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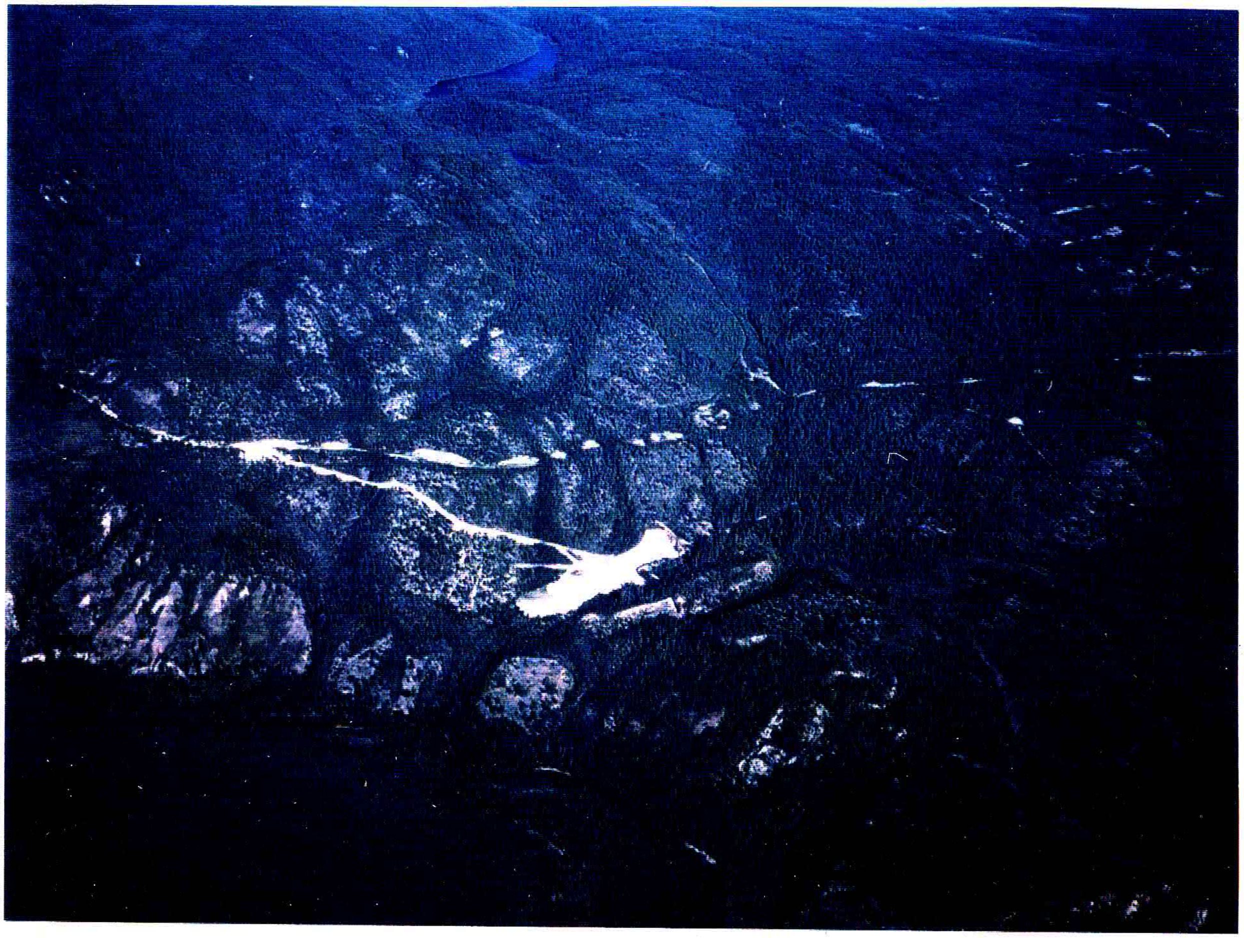

GRIZZLY PROJECT

PRE-FEASIBILITY STUDY

November 1996

Anvil Range
MINING CORPORATION

GRIZZLY PROJECT

PRE-FEASIBILITY STUDY

NOVEMBER, 1996

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1. EXECUTIVE SUMMARY

Surface exploration drilling between 1976 and 1991 revealed the existence of an important mineral deposit at a depth too great for open pit mining from surface. The deposit was called DY, for Dynasty Exploration.

Previous owners have spent an estimated CDN $15 million on drilling, analysing core, evaluating data and producing reserve calculations and reports.

Anvil Range Mining Corporation ("Anvil"), owners of the property, began a review of existing data in January, 1996, and changed the name to GRIZZLY in July, 1996.

