalty as currently composed.

12.5.2 Original Metallurgy – Stratabound Material

(From Simons Report 1995.)

Product	Tonnes / year	WT %	Ag g/t	T Pb %	T Zn %	Fe %	As %	Sb %	Hg ppm
Pb conc	12,227	7.8	400	57.0	4.0	12.0	0.03	0.05	40
Zn conc	27,429	17.4	160	2.0	57.0	3.8	0.00	0.00	570
Total Con	39,656	25.2							
Tailing	117,844	74.8		1.04	0.95	6.9			
Feed	157,500	100.0	61	5.9	10.8				

DISTRIBUTION

Product		WT %	Ag	T Pb	T Zn	Fe	As	Sb	Hg
Pb conc			51.0	75.0	1.4				
Zn conc			45.8	11.8	92.0				
Total Con			96.7	86.8	93.4				
Tailing			3.3	13.2	6.6				
Feed			100.0	100.0	100.0				

No copper concentrate is produced from the Stratabound ore, principally because of the very low levels of copper contained. Otherwise both concentrates produced are highly saleable and demonstrate the clean nature of the Stratabound ore.

Total recoveries of metal to concentrate are reasonable with 75% of the lead going to lead concentrate and 92% of the zinc going to zinc concentrate. No sulphide/oxide analysis splits were undertaken.

12.5.3 Predicted Metallurgy – Post 2000 Testwork

The work carried out to date in 2000 has identified a treatment route which includes the provision of gravity separation to upgrade the feed to flotation and provide three separate flotation streams to prevent cross contamination and maximize recoveries. The effect of gravity separation is to reject 40% to 60% of the feed at around 0.5 mm particle size, while losing only around 5% of the contained metal. This has the effect of doubling the head grade to flotation from around 10-12% to around 18–22% lead and zinc.

From the earlier testwork it is difficult to reconcile whether the grade recovery relationship for Prairie Creek ore is based on constant tail, constant recovery, or more likely a combination of the two. No testwork has been carried out to-date on varying the grade while at the same time measuring recovery. Logically, recoveries can not be constant since, if one feeds 100% material to a flotation circuit then one should recover significantly more than say 67.5% of the contained mineralisation. However due to the presence of locked particles, some metals losses will occur. As a corollary, at low grades and below a certain grade, recoveries to concentrate will be zero.

As a consequence and since no flotation testwork has as yet been reported on gravity concentrate (one flotation test has so far been carried out in December 2000 but assay results are not yet available), estimates will have to be made of the likely recoveries, assuming splitting of the circuits and efficient sulphidisation of oxide mineralisation.

The recoveries postulated off the back of the work carried out in 2000 are summarized below. The proposed Pilot Plant will be used to confirm these predictions and to fine tune the circuits proposed and the costs of treatment within the mill.

12.5.4 Predicted Metallurgy – Gravity Section

The introduction of a gravity circuit allows the mine to mechanise and reduces the concern of dilution on the overall process, since waste material introduced by dilution is rejected in the gravity portion of the mill prior to flotation. This has the effect of reducing the mining cost significantly, due to mechanisation, while only increasing the cost to the mill for crushing and gravity treatment of the extra material. As a consequence, in any calculation of grade and recovery through the gravity plant, dilution is added to the feed grade at 15% of the weight, assuming a grade of 30% of fed material. This has the effect of reducing the head grade to the gravity section of the mill and increasing the overall tonnes.

VEIN ORE

Product	Wt %	Cu %	TPb %	TZn %
Ore in-situ	85	0.40	11.40	12.50
Ore diluted (1)	100	0.36	10.20	11.19

Note: (1) 15% dilution at 30% of feed grade

Based on the earlier testwork by Lakefield, G&T et al. supported by the more recent work carried out by CSMA and SWMS in 2000, the effective response of the gravity section of the proposed mill will be as follows:

Gravity Concentration Stage (minus 32 mesh feed)

Distribution data from Hazen 1997 / tables 10-12

Grades

Product	Wt %	Cu %	TPb %	TZn %
Concentrate	60	0.55	16.32	17.53
Tailing	40	0.06	1.02	1.68
Mill Feed	100	0.36	10.20	11.19

Distribution

Product	Wt %	Cu %	TPb %	TZn %
Concentrate	60	93	96	94
Tailing	40	7	4	6
Mill Feed	100	100	100	100

Similarly the Stratabound mineralisation will be diluted as follows:

STRATABOUND ORE

Product	Wt %	TPb %	TZn %
Ore in-situ	85	5.00	10.30
Ore diluted (1)	100	4.48	9.22

Note: (1) 15% dilution at 30% grade

Subsequent gravity treatment of Stratabound mineralisation should produce the following response:

Gravity Concentration Stage (minus 32 mesh feed)

Grades

Product	Wt %	TPb %	TZn %
Concentrate	60	5.98	12.31
Grav. Tailing	40	0.45	0.92
Mill Feed	100	4.48	9.22

Distribution

Product	Wt %	TPb %	TZn %
Concentrate	70	96	96
Grav. Tailing	30	4	4
Mill Feed	100	100	100

(NB. No gravity testwork has been carried out on Stratabound ore to-date and the above performance figures are estimates only subject to testwork)

12.5.5 Predicted Metallurgy – Flotation of Vein Material

Current predictions on flotation response of the higher-grade feed from gravity, plus the 15% fines bypassing the gravity section, assuming efficient sulphidisation, are as follows:

Flotation Stage

Grades

Product	Wt %	Cu %	TPb %	TZn %
Cu Conc	1.4	23.00	10.00	12.00
Pb Conc	17.8	0.80	75.00	5.00
Zn Conc	27.7	0.10	4.00	57.00
Tailing	53.0	0.10	3.18	1.29
Flot. Feed	100.0	0.55	16.32	17.53

Distribution

Product	Wt %	Cu %	TPb %	TZn %
Cu Conc	1.4	65.5	0.9	1.0
Pb Conc	17.8	25.7	82.0	5.1
Zn Conc	27.7	5.0	6.8	90.0
Tailing	53.0	3.8	10.3	3.9
Flot. Feed	100.0	100.0	100.0	100.0

These responses are hypothetical and require verification by testwork and subsequent pilot trial. They do however represent the most reasonable assumptions that can be made based on testwork carried out to-date and the opinions of the various metallurgists currently engaged and their expectations of a properly run flotation circuit.

12.5.6 Predicted Metallurgy – Vein Material Overall

Combining both the gravity circuit and the flotation circuit produces the following compound recoveries:

Overall

Grades

Product	Wt %	Cu %	TPb %	TZn %
Cu Conc	0.9	23.00	10.00	12.00
Pb Conc	10.7	0.80	75.00	5.00
Zn Conc	16.6	0.10	4.00	57.00
Gravity tailing	40.0	0.06	1.02	1.68
Flot. tailing	31.8	0.10	3.18	1.29
Mill Feed	100.0	0.36	10.20	11.19

Distribution

Product	Wt %	Cu %	TPb %	TZn %
Cu Conc	0.9	55.5	0.8	0.9
Pb Conc	10.7	23.9	78.7	4.8
Zn Conc	16.6	4.6	6.5	84.6
Gravity tailing	40.0	7.0	4.0	6.0
Flot. tailing	31.8	8.9	9.9	3.7
Mill Feed	100.0	100.0	100.0	100.0

Silver, impurities and other assays are presumed to be as per earlier testwork.

12.5.7 Predicted Metallurgy – Stratabound Material Flotation

(From Simons 1995 study.)

Current predictions on flotation response of the higher-grade feed from gravity, plus the 15% fines bypassing the gravity section, based on Simons Study are as follows:

Grades

Product	Wt %	TPb %	TZn %
Pb Conc	11.6	57.00	4.00
Zn Conc	23.8	2.00	57.00
Flot. Tailing	64.6	0.15	1.12
Flot. Feed	100.0	7.16	14.75

Distribution

Product	Wt %	TPb %	TZn %
Pb Conc	11.6	92.0	3.1
Zn Conc	23.8	6.6	92.0
Flot. Tailing	64.6	1.4	4.9
Flot. Feed	100.0	100.0	100.0

These figures are considered reasonable in the light of current knowledge and require confirmation by further test work and in the Pilot Plant. Impurities and other metals contained are assumed to be as per earlier testwork.

12.5.8 Predicted Metallurgy – Stratabound Overall

Combining the flotation and the gravity recoveries produces the following overall figures for the Stratabound mineralisation.

Overall

Grades

Product	Wt %	TPb %	TZn %
Pb Conc	6.9	57.00	4.00
Zn Conc	14.3	2.00	57.00
Grav. tailing	40.0	0.45	0.92
Flot. tailing	38.5	0.15	1.10
Mill Feed	100.0	4.48	9.22

Distribution

Product	Wt %	TPb %	TZn %
Pb Conc	6.9	88.3	3.0
Zn Conc	14.3	6.4	88.3
Grav. tailing	40.0	4.0	4.0
Flot. tailing	38.5	1.3	4.6
Mill Feed	100.0	103.4	100.0

These overall figures for the Vein and the Stratabound mineralisation are those that have been used in the financial evaluation of the property for the purposes of this Scoping Study. Interpretation of results of these tests can be considered subjective and requires confirmation through the operation of a Pilot Plant on site. However, they do indicate the significant benefits to the Project of the addition of a gravity plant to the mill.

12.6 PROPOSED PILOT PLANT PROGRAM

12.6.1 Purpose

Preliminary test work and metallurgical review has indicated that installation of a gravity circuit and thickened or paste backfill circuit will provide a number of significant benefits to the Prairie Creek Mine. These benefits will include:

- Increased throughput
- Improved metallurgical recoveries
- Better water quality
- Reduced power per tonne in the mill

- Reduced reagent consumption per tonne
- Placement of the majority of tailings underground as thickened or cemented fill
- Reduced tailings pond requirements

Because of the obvious potential benefits of the proposed changes to the milling process for the operation as a whole and the environment, it is important that these findings be confirmed by the operation of such a Pilot Plant at the Prairie Creek site. As a consequence, it is proposed to operate a portable 1.5 tonne per hour Pilot Plant within the existing Prairie Creek mill during 2001. The Pilot Plant should be designed in such a way that every aspect of the operation can be investigated so that teething problems which invariably arise during the start-up of the main mill are minimised.

A pilot plant installation which could treat 1 to 2 tonnes per hour on a continuous basis would not only confirm the process philosophy but would be useful in highlighting unforeseen problems that may exist. It should incorporate both gravity and flotation separation as well as having facilities for thickening and filtration equipment. The final design of the main circuit could then be completed with a high degree of confidence. The tailings produced could be used in experimental thickened and paste backfill trials and significant tonnages of concentrates would be available for customers to assess.

The operation of an on-site Pilot Plant is extremely important to the development of the Prairie Creek Mine and will provide operating and financial certainty for the eventual mine design and flow sheet. It will also allow detailed analysis of the likely volumes and quality of the water required for the ultimate milling process and the likely quality and quality of water required to be disposed of, after use in the treatment plant. Additionally, the tailings from the Pilot Plant will allow completion of test runs for paste and thickened backfill and allow accurate design, sizing and placement of any tailings facility that is required beyond placing the tailings underground.

A pilot plant has been located for purchase in the UK and can be airlifted in sections into the minesite (see Plates 8-11). The plant would be placed in the mill building and draw on existing power supplies. The operation would be self-contained and operate at approximately 1.5 tonnes per hour, fed from the existing coarse ore stockpile via the current mill crushing circuit. Tailings from the plant would be stored in the two-onsite 35-foot thickeners inside the mill building to simulate tailings production and recycling of process water.

Fresh water requirement for the Pilot Plant would be approximately 10 gpm extracted from the Prairie Creek Valley aquifer. Reagents would be taken from existing onsite stockpiles or flown in by aircraft in small batches. Power would be supplied from existing on site generators.

It is expected that the plant would be operated for 10-24 hours per day over four months and that the concentrate produced would be stored on site in 2 tonne big bags for test marketing purposes. The total tonnage treated will be in the region of 1,000-2,000 tons of ore taken principally from the surface ore stockpile, or from broken ore underground. Tailings consisting principally of inert sands would be stored in one of the two thickeners

Plate 8 – Pilot Plant - Crusher

And

Plate 9 – Pilot Plant - Mill and Classifier

Over Size photo.

If you would like a copy, please contact the Company at 1-866-688-2001 toll free or 604-688-2001.

Plate 10 – Pilot Plant - Conditioner Pumps and Cells

And

Plate 11 – Pilot Plant - Vacuum Filter and Ancillaries

Over Size photo.

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inside the mill building. Subsequently the tailings would be assessed for suitability for paste or thickened backfill. On start up of the plant proper, these tailings would be returned to the mine as backfill, or to another approved tailings storage site. Water would be recycled between the two thickeners and the plant to study the changes in water chemistry brought about by the milling process. This information will be invaluable for design of the ultimate water treatment process for the operating mine.

A suitable pilot plant has been located and can be airlifted into the site.

12.6.2 Pilot Plant Operation

The plant would be operated 10-24 hours per day for approximately four months and would provide the following information for the full feasibility study:

- Firm metallurgical performance data, including recoveries, grades, grind sizing, cell sizing, etc.
- Firm design and throughput data for the eventual mill overhaul.
- Actual concentrate available to potential customers for evaluation and treatment charges estimates.
- Actual reagent usage data scalable to full size operation.
- Actual operating cost data that can be extrapolated to the full size plant.
- Actual water quality and volumetric data for permitting and eventual plant operation.
- Firm data on tailings quality and suitability for paste or thickened backfill.
- Actual volumes of tailings that can be placed underground and the likely cost and volume implications for any other tailings or settlement pond facility required.

Key operating parameters for the Pilot Plant will include:

- All Pilot Plant components should be air portable or currently available on site.
- Water requirement of 10-15 gallons per minute extracted from the Prairie Creek Valley aquifer by existing well and from recycle water from the 35-foot thickeners, during the operation of the plant.
- Total of approximately 2,000 tonnes of ore from the current coarse ore stockpile or from existing broken ore underground fed to the Pilot Plant.
- All tailings from the Pilot Plant stored within the mill building in the existing 35-foot thickeners.
- Concentrate produced will be stored in the mill building in 2-ton big bags or flown off site for test marketing.
- Power will be generated on site using existing diesel supplies and generators.
- Personnel involved in the operation of the Pilot Plant will be accommodated within the existing camp facility.

12.6.3 Areas of Investigation for the Pilot Plant

The study should be set up to yield information on key aspects of the process.

These would include the following:

- Influence of pre-treatment
- Cut points of pre-treatment
- The possibility of the production of a proportion of lead product by gravity separation
- Milling power requirement
- Liberation size
- Lead sliming
- Order of metal separation
- Required cell capacity
- Reagent regime
- Reagent costs
- Power costs
- Water recycling
- Product quality (including both metal and moisture content)

In addition, the above information will be used to build up a complete financial model of the process operation for the feasibility study.

12.6.4 Pilot Plant Equipment Requirements

The following equipment would be required in order to test the current process philosophy:

- Feed bin (about 10 tonnes capacity)
- Feeder (up to 3 tonnes per hour)
- Feed preparation screen fitted with 1 mm panels
- Roll crusher
- 4 Mineral spirals
- 1 Laboratory scale mineral table
- Classifier
- Ball mill (7 to 10 kW capacity)
- Conditioning tanks (minimum 2)
- Flotation cells (3 banks)
- Thickener
- Vacuum pump c/w ancillary equipment
- Pumps for the above equipment
- Electric control panel
- AA analysis machine and appropriate lab equipment

12.6.5 Cost of Pilot Plant Program

The following is an approximate estimate of costs of the Pilot Plant program.

Ex-works United Kingdom	$ 122,600
Shipping to Fabricator in Canada	10,000
Assembly and testing in Canada (8 weeks)	
Labour	60,000
Material	25,000
Shipping to Site	20,000
Assembly at Site (6 weeks)	
Labour	70,000
Commission Existing Equip.	20,000
Operation of Plant (16 weeks)	
Labour	160,000
Management	45,000
Camp Services	12,000
Airfare	10,000
Laboratory Services (Environmental Testing)	10,000
Total Pilot Plant Costs	$ 564,600

12.7 METALLURGICAL BASIS/PROCESS PHILOSOPHY

Metallurgical test work continues; however, the work carried out to date is sufficient to establish a process philosophy for the plant as a whole and to propose a flow sheet for Scoping Study purposes.

Laboratory studies have indicated that the Vein ore from Prairie Creek Mine is amenable to gravity separation. Three independent studies have revealed that between 38% and 65% of the gangue can be removed from the feed at a size of minus 1 mm with a total metal loss of between 5% and 9% depending on the separating density selected. A feed enrichment factor of 2.5 can be achieved. In addition it has been shown that lead levels in a proportion of the ore can be enriched from 10% in the feed to 54% in a pre-concentrate.

The above facts make possible various possibilities regarding flotation procedures and cell layout and indicate that the ore can be treated in three individual streams. The flow diagrams on the next few pages illustrate the possible arrangements of separate circuits in the mill.

This process philosophy was used to generate the design criteria for the processing of the Prairie Creek ore and is detailed in Section 10 of this study.

12.7.1 Circuit 1

This circuit illustrates how the respective components of the ore can be separated into individual streams because of their specific gravity difference.

In order of descending specific gravities the components of the ore can be arranged as follows:

Mineral	Specific Gravity
Galena	7.4 – 7.6
Cerussite	6.5 – 6.6
Lead/zinc/copper middling	5.5 (approx)
Pyrite	4.9 – 5.2
Tetrahedrite	4.6 – 5.1
Tennantite	4.6 – 5.1
Smithsonite	4.3 – 4.4
Sphalerite	3.9 – 4.1

The process philosophy utilizes the above specific gravity differences in order to create three different flotation streams, plus waste.

- High Density Stream, plus 6.5 specific gravity (containing mostly lead sulphides)
- Middle Density Stream, plus 5.0 specific gravity (containing mostly middling copper/lead/zinc)
- Low Density Stream, plus 3.5 specific gravity (containing mostly zinc sulphides)
- A gravity tailings stream, minus 3.5 specific gravity for disposal to backfill

The flow diagram illustrates only one of the various possible spiral arrangements that could be adopted. The results from pilot scale trials will confirm the optimum layout for this particular ore.

12.7.2 Circuit 2

This stream will process mineral with a specific gravity greater than 6.5. The feed will consist of predominantly lead rich ore with some contamination from zinc and copper middlings.

Flow diagram Circuit 2 illustrates one possible cell and re-grind arrangement, which will accommodate the proposed philosophy. Heavies will be taken from the first spiral cleaner SP2, classified if necessary, and treated in a conventional lead sulphide flotation circuit.

First cleaner tailings will be sent for re-grind before being sulphidised and re-floated in a scavenger cell. Scavenger cons will be returned to the first cleaner cell where newly liberated galena and sulphidised cerussite will be floated. Further cleaning stages will be used until the product has reached an acceptable quality. The final product will be

Figure 17 – Circuit 1 Gravity Separation

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Figure 18 – Circuit 2 Lead Recovery

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sent to a thickener prior to filtration and stockpiling. Overflow water from this thickener will be re-cycled to the pulp density regulation tank at the beginning of the flotation circuit.

Tailings from the lead scavenger cell will be a lead/zinc middling and may contain some amounts of copper. These will be sent to a thickener in order to recover as much of the lead circuit water as possible for internal re-cycling. The thickened lead/zinc slurry plus any "lights" from a second stage of ultra fines separation will be sent to the zinc circuit (Circuit 3). Internal re-cycling of the water in this way reduces the demand on the outside supply as well as preventing water with a high sulphide level being prematurely brought into contact with the oxidized zinc.

12.7.3 Circuit 3

The light fraction from both spiral SP2 and the ultra fines separation will be combined and fed to spiral SP4. This spiral should be capable of separating the lead/zinc/copper middling along with the tetrahedrite and tennantite from the remaining sphalerite and smithsonite. Although normally the spiral would not be capable of separating the minus 75 micron sphalerite and smithsonite on a gravity basis, this mineral will not be lost as it will report to the light fraction in any case because of its size range.

The heavies from spiral SP4 will be sent to Circuit 4. The lights will be conditioned with copper sulphate prior to a first stage of flotation. Tailings from this stage will be reground and sulphidised with sodium sulphide prior to being sent to a scavenger cell. Tailings from the cell will be sent to the thickening and paste backfill circuit. The subsequent froth products will be re-cleaned until a suitable product quality has been achieved.

12.7.4 Circuit 4

The heavies from spiral SP3 will be reground to about 40 micron to liberate any locked lead, zinc or copper. The copper and lead will then be fed to a bulk rougher cell where they will be separated from any remaining zinc.

The option to send tailings from this cell back to the zinc circuit is shown, although it may be necessary to discard this material as a final tailings rather than raise any arsenic or antimony levels in the zinc concentrates because of contamination from residual copper minerals. Cleaning will continue until a satisfactory copper/lead concentrate is achieved. Copper will then be separated from the lead by raising the pH to 10.5 with lime addition.