The Grizzly deposit is located about 20 km from the Faro concentrator in a southeast line from the Faro, Grum and Vangorda deposits. Deposit structure and mineralization are similar to those of the other Anvil Range deposits and contain lead, zinc, silver and gold.

Ore reserves have been calculated by a number of experts and have been expressed as geological inventory in the range of 20 to 40 million tonnes, depending on cutoff grades.

Additional information for a final feasibility study to allow a production decision to be made is required, but can not be obtained from surface drilling. To go underground and obtain this information through underground exploration is, therefore, the next step in the development of the Grizzly project.

The pre feasibility study has, based on the existing information and latest interpretation of the deposit geology, assumed design and operating parameters for the development of realistic figures for production output, and capital and operating costs in order to establish whether Grizzly would make an economic mine. With the right management and the right approach, the Grizzly will make a profitable underground mine.

Pitwall underground mining was successfully used at Faro in 1990/92. This provides valuable experience for Grizzly underground, although depth is 2 to 3 times greater at Grizzly and the mine infrastructure will be more complex. But it does show that underground mining in an Anvil deposit is possible.

It is now necessary to go underground and prove that the orebody can be mined safely and economically. Twin ramps from the Blind Creek Valley, lateral drift development, exploration drilling, data evaluation and metallurgical testing, final feasibility study, and permitting are the next steps in preparing the board for a production decision and bring Grizzly on stream.

In about five years the Grum pit will be depleted, and mill feed will have to come from elsewhere to continue the Faro operation. Grizzly and Grum Underground are, at this time, the only potential suppliers of mill feed.

Twenty seven months have been scheduled for the underground exploration program, the evaluation of data, the final feasibility study, and the production decision. An additional thirty four months are required to put the mine infrastructure in place for production.

Production start up, including training of people, will take time and full production can be expected after three years from production start up. These estimates are realistic but somewhat conservative, to allow for unforseeable occurrences.

The interpretation of the Grizzly orebody developed during the pre-feasibility study is presumed to be sufficiently accurate to allow realistic assumptions for the calculations of capital expenditures and mining costs to be made. Underground exploration will change this picture and require adjustments to mine planning. This will, however, not change the fact that the orebody can be economically mined.

2. INTRODUCTION

2.1 Purpose of the Study

Based on existing information, the pre feasibility study evaluates whether the exploitation of the Grizzly orebody would be economically feasible and whether further expenditures are warranted.

All information concerning the Grizzly deposit is based on the data obtained from surface drilling with holes some 80 to 200 metres apart and around 1,000 metres deep.

Knowledge of tonnage, grade, continuity, expected mining conditions and metallurgy is essential in predicting production rates, mining cost and the economics of the project. Existing information does not sufficiently provide this knowledge. It is, therefore, necessary to go underground and obtain through underground exploration this information, which will be the basis for a bankable feasibility study and the final approval by the board of directors for the production go-ahead.

2.2 General Information

The pre feasibility study has been carried out in house with input of:

• Access Mining Consultants Ltd.	G.A. (Gregg) Jilson
• H.M. Visagie Consultants	H.M. (Rick) Visagie
• Parwest Mining International	D.M. (David) Parkes
	S.L. (Steve) Szabolcsy
	N.R. (Neville) Pease
• Piteau Associates	N.D. (Nick) Rose

The study consists of 19 sections with subsections, plus Section 20 which contains the Appendices.

Much time was spent on defining underground access and interpretation of deposit geology. Two cases, referred to as "Case A" and "Case B", were analysed in detail to establish capital requirements and operating costs. These form the basis on which economic analyses and sensitivities are built.

The picture of the deposit will change as underground exploration progresses. The orebody will only be fully understood once mining ceases. However, available information and assumptions for Case A and Case B are detailed enough to ensure realistic projections of capital and operating costs.

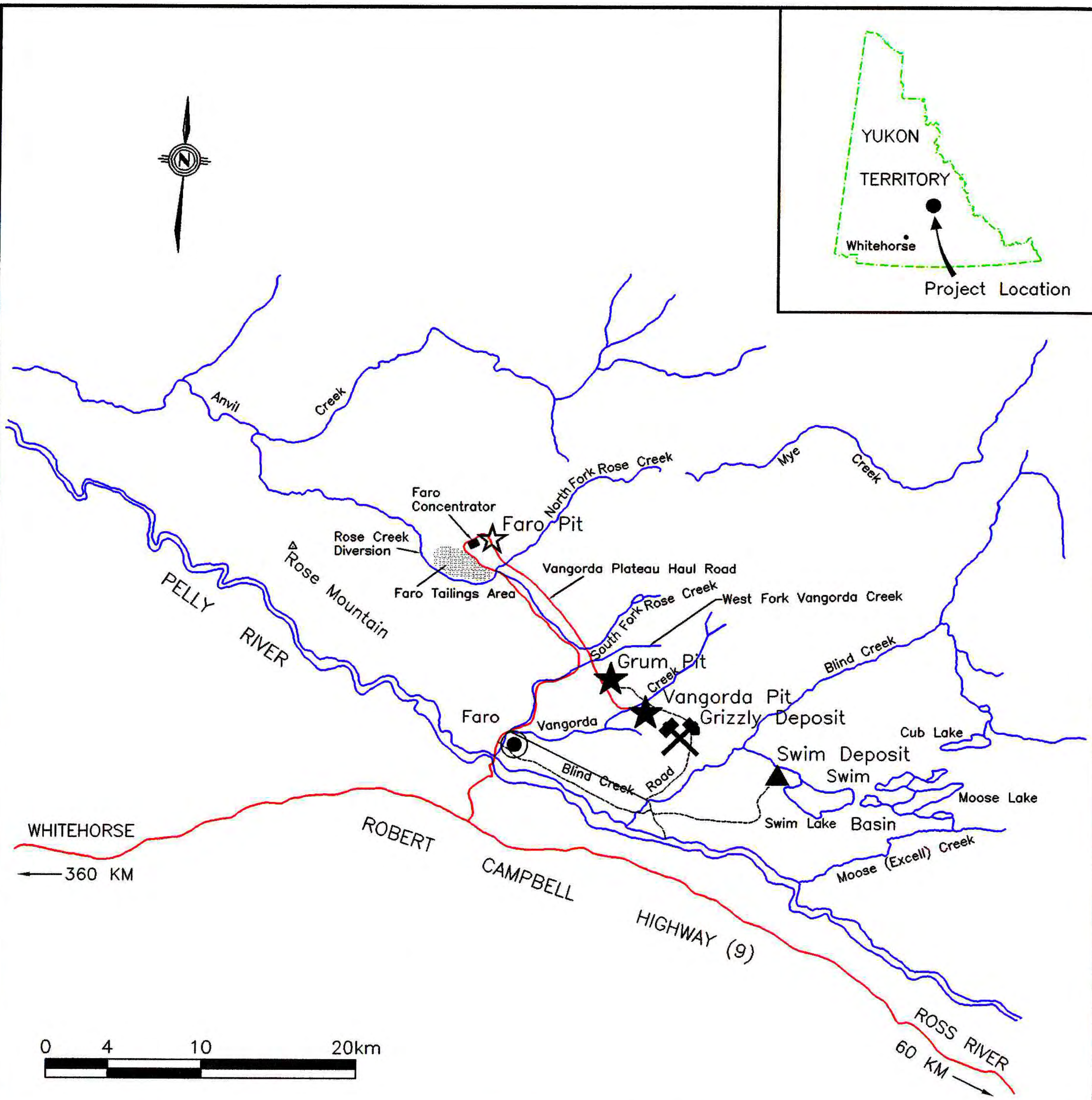
The name of the project was changed from "Dy" to "Grizzly" in July 1996. The deposit is located some 20 km southeast of the Faro concentrator and in line of the Faro, Grum and Vangorda deposits. Since the first surface drill hole intersected mineralization in 1976, a total of 86 holes has been drilled to obtain geological information and geotechnical data. Of these, 56 intersected mineralization within the deposit. The information compiled to date shows the existence of substantial mineral resources at a depth of between 580 and 900 metres below surface.

A drill hole spacing of 80 to 200 metres is too great to provide sufficient information for the satisfactory interpretation of the deposit, and underground exploration must therefore be the next step in the development of the Grizzly deposit.

Ramp and shaft options for underground access were considered by previous owners and have been re-evaluated again as part of this study. A comparison of a vertical exploration shaft, and an exploration ramp from the Blind Creek valley, gives the ramp a cost advantage. For safety reasons and better ventilation, two parallel declines about seven metres apart are favoured as the means of access for underground exploration.

2.3 Project Location and Setting

The Grizzly deposit is located in the Anvil Range lead-zinc-silver-gold district 12 km in a straight line east of the town of Faro (approximately 200 km northeast of Whitehorse), Yukon Territory, with coordinates 62 ° 13' N and 133 ° 08' W (Figure 2.3-01). Specifically, it is situated 6 km southeast of the Grum Deposit on the Vangorda Plateau on the south slopes of Mount Mye. Ground surface in the general area of the deposit is at an elevation of 1,140 m (3,740 ft). The



GRIZZLY PROJECT

UNDERGROUND EXPLORATION PROGRAM

Fig. 2.3-01 Location of the Grizzly Ore Deposit

Anvil Range
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driving distance from the town of Faro along the Blind Creek road to the proposed ramp portal is 16 km.