Figure 19 – Circuit 3 Zinc Recovery

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Figure 20 – Circuit 4 Copper/Lead/Zinc from Middlings

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12.7.5 Backfill Circuit

The tailings from Prairie Creek when combined with the rejects from the gravity pre-concentrator should be suitable for placement underground using thickened or paste backfill. The tailings from the flotation circuit will be thickened using flocculent in one of the two 35 foot diameter thickeners and mixed with rejects cycloned from the tailings stream of the gravity plant. Thickened tailings would be pumped to belt filters that would reduce the moisture content of the tailings stream to approximately 12-18% moisture, from where the thickened tailings would be fed to a pug mixer.

In the pug mixer, the tailings would be mixed with 3-5% cement or fly ash as required and the resultant mixture pumped to the mine using the existing Geho pump or a substitute positive displacement pump. Mix strength and loop pumping tests will be carried out using the Pilot Plant tailings to set final mixes and pumpability parameters, however, it is expected that the vast majority of the tailings can be placed underground in this manner.

12.8 ENVIRONMENTAL TESTING

The Pilot Plant will produce information pertaining to the following environmental concerns:

- Characterisation of effluent water quality
- Characterisation of solids quality
- Acid-Base Accounting test work
- Tailings quality and quantity plus disposal methods

Samples will be collected regularly, preserved and shipped to an environmental laboratory for analysis.

13. PROCESSING

Ore will be delivered to two separate stockpiles, Vein or Stratabound, from the mine. A loader will feed the crushing plant from the stockpiles. The different ores will be batched through the existing crushing plant.

The following components will be used for recovery of metals from the ore: a crushing system, a gravity concentration circuit, flotation circuits, thickening and dewatering, and tailings backfill.

The crushing plant is designed to accept run-of-mine ore from the underground operation and produce a minus 10 mm product. The crushing plant is planned to operate for one 12-hour shift per day and 340 days per year. The mill facility will operate for two 12-hour shifts per day and 340 days per year. To meet the annual production rate of 577,500 tonnes, a design throughput of 178 tph is required for the crushing plant. The milling circuit designed production rate is calculated to 72 tph.

Based on the metallurgical recoveries discussed in Section 12, the mill will produce 70,000 tonnes of lead concentrate, 115,000 tonnes of zinc concentrate and 6,500 tonnes of copper concentrate annually.

The following sections describe the aforementioned systems that will be necessary for producing the final concentrates. The general flow sheet for crushing and gravity concentration is illustrated as Figure 23, the lead and zinc flotation circuit flow sheet is illustrated as Figure 24, and the copper flotation and tailings backfill circuit is illustrated as Figure 25.

13.1 EXISTING FACILITY

The existing plant at the Prairie Creek Mine site, while almost complete, was designed for a somewhat different flow sheet than is now planned. The existing crushing plant is adequate for the 1,750 tonnes per day production rate. The milling circuit will be modified and the mill building extended by two bays to accommodate a gravity concentration circuit prior to flotation. The flotation circuit piping will be modified in order to use as much of the existing equipment already in place. The flotation circuit will produce a lead, zinc and copper concentrate with silver as a by-product compared to the original design of producing silver with lead, zinc and copper concentrates as by-products. The existing primary ball mill and regrind ball mill will remain, with the addition of a second ball mill for zinc regrind and a third ball mill for copper regrind.

Thickening and dewatering will use two high capacity thickeners in the mill extension instead of the existing conventional thickeners. The conventional thickeners will be used for the tailings backfill circuit. The thickened tailings for paste backfill will be pumped underground after dewatering and cement addition in a new backfill section in the mill extension.

13.2 CRUSHING

Ore will be mined as Vein and Stratabound material for processing through the crushing circuit. All ore will be crushed to 80% passing 10 mm in size through the existing two stage crushing and screening facility. The ore will discharge from the crushing plant into a 1,800 tonne storage bin. Scheduling of crushing has been based on crushing Vein ore for two days and then crushing Stratabound ore every third day.

The existing crushing system is of standard and straightforward design and, with properly scheduled preventative maintenance, should be capable of sustaining a high degree of mechanical and operating availability. The following subsections describe the major items of the crushing system.

13.2.1 Primary Crushing

Ore will be delivered to a stockpile area adjacent to the existing ore dump that feeds conveyor Number 1. The ore will be conveyed to a 40 tonne coarse ore bin. The coarse ore will be fed to a grizzly by an apron feeder. A grizzly feeder will screen the minus 100 mm material from the feed and discharge the oversize material into the primary crusher. The crusher product and the minus 100 mm material will be combined on a belt conveyer located below the primary crusher. The throughput for the primary crusher is 1,788 tph. A jaw crusher with a feed opening of 610 mm x 914 mm and a closed side setting of 75 mm will be used to produce an 80% minus 100 mm product. The primary crushing unit will be powered by a 37 kW (50 hp) electric motor.

The combined grizzly undersize and primary crusher product will be conveyed to a 40 tonne surge bin. The surge bin will feed the secondary rod screen. The oversize from the rod screen will feed a 1,676 mm (5 ½foot) short head cone crusher. The undersize material will drop to conveyor Number 7 that feeds the 1,800 tonne fine ore bin.

13.2.2 Secondary Crushing

Ore will be delivered to a 1,524 mm x 4,267 mm double deck rod screen ahead of the secondary crusher. The screen will be fitted with a 25 mm top deck and a 10 mm bottom deck. Oversize material (+25 mm) will pass to the secondary crusher.

The throughput for the secondary crusher is 178 tph; the product will be nominally 80% -10 mm. The secondary crushing unit will have a 150 kW (200 hp) electric motor, an automatic lubrication system and water-cooled bearings.

The discharge from the secondary crusher will be conveyed to the 40 tonne surge bin and subsequently to the rod screen in closed circuit. The undersize material from the rod screen will be conveyed to the fine ore bin. Ore is withdrawn from the fine ore bin and fed to the double deck sizing screen.

13.3 MILLING

The following section discusses the components and design basis to reduce the ore from the crushed product size of 10 mm to 80% minus 10 mesh. The production rate is 1,750 tpd or 79 tph. Figure 21 shows the general mill facility layout.

A conveyor located under the fine ore bin will provide a continuous feed of material to a 1,829 mm x 4,877 mm double deck sizing screen. The sizing screen will be a new modification to the plant. The undersize material will be pumped to the gravity concentrator. The mid-size material (-1 mm +0.5 mm) will be pumped to a rougher spiral to reject coarse waste. The concentrate will be pumped back to the existing mill for size reduction to –0.5 mm. The oversize material will discharge into the existing ball mill for size reduction. The ball mill will be in closed circuit with the sizing screen.

The gravity concentration circuit will use spirals, cyclones and two Falcon concentrators. The purpose of the gravity circuit is to separate the heavier lead/zinc and copper minerals from the gangue. The gangue will report to the tailings backfill plant. The lead, zinc and copper minerals will further be separated through gravity concentration to produce a lead product reporting to the lead flotation circuit. A zinc product will report to the zinc flotation circuit and the lead/zinc/copper product will report to a flotation circuit for separation of the three metals.

Lead concentrate produced in the lead flotation circuit will be thickened, dewatered and stored for shipment to Fort Nelson. The zinc concentrate produced in the zinc flotation circuit will be thickened, dewatered and stored for shipment to Fort Nelson. The copper concentrate will be dewatered and stored in bags at site for further treatment or sale to an appropriate smelter via Fort Nelson.

The tailings from the flotation circuit and the gravity circuit will report to the existing thickeners, then be pumped to belt filters, after which it will be blended in a pug mill mixer and mixed with cement or other pozilans as required and pumped underground by a positive displacement pumping system.

Figure 21 and 22 show the mill general arrangement and mill facility section view.

13.3.1 Grinding

The sizing screen oversize material (+1 mm) will discharge to a 3,048 mm diameter x 4,267 mm long overflow ball mill powered by a 522 kW (700 hp) motor. The slurry will discharge from the ball mill into a sump and then will be pumped back to the sizing screen. The screen will contain a 1 mm opening top deck and a 0.5 mm opening bottom deck. The screen undersize material will discharge into a pump box and feed the gravity circuit. The screen middlings will also drop into a pump box and be pumped to a separate gravity circuit for roughing and the concentrate returned to the mill for reduction to -0.5 mm.

Design criteria for the grinding operation is outlined in Section 10.0 of this study.

13.3.2 Gravity Concentration

The purpose of gravity concentration, prior to flotation, is to utilize the different specific gravities of the lead, zinc and copper minerals from the gangue material. Gravity separation will be performed on the middlings material and the undersize material from the sizing screen.

13.3.2.1 Sizing Screen Middlings

The middlings material from the sizing screen will be pumped to a set of spirals for gravity separation of the coarse ore from the gangue material. The tailings from the spirals will report to the final tailings thickener for subsequent dewatering and pumping to underground as backfill. The concentrate from the spirals will be pumped back to the primary ball mill for further size reduction.

13.3.2.2 Sizing Screen Fines

The fines from the sizing screen, less than 0.5 mm, will be pumped to a cluster of primary cyclones. The oversize material from the primary cyclones will discharge into a launder feeding a group of spirals. The undersize material will be pumped to Falcon concentrator.

The Falcon concentrator will concentrate any fine mineral carried over the cyclones and subsequently feed the lead flotation circuit. The middlings material from the Falcon concentrator will be pumped to the zinc flotation circuit. The tailings from the Falcon concentrator will report to the final tailings thickener.

The undersize material from the primary cyclones will feed to a group of rougher spirals. The concentrate will feed to a second set of spirals (first cleaner spirals). The tailings from the rougher spirals will be pumped to a scavenger cyclone for classification. The undersize from the scavenger cyclone will combine with the undersize from the primary cyclones and feed the Falcon concentrator. The oversize from the scavenger cyclone will be pumped to the final tailings thickener.

The first cleaner spirals will separate the higher specific gravity lead from the lower zinc/copper minerals. The concentrate will report to the lead flotation circuit while the tailings will report to the zinc flotation circuit.

The final result is that the tailings from the gravity concentrator are made up of the middlings spiral tailings, the Falcon concentrator tailings and the scavenger cyclone oversize material. The zinc flotation circuit will receive feed material from the Falcon middlings and the first cleaner spirals tailings. The lead flotation circuit will receive feed material from the Falcon concentrate, first cleaner spirals concentrate and the middlings spiral concentrate.

Figure 21 – Concentrator General Arrangement

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Figure 22 – Concentrator General Arrangement Showing Additional Two Bay

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13.3.3 Flotation Circuit

The flotation circuits will work as independent pseudo-parallel circuits compared to the traditional series philosophy. The pseudo-parallel scenario is obtainable through gravity concentration prior to flotation caused by the significant differences in specific gravities of the minerals. The solution management allows the ability to reduce the quantity of fresh water required, cross-contamination of reagents in reclaim water and tailings water storage.

13.3.3.1 Lead Flotation

The lead flotation feed material will be combined and process solution added for slurry density control. The combined material will directly feed the first cleaner flotation cells. Dowfroth 250 and Cytec 3418A will be added for lead flotation. The concentrate from the first cleaner will feed second cleaner cells. Cytec 3418A will be added and a final lead concentrate produced. The final concentrate will report to a newly installed high capacity thickener for dewatering and storage.

The tailings from the second cleaner cells will return to the first cleaner cells. The tailings from the first cleaner cells will feed the existing regrind mill 56kW (75 hp). Sodium sulphide will be added to the discharge from the mill in order to sulphidise the lead oxide material. The sulphidised material will be pumped to a cyclone. The oversize from the cyclone will discharge into the regrind mill feed chute. The undersize material will feed a scavenger cell. Cytec 3418A will be added. The concentrate from the scavenger cells will combine with the feed for the first cleaner cells. The tailings from the scavenger cells will report to the zinc flotation circuit.

13.3.3.2 Zinc Flotation

The feed to the zinc flotation circuit will comprise of gravity material and the lead scavenger cells tailings. The material will be pumped to a set of spirals for separation of the clean zinc ore and the combined lead/zinc/copper ore. The tailings from the spirals will report to the copper flotation circuit. The concentrate from the spirals will report to the zinc flotation circuit.

The flotation feed material will be conditioned with lime and copper sulphate prior to flotation. The material will feed the first zinc cleaner cells and be concentrated using Cytec 3418A and SIX for Stratabound material and Vein material respectively.

The concentrate from the first zinc cleaner cells will feed the second cleaner cells. The tailings from the first cleaner cells will report to a new zinc regrind ball mill, 75 kW (100 hp). Subsequent sulphidisation of the ball mill discharge will occur using sodium sulphide. The discharge will be pumped to a cyclone for classification of the material. The oversize material will return to the regrind mill. The undersize material will feed the zinc scavenger cells. The concentrate from the scavenger cells will combine with the feed for the first cleaner cells. The tailings will discharge to the final tailings thickener.

Figure 23 – Crushing/Grinding & Gravity Circuit Flowsheet Drawing

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Figure 24 – Pb & Zn Flotation Circuit Process Flowsheet Drawing

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Figure 25 – Cu/Pb/Zn Flotation Circuit & Backfill Process Flowsheet Diagram

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The tailings from the second zinc cleaner cells will return to the first cleaner cells. The concentrate from the second cleaner cells will produce the final zinc concentrate. The concentrate will be dewatered and stored.

13.3.3.3 Copper Flotation

Feed material for the copper flotation circuit will be provided from the zinc flotation feed spirals. The material will be ground in closed circuit with a cyclone with the oversize reporting back to a new regrind mill, 37 kW (50 hp). The undersize will discharge to the copper/lead bulk rougher. The tailings from the bulk rougher will return to the zinc flotation conditioning tank.

The concentrate from the bulk rougher will report to the copper/lead cleaner cells. The tailings from the cleaner cells will combine with the bulk rougher tailings and return to the zinc flotation conditioning tank. The concentrate from the copper/lead cleaner cells will be conditioned with lime to a pH of 10.5. The pH will allow for lead and copper separation. Separation will occur in final cleaner cells that will produce a copper concentrate and a lead tailings. The copper concentrate will be dewatered and stored in bags.

The lead tailings will be pumped to the lead concentrate thickener. This circuit will produce a copper concentrate only when Vein ore is being processed. When Stratabound ore is being processed, the final copper/lead cleaner will not be used. The first cleaner will produce a lead concentrate that will be directed to the lead concentrate thickener.

13.3.4 Thickening and Concentrate Storage

13.3.4.1 Lead Concentrate

The lead concentrate from the flotation circuit will be collected in a newly installed 4.6 meter diameter high capacity thickener. The underflow from the thickener will feed the surge tank. The surge tank will feed either of the two existing Larox filters. The Larox filters are currently PF19 style and will be upgraded to PF25 style. The increase in plates will allow 12 tonnes per hour of concentrate production from the current 9 tonnes per hour.

The filter cake will discharge onto a belt conveyor leading to the lead concentrate storage building. The concentrate storage building will allow for eight weeks of concentrate storage. The building will be covered with a “Cover-all” system and cover a concrete pad with approximate dimensions of 25 meters x 40 meters.

13.3.4.2 Zinc Concentrate

The zinc concentrate from the flotation circuit will be collected in a newly installed 4.6 meter diameter high capacity thickener. The underflow from the thickener will feed the surge tank. The surge tank will feed either of the two Larox filters.

The filter cake will discharge onto a belt conveyor leading to the zinc concentrate storage building. The concentrate storage building will allow for eight weeks of concentrate storage. The building will be covered with a “Cover-all” system and cover a concrete pad with approximate dimensions of 35 meters x 50 meters.

13.3.4.3 Copper Concentrate

Copper concentrate will be collected and dewatered in a small vacuum filter. The dried concentrate will be stored in 1,000 kg bags for either further treatment to reduce the mercury content, or sale to an appropriate smelter.

13.3.5 Tailings and Paste Backfill

Tailings material will be collected in the two existing 10 meter (35 foot) diameter conventional thickeners. One thickener will collect the gravity tailings material. The solution collected from this thickener will be re-cycled to the primary grinding and gravity circuits as it will contain no flotation reagents. The flotation tailings will be collected in the second thickener. Solution from this thickener may be re-cycled to the flotation circuit or discarded to the tailings impoundment. Storage of process solution in the tailings impoundment will allow for degradation of the flotation reagents and subsequent re-cycle of solution where required.

Underflow from the two thickeners will feed two new belt filters. The slurry will be dewatered to approximately 85% solids. The filtered solids will discharge into a pug mill mixer, 112 kW (150 hp). Cement or other pozilans (pulverized fly ash or PFA) will be added to the pug mill at a rate of 20 to 40 kg per tonne of tailings as required. The prepared tailings will be pumped to the underground mine as paste backfill, or thickened fill if no binder is required. Due to the high percent solids of the tailings, a 112 kW (150 hp) positive displacement pump will move the 1,200 tonnes per day paste.

13.3.6 Reagent Handling

This system includes the required equipment to mix the flotation reagents, Cytec 3418A, copper sulphate, Dowfroth 250, sodium sulphide and SIX.

All reagents will be mixed in mixing tanks and then pumped to a holding tank. The holding tank will feed Clarkson feeders by gravity. Reagent additions will be monitored by the operators and adjusted accordingly.

Lime will be mixed in a slaker and distributed to the appropriate addition point.

13.3.7 Water Treatment

The ability to treat process solution before discharge to the environment has been incorporated in the design of the plant. The total water to be treated and the metals to treat for will be best determined from the Pilot Plant study. Based on this type of mill layout, it is likely that only PH adjustment will be necessary to remove dissolved metals prior to discharge. Consequently, the only treatment of water will consist of lime addition for pH control. Water will be treated to adjust pH prior to discharge from the tailings dam, or the site sedimentation pond.

This water treatment proposal will be confirmed during operation of the Pilot Plant.

Figure 26 – Concentrate Storage Site Plan

Over Size photo.

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Figure 27 – Zinc Concentrate Storage Foundation Plan and Details

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Figure 28 – Lead Concentrate Storage Foundation Plan and Details

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Figure 29 – Site Plan

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14. INFRASTRUCTURE

14.1 EXISTING FACILITY

14.1.1 Mine

A number of facilities are available underground in the mine. These are listed in Section 11, Mining.

Propane for mine air heating is supplied from two large propane tanks at the 930 meter portal. These tanks are 70% full. A tool crib and shifter's office is located on the 930 meter level.

Principle items of mine equipment available on site are listed in the table below:

Mining Equipment and Supplies:	**Location**
5 Wagner ST 2B scoops, generally good appearance, need rubber & hoses – available on site	North Yard
4 Eimco 630	Bone Yard
1 Eimco 630	North Yard
4 Getman Ore Hauler chassis, two with buckets installed	North Yard
1 Getman Ore Hauler chassis	S Mill Area
3 Getman Ore Hauler buckets	North Yard
2 Atlas Copco Cavo Autoloaders	North Yard
2 Grangesberg Industries G&A DHS 40 diesel loci – 36 inch gauge	Portal
2 Grangesberg Industries G&A DHS 40 diesel loci	Shop
4 Granby 5 ton cars	Portal
2 Granby 5 ton cars	North Yard
2 Granby 5 ton boxes, chassis may have been under snow	Bone Yard
3 2 ½ton Granby boxes, chassis may have been under snow	Bone Yard
6 2 ½ton Wabi cars	Bone Yard
6 timber cars	Portal
1 gear truck (for drifting)	Bone Yard
1 man car	Bone Yard
2 car dumpers	Portal
24 joy jacklegs	930 m Level
24 joy stoppers	930 m Level
Spare legs	930 m Level
Spare drill parts	930 m Level
2 Joy electric slushers, 50hp, with 36in scrapers	North Yard
24 in scraper	Bone Yard
100 lengths of 60 # rail	Portal
Assorted 60 pound rail, splice bars, track bolts	North Yard
Stock of bull horns	Portal
6 undercut arc gate chute assemblies	Portal
1 undercut arc gate chute assembly	Airport
Air receiver on 6 in air line	Portal
1 Joy 75 hp fan	Bone Yard
1 underground sub station	Bone Yard
2 underground sub stations	Mill Area
16 in air driven fan for underground	Bone Yard
Split set rock bolts	S Mill Area

Air and water hose, assorted	South Yard
Chain various sizes	North Yard
200 + 5 ft diameter culvert manway sections for shrink stopes	North Yard
50 + 5 ft diameter culvert manway sections	Bone Yard
200 + 5 ft diameter mild steel mill hole liners for shrink stopes	North Yard
50 + 5 ft diameter mild steel mill hole liners	Bone Yard
Round and square timber, 3 in plank, barrels of wedges	Portal
Mine timber, various sizes, some possible salvage	North Yard
Round mine timber, some salvage	Bone Yard
Flat lagging some salvage	Bone Yard
Rolls and sheets of screen for underground, badly rusted	North Yard
Sheets of galvanized screen	North Yard
Vent tubing	Cold Storage
Ug tools – vic fittings, drill steel, anfo loaders, rock bolt shells, picks, shovels, jacks, track hammers, fire extinguishers, saws, acetylene	Mill Area #3 trailer
4 racks for mine lamps	S Mill Area
Control chain in barrels	S Mill Area
Bit grinder	930 m Level
Drill steel	930 m Level
6 lamp charger	930 m Level
5 semi trailer magazines rated at 15,000 pounds each	S of Site
Fuse magazines – 16x32 feet	S of Site
Misc. fans for ug, may be some salvage	Bone Yard
Misc. scoop parts, may be some salvage	Bone Yard
24 inch galvanized vent ducting, useable	Bone Yard
Reels of messenger wire	Bone Yard
Reels of used wire rope (hoist rope?)	Airport

14.1.2 Mill

The mill is approximately 90% completed. The coarse ore bin, apron feeder, jaw crusher (Kue Ken 24x36 inch), the cone crusher (5 ½ foot Nordberg shorthead), the screen (Tyrock), dust collection system and conveyors are tagged as operational. The crusher bay is equipped with a bridge crane, however the crane travel does not cover the entire area.