Surface exploratory drilling to delineate the deposit was carried out periodically at Grizzly from 1976 through 1991. Published drill indicated mineral inventory is estimated to be 21 million tonnes grading 5.8% lead and 6.8% zinc at a cut-off grade of 9% combined lead plus zinc. In addition, the ore contains 83.0 g/t of silver and 0.94 g/t of gold. Dilution and mining reocvery will affect these numbers and change them (see Section 6, Geology and Ore Reserves). Mining of the Grizzly deposit will be strongly affected by geology and tectonics. The cutoff grade will not be controlled by economic considerations, but by mining. The deposit remains open to extension in several directions. It can be expected that additional reserves will be found.

The proposed site for the surface facilities required at the ramp portal to conduct the underground exploration program is located on a southeast facing slope in the valley of Blind Creek, an important salmon spawning river. All precautions must be taken to avoid any spillage of acid water or chemicals into this river. In 1991, overburden was removed at this site in preparation for a ramp portal.

Access to the portal is from the northwest via a secondary road which is an extension of the main mine access road serving the Faro, Grum and Vangorda open pits. Access can also be gained from the southwest from the town of Faro via the Blind Creek Road (Figure 2.3-01).

2.4 History

The initial mineral discovery in the Anvil Range was the Vangorda deposit, which was first drilled between 1953 and 1955 by Prospector Airways. Systematic geological mapping of the Anvil District was not carried out until 1961 by Roddick and Green of the Geological Survey of Canada. The discovery of the Vangorda deposit was followed by the finding of the Faro (1964), Swim (1964), Grum (1973) and Grizzly (1976) deposits.

The Faro deposit was the first of these ore bodies to be developed and brought into production. Mining of the Faro open pit commenced in 1969 under the auspices of the Anvil Mining Corporation, later Cyprus Anvil Mining Corporation (CAMC). Under CAMC's management, mill feed reached rates of up to 10,000 tonnes per day. In the mid 1970's, CAMC embarked on a program of expansion which included both an aggressive exploration program, resulting in the Grizzly discovery, and the acquisition of mineral deposits and claims on the Vangorda Plateau and Swim basin held by Kerr Addison Mines Ltd. (successor to Prospector Airways), including the Grum, Vangorda, and Swim deposits. The objective of the acquisition was to bring the other Vangorda Plateau deposits into production to supplement the Faro mill feed.

Depressed base metal prices, coupled with low productivity and high operating costs at Faro, and the added burden of the debt load brought about by expansion, led to a major slowing down of production at Faro and closure of the concentrator by CAMC in 1982. Some open pit waste stripping operations were carried out between June and October 1983, but mining had ceased completely by the end of 1984.

In November 1985, Curragh Resources (later Curragh Resources Inc. and Curragh Inc.) acquired the holdings of Cyprus Anvil Mining Corporation and reactivated the Faro operation in January 1986. Concentrator operations resumed in June 1986, and the first concentrates were shipped in July 1986. In 1989, development of the Vangorda Plateau was begun with stripping of the Vangorda and Grum deposits. Ore mined from the Vangorda pit was used to supplement and, in turn, replace mill feed from the Faro pit. Between January 1990 and October 1992 underground pit wall mining was successfully carried out at Faro. The property was held on a care and maintenance basis from 1993 to 1994 following the demise of Curragh Inc.

In 1994, Anvil Range Mining Corporation acquired the Faro holdings of Curragh Inc. and mining resumed in the Vangorda and Grum pits. The mined-out Faro pit was used for tailings disposal.

The concentrator at Faro is currently processing approximately 13,000 tonnes of ore per day. The concentrator produces two concentrates: a lead concentrate which includes payable quantities of gold and silver, and zinc concentrate. The concentrates are transported in custom-designed, sealed containers via road to Skagway, Alaska, where they are bulk loaded on ships for markets in Europe and Asia. Proven open pit mineable ore reserves indicate a mine life of approximately five years. Successful development of the Grizzly deposit would ensure the continuity of mining operations and contribute to the longevity of Anvil Range Mining, thus maintaining the current economic benefits to Faro and the Yukon from operations at the Faro mining complex and provide resources to finance the overall district closure plan as well.