The main mill is equipped with a bridge crane that covers almost the entire mill grinding and flotation floor. The ball mill is in place and on timber cribbing waiting to be set on its bearings and charged. The bottom level consisted of concrete floors, collection ditches and sumps; however the concrete is cracked in a number of locations, probably due to frost action, and will require some form of grouting or epoxy coating. There is a large supply of parts awaiting installation. The main mill equipment in place including two Larox presses, two thickeners, a 10x14 Dominion ball mill and flotation cells, etc. The mill offices, mill dry and mill control room are about 80% completed.

The generator room is separated from the plant by solid wall. The generator's switch gear and controls are installed. The powerhouse is equipped with four Cooper Bessemer 1,150 kW, 1,540 hp dual fuel gen sets. It was reported that only one of the four generators was made operational. One unit did not have a muffler system or radiator installed. Each gen set is equipped with a crawl beam and chain hoist. Two

Sullair compressors are located on the mezzanine floor of the powerhouse to supply mill air.

It had been proposed that a heat recovery system be used to salvage waste heat from the generators to heat the mill and process solutions. There is sufficient space in the grinding bay to install two additional regrind mills as required.

The construction work and installations on site are of high quality for the work completed. As designed and erected this will be a first class, operator friendly plant.

The assay office building beside mill had been used as a generator and storage building. It contains a small (50 kw) generator, Herman Nelson, assorted spare parts, deep freezers and a six lamp charging rack. The building was 50% completed inside with some dry wall, radiators and services to the mill connected.

Adjacent to the mill is an ore stockpile estimated to contain up to 50,000 tons of material mined in the eighties.

Additional Mill Equipment and Supplies	**Location**
Ball charges for all ball mills	North Yard
60 pallets of soda ash	Crusher Bay
40 pallets of soda ash	North Yard
20 pallets of soda ash	Shop
100 pallets of soda ash	North Yard
Sodium Cyanide – 132 pallets	S of Site
Copper Sulphate – 196 pallets	S of Site
Sodium Isopropyl Xanthate – 84 pallets	S of Site
MIBC – 4 pallets	S of Site
Dowfroth – 4 pallets	S of Site
Methanol – 1 pallet	S of Site
20x20 ft lengths 4 in Sclair pipe with heat trace mouse hole	North Yard
Rolls of conveyer belt	North Yard
Flotation launders	North Yard
Bull gear segments	North Yard
Bearing caps for ball mill	North Yard
Pump boxes	South Yard
Catwalks, stairs, railings, grating	North Yard
Electric control panel	North Yard
Heat exchanger components ?	North Yard
Pallet of insulation jackets for Sclair pipe	South Yard
Assay lab – much weather damage – some salvage	North Yard
Various crates of equipment	North Yard
Fume hoods	North Yard
Shelving and racks	North Yard
Doors	North Yard
Cabinets	North Yard
Insulation	North Yard
Counters with black laminate tops	North Yard
Splitters and pans	North Yard
Various test equipment	North Yard

Jaw crusher	North Yard
Sieve shaker	North Yard
Electrical gear	North Yard
Racks for test equipment	North Yard
Pump assemblies and pump parts	South Yard
Pumps, gate valves, conveyor idlers, switch gear	Cold Storage
Electrical fittings, florescent lights, gate valves, trailer #4	Mill Area
Catwalks, railings	S Mill Area
Pump liners	S Mill Area
Bulk rougher conditioning tank, flotation launder, 42 inch conveyor idlers, Denver float cells stairs, pallets of 24 inch conveyor, conveyor frame, cleaner cell launder	S Mill Area
Cone and mantle for 5 ½ foot shorthead	N Mill Area
3 Sala vertical pumps	N Mill Area
24-inch conveyor frame and idlers. Plow assembly	Bone Yard
Pump box	Bone Yard
Roll of 24 inch conveyor belt	Bone Yard
Pump box fittings	Bone Yard
42 inch conveyor tail pulley	Bone Yard
Mercury vapor lamp, reflectors damaged	Bone Yard
Pallet of reflectors for mercury vapor lamps	Bone Yard
Metering pumps, electric valves	Bone Yard
Exhaust parts for Cooper Bessemer generator	Bone Yard
Mobile Equipment On Site	**Location**
American Model 599 50 ton crane/drag line, 60 ft stick with jib	North Yard
P & H 20 ton hydraulic crane, cab crushed	Portal
Cat 966C loader dismantled	Shop
Cat 950 loader	Shop Area
Cat 941 crawler loader	Portal
Cat 922B loader with quick change bucket and forks	Shop Area
Volvo BM4600 Loader	South Yard
Hitachi UH122 backhoe	Portal
Cat 815 sheepsfoot Roller	Shop Area
2 Cat 627B Scrapers	Shop Area
Cat D8 with ripper, final gear torn down	South Yard
Cat D8 frame mostly cannibalized for parts	Bone Yard
Cat D6 older model	Shop Area
Cat 14G grader	Shop Area
Cat 14 E grader	Shop Area
2 Kenworth/Challenge 7 yd cement mixers 1980 vintage	Shop Area
3 – Volvo 5350 rock wagons	Shop Area
2 Cat 16 yd rock trucks 1940's vintage	Bone Yard
Tire service truck	Shop Area
Lube truck	Shop Area
5 ton truck with fuel / water tank	Shop Area
Low boy trailer	Shop Area
3 flat deck trailers from Ritchie Brothers	S Mill Area
40 ft van trailer used for diamond drill parts and supplies	S Mill Area
Star tractor	Portal
International tractor	Portal
3 x ¾ ton crew cab	Shop Area
2 ton with welder	Shop Area

2 ton with fuel tank	Shop Area
Suburban	Shop Area
6 x 16 ft wagon trailer	S Mill Area
2 Joy 1200 cfm compressors	S Mill Area
2 Joy 600 cfm compressors	S Mill Area
Roofing tar pot	S Mill Area
1 Gardner Denver 250 cfm	S Mill Area
Chevrolet school bus 32 seats, 4 rows removed for baggage	Shop Area
Tail gates for tandems	Bone Yard
2 rock box for 16 yd tandem	Bone Yard
Ford 9000 frame	Bone Yard
Flat deck for 2 ton	Bone Yard
Tandem tailgate mount sand spreader	Bone Yard
6 ½ton truck frames	Bone Yard
Snow plow for tandem	Bone Yard
Concrete batch plant	S of Site
Cedar Rapids for crusher	Airport
Power unit for crusher	Airport
2 stacking conveyors	Airport
Sand wash plant	Airport
Construction Equipment and Supplies On Site	**Location**
1 air track	Portal
12 Herman Nelson heaters	South Yard
2 Herman Nelson heaters	Shop
1 Herman Nelson heater	Assay Office
1 Herman Nelson heater	Core Shack
Journeyman re-bar bender and cutter	Shop Area
2 concrete bucket	Bone Yard
½ yd cement mixer	Bone Yard
Pallets of roofing felt and dozens of kegs of roofing tar	North Yard
Electrical cable tray	North Yard
Electrical cable tray	South Yard
Electrical cable all sizes	North Yard
Rolls of Teck cable	Airport
100 hot water radiators, 4 ft long by 3 ft high by 6 inches thick	North Yard
FPE single phase transformers	North Yard
Several hundred sections of steel scaffolding, platforms	North Yard
50 sections of scaffolding	Airport
Plastic pipe 3,6,8 inch, assorted lengths	North Yard
4,6 inch insulated Sclair pipe	Airport
48 lengths of uninsulated Sclair pipe	Airport
Galvanized iron roofing material	North Yard
Duct work, heaters and blowers all galvanized may be salvage	North Yard
Plumbing and electrical parts, old fuel tank #1	South Yard
Electrical parts, old fuel tank #2	South Yard
Plastic pipe, Sclair pipe, iron pipe, expansion joints, gate valves	South Yard
24 2 and 3 foot culverts, 20 foot lengths c/w couplings	Airport
Several skid mounted propane tanks	Airport
Steel frame for building and rigid insulation panels	Airport
Steel concrete forms	Airport
200 2 ton bags of portland cement al hard	Bone Yard

Pipe On Site	**Location**
2,4,6 and 8 inch mild steel pipe, sched 40, some sched 80 and lightwall	North Yard
4 inch mild steel sched 80 and 2,3 and 4 inch victaulic lightwall	South Yard
Small diameter sched 40 threaded with couplings	South Yard
2,4,6,and 8 inch, sched 80, 40 ft lengths, vic and straight	S Mill Area
Vic pipe, light wall, mixed sizes	S Mill Area
2,4,6 and 8 inch mild steel elbows, sched 80, 45* and 90* weld in units	North Yard
Mixed mild steel elbows, sched 80, 45* and 90* weld in units	South Yard
Barrels of vic fittings	South Yard
Vic fittings scattered in yard	South Yard
Barrels of vic fittings	S Mill Area
Vic fittings scattered in yard	Bone Yard
Barrels of 2,4 and 6 inch flanges and vic fittings in Old Fuel tank #3	South Yard
Ductile sewer pipe	S Mill Area
Steel On Site	**Location**
Plate, assorted thickness and size	North Yard
Plate rack with quantity of assorted plate	South Yard
Steel rack with quantity of assorted steel	South Yard
I beams, assorted sizes and lengths	North Yard
Re-bar, large stock badly rusted and driven over by Cat	North Yard
Re-bar large stock assorted sizes heavily rusted	Shop Area

14.1.3 Offices, Mine Dry, First Aid Facility and Warehouse

The 22 meter x 35 meter, two-story, steel, prefabricated building includes a limited warehouse facility, a mine rescue, mine dry and first aid facility on the ground floor. The second floor contains mine offices, a recreation facility for the camp as a whole and a mine drafting area. The building is generally in good condition, however, surficial repairs to wall finishes, pipe work, etc. will be required. The offices are totally adequate for the operation of the mine as contemplated.

14.1.4 Camp

The kitchen is generally serviceable but will require substantial roof repair and leveling. The Fabco bunkhouse unit is in poor shape with many roof leaks. It could be salvaged and used for construction but is no longer suitable for a permanent camp. The Atco bunkhouse units have some damage and roof leaks, and repairs should be minimal. The Territorial bunkhouse units have survived the best and will require only modest repairs.

Because it is expected that this camp will be used for an extended period, a snow roof should be added to some units to increase their longevity. Snow roofs will protect the rather fragile trailer roofs from deterioration.

Miscellaneous Equipment On Site	Location
12 ft satellite telephone dish	Camp
5 – 10 room Territorial bunkhouse units	Camp
10 – 10 room Atco bunkhouse units	Camp
1 – 40 man Fabco bunkhouse trailer unit	Camp
Atco kitchen complex	Camp
1 entrance unit	Camp
4 dining hall units	Camp
3 kitchen units	Camp
2 walk in cooler/freezer units	Camp
40 ft pantry trailer	Camp
2 Garland ranges – with 4 burners, 2 grills and double ovens	Camp
1 Garland range – with 10 burners and double oven	Camp
1 double Moffet deep fryer	Camp
1 convection oven	Camp
Steam table, stainless steel counters	Camp
Dishwasher	Camp
Gas toaster	Camp
Potato peeler	Camp
2 baker's tables	Camp
Assorted pots, pans, bakers sheets, glassware, trays, tables	Camp
Milk cooler	Bone Yard
Commercial dishwasher	Bone Yard
Sewage treatment plant	South Yard
4 trailers from original camp used for storage	Mill Area
3 trailers from original camp used for storage	South Yard
1 trailer from original camp	Core Shack
Beds and camp repair parts stored in original camp trailers	South Yard
Potable water well	Camp
Wash car from original camp, used for paint storage	South Yard
Original camp dry, lockers, hooks, assorted spare parts	South Yard
Cases of CV100 universal catalyst, in dry building	South Yard
Shed containing several engines	South Yard
Shed containing steel shelving	South Yard
Original camp office	South Yard
Incinerator – requires spark arrester	South Yard
Propane tanks and vaporizer in shed	Camp
Tires – scoops, trucks, ½ton, tandem	South Yard
Propane tanks and vaporizers	South Yard
Cutting edges and teeth for Cat equipment	South Yard
Loader bucket	South Yard
Cat canopy	South Yard
Cat wishbone and idlers	South Yard
Mattresses and pipe insulation stored in Atco trailer	South Yard
Parts for small diameter Sclair pipe	South Yard
Cat drive sprockets and assorted other parts	Shop
Truck differentials and axles	Shop
Skidoo trailer	Shop
Camper set up as ambulance	Shop Area
Genset, compressor, Lincoln welder, plumbing parts, pumps and motors, Hobart welder, pillow blocks, Bean pump, slings and chokers, air receivers, assorted engines, drill parts, ½ton truck	S Mill Area

box trailer, 2 diamond drills	
Trailer #1 electrical parts, pipe fittings, toilets, fire extinguishers	Mill Area
Trailer #2 electrical parts, concrete test equipment	Mill Area
Core shack (carpenter shop)	North Yard
16 inch diamond saw	North Yard
2 Longyear 38 drills	Portal Area
Muskeg with full cab for use with diamond drills	Shop Area
Three floats for diamond drills	Mill Area
100,000 ft of drill core in timber racks	North Area
Cat tracks	Bone Yard
End and wall panels for Atco fold away building	Bone Yard
Diesel engine used for parts	Bone Yard
Misc. cat parts	Bone Yard
Misc. scoop parts	Bone Yard
Chevrolet c60 flat deck 2 ton service truck	Bone Yard
Misc. cable various sizes from 2x14 to 00	Bone Yard
Reels of temporary lighting for construction complete with light bulbs	Bone Yard
Steel bridging sections approximately 60 ft long by 6 ft high	Bone Yard
Assorted steel form above bridge	Bone Yard
Deutz diesel engines	Bone Yard
Many cable reels	Bone Yard
24 in I beam and columns – suitable for mill	Bone Yard
Shed of assorted electrical parts	Bone Yard
Fan forced hot water heaters 16 @ 24 in	Bone Yard
Shed of parts for ¾ ton trucks, cat parts	Bone Yard
Bobcat bucket assembly	Bone Yard
Dozen telephone poles	Airport
Pan Abode house – complete kit for 2000 square feet	Airport

14.1.5 Shop

The shops are well equipped with:

- Welding tables
- Welders
- Mechanics work benches
- Lathe – Yam 1,250 G, approximately 4 ft x 8 in
- Lathe – Smith, approximately 8 ft x 24 in
- Hydraulic press
- Radial drill
- Shaper
- Power hacksaw
- Cutoff saw
- Grinders
- Compressor
- Chargers
- Acetylene units
- Large quantity of small tools
- Tool crib

- Modest inventory of spare parts

One of the old camp trailers is set up beside machine shop as a tool crib, which contains spare parts and specialty tools.

14.1.6 Cold Storage

There is one Atco Foldaway building for cold storage. This building appeared inadequate for requirements. Cold storage capacity should be at least doubled in size.

14.1.7 Fuel Storage

The tank farm for diesel consisted of four tanks of 10,500 barrels capacity each. These tanks are surface rusted and require painting. Fuel on hand consisted of tank #1 – half, tank #2 – 1/8, #3 and #4 – 1/4. The fuel dump is equipped with a fuel load out area, pump house with two pumps for diesel and one smaller pump for gasoline. There are two smaller tanks for gasoline, a metered pump for gas and an overhead rack for filling fuel trucks.

The fuel area is well bermed, and has been recently cleaned and regravelled. Drums of lubricant and 5 gallon pails of lubricant are stored separately in the fuel area in a mobile trailer.

Fuel lines were installed to the plant with expansion joints and insulation near tanks but the installation was not completed.

14.2 CAMP COSTS AND UPGRADES

As can be seen from the above Section 14.1, a number of areas within the camp facilities require upgrade. The two main areas requiring additional work are the camp kitchen and the sewage treatment plant. These are summarized as follows:

14.2.1 Camp Overhaul

While a large proportion of the existing accommodation units are currently habitable, changes in regulations, particularly with regard to fire protection, heating methods and construction materials will require that the existing units be overhauled. In addition, the existing dining room unit requires work including a new roof. As a consequence, a budget figure of $600,000 has been applied to the overhaul of the camp facilities.

14.2.2 Sewage Treatment

A mechanical sewage treatment plant is in place at the minesite. It is not clear what condition this plant is in although it was run during construction. As a consequence, an allowance of $150,000 has been made for overhaul of the sewage treatment plant.

14.3 TAILINGS IMPOUNDMENT

The existing tailings impoundment was designed by Golder Associates and constructed in 1982 in conjunction with mine development. A total of nine alternative locations were evaluated prior to recommending the current location adjacent to Prairie Creek immediately upstream of the plantsite. The design capacity of the impoundment was set at 1.5 million tonnes, sufficient to support 7-8 years of production at the planned 1,000 tonnes per day mill throughput. The downstream embankment adjacent to Prairie Creek is heavily armoured and designed to withstand the probable maximum flood.

The tailings containment area design was approved and formed an integral part of Water Licence N3L3-0932 issued by the Northwest Territories Water Board in 1982 authorizing the use of water and disposal of waste associated with mining and milling operations at the Prairie Creek Mine.

Shortly after construction, the interior slopes of the embankments and the backslope of the impoundment area experienced minor instabilities and a degree of sloughing. The embankments and backslope have remained stable and essentially unchanged since that time.

BGC Engineering undertook a field assessment of the impoundment in 1994-95 and in 2000 was requested to provide follow-up recommendations and a cost estimate for measures required to re-commission the impoundment for use in support of future operations (BGC, December, 2000). The “most likely” case for re-commissioning is based on adopting the current slope configurations at a rehabilitation cost of $0.75 million. The “worst case” scenario, based on the application of very conservative design assumptions, involves reconfiguring upstream and downstream slope, at a rehabilitation cost of $1.23 million. Both estimates include provision for further pre-engineering geotechnical evaluation to confirm design assumptions.

Under the current mine plan, provision for maximizing tailings disposal underground through operation of a paste backfill plant will minimize use of the tailings impoundment for this purpose, relegating it to primarily a process water settling pond and greatly extending its usable life.

14.4 POWER SUPPLY

14.4.1 Power Generation Requirements

CZN’s Prairie Creek Mine has an estimated total connected horsepower of 4525 HP, or 3376 kW plus 464 kW of other loads, such as lighting and heating. The largest single connected horsepower is a 700 HP synchronous motor for the Ball Mill. The typical operating kilowatts are estimated to be 2700 kW. The typical running kilowatts are based on a yearly Operating Time Factor plus a Demand Factor, which averages to approximately 70% of the total plant anticipated load.

Table 6 – CZN Power Report

Page 1 of 5

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Table 6 – **CZN Power Report**

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The preceding CZN Power spreadsheets (Table 6) give a breakdown of the individual operating areas' power consumption and costs.

Below is a summary of the Power Consumption by Operating Area. The annual power consumption is based on operating 340 days per year.

Area	Installed Hp	Annual Power Consumption (kWh)
Underground Operation	430	1,840,818
Crushing / Grinding	1,621	5,947,419
Flotation	663	1,867,228
Concentrate Storage & Handling	345	168,011
Backfill Tailings Disposal	537	1,075,454
Reagents	76	16,071
Buildings / Air Compressors	707	2,305,891
Power House	76	98,006
Ancillary Loads	120	1,337,958
TOTAL	**4,575**	**14,660,506**

14.4.2 Annual Power Generation Costs

UEE of Penticton, a specialist mining electrical contractor, based the fuel and power generation costs on removing the four existing Model No. LS-8-GDT Cooper Bessemer 1150 kW Diesel Generators. The justification for removal is based on discussions with power system operators, parts suppliers and maintenance people.

- Cooper Bessemer Diesels are no longer manufactured.
- Parts are very difficult and expensive to source.
- Personnel with experience are difficult to hire.
- These machines are more difficult to switch on and off line.
- Newer engines give better fuel economy.
- Cooper Bessemer engines require significant ancillary driven equipment.
- Initial rebuild costs would be prohibitive, if parts are available.
- Factory engine updates may not be available.

UEE recommend they be replaced with five Caterpillar Model No. 3512 Diesel Generators operating at 1200 RPM and rated at 750kW each. Caterpillar Model No. 3512 Diesel generators typically consume 0.2800 liters of fuel per kilowatt Hour. Typically four generators would be on-line during full production hours and one generator would be stand-by. During operating hours when no underground backfill and no crushing are required, three generators would be on-line.

Caterpillar Model 3512 was recommended for the following reasons:

- They have an excellent operating record.
- Parts are readily available.
- Caterpillar Dealer has a Core Replacement Program.

- Low overhaul costs

Power Generation Cost Criteria used:

- fuel cost to Fort Nelson of $0.4837/litre
- plus, $0.0085 per kWH maintenance, including labour
- plus, $0.05 per liter transportation cost from Fort Nelson to the site
- minus $0.04 per liter Federal Excise Tax rebate

This converts to a cost of $0.1544 per kWH. Based on the total annual power consumption of 14.4 Million kWH required for the mine, mill and ancillary equipment, an energy cost in the magnitude of $2,219,000 can be expected.

14.4.3 Power Generation Capital Costs

It is possible to recover some capital cost from the sale of the existing Cooper Bessemer engines, if a user can be found that requires spare parts. The recovery amount is very difficult to pinpoint at this time.

As discussed above, this capital cost estimate is based on the recommendation to remove the existing Cooper Bessemer Diesel Generators and install Caterpillar 3512 Diesel Generators. Based on a request for a budget figure from Finning, Vancouver, a capital expenditure of $656,000 is required for each new Model 3512 Diesel generator with a complete Heat Recovery System (HRS) installed. The HRS would need to be designed in conjunction with the plant heating system. Therefore a capital expenditure of $3,280,000 would be required. This power can be leased to reduce the up front capital requirements of the operation.

Assuming it is decided to change the power supply system, Caterpillar has supplied a lease option over 60 months with a residual payment of $240,300. Equal monthly payments of $30,372.64 over the 60 month period will be paid. This option would reduce the initial capital requirements, but increase the initial working and operating costs of the Project.

However, in discussions with used diesel generator suppliers, there are often Model 3512 Caterpillar Diesel Generators available on the market. The typical price for a complete rebuilt or low hour machine is approximately $375,000. The capital investment required for this scenario is $1,875,000. This is the preferred option as the Heat Recovery System could be incorporated into a total package with the mine’s needs addressed with a possibly significant savings benefit.

14.4.4 Heat Recovery System

A capital estimate of $300,000 is expected to design, purchase and install a heat recovery system. This cost estimate is conditional on the extent to which the Heat Recovery System is applied throughout the plant. If the system is to extend throughout the total site, additional capital may be required. A true capital estimate should be prepared in conjunction with the design of the Heat Recovery System. There is an estimated kilowatt of waste heat for each kilowatt of power generated from each generator running at peak power of 750 kW. Using a conservative heat exchanger efficiency of 60%, this equates to approximately 450 kW of heat available per operating generator, which would otherwise be lost to the atmosphere.

14.4.5 Power Distribution

Power distribution will utilize the existing system and equipment to the extent possible. Changes may be required to meet the pertinent parts of CAN/CSA-M-421-93, Use of Electricity in Mines. A very important issue with any generation system is Power Factor Correction. A capital estimate of $20,000 should be included in the design and engineering phase to design the correct amount of and the location to install Power Factor Correction capacitance. The Ball Mill Synchronous Motor will have a positive effect on the power factor correction and would be an integral part of this design. The cost to supply and install a capacitive system will be dependent on the location(s) that the equipment is connected on the power grid. If it is deemed to be most beneficial on the 600 volt system as opposed to the 2300 volt system, then the costs would be lower and most of the switchgear would be available in the existing equipment.

14.4.6 Hydro Power Generation

A very brief exercise was carried out to examine the feasibility of generating supplemental power by use of low pressure turbines. Assuming a drop of 30.5 meters per kilometer can be identified on Prairie Creek or one of its tributaries near the minesite, then the installation of a small dam and 4 kilometers of pipe would allow the mine to generate up to 2 MW of hydro power.

The following table outlines the economics of such an installation and demonstrates that further work can be justified to detail the possibilities of supplemental hydro generation.

Assumptions

- 30.5 meter drop per 1 kilometer over 4 kilometers
- Installation will include 4 km of 48 inch pipe and a 2.5 MW turbine installation which would include two 1 meg units and 1 500 kw
- Power generated - 2.043MW
- Flow required - 2.56 cubic meters per second

Major cost items

Pipe 4km	$100/ft x 13,000 ft	$1,300,000
Transport of pipe to site	13,000/40 = 325 lengths	$400,000
Powerhouse		$40,000
Turbines		$1,500,000
Generators		$500,000
Switchgear		$300,000
Accessories		$150,000
construction mobile equipment		$300,000
power line		$200,000
transport of misc equipment		$250,000
Labour		$1,000,000
		$5,940,000

operating and maintenance cost estimated at $200,000.00/year		
KWH produced per year estimated at 2meg for 6 months and 1 meg for 2 months will be		10,080,000
average daily KWH produced		27,616
estimated operating cost per KWH produced by Hydro	**$**	**0.0198**
Estimated total site power consumed per year		14,371,000
Estimated Power produced by diesel component		4,291,000
Cost of Diesel Generation		**$0.1544**
Average cost of power using hydro/diesel combination	**$**	**0.0600**
Estimated Annual Saving	**$**	**1,357,000**

14.5 ACCESS ROAD AND AIRSTRIP

Historically, road access into Prairie Creek has been by winter road only. The original winter road connected the property to the Mackenzie Highway near Camsell Bend approximately 69 kilometers north of Fort Simpson. The road was approximately 288 kilometers long crossing the Ram Plateau and following the Ram and North Nahanni River valleys down to the Mackenzie.

During construction of the mine in the early 1980’s, the original winter road was abandoned in favour of a new 170 kilometer winter road alignment connecting the minesite to the recently completed Liard Highway near Blackstone about 20 kilometers east of Nahanni Butte.

A 1,000 meter gravel airstrip was constructed adjacent to Prairie Creek approximately 1 kilometer north of the minesite to provide for air access into the site in support of exploration activity and mine development.

Figure 30 – Proposed All Weather Access Road

Over Size photo.

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The current mine re-development plan proposes construction of an all weather road connecting the minesite to the Liard Highway near Nahanni Butte and a 1,500 meter gravel airstrip approximately 40 kilometers to the east of the minesite, between the Mackenzie Mountains and the Ram Plateau.

The provision for an all weather road will facilitate year round transportation of product to market, as well as backhaul of supplies to the minesite. A winter road only supported operation would necessitate shipment of the full year’s production of concentrate over an approximate 100 day winter road window between January and March of each year. This in turn would require significant provision for additional concentrate storage capacity both at the minesite and railhead.

14.5.1 All Weather Road

Ker Preistman & Associates laid out the alignment for the proposed all weather road in 1982. The route followed the same corridor as the existing winter road with minor deviations to improve grades, minimize stream crossings and minimize construction through difficult terrain, such as muskeg, permafrost and unstable slopes. Wildlife habitat assessments of the road corridor and fisheries assessments of the stream crossings were completed by Beak Consultants in 1981 and again by Rescan Environmental in 1994. Delcan provided engineering assessments of the proposed all weather road in 1983 and again in 1994.

More recently, BGC Engineering has provided an updated preliminary construction cost estimate for the road in September 2000, including an alternative route turning south from Grainger Gap, running along the Nahanni Range to cross the Liard River about 6 kilometers east of Nahanni Butte. The alternate route is slightly shorter at 159 kilometers as compared to 162.5 kilometers for the original all weather route, however the cost estimates for each of the routes are approximately the same at just over $17 million.

The advantages to the alternate route along the eastern flank of the Nahanni Range include the fact that the Nahanni Butte Dene Band would be more able to control access into their traditional territory due to the proximity to their own access road and, as well, this location would facilitate sharing of the ferry with the community of Nahanni Butte.

14.5.2 Airstrip at Sundog

The location of the current airstrip 1 kilometer north of the minesite poses limits to its usefulness due to topographic constraints. Approaches are confined to the narrow Prairie Creek valley located within the steep topography of the surrounding Mackenzie Mountains and permits landings under visual flight rules only.

CZN proposes to construct a 1,500 meter gravel airstrip on more open terrain about 45 kilometers east of the minesite between the Mackenzie Mountains and the Ram Plateau. This location will allow operation of the airstrip under instrument flight rules to support orderly movement of operating personnel into and out of the site on a shift rotation basis.

Capital cost of construction of the airstrip has been estimated at $0.75 million (Simons, 1995).

14.5.3 Liard River Crossing

The southern terminus of the proposed all-weather road requires provision for an approximate 600 meter crossing of the Liard River prior to connecting with the Liard Highway. It is proposed that the crossing be accomplished through a combination of an ice bridge over the period from December through April and ferry from May through November. Operations would be interrupted for an approximate one month period in the spring during breakup and again in the fall during freeze up.

Capital costs associated with purchase of the barge/tug combination and construction of berthing and access ramps were estimated by Simons (1995) at $1.45 million and by Procan in 1983 at $0.5 million. For the purposes of this exercise a figure of $1.0 million has been used.

14.5.4 Maintenance Costs

Annual maintenance and operating costs associated with the all weather road, airstrip, ice bridge and ferry, including manpower and materials, have been estimated at a lump sum contract of $1.0 million (Simons, 1995). This estimate contains provision for a road crew of six operators and a ferry crew of four operators, plus one supervisor.

15. CONCENTRATE QUALITY AND SALES

15.1 CONCENTRATE QUALITY

The concentrates produced from the Prairie Creek Deposit are known to be of intermediate quality, containing concentrations of penalty elements such as mercury (copper and zinc concentrates) and antimony and arsenic (copper concentrate). Both the copper and the lead concentrate have significant quantities of silver, and the lead and the zinc concentrates are of relatively high grade.

The copper concentrate in particular can be considered as a “dirty” concentrate and will be penalized appropriately at the smelter. However, production of such a concentrate is beneficial to the operation, despite the low Net Smelter Return (NSR), because it contains half of the mercury and almost all of the antimony and arsenic produced at the mine, thus depleting the other concentrates of penalty elements and improving overall marketability. However, because of the high silver levels and low tonnage, it is quite likely better than standard terms can be obtained for this concentrate, making it more worth shipping.

The lead concentrate from the operation is of high grade and highly marketable, especially with it’s high silver content. Zinc concentrate quality is variable since despite it’s high grade, concentrate from the Vein attracts a mercury penalty of around US$30 per tonne of concentrate while concentrate from the Stratabound is relatively free of mercury and will attract only minimal penalties (US$15/tonne concentrate).

Assays of concentrate produced by G&T Metallurgical of Kamloops and assayed by Chemex of Vancouver in 1994 produced the following grades and impurity levels:

Table 7 – Concentrate Specifications

	Zinc		Lead		Copper
Element	**Vein**	**Stratabound**	**Vein**	**Stratabound**	**Vein**
Copper	0.11%	0.11%	1.09%	0.03%	23%
Zinc	57%	57%	5%	4%	12%
Lead	4%	2%	75%	57%	10%
Silver (oz/ton)	2	5	20	14	250
Iron	0.7%	4%	1.2%	12%	0.7%
Arsenic	0.02%	<0.01%	0.2%	0.03%	5%
Cadmium	0.38%	0.17%	0.04%	0.02%	0.15%
Antimony	0.05%	<0.01%	0.5%	0.05%	12%
Mercury (ppm)	2,000	570	250	40	3,200
Total Sulphur	28%	32%	16%	26%	24%
Silica	2%	2%	<1%	5%	4%
Nickel	0.004%	0.002%	0.001%	0.001%	0.004%
Cobalt	0.001%	0.001%	<0.001%	<0.001%	<0.001%
Manganese	0.01%	0.01%	<0.01%	0.01%	<0.01%
Gold (oz/ton)	0.002	0.006	0.001	0.005	0.006

Magnesium oxide	0.03%	0.34%	<0.01%	0.37%	0.02%
Fluorine (ppm)	110	10	120	10	140
Chlorine	0.01%	0.01%	0.01%	0.01%	0.01%
Annual Dry Tonnes	58,000	25,700	37,500	12,500	3,000

Note: 1. In each of the three concentrates bismuth, molybdenum, uranium and thorium are all <0.01%.
2. Silver and gold are quoted as ounces per short ton.

15.2 ANNUAL CONCENTRATE TONNAGES

Approximate dry tonnages of concentrate to be produced based on a throughput of approximately 1,500 tonnes per day undiluted ore are:

Dry tonnes Zinc Concentrate: 88,000 tonnes per annum
Dry tonnes Lead Concentrate: 54,000 tonnes per annum
Dry tonnes Copper Concentrate: 4,000 tonnes per annum

This equates to approximately 158,000 tonnes of concentrate shipped per annum at 8% moisture.

15.3 METALS PRICES, SMELTER TERMS AND TREATMENT CHARGES

Neil S. Seldon of Neil S. Seldon & Associates Ltd. was retained by the Company to advise on “Metal Prices and Smelter Terms relative to Assumptions for the Scoping Study.” Mr. Seldon has extensive experience of the concentrate market and has advised CZN in the past on the marketability of Prairie Creek concentrates. Mr. Seldon provided metal price and treatment charge estimates for concentrates from Prairie Creek based on his extensive experience of the concentrate market and discussions held by him during 2000. Selected highlights of Mr. Seldon’s paper are as follows:

15.3.1 Copper Price and concentrate treatment terms

For 2001, an average price of US$0.95/lb is assumed, with the real prospect of higher prices in the following two years. At this time, in constant 1999 US Dollars the consensus 10-year outlook is in the US$0.85 to US$0.95 range. An assumption of US$0.92 as a constant dollar price is a reasonable base case. Indicative prices in dollars of the day for the period 2000 to 2008 are:

Year	2000	2001	2002	2003	2004	2005	2006	2007	2008
US$/lb	0.82	0.95	1.10	1.30	1.10	0.92	0.80	0.90	0.92

At this time, in constant 1999 US Dollars the 10-year Treatment Charge/Refining Charge ranges are in the US$80/85 and US$0.80/0.85 ranges. An assumption for TC of US$0.82 and RC of US$0.82 in constant dollars is a reasonable base case. In summary, for this period, a possible TC/RC scenario in dollars of the day for the period

2001/2008 (TC is US$/dmt and RC is US¢/lb with price participation of +/-US$0.10 at US$0.90) is:

Year	2001	2002	2003	2004	2005	2006	2007	2008
TC	$75	$80	$88	$99	$99	$85	$85	$90
RC	$0.075	$0.08	$0.088	$0.099	$0.099	$0.085	$0.085	$0.09

15.3.2 Lead Price and concentrate treatment terms

Lead for 2001 is forecast as an average of about US$500 per tonne (US$0.2268). The consensus for long term prices in constant 2000 dollars is in the range of US$500 to about US$600 (US$0.2722). For planning prices in constant dollars US$525 (US$0.238) is suggested. In summary, for the period 2001/2008, a possible price scenario in dollars of the day is:

Year	2000	2001	2002	2003	2004	2005	2006	2007	2008
US$/lb	0.207	0.227	0.261	0.295	0.283	0.272	0.253	0.261	0.268

The environmental costs for lead hygiene have added considerably to smelting costs and for this reason, long-term charges are expected to be nearer the levels of the late 1990's rather than the earlier period. A consensus view for a constant dollar treatment is in the US$175 to US$185 range. A potential annual lead treatment charge scenario is, assuming a relatively steady basis price:

Year	2000	2001	2002	2003	2004	2005	2006	2007	2008
TCUS$/dmt	185	150	165	180	190	185	175	180	185
Basis price	550	500	500	550	550	550	550	550	600

15.3.3 Silver Price

Statistically silver has shown a supply deficit for many years and higher prices are on the horizon. However, again the strength of the US dollar is restraining factor to upward movement. The major silver producers tend to use a planning price for the long term of US$5.00 to US$5.25 in constant 2000 US dollars. For 2001 the price is likely to average around US$5.00. Background to the market and price indications for the 2000 to 2008 period follow. For this period, assumed prices are:

Year	2000	2001	2002	2003	2004	2005	2006	2007	2008
US$/oz	4.90	5.00	5.15	5.35	5.55	5.65	5.75	5.85	6.00

15.3.4 Zinc Price and concentrate treatment terms

It seems probably that the price will start weak in 2001 but with some recovery as the year progresses. Zinc is likely to benefit as the copper market strengthens and given the normal cyclical behaviour of zinc, it would seem timely to expect one or two relatively good years, probably in 2003/2004. In summary, for the period 2001/2008, a possible price scenario in dollars of the day is:

Year	**2000**	**2001**	**2002**	**2003**	**2004**	**2005**	**2006**	**2007**	**2008**
US$/lb	0.515	0.52	0.54	0.62	0.57	0.52	0.50	0.52	0.56

The current upward trend in TC's is likely to continue through 2001, stabilize and then reverse. At the same time, indications are that zinc metal supplies are tightening and many forecasters are viewing the next year or two as high points in the zinc price cycle. Prices in the range of US$1,250 to US$1,500 are talked about as being possible. This is very important in relation to treatment charges.

On the basis of the assumption that new mines will lag concentrate demand, it is likely the TC will move back to more historical levels relative to payable and the following scenario reflects this:

Year	**2000**	**2001**	**2002**	**2003**	**2004**	**2005**	**2006**	**2007**	**2008**
TC-US$/dmt	190	190	200	205	200	170	170	172	199
Basis price	1,000	1,000	1,100	1,200	1,200	1,000	1,000	1,000	1,100
TC-US$/dmt	190	190	186	177	172	170	170	172	185
US$1,000 basis	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000

The above charges at a basis of US$1,000 represent about 39.9% of the payable at 52.5% zinc. For purposes of background, the focus has been on the changes and trend in zinc treatment charges. In reality, there are a number of contract terms, which vary somewhat with the market as with copper and lead concentrates**.**

15.4 SUMMARY OF SCOPING STUDY PRICES AND TREATMENT CHARGES

Scoping Study Assumptions used in the model:

15.4.1 Prices

The model used fixed long-term prices and exchange rates to calculate project revenues. These were

Copper - US$0.90 per Pound
Zinc - US$0.50 per Pound
Lead - US$0.25 per Pound
Silver – US$5.50 per Ounce
CAN$ = US$0.66

15.4.2 Long-term Smelting, Refining and Other Charges

15.4.2.1 Zinc Treatment Charges (constant dollars)

TC-US$/dmt	175.00
Basis price	US$1,000 basis

Note: Escalators of plus 16% and minus 14% to be applied. Using constant dollars for the zinc treatment charge is complicated by the price escalation, but about 39% of the payable or about US$175 CIFFO main Japanese port per dmt at basis price of US$1,000 is reasonable.

Payable Metals

Zinc pay for 85% of content (minimum deduction 8%)

Silver deduct 3.5 ozs per dmt from content and pay 70% of balance

Cadmium deduct 0.2% and pay 60% of balance less a refining charge of US$1.50/%1.25 per lb, with negative value possible applicable – effectively a penalty. For planning for cadmium use a flat penalty of US$1.00 to US$3.00. As the Basel convention signatories move to place Cadmium on the Red list this penalty may increase

Zinc Price Participation

Plus 16% and minus 14% < / >US$1,000 (Note: scale may often be varied with price breaks, but for purposes of the Scoping Study, simple scale assumed)

Penalty ranges in Zinc Concentrate Terms

The penalties included in the generic terms reflect the average ranges below, but are subject to significant changes (both increases and decreases) as smelters needs and market availability shift.

Iron US$1.50 to US$2.50 per 1% over 7.5-8.5%

Mercury Typically US$0.30 to US$0.50 per 10 ppm > 30 ppm < 100 ppm and then US$0.50 to US$1.50 per 10 ppm over 100 ppm are typically applied. However, with very high mercury concentrates, such penalties are likely to be in the range of US$20 to US$30 per dmt max. In the case of Prairie Creek Vein, it is assumed that the penalty will be US$30/dmt. For Stratabound it is assumed that the maximum penalty will be US$15/dmt.

Arsenic US$2.00 to US$2.50 per 0.1% over 0.1%.

Magnesia US$1.00 to US$2.00 per 0.1% over 0.3% (threshold may be raised at times)

Payment

Provisional 80% on arrival

Final balance when all facts known

15.4.2.2 Lead Treatment Charges (Constant dollars)

TC-US$/dmt	175
Basis price	500

Realised charges would be higher as the effect of escalation is felt.