3. INFORMATION BASE

3.1 Background Information

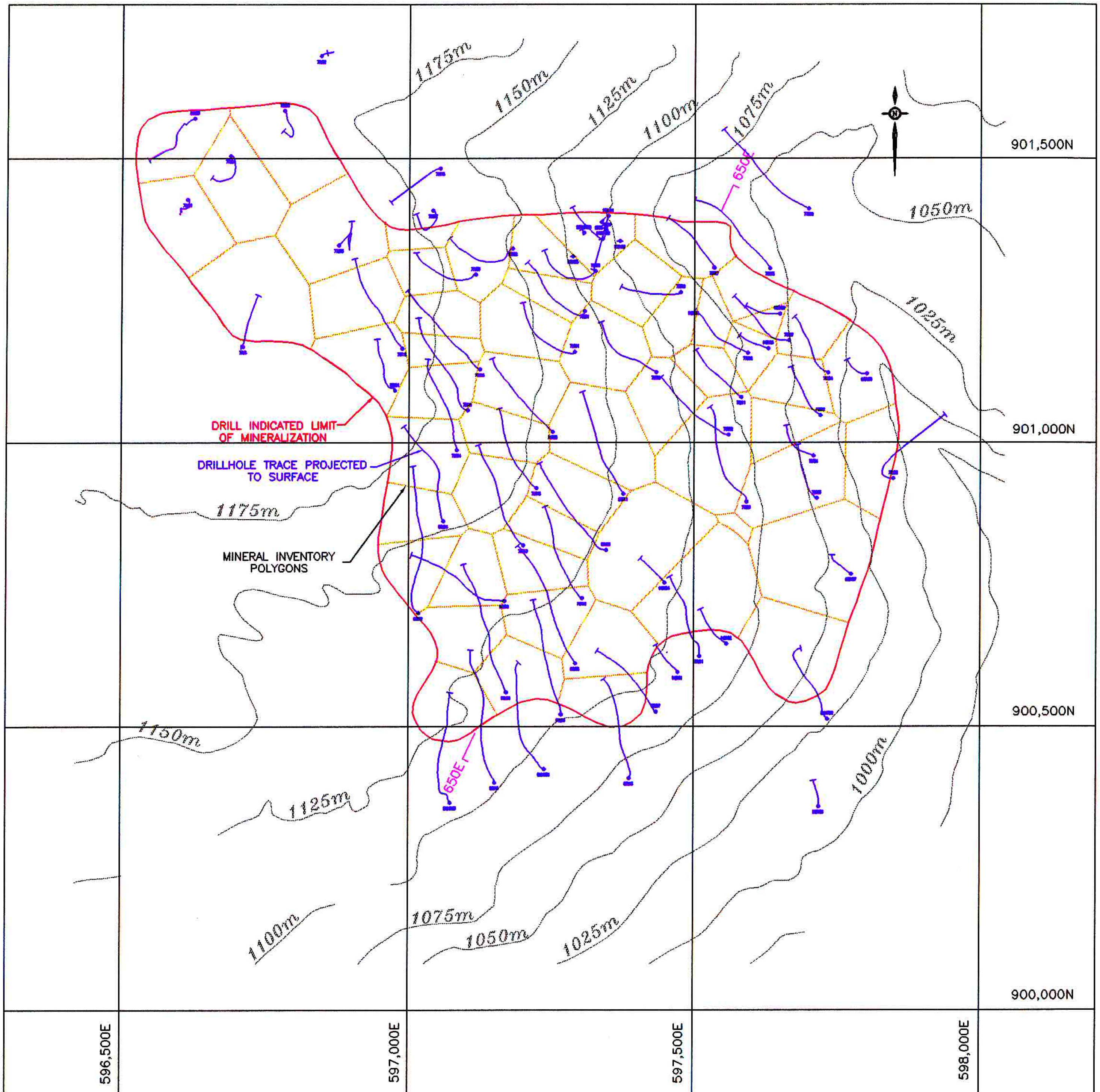
All information on the Grizzly deposit is based on the findings of surface drilling that took place between 1976 and 1991. A total of 86 holes was drilled during this period. Of these, 63 tested the main orebody; 56 intersected mineralization. Most of the holes are around 1,000 metres deep (see Figure 3.1-01).

Core from these holes is stored in core sheds at the Grum mine site and is available for inspection. Core logs, analyses, assay composites, and all other information used for the calculation of ore reserves, are in the Anvil Range Mining Corporation files at Whitehorse and have, together with other reports on the Grizzly, been extensively used for compiling this report.