Treatment - Charge in constant dollars

175 (DES world port) per dmt basis price of US$500 per tonne with price participation of plus 15% over US$500

Payable Metals

Lead pay 95% of content (minimum deduction 3%)

Silver pay 95% of content (minimum deduction 50 grams per dmt) with refining charge of US$0.35 per ounce

Gold deduct 1 gram per dmt from content and pay 95% of balance with refining charge of US$6 per ounce

Penalty ranges in Lead Concentrate Terms

The penalties below reflect average ranges, but are subject to significant changes (both increases and decreases) as smelters needs and market availability shift.

Mercury US$0.50 to US$1.00 per 10 ppm > 30 ppm < 100 ppm and then US$1.00 to US$1.50 per 10 ppm over 100 ppm.

Cadmium Flat penalty of US$1.00 to US$3.00 per dmt. As the Basel convention signatories move to place Cadmium on the Red list this penalty may apply.

Arsenic US$2.00 to US$2.50 per 0.1% over 0.2%.

Antimony US$1.50 to US$2.00 per 0.1% over 0.2 to 0.5% (Antimony and Arsenic may be a combined penalty in which case the threshold may be increased to the higher level of 0.5%).

Bismuth US$1.50 to US$2.50 per 0.01% over 0.01 to 0.1% (threshold may be varied depending on the type of lead refinery. Pyro or Electrolytic).

Payment

Provisional 85% on arrival

Final balance when all facts known

15.4.2.3 Copper Treatment Charges (constant dollars)

TC	$82
RC	$0.082

Treatment Charge - Constant dollars

Range US$75/ US$85 dry tonne CIFFO main Japanese port
Base case US$82

Payable Metals

Copper unit deduction 1.0 for grades less than 30%

Gold pay 90% to 98% in Japan according to grade

Silver pay 90% to 95% in Japan according to grade

Refining Charges - Constant 1999 dollars

Range - Copper US$0.075 to US$0.85 per pound
Base case US$0.82

Range - Gold US$4 .00 to US$6.00 per payable ounce
Base case US$5.00

Range - Silver US$0.35 to US$0.45 per payable ounce
Base case US$0.40

Copper Price Participation

Plus/minus 10% of the copper price, above / below US$0.90 to US$1.00 per pound

Penalties

Arsenic	US$2.50/3.00 per 0.1% over 0.1%/ 0.2%
Antimony	US$2.50/3.00 per 0.1% over 0.1%/0.2%
or	Arsenic plus Antimony US$2.50/3.00 % over 0.1%/ 03%
Lead	US$2.50/3.00 per 1% over 1%/2%
Zinc	US$2.50/3.00 per 1% over 2%/4%
or	Lead plus Zinc US$2.50/3.00 % over 3%/ 6%
Mercury	US$0.10/0.20 per ppm over 10/20 ppm
Bismuth	US$3.00 per 0.1% over 0.05%

Payment

Provisional	90% from CAD up to agreed number of days after arrival, subject to negotiation
Final	balance when all facts known

15.5 POSSIBLE IMPROVEMENT IN NET SMELTER RETURN (NSR)

A number of possible areas exist for improvement to overall Project NSR. These are reduction in penalty payments and improvement in revenue from contained metals.

The two principal areas that will be addressed on receipt of concentrates from the Pilot Plant are the mixing of Vein and Stratabound zinc concentrates to reduce the mercury penalty and the possible leaching of the copper concentrate to remove the contained silver and copper on site. While it is not prudent to plan a mine on these possible and currently hypothetical assumptions, they must be considered possible upsides and dealt with as soon as concentrate is available for testing.

16. LOGISTICS AND CONCENTRATE SHIPPING

A significant aspect of operations on the mine at Prairie Creek will be the transportation of approximately 205,000 tonnes of concentrate per annum from the minesite to the smelter via road and rail. Operating supplies and the use of paste backfill requiring cement, would result in a backhaul to the minesite in the region of 75,000 tonnes per annum. This would include diesel fuel, explosives, cement, mill supplies, food and perishables, etc. Because of the significant volume of material to be transported and the high cost of working capital and storage facilities for the concentrate, an all-weather road was proposed for the site and designed in 1982. Revisions to this design in 2000 by BGC Engineering Inc. indicate that a road can be constructed for approximately $17 million.

16.1 ROUTES

Three possible routes for getting concentrate to market exist (see Figures 31 and 32). These would be:

- by road to Stewart in British Columbia and then by ship to smelters in Japan, Korea, China and Europe;
- by road to Fort Nelson and then by B.C. Rail to Vancouver or Prince Rupert, then by ship; or
- by road to Hay River and Enterprise and then by C.N. Rail to smelters at Flin Flon, Manitoba or Trail, British Columbia.

Arrow Transportation Systems were contacted to provide budget quotations for the road haulage portion of these contracts. Trucking rates were quoted as follows:

- Mine to Fort Nelson – approximately $40/tonne during unrestricted periods. During road restrictions the rate would be $58.50/tonne.
- Mine to Hay River – approximately $58.29/tonne during unrestricted periods. During road restrictions the rate would be $93.08/tonne.
- Rates for general freight as backhaul – approximately $1,600 per trip from Fort Nelson to the minesite and $2,450 from Hay River to the minesite.

For the purposes of this Scoping Study, it has been assumed that a concentrate shipping facility would be established at Fort Nelson. Preliminary drawings for this facility have been prepared and cost estimates for its construction have been completed.

In accordance with the Prairie Creek Development Cooperation Agreement, the trucker involved in the concentrate haulage will be encouraged to enter into a partnership with the relevant First Nation communities in the region. There may be some benefit to carrying out trucking activities completely within the Northwest Territories and in that case operation of the concentrate shipping facility may be moved to Enterprise (Hay River), Northwest Territories.

Figure 31 – Concentrate Transportation Routes

Over Size photo.

If you would like a copy, please contact the Company at
1-866-688-2001 toll free or 604-688-2001.

Figure 32 – Possible Transportation Routes

Over Size photo.

If you would like a copy, please contact the Company at
1-866-688-2001 toll free or 604-688-2001.

Figure 33 – Concentrate Storage and Load Out Facilities

Over Size photo.

If you would like a copy, please contact the Company at
1-866-688-2001 toll free or 604-688-2001.

16.2 RAIL SHIPMENT

Preliminary indications from B.C. Rail in 1997 for the cost of shipping up to 200,000 tonnes of concentrate by gondola from Fort Nelson to Vancouver were $36.24 per tonne of concentrate. The figure of $37.00 per tonne has been used for the purpose of the Scoping Study. B.C. Rail are experienced in shipping concentrate, having handled significant volumes of concentrate from Gibraltar Mine and from other mines in central British Columbia.

For the Scoping Study, it is presumed that the lead concentrates and the Stratabound zinc concentrates produced by the operation will be sold in Canada, either to Cominco at Trail or to Hudson Bay at Flin Flon. Rail costs to these two points have been increased by $14/tonne of wet concentrate to $51/tonne to include loading and unloading, backhaul of freight, etc.

It may be economically better to ship from Enterprise to Flin Flon or to Trail direct, in which case rail transport would be by the CN system.

16.3 PORT TRANS-SHIPMENT

Three possible ports exist for trans-shipment of Prairie Creek concentrates. These are: Stewart, which is non-union but road access only; Prince Rupert, which is rail access and unionized; and Vancouver Wharves, which is also rail access and unionized. Typical concentrate trans-shipment terms for Stewart are $7/tonne of concentrate. Costs at Prince Rupert and Vancouver are likely to be very similar; currently costs including amortization of storage facilities at Vancouver, are running in the $13-15/tonne of concentrate handled. For the purposes of the Scoping Study the Company has used a figure of $14/tonne of concentrate. These figures and final routes would be confirmed as part of the Feasibility Study.

16.4 OVERSEAS SHIPPING

Shipping rates vary, dependent entirely on the availability of ships and economic pressures on ship owners. Current quotations from Vancouver to Japan are in the region of US$21 to US$25. For the purpose of this study the Company have used a figure of US$23 per tonne of concentrate.

16.5 OPERATION OF THE LOAD-OUT FACILITY

Loading and unloading of concentrate onto trucks and into rail cars would be handled by the haulage contractor. Total cost for handling concentrate in the load-out facility would be between $3-4/tonne. As part of the operation of the load-out, freight would be forwarded and handled by the haulage contractor, approximately $14/tonne of concentrate sold in Canada has been allowed to cover backhaul of supplies (approximately $1 million/annum). Back haulage of cement using an on-site silo and big bags is estimated at approximately $15/tonne to the mine.

17. ENVIRONMENTAL AND PERMITTING

17.1 INTRODUCTION

At the time of construction in 1982, the Prairie Creek Mine had been fully permitted for full scale mining and milling operations. Permitting had been undertaken under the regulatory regime of the day, which involved a comprehensive environmental assessment and public review before the Northwest Territories Water Board. A considerable number of technical and baseline studies describing the proposed development and the physical and biological environment were undertaken at that time.

Water Licence N3L3-0932 was issued by the Department of Indian Affairs and Northern Development on July 1, 1982 pursuant to the Northern Inland Waters Act and Regulations authorizing use of up to 1,150 m^3/day and 420,000 m^3/year of water from the Prairie Creek Valley Aquifer and setting standards for discharge of process effluent to Prairie Creek. Land Use Permit N80F249 was issued July 2, 1980 for the winter road connecting Prairie Creek to the Liard Highway, and Surface Leases were issued for the minesite area and airstrip.

The Water Licence and Land Use Permit have since expired, necessitating the requirement for new ones to be issued in support of future operations.

Further baseline studies were undertaken in 1994 in support of planned re-development and permitting activity at that time. While the re-permitting process was subsequently discontinued in 1995, these studies represent a significant addition to the information database in support the Project.

17.2 ACID ROCK DRAINAGE

The Prairie Creek Mine benefits greatly from the fact that the mineral resources are hosted in carbonate rocks. Rescan Environmental of Vancouver, B.C. undertook a detailed analysis of the acid generating characteristics of all dominant rock types at the Prairie Creek Mine in 1994. The results indicated an overwhelming dominance of acid neutralizing minerals with acid neutralizing carbonate minerals exceeding the total capacity to generate acidity by an average factor of almost 200. Initial analysis of flotation tailings generated from metallurgical testwork has indicated a similar excess of neutralization potential.

The low sulphide values and high excess neutralization potential of the host rocks and tailings products indicate that these materials will pose no long term hazard to the environment through sulphide oxidation processes.

17.3 ENDANGERED SPECIES

The Committee on the Status of Endangered Wildlife in Canada (COSEWIC) lists only two species in the area of the Prairie Creek Mine. These are the Grizzly Bear (Ursus arctos) and the Wolverine (Gulo gulo), both of which are listed in the Special Concern category.

In areas removed from the minesite, COSEWIC lists the Anatum Peregrin Falcon (Falco peregrinus anatum), the Woodland Caribou Boreal population (Rangifer tarandus caribou) and the Wood Bison (Bison bison athabascae), each of which are considered Threatened.

Detailed field studies of wildlife populations and wildlife habitat in the area of the Prairie Creek Mine and the access road were conducted by Beak Consultants in 1980-81 and again by Rescan in 1994. None of the listed species and no critical habitat, such as denning or nesting areas, were identified in the area of the mine.

Grizzly bears and wolverine have been observed or encountered only very infrequently in the area surrounding the mine over the past 20 years.

Caribou populations and potential caribou habitat have been identified in areas removed from the minesite to the north and east in the Mackenzie mountains. Potential impacts to these populations are primarily transportation related and can be mitigated through standard road safety practices.

Specific surveys of potential Peregrine falcon nesting habitat have identified no nesting sites in the area of the minesite.

Wood bison were re-introduced into the Nahanni Butte area, 90 kilometers to the southeast of the Prairie Creek Mine, in 1980 with additions to the herd made in 1989 and again in 1998. As with caribou, potential impacts to these populations are primarily transportation related, in this case primarily in the area of the Liard Highway, and can be mitigated through standard road safety practices.

No rare or highly valued species of vegetation or plant communities have been identified in the area. COSEWIC does not list any plant species as endangered, threatened or of special concern in the area of the Prairie Creek Mine.

17.4 PROTECTED AREAS

The Prairie Creek Mine, located adjacent to Prairie Creek, is within the watershed of the South Nahanni River, 32 kilometers upstream of the point where Prairie Creek crosses the boundary of the Nahanni National Park Reserve and 48 kilometers upstream of the point where Prairie Creek joins the South Nahanni River.

The Nahanni National Park Reserve was created in 1972, following a canoe trip down the river by Pierre Elliot Trudeau, specifically for the purpose of setting aside the South Nahanni River for wilderness recreational purposes. The South Nahanni River is 500 kilometers in length of which 300 kilometers are contained within the Nahanni National Park Reserve. The confluence of Prairie Creek and the South Nahanni River is 65 kilometers upstream of the point where the South Nahanni River leaves the Nahanni National Park Reserve crossing its downstream boundary.

The South Nahanni is highly valued as a wilderness recreation river and is regularly used for canoeing trips during the summer months. These wilderness adventure tours, of which there were 58 such private trips in 1999, are supported by a number of outfitting companies from as far away as Ontario.

Through the efforts of Parks Canada the South Nahanni River and the Nahanni National Park Reserve have been designated as a Canadian Heritage River and a World Heritage site, respectively.

The Company recognizes the highly valued wilderness attributes of the South Nahanni River which have led to these designations, and is committed to working closely with the Deh Cho First Nations, in whose traditional territory the Nahanni National Park Reserve lies, Parks Canada and the Nahanni River Outfitters Association to ensure that operations at the Prairie Creek Mine in no way impact upon this special place put there by the Great Creator.

17.5 PERMITTING

Being located within the Mackenzie Valley in the southwest corner of the Northwest Territories, all present day permitting activity relating to land and water uses at Prairie Creek will fall under the recently proclaimed Mackenzie Valley Resource Management Act (MVRMA).

The MVRMA evolved out of the Gwich'in and Sahtu Comprehensive Land Claim Agreements, which required the establishment of land and water boards as institutions of public government within an integrated and coordinated system of land and water management in the Mackenzie Valley, and received royal assent on June 18, 1998.

The Mackenzie Valley Land and Water Board (MVLWB) was created on March 31, 2000. The MVLWB has jurisdiction in all uses of land or water or deposits of waste in the Mackenzie Valley for which a permit is required.

The MVLWB and its associated regional boards take over regulatory functions previously performed by the Department of Indian Affairs and Northern Development (DIAND), the Northwest Territories Water Board, and the Government of the Northwest Territory's Department of Municipal and Community Affairs on Commissioner's Lands. As well, the Canadian Environmental Assessment Act no longer applies in the Mackenzie Valley except under very specific situations.

Under the MVRMA, the public boards are responsible for:

- preparing regional land use plans to guide the development and use of land, waters and other resources;
- regulating all uses of land and water; and
- carrying out the environmental assessment and review process.

As stated in the MVRMA, "the purpose of the establishment of boards by this Act is to enable residents of the Mackenzie Valley to participate in the management of its resources for the benefit of the residents and of other Canadians."

The stated objective of the MVLWB is to "regulate the use of land and waters and the deposit of waste so as to provide for the conservation, development and utilization of land and water resources in a manner that will provide optimum benefit to the residents of the settlement areas and of the Mackenzie Valley and to all Canadians."

The MVLWB has three main functions:

1. Processing trans-boundary land use and water use applications in the Mackenzie Valley;
2. Ensuring consistency in the application of the legislation throughout the Mackenzie Valley; and
3. Issuing land use permits and water licences outside settled land claim areas in the Mackenzie Valley.

All Applications for a land use permit or a water licence in relation to a development in the Mackenzie Valley are made to the Board or one of its regional boards, as determined by the location of the development. In the case of Prairie Creek, being located within the Deh Cho First Nations region, for which a settlement agreement has not as yet been reached, applications will be processed by the MVLWB with representation by the Deh Cho.

These Applications will require the inclusion of certain baseline and other technical information to allow them to be appropriately assessed and processed. Information provided with an Application will be used for undertaking a preliminary screening and for regulatory review of the Application. Much of this work has previously been completed in support of permitting initiatives in 1982 and again in 1994, however additional work to supplement the existing database will likely be required to support current development plans and a new application.

The preliminary screening will determine if the Project will be referred to the Mackenzie Valley Environmental Impact Review Board (MVEIRB) for an environmental assessment (EA) or whether it will proceed directly to a regulatory review for the issuance of licences and permits.

If referred, which is most likely for a mining project such as Prairie Creek, the MVEIRB will develop a work plan and terms of reference in order to conduct the EA. Upon completion of the EA the MVEIRB will recommend terms and conditions to be attached to the permit or licence. The MVEIRB's recommendations will then be forwarded to the Minister of Indian Affairs and Northern Development prior to issuance of permits and licences by the MVLWB. The MVEIRB may also recommend the proposal undergo an environmental impact public review before a panel.

The Company initiated preliminary discussions with the regulatory authorities in Yellowknife in August, 2000 with respect to re-development and re-permitting of the Prairie Creek Mine. A follow up presentation was made to the Mineral Development Advisory Group in November to elicit specific feedback from each of the regulatory agencies on the information requirements necessary for them to fulfill their roles in review of an Environmental Assessment Report to be submitted in support of applications for permits and licences authorizing full production of the mine.

The EA process for re-permitting of the Prairie Creek Mine, including the collection of additional baseline data to support preparation and submission of the EA report, is expected to take up to two years to complete.

18. RECLAMATION

A conceptual reclamation plan for the Prairie Creek Mine and associated facilities will be developed in conjunction with the Environmental Assessment Report to be prepared in support of permit and licence applications. Preliminary cost estimates for reclamation will be determined as a component of the plan and will form the basis for discussion with regulatory authorities as to the level and timing for deposit of reclamation security.

The key components of the development for consideration in the reclamation plan will be:

- camp and plantsite facilities;
- tailings impoundment;
- waste rock disposal areas;
- underground workings;
- exploration roads and drill sites;
- airstrip; and
- access road

As an underground mine with surface facilities already in place, including the mill, camp, engineering and service buildings and tailings impoundment, little further disturbance in the area of the minesite is anticipated in conjunction with the proposal for re-development. As a result, the present minesite disturbed area of about 20 hectares will change very little as a result of the planned re-development.

Additional disturbance will relate primarily to development of the all weather access road and airstrip totaling approximately 135 hectares. The proposals put forward for development of these components as a Deh Cho Initiative and the potential for long term use of these components by the Deh Cho First Nations beyond the mine life may preclude the need for inclusion of them in the overall reclamation plan.

The objective of the Reclamation Plan will be to provide a systematic approach to decommissioning the mine and returning all disturbed areas to an acceptable condition. The primary considerations in achieving this objective will be as follows:

- the long-term preservation of water quality within and downstream of decommissioned operations;
- the long-term stability of erodable material including particularly waste rock disposal areas and the tailings impoundment;
- the removal and proper disposal of all structures and equipment not required beyond the end-of-mine-life;
- the natural integration of disturbed lands into the surrounding landscape, to the greatest possible extent, after mining ceases;
- the establishment of a self-sustaining vegetative cover consistent with existing land uses;
- the promotion of natural regeneration of vegetation; and
- the re-establishment of land use consistent with pre-mining conditions.

A significant consideration from a reclamation perspective is the fact that the mineralisation at Prairie Creek is hosted within carbonate dolostones and shales. Testwork conducted to date has demonstrated a substantial excess of neutralizing capacity as compared to acid generation capacity indicating that acidic drainage will not be a concern at Prairie Creek either in waste rock or tailings. This fact will significantly limit exposure to reclamation liability.

Also significant is the plan for disposal of the bulk of the tailings underground as a paste backfill. This will limit utilization of the tailings impoundment primarily to that of a process water settling pond. While chemical and geotechnical stability issues relating to the tailings impoundment will remain paramount from a planning perspective, the consequent risks associated with the facility will be much lower.

19. LABOUR AND ADMINISTRATION

19.1 LABOUR

Labour costs have been separated out from the other operating costs and broken down by area and function. The table below presents a wage listing for process and associated general and administrative personnel. Salaries and hourly rates are based on information supplied from similar operations. The annual salary rates and the hourly rates include a 40% burden for overtime, medical, unemployment and contributions to the Canada Pension Plan. The wages do not include catering costs or transportation costs to and from the job site.

Table 8 – CZN Manpower

Position	On-site Number	Total Number	Salary Rate, $/year	Salary Rate, $/hour	Total Cost Before burden ($)	Days Work
MANAGEMENT						
Site Manager	0.5	1	130,000		130,000	365
Mill Manager	0.5	1	95,000		95,000	365
Mine Manager/Chief Engineer	0.5	1	95,000		95,000	365
Bookkeepers	1	2	55,000		110,000	365
SUBTOTAL MANAGEMENT	**2.5**	**5**			**430,000**	
PROCESS						
Mill Foreman	1	2	85,000		170,000	365
Metallurgist	1	2	80,000		160,000	365
Shift Boss	2	4	75,000		300,000	365
Crushing Operator	1	2		29.39	119,911	340
Crushing Helper	1	2		21.66	88,373	340
Grind Operator	2	4		29.71	242,434	340
Flot. Operator	2	4		29.71	242,434	340
Labourers	2	4		19.21	156,754	340
SUBTOTAL PROCESS	**12**	**24**			**1,479,905**	
LABORATORY						
Chief Assayer	1	2		29.71	121,217	340
Assayer	1	2		21.66	88,373	340
Sample Buckers	2	4		19.21	156,754	340
SUBTOTAL LABORATORY	**4**	**8**			**366,343**	
ENGINEERING						
Geologists	1	2	75,000		150,000	365
Surveyors	2	4		26.75	218,280	340
Engineers	1	2	75,000		150,000	365
SUBTOTAL ENGINEERING	**4**	**8**			**518,280**	
SITE OFFICE						
Environmental Coordinator	0.5	1	60,000		60,000	365
Warehouseman/First Aid Attendant	1	2		29.71	121,217	340
Clerical	2	4		19.21	156,754	340
Safety Coordinator	0.5	1	60,000		60,000	365

SUBTOTAL SITE OFFICE	**4**	**8**			**397,970**	
MAINTENANCE						
Maintenance Foreman	1	2	80,000		160,000	365
Millwright/Welder	2	4		30.75	250,920	340
Pipefitters	2	4		29.21	238,354	340
Electrician	2	4		30.75	250,920	340
Labourers	4	8		21.66	353,491	340
SUBTOTAL MAINTENANCE	**11**	**22**			**1,253,685**	
GRAND TOTAL Salary before burden, camp etc	**37.5**	**75**			**4,446,183**	

NB. Mining manpower separate and within mining contract price excluding camp costs.

All on-site personnel will work a four week period at the site and then have four weeks of time off. While on-site the processing and mining personnel will work a rotating 10-12 hour shift on a 24 hour per day basis as allowed by the local regulations. The crushing personnel will work a 12 hour shift, seven days a week for the four week period.

19.2 EMPLOYEE TRANSPORTATION

It is anticipated that a chartered aircraft from a major center, to be determined, will be used to transport employees to the minesite for a cost of approximately $40 per man-day. The total number of employees for all areas except for mining is 75. Only half of the 75 will be traveling once every four weeks. . Mining contractor personnel will total 80, with 40 traveling once every 4 weeks. The estimated travel cost is $496,000 per annum, or $18 per day on site.

19.3 CAMP CATERING

Camp catering costs were supplied from similar operations at $40 per man-day. For the 37.5 CZN employees, plus 40 mining contractors on site at one time, the catering cost is $1,100,000 per annum.

19.4 GENERAL AND ADMINISTRATIVE

General and administrative (G&A) costs have been estimated based on information supplied by Canadian Zinc Corporation. The G&A costs include the following:

- Road and ferry maintenance
- Corporate Office Costs
- Property Insurance
- Property Tax
- First nations training and benefits initiatives

Based on the above items and supplied costs, an annual G&A cost is estimated to be in the range of CAN$ 1.8 million.

20. CAPITAL COSTS

20.1 INTRODUCTION

CZN has prepared the capital cost estimate presented in this section based on preliminary process design modifications. Included in these modifications is the addition of a gravity section and a paste backfill section within a two bay extension to the mill on the east side.

Mining capital development costs were determined from contractor quotes to perform the work.

All capital cost estimates are given in Canadian currency. Quotations received in other currency were converted to Canadian currency at an exchange rate of US$0.66 equal to CDN$1.00.

20.2 COST BASIS

20.2.1 Mining

CZN, in co-operation with Procon Mining and Tunnelling Ltd. of Burnaby, determined the mine development requirements for the underground mine operation. No capital investment is required by CZN for the mining fleet since this will be supplied by a contractor. The contractor will be responsible for providing all necessary equipment and maintaining the equipment. CZN will supply fuel and power to the contractor and the cost is excluded in the contract bid price. Catering costs and transportation costs are included in the CZN labour cost for contractors personnel.

An allowance of $1.7 million has been allowed for mining capital expenditure prior to stoping from existing workings commences.

20.2.2 Site Access

The cost estimate for the all weather access road was provided by BGC Engineering Inc. as a budget estimate based on the B.C. Forest Practices road construction database. Estimates for the cost of the ferry and ferry landings were based on estimates by Simons Engineering in 1995. The cost of the new strip at Sundog was based on an earlier estimate by Simons in 1995.

20.2.3 Processing

Processing capital costs are based on quotations provided by equipment suppliers. Concrete, structural steel and infrastructure, and support costs were determined from supplier quotes and design criteria. Power generation, electrical distribution and heat recovery system quotes were provided by Unit Electrical Engineering Ltd. (UEE).

20.2.4 Spare Parts

A spare parts allowance has been included and is based on 10% of the total material costs.

20.2.5 Freight

Port of entry for imported goods is Vancouver, British Columbia, Canada. Shipping costs were based on 7% of the equipment and material price for the majority of equipment and materials.

20.2.6 Installation

Process equipment installation costs were estimated based on information supplied by contractor quotes and estimates from projects located in the same general area.

20.2.7 Taxes

A Goods and Services Tax (GST) of 7.0% is collected by the federal government for purchase of equipment and materials and services provided. The GST is recoverable by CZN and, as a result, has not been included in the capital cost estimate.

There is no territorial sales tax.

20.3 ENGINEERING, PROCUREMENT AND CONSTRUCTION MANAGEMENT

The engineering, procurement and construction management (EPCM) required for the construction of the Prairie Creek Project is estimated at 10% of the total installed cost, not including the contingency, based on typical EPCM costs for other similar projects.

20.4 CONTINGENCY

Considering the accuracy of the overall estimate, a contingency of 15% of the total installed costs, not including EPCM costs.

20.5 WORKING CAPITAL ALLOWANCE

A working capital allowance has been made in the model by moving cash flow from year 1 to year 10 of the model. This is the equivalent of a working capital allowance of $5 million dollars.

21. OPERATING COSTS

21.1 INTRODUCTION

The operating costs developed in this section were based on processing approximately 540,000 total tonnes of ore per annum with a split of approximately 1/3 Stratabound ore and 2/3 Vein ore. As a result, the power, labour and supply costs are calculated separately for Vein and Stratabound ore and reflect the aforementioned split.

Unit costs for reagents and other miscellaneous supplies were based on current information supplied by CZN or obtained from suppliers' quotations. Unit power costs were calculated to be $0.1544/kWh based on a delivered fuel cost and consumption of diesel supplied by Unit Electrical Engineering Ltd. (UEE).

It is assumed that all Goods and Services Tax (GST) taxes paid for the purchase of materials, equipment and services will be recaptured by CZN, consequently GST taxes are not included in the operating costs. It is assumed that no Provincial Sales Tax (PST) is paid in the Northwest Territories or is included in the delivered price of materials supplied.

Operating and maintenance supplies for the processing facility are assumed to be minimal and an annual cost is estimated for repair of minor piping, equipment, etc.

The tables on the next three pages summarise the operating costs for the operation split by one type and overall.

21.2 LABOUR

Labour costs have been separated out from the other operating costs and broken down by area and function. The breakdown is located in Section 19 of this study and is included in the operating costs.

21.3 MINING

Mining will be performed by a contractor. Development costs have been estimated by Procon Mining and Tunnelling Ltd. of Burnaby, British Columbia. Ore mining costs are estimated by ore type, as advised by Procon, based on typical costs of similar methods elsewhere. Costs for exploration, backfill, services, etc. are supplied by CZN on a manpower or a per tonne basis. The breakdown is located in Section 11.

21.4 CRUSHING AND GRINDING

The annual power consumption and cost per kWh was estimated by UEE. Consumption of grinding media was estimated from literature on similar operations. Supplies were estimated using 5% of the material annual operating costs minus power and labour.

Table 9 – Operating Costs – Overall Cost per tonne

Over Size photo.

If you would like a copy, please contact the Company at
1-866-688-2001 toll free or 604-688-2001.

Table 10 – Operating Costs – Vein Mineralisation

Over Size photo.

If you would like a copy, please contact the Company at
1-866-688-2001 toll free or 604-688-2001.

Table 11 – Operating Costs – Stratabound Mineralisation

Over Size photo.

If you would like a copy, please contact the Company at
1-866-688-2001 toll free or 604-688-2001.

21.5 FLOTATION

Flotation power costs were derived on the basis that all equipment was to be used at an associated operating load averaged to 70%. Reagent consumptions were based on information supplied by Westcoast Mineral Testing Inc., based on testwork results to date. Reagent costs were supplied from suppliers. Supplies were estimated at 5% of the material annual operating costs minus power and labour.

21.6 CONCENTRATE STORAGE AND HANDLING

The Larox filter cloth will be changed once per month. Limited flocculent will be required for the concentrate high capacity thickeners and most will be used in the tailings backfill thickeners. A loader will be required for loading the concentrate trucks. The truck operators will load their own 40 tonne concentrate truck in 30 minutes. Using the previous numbers, a total of 1,500 hours of loader time will be used in loading concentrate trucks.

21.7 TAILINGS BACKFILL DISPOSAL

Between 30 and 50 kg of cement per tonne of dewatered tailings is required to make a good paste backfill. A consumption of 40 kg of cement per tonne of tailings is used for this Scoping Study. Only 60% of the fill placed underground will be cemented and only 90% of all tailings will be placed underground in this estimate. The actual quantity of tailings suitable for backfill and the quantity of cement or other pozilans required will be determined from the Pilot Plant test work in 2001.

21.8 POWERHOUSE AND AIR

The power requirements for the powerhouse and air are based on literature collected from CZN files. Supply cost is calculated at 5% of the total annual power costs of all areas.

21.9 SERVICE AND SUPPORT

21.9.1 Employee Transportation

This cost is discussed in Section 19 of this study and included here for completeness of reporting.

Operating costs associated with travel of the contractor employees and consulting personnel is included in the CZN labour costs.

21.9.2 Camp Catering

This cost is discussed in Section 19 of this study.

21.9.3 Laboratory Supplies

The laboratory will provide mine sample assay service for grade control and mine planning purposes as well as assay services for the process plant. The costs are included in operating supplies and are based on similar operations and the monthly assaying load.

21.9.4 Concentrate Shipping

The haulage cost from the site to the load-out facility in Fort Nelson is quoted at $40/tonne of concentrate. An additional $14 per tonne of concentrate shipped to Canadian smelters has been added to cover back haulage to the site of reagents, cement and other necessary supplies (approximately $1 million/annum).

Concentrate will be railed from Fort Nelson to Vancouver or to the smelter at Trail. The cost quoted from B.C. Rail is $37 per tonne of concentrate. An additional $14 per tonne of concentrate will be applied for storage and trans-shipment at Vancouver Wharves. Agency fees are set at $1.00 per dry tonne of concentrate.

Other alternatives and options pertaining to the end purchaser of the concentrate needs to be determined before transportation costs are finalized.

21.9.5 Supplier References

Table 12 – Supplier References

Supplier	Title	Date
Unit Electrical Engineering	Power Generation Requirements	January 2001
Graymont Western Canada Inc.	Quotation on Bulk Crushed Quicklime	December 2000
Molycop Canada	Grinding media Quotation	December 2000
LaFarge	Cement Delivery Quotation	November 2000
Great Western International	Flotation Reagent Quotation	January 2001
Mount Polley Mining Corporation	Processing Power Cost Summary	January 2001
Goldstream Mine	Copper Sulphate Addition Rate	January 2001
JN Davies Electric	Golden Bear Mine Power Generation Operating and Maintenance Costs	December 2000
Mount Polley Mining Corporation	Transmittal on Larox Upgrade Costs and High-Capacity Thickener Costs	November 2000
Golden Bear Mine	Generator Purchase and Lease Plan Quotes	December 2000
Aaron Equipment Company	Quotation for Horizontal Belt Filter	November 2000
Nelson Machinery and Equipment Ltd.	Quotation for Pugmill Mixer	December 2000
Nelson Machinery and Equipment Ltd.	Quotation for Horizontal Belt Filter and Geho Pump	November 2000
CSFM Engineering Ltd.	Cement and Structural Steel Quantities for Concentrate Load-out and Mill Expansion	November 2000
COVER-ALL Building Systems	Quotation for Covered Structure over Zinc and Lead Concentrate Storage Areas at Site	December 2000
Golden Bear Mine	Labour Rates Plus Burden, Employee Transportation and Catering Costs From Actual 2000 Operating Costs	October 2000
Petro-Canada	Fuel Price FOB Fort Nelson	December 2000

22. ECONOMIC ANALYSIS

The purpose of a Scoping Study is to identify possible alternatives for development of a project and to propose a program, which will allow the reasonable and systematic exploration, development and permitting of the project to production. The analysis of any future operation at Prairie Creek is therefore complicated by a number of, as yet incompletely defined, alternatives. However in order to bring some order to this situation, a systematic analysis of the opportunities presented by the project has been attempted using a computer generated financial model.

No single solution to the development of Prairie Creek would be the definitive answer at this time. We have therefore attempted to present a number of different scenarios, which will result in a range of possible outcomes for the project; all of which are extremely positive under any foreseeable metal price scenario.

Please note that all references to dollars are to Canadian dollars unless otherwise specified and all modeling is pre-tax, and before possible partner participation and financing.

22.1 NET SMELTER RETURN (NSR) CALCULATIONS FOR MODEL

Because of the need to split revenues into Vein and Stratabound ores due to their different metallurgies, two smelter models were created; one for Vein mineralisation and one for Stratabound mineralisation. In order to allow the metal prices to be flexed, without setting up a smelter model for each ore type for each year, average metal prices have been applied to each of the two smelter models and percentage revenues and penalties calculated in each case. These percentage revenues and charges were then compared with industry standards to ensure representative modeling.

Prices and exchange rates entered into the models were as follows:

METAL PRICES	($Can/tonne)	US$/t	Cent US/lb
Copper	3206.69	2116.42	**90**
Zinc	1803.76	1190.48	**50**
Lead	801.67	529.10	**25**
Silver ($/oz)		**5.50**	
Gold ($/oz)		**300.00**	
EXCHANGE RATE		**0.66**	

These represent the Company's long term view of metals prices over the operating life of the mine and compare well with Neil S. Seldon & Associates estimates for the same period. Similarly, smelter terms were adopted based on NSA's long-term estimates of smelter charges for the period. These terms were combined into separate models; specific details of which are listed by ore type below.

22.2 VEIN MINERALISATION

Based on the above prices, diluted Vein mineralisation grades and the predictive metallurgy outlined earlier in the report, then the gross contained metal value for diluted Vein ore is US$217.69. After taking into account trucking, rail, shipping and smelting, total NSR values for Vein mineralisation at the mine gate is thus US$65.80. Within this deduction, the freight charges per tonne of mill feed represent US$25.46/tonne of diluted ore.

Taking into account the contribution of each concentrate to the mine, the following mine gate valuation per tonne of Vein material processed (ore, plus 15% dilution at 30% grade) can be prepared for valuation purposes:

	US $/t ROM	Can$/t ROM
COPPER CONCENTRATE	6.19	9.38
ZINC CONCENTRATE	36.92	55.94
LEAD CONCENTRATE	22.69	34.38
Total value per tonne of mill feed		**99.70**

As can be seen from the above, the operation is in reality a zinc mine. Silver acts as a credit in both the copper and the lead concentrates.

Summarising by Concentrate, we can comment as follows:

Copper Concentrate

	Value in US$/ tonne	Percentage of gross
Gross metal values	**$2018.75**	**100.0%**
Paid as Revenue	1742.76	86.3%
Less Treatment Charges	82.0	4.1%
Less Refining	75.40	3.7%
Less Transport/Freight	90.42	4.4%
Less Penalties	779.00	38.6%
Net Smelter Return	**715.95**	**35.5%**

Zinc Concentrate

	Value in US$/ tonne	Percentage of gross
Gross metal values	**$663.35**	**100.0%**
Paid as Revenue	534.07	80.5%
Less Treatment Charges	191.37	28.8%
Less Refining (Silver)	0.00	0.0%
Less Transport/Freight	90.42	13.6%
Less Penalties	30.00	4.1%
Net Smelter Return	**222.28**	**33.5%**

Lead Concentrate

	Value in US$/ tonne	Percentage of gross
Gross metal values	**594.36**	**100.0%**
Paid as Revenue	481.53	81.0%
Less Treatment Charges	182.67	30.7%

Less Refining (Silver)	7.00	1.2%
Less Transport/Freight	51.48	8.7%
Less Penalties	28.50	4.7%
Net Smelter Return	**211.88**	**35.6%**

22.3 STRATABOUND MINERALISATION

Once again, based on the above prices, diluted Stratabound mineralisation grades and the predictive metallurgy outlined earlier in the report, then the gross contained metal value for diluted Stratabound ore is US$136.05. After taking into account trucking, rail, shipping and smelting, total NSR values for Stratabound mineralisation at the mine gate is thus US$ 47.81. Within this deduction, the freight charges per tonne of mill feed represent US$19.41/tonne of diluted ore.

Taking into account the contribution of each concentrate to the mine, the following mine gate valuation per tonne of Stratabound material processed (ore plus 15% dilution at 30% grade) can be prepared for valuation purposes:

	US $/t ROM	Can$/t ROM
COPPER CONCENTRATE	0.00	0.00
ZINC CONCENTRATE	40.26	61.01
LEAD CONCENTRATE	7.55	11.43
Total value per tonne of mill feed		**72.44**

As can be seen from the above, the only real value in the Stratabound ore is in zinc production. Without the large silver credit in the Vein, the lead concentrate struggles to overcome the significant cost of shipping the concentrate and poor payment terms from the smelter.

Summarising by Concentrate, we can comment as follows:

Copper Concentrate

No concentrate produced

Zinc Concentrate

	Value in US$/ tonne	Percentage of gross
Gross metal values	**$669.02**	**100.0%**
Paid as Revenue	540.26	80.8%
Less Treatment Charges	191.37	28.6%
Less Refining (Silver)	0.45	0.1%
Less Transport/Freight	51.48	7.7%
Less Penalties	15.00	2.2%
Net Smelter Return	**281.96**	**42.1%**

Lead Concentrate

	Value in US$/ tonne	Percentage of gross
Gross metal values	**$435.85**	**100.0%**
Paid as Revenue	355.89	81.7%
Less Treatment Charges	182.67	41.9%

Less Refining (Silver)	4.90	1.1%
Less Transport/Freight	51.48	11.8%
Less Penalties	8.00	1.8%
Net Smelter Return	**108.84**	**25.0%**

Copies of the two smelter models, including the indicative metallurgy using the gravity plant, are attached to this section as Vein Smelter and Strata Smelter.

22.4 MODELING OF THE SMELTING PROCESS

Because of the need to vary the metal prices and the differing contributions of each metal within the cash flow model, factoring of the revenues and deductions were generated from the individual smelter models to account for the revenue from each metal and the respective charges applicable to the concentrate dependant on metal price. These were applied to the individual concentrate revenues, which were increased or decreased by the appropriate metal prices. I.e. for the Vein zinc concentrate, the payable revenue at the above price, (50 cents per pound) for zinc within this conc. is say, US$ 587.48. Treatment charges, price participation etc are say $209.01 and penalties are a further US$ 30.00/t. Consequently, assuming the price goes to 60 cents per pound and that the take by smelters, including price participation and deductions, is relatively constant, this figure is flexed by increasing the net revenue figure to 587.48*60/50 or US$ 704.98. At the same time, the treatment charges etc are increased by the same factor, however, penalties are deducted at a fixed rate of US$ 30.00/t and transport is fixed at US$ 90.34/t. Whilst somewhat simplistic, this method reflects relatively accurately the fixing of net revenue percentage by the variation of smelter terms according to price.

This methodology has been followed in each case on the REVENUES sheet of the model, to allow the generation of differing smelter returns based on annual price variation and to allow the model to be flexed, to look at the effect of varying metals prices and the US dollar overall. Revenues for each concentrate are reported by ore type to aid the analysis.

22.5 VARIATION OF NET SMELTER RETURN WITH METAL PRICES AND EXCHANGE RATES

Taking the above smelter models, and assuming a one third, two thirds split, Vein and Stratabound to the mill, then the NSR at the mine gate drops to CDN$96.62 per tonne of mixed feed.

For every:

- **1 cent increase in the price of Zinc**, over the 50 cent base,
  the NSR increases by approximately CDN$ 1.66/tonne.
- **1 cent decrease in the price of Zinc**, below the 50 cent base,
  the NSR decreases by approximately CDN$ 1.65/tonne.
- **1 cent increase in the price of Lead**, over the 25 cent base,
  the NSR increases by approximately CDN$ 0.99/tonne.