The backbone of all information available on the Grizzly deposit is formed by the reports on geology and ore reserves. The knowledge gained from the reports listed below forms the basis for understanding the Grizzly deposit as we see it today, and has been used in the review of ore reserves and all reports, including this presentation:

- Hall, 1981, Cyprus Anvil Mining Corporation
- Rollings, 1982, Cyprus Anvil Mining Corporation
- Coltas, 1989, Kilborn Ltd.
- Chernoby and Reed, 1992, Curragh Resources Incorporated
- Jilson, 1991, 1992, and 1993, Curragh Resources Incorporated

Mineralized intersections of drill logs and assay grades used in the interpretation of the deposit are shown in Appendix 6. These intersections give in graphic form % Pb+Zn to the right of the hole and the thickness of the mineralized intersection in metres to the left. In addition Pb%, Zn%, Ag (gr/mt) Au (gr/mt) and thickness of composite intervals (in metres) are shown. Intersections are reproduced from longitudinal sections.



GRIZZLY PROJECT

UNDERGROUND EXPLORATION PROGRAM

Fig. 3.1–01 Surface Drillhole Plan

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Records and reports from the Faro underground (Leo Hwazdyk, *Pitwall Mining Underground at Faro*, 1992) and open pit operations, as well as the experience from Grum and Vangorda, provided additional background information for this study.

In the past, a number of reports have been developed on the Grizzly deposit by various companies, such as Wright Engineering, Kilborn, Rescan, and others.

Between June 1988 and December 1992, Canadian Mine Development (CMD) submitted seven different proposals for underground access and underground exploration. To get underground, do underground exploration and bring the project to the decision point for go-ahead was estimated to cost around CAD $28,000,000. The estimate in February 1991 for driving a single ramp, 6.20 m wide by 4.16 m high, and 1,730 m long at -20% from the Blind Creek valley to a depth of 490 metres was CAD $11,800,000, which did not include the price of a conveyor.

Valuable information is found in the report produced by N.D. Rose, Fox Geological Consultants, in 1992 with an Appendix, here enclosed, written by Drs. Mathis and Page of Steffen, Robertson and Kirsten of Vancouver.

3.2 Existing Infrastructure and Setting

The existing infrastructure includes:

- Faro concentrator with a capacity of about 13,000 tpd mill feed
- Operating organization
- Faro town site
- Skilled mill operators
- Some people in the present workforce interested in being trained for underground work
- Operating and administration staff
- Administration office
- Electricity
- Roads
- Concentrate handling

- Marketing

The Faro concentrator is located about 13 and 20 km north west of the Grum and Grizzly deposits respectively. Dipping to the north west, the Grum deposit extends beyond possible open pit mining to the north west. Underground mining of this portion of the Grum is anticipated once open pit mining ceases.

According to the Anvil Range Mining long-range plan of 1994, this will occur in November of the year 2,000. Additional surface ore is expected to be found, allowing this date to be extended. Planning for underground mining at Grum is at its preliminary stage. It is, however, reasonable to assume that production from Grum underground will have come on stream by the time open pit production ends, to add to the mill feed from Grizzly.

3.3 Assumptions

It is probably possible to start a portal for underground access at Grum in the north pit wall while the open pit is still in operation. This would create an opportunity to train people and develop Grum underground for production. Grum underground will be developed by pit wall mining in a similar fashion to Faro underground, and is expected to produce at a rate of 600,000 tonnes per year or more.

It is assumed that production from the Grizzly will commence prior to cessation of production from the Grum open pit.

Grizzly is planned to produce at a rate of 1.5 million tonnes per year three years after production start up. It is, therefore, reasonable to assume that a mill feed of 2.5 million tonnes is possible from Grizzly plus Grum underground combined in the year 2004.

At the time of writing this report, the proposed drilling of Grizzly East has not yet started. There is a good chance of finding the extension of the Grizzly deposit that has been cut off by the East Fault. This new deposit could be expected east of the known Grizzly deposit at a depth of around 300 meters or more. Underground mining would be required to exploit this orebody.

It is possible that present exploration efforts will bear fruit and lead to the discovery of additional ore reserves. This would add to the mill feed from Grizzly and Grum underground.

3.4 Design and Operating Parameters

Design and operating parameters have been developed for Case A and Case B and are shown in the summary at the end of this section.

Case A assumes a production rate of 1,2 million tonnes of ore per year. The production shaft is located west of the orebody. Main underground development is in waste rock in the footwall. Crushed ore is brought on belt conveyors to the surge bin and to the shaft.

Case B assumes a production rate of 1,5 million tonnes of ore per year. The production shaft is located in the centre of the orebody, main haulage is with diesel trucks to a centrally located underground crusher and from there by belt conveyors to the surge bin and to the shaft. Main underground development is in ore.

Case A and Case B are based on different assumptions and cannot be compared directly. However, each case has been worked out in detail to estimate capital requirements and calculate operating costs. A juxtoposition of the two cases is shown in the Table 3.4-03.