- **1 cent decrease in the price of Lead**, below the 25 cent base, **the NSR decreases by approximately CDN$ 0.98/tonne**.
- **10 cent increase in the price of Silver**, over the US$ 5.50 base, **the NSR increases by CDN$ 0.48/tonne**.
- **10 cent decrease in the price of Silver** below the US$ 5.50 base, **the NSR also decreases by CDN$ 0.47/tonne**.
- **1 cent increase in the price of Copper**, over the 90 cent base, **the NSR increases by approximately CDN$ 0.04/tonne**.
- **1 cent decrease in the price of Copper**, below the 90 cent base, **the NSR decreases by approximately CDN$ 0.04/tonne**.
- **1 cent increase in the Canadian dollar exchange rate**, over the 66 cent base, **the NSR decreases by approximately CDN$ 1.82/tonne**.
- **1 cent decrease in the Canadian dollar exchange rate**, below the 66 cent base, **the NSR increases by approximately CDN$ 1.89/tonne**.

As can be seen from the above analysis of NSR, the mine, along with the rest of the Canadian mining industry, is highly sensitive to the strength of the Canadian dollar. Otherwise, the principal revenue generator is zinc, followed by lead. The effects of variation in the silver price are less marked, and copper has almost no effect on the revenues from the property.

The effects of variation of the prices and exchange rates, along with a number of other variables are illustrated on the two spider diagrams accompanying this report.

Of significantly more pertinence is the break-even cash cost of production expressed in terms of zinc price, by estimation of the cost of production per payable pound of zinc metal taking into account by-product credits. This figure effectively defines the zinc price at which the operation breaks even, assuming all other prices are as per estimates. Payable zinc is defined as the total number of pounds of zinc per tonne of ore that will be paid for by the smelter in concentrates after standard deductions. At full production the operation will produce 95 million pounds of payable zinc per annum. This is split 65M lbs from the Vein and 25M lbs from the Stratabound ore annually averaged over the first 10 years of the 18 year mine life.

Contained pounds of payable zinc, per tonne of ore, in each case are 177 pounds per tonne of diluted Vein ore and 153 pounds per tonne of Stratabound ore. Based on the first ten years of production and constant metals prices, then the break-even cost of production, taking into account by-product credits for the current model is calculated at US$ 34 cents per pound of saleable zinc dependant on final mill design and throughput.

22.6 POSSIBLE AREAS OF IMPROVEMENT OF PROJECT ECONOMICS

A number of areas exist in the current proposals that offer real opportunities for improvement to project economics. These are difficult to quantify at this time due to limitations of available information. However, as with all scoping studies, the purpose of this exercise is to target areas that require further study, as part of the work up from Scoping Study to Feasibility Study. These areas can be summarised as follows:

22.6.1 Increased Throughput by feeding more tonnes to the mill

Like all mining operations, a big part of the total cost of the operation is labour and site costs, which are to some extent fixed by virtue of site costs and manning levels. Increasing production levels can be achieved by increasing the proportion of Stratabound ore fed to the mill, by increasing the amount of Vein ore fed to the mill, or by increasing the proportion of both types of ore fed to the mill but at the same time increasing the rejection level to the gravity section of the mill. A straightforward increase in throughput will eventually require an increase in mill capacity.

- **Feed more ore.** - The Current mill design can likely be stretched to accommodate at least 1,800 tonnes per day as currently planned. Assuming no other changes are made to the proportions of Vein and Stratabound fed to the mill and that the mine can deliver these tonnages; this would have the effect of **decreasing the cash operating cost per pound of zinc by around 1.1cents.**

- **Increase rejection in the Gravity plant.** - Heavy liquid testwork at CSMA (Section 12.4.1) indicated that rejection levels in Vein material at an SG of 2.9 could be as high as 68% of feed at –1mm to +0.075mm. Metal losses at this level were 7.5% of the lead and 11.2% of the zinc. By increasing the rejection levels in the gravity section and by using up excess grinding capacity in the primary mill, it is quite possible that the mill throughput can be increased to 2000 tonnes per day if the mine can supply the tonnage. Optimising the rejection rate from the gravity section is one of the primary aims of the pilot plant program, along with firming up of metallurgical performance. Increasing the rejection rate by 20% to 60% and also increasing the tonnes fed to the mill to these levels results in a further **decrease in cash operating cost per pound of zinc by 1.4 cents**.

22.6.2 Increased Grade from the mine

Obviously any improvement in grade fed from underground will have an improvement on costs. While this is to an extent masked by the operation of the gravity plant rejecting waste before the milling process, an improvement in the head grade for the same mining cost will improve the economics of the operation. The current MRDI resource can be considered a diluted resource statement when compared with the in situ grades of the two ore types. The current model further dilutes this material with 15% dilution at 30 % grade. There are therefore a number of areas of much higher grade that are available to the mine that can be fed to the mill for an earlier payback and improved economics.

Increasing the grade fed from underground can thus be achieved through a number of methods:

- **Reduction of Dilution.** – Unlikely in the current circumstances.

- **Mining of higher-grade zones.** –This is easily possible. The Stockwork zone for instance contains 1.05 million tonnes of material grading 0.4% Cu, 5.9% Pb and 15.8% Zn, with 178 gpt Ag (MRDI 1998 Resource statement.) It is however ill defined; (two drill holes and exposure on the 930m. level) and needs further drilling to raise confidence in its availability. Similarly, channel sampling of the in situ Vein on the 930m. Level showed 600 meters of backs grading 0.5 % Cu, 12.4% Pb, 14.6% Zn and 242gpt Ag. Raising the grade of material fed to the mill by 10% to 0.29 % Cu, 9.27% Pb, 11.64% Zn and 151gpt Ag **decreases the cash operating costs per pound of zinc by 3.6 cents** and **increases** the pounds of **zinc produced to 108 million pounds per annum**, creating a double benefit.

- **Increasing the percentage of Vein fed to the mill over Stratabound.** - The NSR of typical diluted Stratabound ore is approximately CDN$80.18/t. of mill feed and the operating cost per tonne of Stratabound runs at around $ 53.08 per tonne. Vein material has an NSR of CDN$108.91/t, with an operating cost per tonne of $62.17. No Stratabound mineralisation is fed to the mill until late on in year two, and after that it is fed approximately 1/3 Stratabound, 2/3 Vein. Increasing the volume of Vein fed to the mill over Stratabound to 75/25 **decreases the cash costs of operation by approximately 1.0 cents per pound of zinc.**

22.6.3 Decreasing Mining Costs

The area of mining cost represents one of the larger unknowns for the project. In order to present a reasonable level of mining cost for the project, capital development and mining development in waste were estimated for a known block of ore. Mining development was then linked back to the available tonnage of ore from that block and a required amount of development estimated per tonne of recoverable ore for each ore type. These amounts were then applied to the ore tonnages mined and fed to the mill. Similarly, costs per tonne mining were developed based on knowledge of other operations and applied on a per tonne delivered to the mill basis. The end result was then blended into an overall mining cost per tonne and this was judged against other operations.

On this basis there are a number of assumptions that can be challenged and would reduce the operating costs underground and thus per pound of zinc. These could include:

- **Installation of a crusher below the level of the Stratabound ore. -** Fed by ore passes from both the Stratabound and the Vein and conveying of the ore from the crusher to the mill. This will require a separate costing exercise, but would significantly reduce the operating cost of hauling ore from underground. It would also reduce the ventilation requirement for the mine and also the air heating requirement in winter. On the ball park presumption that each miner costs $100,000 plus per annum and that 4 trucks would be required to haul ore from underground to the mill, then a personnel reduction of 4 miners per shift, or 16 miners should be able to be discounted against 1.5 extra crushing and conveying operators, resulting in a net saving of 13 miners. Operating cost savings for the trucks could be offset against the cost of running the crusher and belt underground and result in a partial saving of these costs. These trucking costs will increase as the mine gets deeper. Capital costs to the operation for the belt and crusher would increase by the difference between the cost of the 4 trucks and the cost of a belt and crusher installation and the additional ore pass development required. (Likely to be in the region of $3-4 million) This installation cannot take place until the decline is completed to below the Stratabound ore and relies on the decline being mined large enough and straight enough to take a belt, but the crusher could be in place as early as year 2. A very simple analysis of the likely savings suggests a **decrease in mining cost overall of around $2.0 per tonne, or 1.4 cents per pound of zinc.** The installation could pay back within 4 years at this level of capital cost and it is intended to examine this further early in the Feasibility Study process.

- **Increased Tonnage of Stratabound Ore. -** The current Stratabound resource stands at 1.4 million tonnes grading 10.3% Zn, 5.3% Pb and 53gpt Ag. The deposit is open ended in most directions and is only constrained by lack of drilling. While, as earlier stated, this ore, when diluted, carries much lower values than the Vein, it also costs less to mine. Assuming significantly more tonnage can be found in these thicker sections, it is quite likely that the mining costs of this material will be further reduced; particularly when combined with the installation of a belt and crusher combination. While these savings are impossible to quantify at the moment they are worthy of further study, when more drill data becomes available.

- **Decreased cost of Vein Mining.** - Vein mining is expensive in this orebody due to its relatively narrow width and need for a footwall drive on each of the gathering levels. The introduction of Alimak stoping, if feasible, or the driving to the end of each production panel and retreating without the use of a foot wall drive could reduce the mining cost in Vein ore. It is intended to examine the suitability of the Vein ore for Alimak stoping as part of the feasibility process.

22.6.4 Reduction in Milling Costs

Reduction in milling costs for this operation would center on the most efficient operation of the gravity plant to reduce grinding and flotation costs and reduction of reagent costs.

- **Increased rejection by gravity.** - This operation has to be balanced against metal losses to gravity tails and requires optimization. This possibility is further discussed in the section on increased throughput above and assuming an increase in rejection to 60% as per CSMA, but without increasing the throughput, has the potential **to reduce milling costs by 0.31 cents per pound of zinc produced.**

- **Reduction in reagent and cement costs.** – The reduction of reagent costs is possible but unlikely to make a serious difference to the costs of operating the mill. These costs are taken into account if the percentages of Vein vs. Stratabound ore are varied, since reagent costs for Stratabound are significantly higher than those for Vein, due to the addition of lime to depress pyrite. Cement costs could be reduced by the use of pozilans such as pulverised fly ash (PFA) assuming they are available at reasonable cost. No attempt has been made to quantify this saving.

22.6.5 Power costs

Power costs form a significant part of the costs of operation at Prairie Creek. Every tonne of ore processed at Prairie Creek requires approximately 25 kilowatt hours of power. The possibility exists to reduce this number, however this saving is un-quantifiable at this time.

- **The use of diesel generation: -** effectively ties this cost to the cost of fuel, which is at, or close to, an all time high. The costs of diesel generation on the site has been estimated at 15.44 cents per kilowatt hour, based on a rack price of $0.4837 per liter delivered Fort Nelson. **A reduction in rack price of 5 cents Canadian will result in a reduction per pound of zinc produced of 0.23 cents US.**

- **Alternative sources of energy.** - The opportunity exists to install a small (2.5MW) hydro scheme in Prairie Creek or its tributaries. While no work has been done on the possible sites for such a scheme, a brief review of the costs of hydro power on the site indicates that the cost per kilowatt hour of power generated on the site could be reduced from 15.44 cents to approximately 6.00 cents. **This represents a saving of 0.95 cents US per pound of zinc,** against a capital cost estimated at CDN$5.94 million, giving a payback in the 3-4 year range.

- **Power generation using natural gas** – Canada's most prolific gas well lies within 170 km of the site and a number of smaller non commercial gas wells were drilled within 150 km of the site in the 1960's. Since that time exploration north of the Fort Liard highway has been curtailed as part of the land claims process. Exploration interest in the area to the north of the highway is high and a number of companies are positioning themselves to commence exploration within 100 km of the mine within the next year. Additionally, the Mackenzie Valley pipeline, if built, would pass within 100 km of the site. All of these activities lend themselves to the opportunity to install a small low pressure gas line to the site to provide fuel for generation purposes. No attempt has been made to quantify these possible benefits, although the company does hire its camp and facilities to oil and gas companies during their exploration efforts. This opportunity is worth keeping in mind for the long term.

22.6.6 Transport and Shipping Costs

A number of opportunities exist to make significant savings within the charges for transportation of concentrates. These can include shipment through lower cost ports, reduced fuel and truck operating costs through operation of a DCFN trucking company, etc.

- **Shipment through a lower cost port**. – Stewart in Northern BC has a reputation as a low cost port for the shipment of concentrates. Port charges at Stewart are currently $7 per tonne of concentrate, compared with $13-15 at Vancouver. Assuming that the cost of trucking and shipping concentrates through Stewart is the same with the exception of port handling costs, then this could represent a saving of $1.4 million dollars per year. It could also save the cost of constructing and operating a trans-shipping facility at Fort Nelson. Capital cost for this facility is estimated at $0.775 M and operating cost to trans-ship the ore is estimated at $1-3/tonne depending on volume. Instituting shipping through ports other than Vancouver has not yet been researched, but if the above savings can be realised, then they **would represent 1.5 cents US per tonne of zinc produced.**

- **Operation of the concentrate haulage as a DCFN company**. – If the Prairie Creek mine and access road were presumed to be on First Nations land and the load out were moved to Hay River on Native lands and then shipped on rail on the CN system, then a case could be made for relief from fuel tax for the haulage from the mine. This opportunity is encompassed within the Development Co-operation agreement with the Nahanni Butte Band of the DCFN, but no evaluation of the possible benefits of such an arrangement has as yet been completed.

22.6.7 Reduction of Smelter Charges

A number of possibilities exist to reduce operating cost by reducing smelter charges. These center on reduction of penalties for mercury, antimony and arsenic in the copper concentrate and reduction of mercury in the Vein zinc concentrate. A number of paper studies were carried out in 1997 by Patrick Downey and Associates Ltd. on the possible pyro and hydro-metallurgical treatment of both the copper concentrate and the zinc concentrate.

- **Treatment of the copper concentrate** - included autoclaving followed by hydro-metallurgical treatment to remove and produce a mercury product and then to remove and produce a silver product. Summaries of this study, concluded that the project was feasible and would produce up to 500,000 ounces of silver annually. In addition the project would produce a mercury product for sale to refiners. Waste material containing the arsenic, antimony and unrecoverable copper and mercury could be placed back underground as cemented backfill. It is difficult to justify such an installation with the very limited information currently available at this time; however, there is US$1350 worth of silver in every tonne of copper concentrate, which for a typical years production of 3000 tonnes represents revenue from the silver sales alone of US$ 4.59 million. This compares with only US$ 2.52 million revenue for the copper concentrate as a whole after penalties, shipping etc. Assuming a 90% recovery of silver and a saleable mercury product, then Downey postulated a 41.0% rate of return on an investment of $10.0 million for this plant. This would equate to an increase in by product credits equivalent to a **reduction in production costs of 2.11 cents US per pound of zinc**.

- **Treatment of the Vein zinc concentrate** – to reduce the mercury count would have a number of benefits. It would reduce the penalties for mercury, but more importantly it would increase the marketability of the concentrate and allow the operation to take advantage of concentrate sales to Canadian smelters. This has innumerable benefits, including reduction of overseas shipment costs and the protection of the zinc concentrate from exposure to an increase in the value of the US dollar. The proposal put forward in 1997 was to transport the concentrate to a source of cheap energy such as Fort Nelson, where a facility could be established to autoclave the concentrate, removing a mercury product and producing a zinc oxide for sale to the smelters. This report also suggests the possibility of processing the zinc oxide into saleable metal through a leach and electro-winning process. The process is energy intensive and environmentally sensitive and as such has not been studied in any detail at this time. However it cannot be ignored over the longer term, as it represents a significant upside on the sale of a total of around 58,000 tonnes of Vein zinc concentrate.

- **Raise the grade of Strata lead concentrate**. – Currently because of metallurgical difficulties and insufficient testwork, the grade of Strata lead concentrate is set at 57% compared with 75% in the vein zinc concentrate. Insufficient data exists to confirm this relatively low grade of concentrate and further work is required to confirm this grade, or recognise significant gains available by raising the concentrate grade. Increasing the concentrate grade to 75% as per the Vein concentrate would increase the net smelter return from approximately US$70.4 to US$190 per tonne, while reducing the tonnage from 12,500 to 9,500; a net gain of US$925,000 per annum, or CDN$1.4 million equivalent to **a reduction in production costs per pound of zinc of 0.56 cents US per pound.**

22.6.8 Other possible methods to reduce operating costs

Two other more radical proposals can be envisaged which may have the ability to reduce the operating costs and improve the return to the shareholders. These are operation of a winter road only and seasonal operation of the mine. While it is difficult to envisage operating the mine on a winter road only, it has the advantage of significantly reducing the costs of road maintenance. Disadvantages are increase in rail haulage cost, increased need for storage at the mine and railhead, shortage of winter transport, since all the ice roads in the north are operating at the same time, working capital difficulties etc. etc.

However the possibility of operating the mine and mill during the 8 months of warmer weather and then hauling the resulting concentrate out over a winter road cannot be ignored. It is possible that the significant reductions in operating cost and difficulty brought about by warm weather working could offset the disadvantages of expending approximately $20 million to build a road and ferry. This would make the mine a very cheap operation to start up and produce say 65% of the annual throughput of a full size operation. Capital costs could be as low as $18 million to get the mine producing at 350,000 tonnes per year. This possibility, however remote should be kept in mind in the event that capital is difficult to obtain and the market for zinc is strong.

22.6.9 Enhancements to project Internal Rate of Return

The above items will require clarification before a Final Bankable Feasibility Study can be completed and project construction can be justified. However, returns to the equity investor can be improved using a number of traditional and not so traditional methods. Reduction in equity input and enhancement of Internal Rate of Return can be achieved along a number of possible avenues. In the case of Prairie Creek these can include:

- **The obtaining of Federal and Territorial training and infrastructure grants.** – Grant aid is available for training and infrastructure development. The development of the road to the mine ($18 million) could qualify for such grant aid. The provision of over 120 jobs in an area such as Fort Liard/Nahanni Butte would also qualify the operation for training and employment grants. In addition, by linking the operation of the ferry to Nahanni Butte, grant aid should be available for part or all of the ferry cost, estimated at $1million. These possibilities will be further investigated as part of a Feasibility and Funding exercise.

- **Bank Loans** – Like almost all mining developments, the application of bank or supplier loans to the project would be used to enhance the return to shareholders. Assuming the project could qualify for 60% loans against the project, this would improve the IRR of the project by effectively doubling the return on equity employed.

- **Construction and operation of the road and ferry access by the DCFN** – Once again, assuming the road and ferry link can be owned and operated by the DCFN, then, assuming road usage would be charged at a commercial rate, then this would have the effect of removing approximately $18-20 million dollars from the capital cost of the operation, at an effective interest rate of say12%, reducing shareholder risk while providing employment and a steady form of income to the DCFN. Funding for this type of investment for the DCFN could come from the Federal government as part of the current land claims settlement.

- **Operation of the mine over a winter road**. – Once again as in section 22.6.8 above, the operation of the mine over a winter road would reduce the capital cost and thus the risk and return profile to the shareholders by around $16-18 million dollars. Larger concentrate stores would be required at both the mine and the rail load out and this would affect the savings overall. Unfortunately, operation over a winter road with its reduced shipping period comes with its own drawbacks in the form of increased working capital requirements and increased haulage costs, however this case should be fully examined and costed as part of a Feasibility study.

- **Purchase of the Titan Royalty** – The current 2% royalty to Titan Pacific capped at $8.2 million could be purchased up front for significantly less than the residual value. While this would be an up front payment, it would represent **a reduction in operating costs of 0.75 cents per pound of saleable zinc**.

22.7 SUMMARY OF FINANCIAL PERFORMANCE

Because of the many variables in the financing and operation of the mine at Prairie Creek, it is difficult to accurately predict the final outcome of the operation. However, the base case modeled in this Scoping Study indicates a **break-even cash cost** of production **of US$34.5 cents per pound of saleable zinc** after by-product credits, but **before financing and taxation**. The operation will take advantage of the existing mine and mill infrastructure put in place by the Hunt Brothers in 1982, at a cost of CDN$67 million, but never operated. The replacement cost of this mine and mill can be estimated at $100 million in today's dollars.

Capital costs for the new operation are indicated to range from **CDN$40.5 million** to CDN$21.7 million depending on the final configuration of the operation and the construction of an all weather, or winter only, access road. A number of additional upside scenarios exist for the operation and these will be examined further during the feasibility study process.

The base case financial model indicates that the operation at a capital cost of CDN$40.5 million would have a **pre-tax and financing IRR of 45.6%** and an **NPV at 10% of CDN$97.2 million dollars** over the first ten years of a minimum 18 year mine life. The study used long term metal prices of $0.90 per lb – Cu, $0.50 per lb – Zn, $0.25 per lb – Pb and $5.50 per ounce Ag. The Canadian dollar was kept constant at US$0.66. On this basis and with the mine capable of producing just over 95 million pounds of payable zinc per year; for every cent the Zn price is over the break-even production cost of US$ 34 cents per pound, pre tax and financing cash flow increases by around US$0.64 million per annum.

No attempt has been made to complete federal or territorial taxation calculations; no calculation of payments to, or assumption of future participation by the Nahanni Butte Dene has been included and all figures are before taxation and finance.

The capital cost of the operation could be as low as $22 million if a winter road option was selected, however there would be an increasing demand on working capital and operating cost. Construction of an all weather road will add $18 million to the project cost, but significantly eases the operating costs and working capital requirements.

Additional capital can be expended to further improve the project economics by the installation of an underground crusher and belt haulage, and the possible treatment of the copper concentrate by hydro-metallurgical or pyro-metallurgical means to recover the silver on site

A number of significant upside opportunities have been identified which will further enhance project returns. These opportunities require additional work as part of the ongoing feasibility study, however for the purposes of this study they are summarised below:

Table 13 – Scoping Study Variables

Item	**Capital Cost in CDN$**	**Potential for Success**	**Change in Production cost per lb of Zinc in US cents**
Basic operation at 530,000 diluted tonnes per year	$40 million inc. all weather road and contingency	High, subject to Feasibility	34.5 cents per lb break even cash cost including smelter participation all royalties etc.
Operate mine and mill over winter road	Decrease in Cappex by $18 million,	Requires large increase in working Capital	Increase in cash cost of metal of around 1-2 cents
Operate mine for 8 months a year over winter road	Decrease of Cappex by at least $18 million	Requires much smaller increase in working capital	Increase in cash cost of metal of around 1-2 cents
Increase Grade from mine	Nil, high grade from existing development	High, Much higher grades available	Reduces cash cost by 3.6 cents for 10% increase
Install crusher and belt below Stratabound	$3-5 million	High, but adds to capital cost	Reduces cash cost by 1.4 cents US per lb Zinc
Increase rejection in gravity plant and increase throughput to 615,000 tonnes per year	No cost	High, as above	Reduce cash cost by 1.4 cents per lb Zinc
On site treatment of Copper Conc. For Ag and Hg	Around $10 million	Unknown, requires further testwork	2 – 4 cent reduction in cash cost dependent on products made
Purchase of Titan royalty	Say $1-2 million	High	0.75 cent reduction in cash cost.
Increase percentage of Vein to mill	Nil.	Simple for first two years then more difficult	Minor increase in Vein %age reduces cash cost by 1 cent
Decrease cost of Vein mining	Nil if by contractor	Good, but depends on orebody configuration	Unquantifiable
Install hydro generation	Approximately $6 million.	Depends on identification of good stream	0.95 to 1 cent per lb reduction in cash cost of Zinc

Based on the base case cost of production and the assumption that a number of the above savings can be realised, the mine should be capable of producing in the low US$0.30's break even cash cost of production for saleable zinc, assuming the Company's long term price scenarios are met.

The base model has been flexed to demonstrate the effect of changes in various operating parameters on the IRR for each scenario.

If a low cost entry-level mine is required, the possibility of starting the mine over a winter road with seasonal working should not be ignored. In this scenario, a smaller mine could be put into production for around $20 million, within 3 months of establishing the winter road.

All cost figures should be considered order of magnitude estimates (+25% -15% order of accuracy) and will require verification by more detailed study and pilot scale operation in order to convert this Scoping Study into a bankable Feasibility Study.

It should be noted that the Economic Assessment in the Scoping Study is preliminary and partially based on Resources that are considered too speculative to be categorized as Reserves in accordance with National Instrument 43-101. In addition, the Scoping Study is preliminary in nature and despite the existing underground development in the ore body and the on-site mill; the assumptions made within the Scoping Study and its subsequent results may not be attained. It is for this reason that the Company has put forward a development program designed to lead to a full bankable feasibility study and obtaining of the applicable permits, prior to final development taking place.

It is the Company's intention to carry out work in 2001 to increase the confidence levels in the above data through a program of Pilot Plant operation on site, geological and geotechnical examination and delineation drilling. In order to continue with the development of this project and to provide certainty to the future funding partners, or banks and other stakeholders in the project, it is also necessary to complete a bankable Feasibility Study to confirm the final mine design, metallurgical and financial performance, etc. and allow the construction of an operation capable of generating a reasonable return to shareholders and to support significant levels of debt funding.

However, based on the above figures, it appears that a mine at Prairie Creek could have a long and profitable lifespan, with current resources giving a mine life in excess of 18 years. When combined with the exploration potential for the area, this could lead to the development of a significant mining camp and a major profit center for CZN, the Nahanni Butte Dene Band, Northwest Territories and the Deh Cho First Nations.

Figure 34 (Figure 1) – Effect of Changed Economic Variables on Project Return

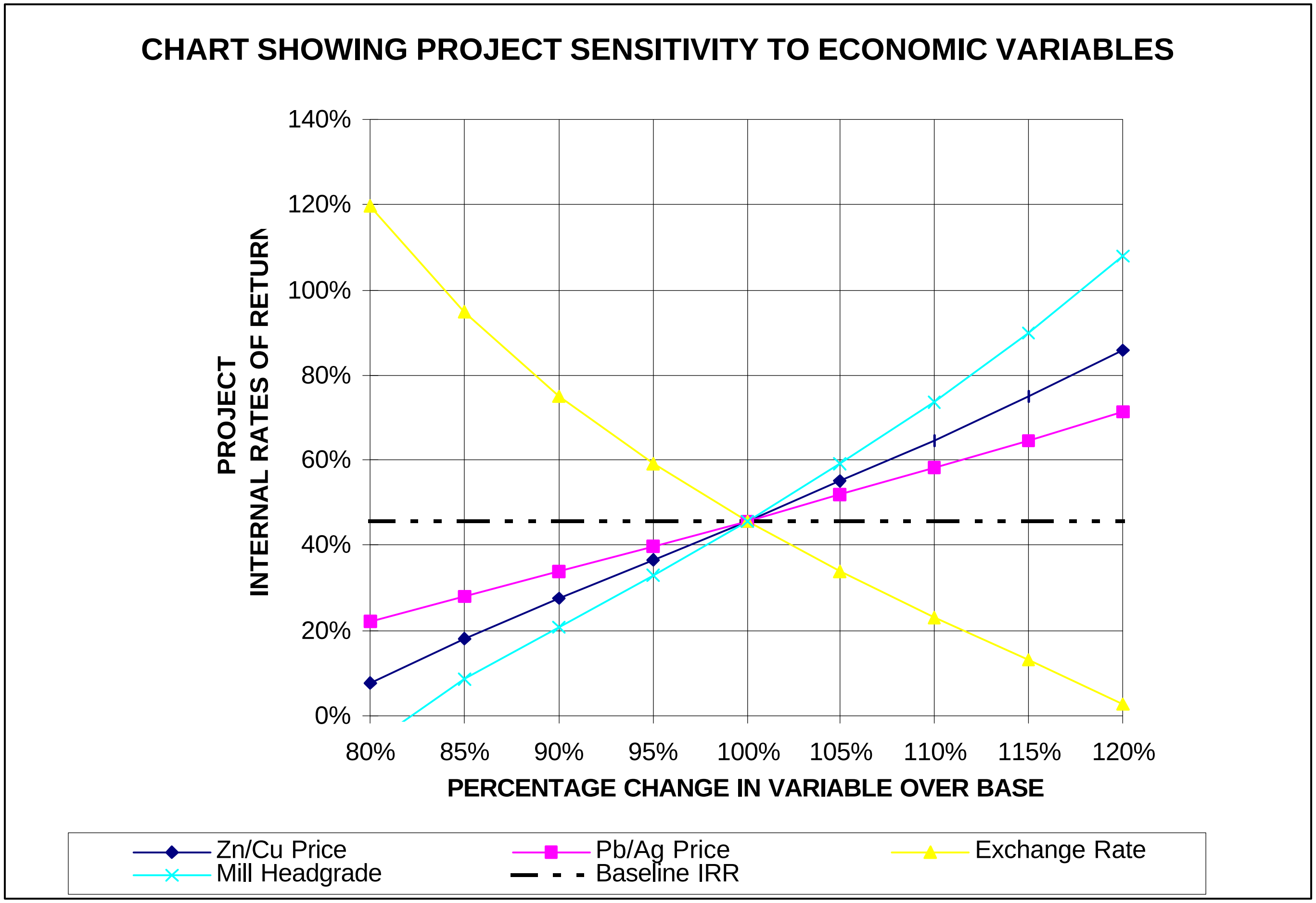

Figure 35 (Figure 2) – Effect of Changed Operating Variables on Project Return

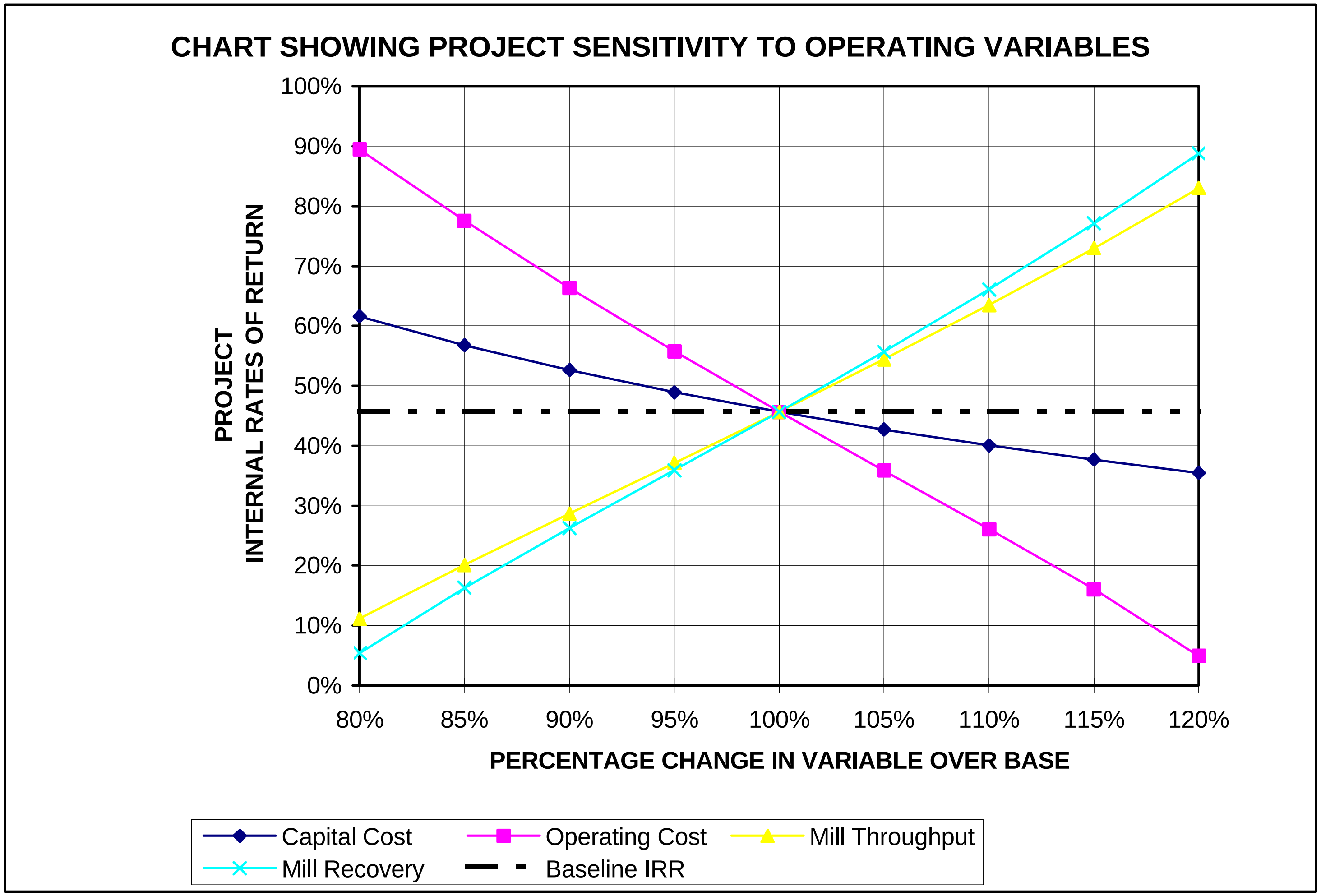

Table 14 – Base Case Cash Flows

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23. PROJECT ORGANIZATION AND SCHEDULE

Work at the Prairie Creek Mine is currently seasonal and starts in May, running through to the end of October. Access can be made earlier using ski equipped planes and then the runway can be cleared using the on site Cats. Accommodation for all on site activities are provided in the existing buildings and a cook and camp help is imported as required. First aid and supervisory services are provided as required by the Northwest Territories Workman's Compensation Board (WCB).

23.1 PERMITTING AND MINE DEVELOPMENT

Because of the current pace of permitting in the Northwest Territories, it is the Company's intention to apply for permits early in 2001 to carry out a number of activities in the 2001 season. This will include the operation of a Pilot Plant on the site, delineation drilling from surface, the driving of an exploration decline and underground delineation drilling, and surface exploration permits elsewhere on the property. This will hopefully result in permits being available to carry out the work when the season opens in 2001.

Following a meeting with the Yellowknife Mineral Development Advisory Group (MDAG) in November of 2000, it is also the Company's intention to work on any deficiencies in the environmental database during the 2001 season, compile the Project Report and to apply for a water licence to operate the mine during 2001. Assuming a further year to complete the permitting and financing, then construction work would commence in early 2003 with the construction of a winter road to the site, along the line of the proposed summer road. Equipment would be imported along the road to allow construction to commence at the mine and mill, concurrently with the construction of the all weather road. Personnel would be flown in to the mine airstrip until such time as the new strip at Sundog was complete, allowing larger loads to be flow in. Mine construction could be complete within 6 months of issuing of the water licence. Road construction would be concurrent with this schedule and would result in the first haulage on the road in midyear (see Figure 36).

Underground development by contractor would start immediately, supported on the winter road. Additional mining gear, explosives, fuel, etc. would be hauled in on the winter road.

Construction of the concentrate handling facilities and freight forwarding depot at the railway would be commenced three months before completion of the road, to take advantage of the better weather.

23.2 PILOT PLANT

During 2001, the Company will run a Pilot Plant on site in the existing mill building to simulate the operation of the mill and produce data for the full bankable feasibility and environmental data for the water permit application. The Pilot Plant will be flown in during early May, having first been set up and water tested in Penticton, British

Columbia in February/March. It will operate throughout the season and treat in the order of 1-1.5 tonnes per hour, or around 2,000 tonnes of ore from the existing surface stock pile, or from underground from the existing levels and if possible, from the new decline.

The existing on site generators, burning on site fuel, will provide power. The crushing plant within the existing mill will be commissioned to provide crushed feed, and water will be provided from existing wells. Tailings will be stored in the thickeners in the mill to simulate the operation of the tailings impoundment.

23.3 SURFACE DELINEATION DRILLING

A program of surface delineation drilling will take place during 2001 below the existing 830 meter level, aimed at near surface extensions of the Stratabound ore and extensions of the Vein below the current lowest level. This drilling will use the two existing Super 38 drills on site and provide detailed drill information for the establishment of mine development below the 870 meter level. This drilling will be supplemented by an underground decline development and underground drilling to further outline the Stratabound mineralisation to the northeast of the current portals.

23.4 UNDERGROUND DRILLING DECLINE

Assuming funding permits, it is intended to drive a decline in the hangingwall of the Vein over the anticline of the Stratabound mineralisation, to provide a platform for underground delineation drilling and to provide a fresh sample of Vein ore from below the current lowest level in the mine. Quotations have been received from contractors to carry out this work by flying in a small jumbo drill to augment the equipment already on site. This work can be carried out early in 2001 subject to receipt of permits and will provide excellent drill intercept angles to increase the confidence in the existing Vein and Stratabound resources and outline addition Stratabound mineralisation outside the current know limits of mineralisation.

A project timeline has been appended indicating the above programs in their broadest terms (see Figure 36).

23.5 BANKABLE FEASIBILITY STUDY

As work progresses during 2001 on the above items, the Company will move to convert this Scoping Study into a bankable feasibility study; covering off the points highlighted within this report. The bankable feasibility study will be carried out under the supervision of an independent consultant. This study will draw on work carried out within the Pilot Plant and elsewhere on the site and will allow the Company to make a positive financing decision. The completion of a bankable feasibility study will trigger a construction decision, subject to permitting, and also subject to the Agreement, will require an investment decision from the Nahanni Butte Dene Band of the Deh Cho First Nations.

The Company will use the bankable feasibility study to arrange project financing and other forms of financing for the completion of the mine and mill.

23.6 ALTERNATIVE SOURCES OF FUNDING

During the next year, the Company will work in partnership with the Nahanni Butte Dene Band and the Deh Cho First Nations to explore possible employment related funding initiatives with the Government of the Northwest Territories and the Federal government. The provision of in excess of 200 jobs in a remote area should be capable of attracting significant federal and territorial funding. Such funding will provide a solid base for the Project and further reduce risk to the participants.

The Company will also use 2001 to pursue other sources of funding for the Project from banks, industry partners and downstream consumers of base metal concentrates.

23.7 PROGRAM COSTS

The Project timeline has been appended indicating the above program in its broadest terms. Approximate costs have been estimated based on quotes from likely contractors and previous experience. These indicate an expenditure in the region of $6 million will be required to progress the Project from Scoping Study to bankable feasibility in 2001.

Figure 36 – Prairie Creek Project Timetable

Oversize Photo

To view, please call the Company at 1-866-688-2001

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January 29, 2001

APPENDIX I

Methodology of Mineral Resource Estimation, extracted from MRDI 1998

For a full copy, please contact the Company at 1-866-688-2001

January 29, 2001

APPENDIX II

Financial Model of Prairie Creek Mine using long term metal prices

For a full copy, please contact the Company at 1-866-688-2001

APPENDIX III

Certificate of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

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I, *Alan B. Taylor*, P.Geo., am a Professional Geoscientist presently employed as Vice President – Exploration, of Canadian Zinc Corporation of 1202-700 West Pender Street in the City of Vancouver, in the Province of British Columbia V6C 1G8; telephone 604-688-2001, fax 604-688-2043, email alan@canadianzinc.com .

I am currently:

- A member of the Association of Professional Engineers and Geoscientists of British Columbia.
- A fellow of the Geological Association of Canada.
- A member of the Canadian Institute of Mining, Metallurgy and Petroleum.

I graduated from Brock University of St. Catharines, Ontario with a Bachelor of Science (with Honours) degree in Geology in 1979, and subsequently obtained a Master of Science degree in Geology from the University of Western Ontario in 1983.

I have actively practiced my profession as a Geologist since 1978. Since 1978 I have been involved in base metal and gold mineral exploration across North America. Specifically I have been involved in various exploration aspects of the Prairie Creek Project since 1993 during which time I actively directed exploration programs for San Andreas Resources Corporation and Canadian Zinc Corporation. Programs included implementation and supervising of geochemical, geophysical, geological surveys and diamond drill programs.

As a result of my experience and qualification I am a Qualified Person as defined in N.P. 43-101.

The mineral resources quoted in the Scoping Study Report on Prairie Creek Project dated January 29, 2001 by Canadian Zinc Corporation were estimated by MRDI Canada (a wholly owned subsidiary of AMEC E&C Services Limited) in 1998 under my guidance in consultation with MRDI technical specialists who are Independent Qualified Persons who visited the Prairie Creek mine site on November 3rd and 4th , 1997. The Independent Qualified Persons employed by MRDI Canada were Mr. Keith Durston Chief Engineer, and Mr. Tim Maunula, Principal Geologist who have many years of experience on computer ore reserve estimation, computerized geological modeling, grade control, mineral exploration and development.

As a Qualified Person representing Canadian Zinc Corporation, I do so state that it is my opinion the categories of measured, indicated and inferred mineral resources in the Australian Code for Reporting of Mineral Resources and Ore Reserves used by MRDI in their 1998 report Prairie Creek – Zone 3 Report are substantially equivalent to the categories of measured, indicated and inferred mineral resources in the CIM Standards on Mineral Resources and Reserves – Definitions and Guidelines as incorporated in National Instrument 43-101.

It should be noted that the Scoping Study is preliminary and partially based on Resources that are considered too speculative to be categorized as Reserves in accordance with National Instrument 43-101. In addition, the Scoping Study is preliminary in nature and despite the existing underground development in the deposit and the on-site mill, the assumptions made within the Scoping Study and its subsequent results may not be attained. It is for this reason that the Company has put forward a development program designed to lead to a full bankable feasibility study and obtaining of the applicable permits, prior to final development taking place.

Dated at Vancouver, British Columbia, this 29th day of January 2001.

Alan B. Taylor P.Geo. Qualified Person