Table 3.4-03

	Case A	Case B
Tonnage/year	1.2 million	1.5 million
Tonnage/day	3,500	4,300
Shaft Location	Southwest of orebody	Central “Barren Zone”
Shaft Diameter	18 feet	18 feet
Vein Haulage	Conveyors	Diesel trucks, 40 ton
Shaft Depth	778 metres	690 metres
Ventilation Requirement	400,000 cfm	500,000 cfm
Fan HP	1,700	2,300

The final feasibility study will establish which alternative of location of the production shaft and which combination of mine infrastructure, mining methods, and mining equipment is best suited to the conditions at Grizzly, and offer the most favourable economic benefits.

SUMMARY OF DESIGN AND OPERATING PARAMETERS FOR CASE A

3.4.1 OVERVIEW

Production	1.2 Mio tpy at 9% Pb+Zn cutoff plus mining mix
Workforce	Anvil Range Existing bargaining unit Commitment to training Mining contractor for the first two years might be considered Camp will be available a.) Contractors equipment
Access	Decline (Blind Creek valley) plus Shaft (production, south west of orebody)
Development	In the footwall (waste)
Mining	Two separate horizons in the AB-Zone
Method	Room and Pillar with pillar robbing in retreat Benching (horizontal or vertical) Bulk Mining Methods where possible Contour Mining is being considered Backfill for mining height over 6.5 metres
Flow of Ore	Scoop trams

	Ore passes
	Crushers
	Conveyors
	Surge bin
	Loadout conveyor
	Loading pockets
	Skip hoisting
	Raw ore storage
	Truck transport to concentrator
Waste Handling	Decline conveyor to surface (initially)
	Underground backfill
Planning Concept	Simple
	Computerised mine planning
Operating Philosophy	Simple
	High level of automation
	Use of latest technology
	Minimum number of people, well trained
Ventilation System	Fresh air intake and main air heating at the production shaft
	Exhaust air through decline
Pumping	Main Pump station at mining level
	Second pump station in mid shaft
	Pump discharge line in the shaft

3.4.2 SURFACE ARRANGEMENTS

Dry and change room	:	At the production shaft
Mine office	:	Existing Grum Facilities
Warehouse	:	Existing facilities at the Faro concentrator

		Open warehouse near underground shop
Underground lamps	:	Shaft collar
Shifters Office	:	Shaft collar
Check in	:	Shaft collar

3.4.3 PRODUCTION SHAFT

Location:

West of A-Zone, outside the influence of mining

Coordinates	900880 N	(approximately)
	596720 E	(approximately)

Depth (preliminary):

Elevations	Collar	1,175 m
	Station	400 m
	Bottom	
	Depth	

Function

ore hoisting

transport of people at scheduled times only

fresh air supply

mine air heating

pump line for mine dewatering

supply of electric energy through main power cable

supply pipe lines for

- Anfo
- diesel fuel
- hydraulic oil

Operation

Hoist will have automatic control for hoisting as well as for transportation of people no special hoist operator

Trained hoist operator for shaft inspection and special trips only

Hoist controls are located in the central control room at the shaft collar

Central control for:

hoisting

pumping

ventilation

mine air heating

underground crushers and conveyors

surge bin

reclaim conveyor

loading pockets

headframe bins

electric stench gas warning system

all communications

automatic greasing of

head ropes

guide ropes and

rubbing ropes

manual greasing of tail ropes

Layout

concrete or steel headframe

tower mounted Koepe hoist

Thyrister control

multi rope installation (probably six headropes, no deflection sheave)

cage over skip combination

rope guides

rubbing ropes

cheese weights

clean-up ramp to shaft bottom

pumping from shaft bottom to pump station

fresh air fans

mine air heating

3.4.4 DECLINE

Function

Air exhaust

Moving large equipment underground

Transporting bulk materials

Conveyor haulage for development muck

Operation

Exhaust air

The haulage conveyor will be left in place until all main development work is complete and sufficient room is available underground for placing waste as backfill

No pumping

Portal

All temporary facilities will be remove by the contractor

Core sheds

Screen gate - normally locked

All acid generating material and ore will be removed

Seeding and tree planting

3.4.5 Underground Infrastructure

Refuge stations

Electric stench gas warning system

Lunch rooms

Shifters office

Sanitary facilities

Underground communications

Power supply

- Compressed air supply
  - stationary electrical compressors
  - mobile diesel and electric compressors
- Ventilation system
  - auxiliary fans
  - ventilation ducting
  - bulkheads
  - overpasses
- Underground shop
- Underground warehouse (open)

3.4.6 Development Mining

- Main conveyor drifts in the footwall
- Drop lines from surface for supply of backfill material

3.4.7 Production Mining

- Orebody Characteristics
  - Depth
  - Strike
  - Dip
  - Number of mineable horizons
  - Uniformity
  - Thickness
  - Grade
  - Strength of ore
  - Strength and shape of hanging wall
  - Strength and shape of footwall
  - In-situ stresses

Methods

- Room and pillar
- Contour Mining
- Pillar robbing in retreat
- Benching (vertical or horizontal with or without backfill)
- Bulk mining where applicable

Design Parameters

- Opening size
- Pillar size
- Extraction rate
- Ground support

3.4.8 Backfill

Application	Only in areas of thick and high grade ore
Material	Gravel and rocks
	Some tailings instead of fly ash
Distribution	Drop lines from surface
	Hopper underground
	Dump trucks
Cement	Slurry plant is located on surface
	Slurry will be added at the dump point at the stope

3.4.9 Roof Support

- Split Sets
- Rebar bolts
- Straps
- Mesh
- Cable Bolts

3.4.10 Environmental Issues

Permits

Constraints

No discharge of water or waste into Blind Creek

3.4.11 Ross River Kaska Dena

Good contact

Training

Contracts

3.4.12 Other Matters

- Safety training
    - ongoing
    - safety meetings
    - mines act
    - "Incident Recall System"
    - "Management of Total Loss Control"
- First Aid Training
- Mine Rescue Training
- Development of standard procedures for
    - operating
    - maintenance
- Development of cost accounting

SUMMARY OF DESIGN AND OPERATING PARAMETERS FOR CASE B

Same as for Case A except as noted below:

3.4.13 Overview

Production 1.5 Mio tpy at 9% Pb+Zn cutoff

Workforce Underground development only partially with contractor, the rest with own crews

Access Shaft will be located in the centre of the orebody

Development In ore wherever possible

3.4.14 Same as Case A

3.4.15 Production Shaft

Location:

Centre of the orebody

Depth (preliminary):

Elevations	Collar	1,140 m
	Station	
	Bottom	
	Depth	

4. SUMMARY, CONCLUSIONS, AND RECOMMENDATIONS

Anvil Range Mining Corporation owns the lead, zinc, silver, gold deposit called GRIZZLY, which is located some 20 km southeast of the Faro concentrator in line with the Faro, Grum and Vangorda deposits.

Surface drilling between 1976 and 1991 outlined the existence of a sizeable ore deposit that, due to its depth, must be considered for underground mining.

A total of 86 holes were drilled, of which 56 intersected mineralized zones. Drill spacing (up to 80 and 200 m between holes at the ore intersections), depth of the holes (mostly over 1000 m), and hole deviation make a proper interpretation of the deposit difficult.

The next step in the development of this deposit, therefore, is going underground to carry out underground exploration.

An underground exploration program has been designed and should be carried out to confirm continuity of the ore zones and enhance the certainty of reserve estimates.

A pre feasibility study has been executed in house, with the assistance of outside experts, to investigate whether the exploitation of the Grizzly orebody would be economically feasible and whether further expenditure is justified.

An analysis based on existing information, combined with realistic and somewhat conservative assumptions, concludes that the Grizzly would make an economically viable mine. Key data, mainly on mining conditions, is still missing and can only be obtained from going underground.

It is the conclusion of the Pre Feasibility Study to recommend to the Board of Directors of Anvil Range Mining Corporation to move forward with the development of the Grizzly Project to reach the point of a production decision, which means the completion of the underground exploration program to prove that the deposit can be mined economically.

This includes expenditures for:

- Going underground with twin ramps from the Blind Creek Valley
- Lateral development underground
- Underground exploration drilling
- Collecting a bulk sample for metallurgical testing
- Technical evaluations
- Final Feasibility study
- Permitting
- Presentation to the Board for a production decision

In case the underground exploration program proves positive, the twin ramps will be used for access and ventilation of the future mine. Underground development for production can be carried out through the ramps and the lateral drifts can be used for exploration drilling.

Two mining methods have been detailed in the pre-feasibility study in order to get valid mining cost.

Once the orebody is better understood, other methods will be considered. Contour mining is a possibility; cut and fill mining with skin pillars might be applicable in certain areas (to save cement costs and reduce development work); bulk mining methods in other areas is desirable.

Mining will be strongly controlled by geology; cutoff grades will be governed by mining.

Mine engineers and mine operators must have a good understanding and a good feel for the ore deposit and its geology and structural features.

5. ACTION PLAN

5.1 Options to be Considered for the Final Feasibility Study

5.1.1 Production Access

- Shaft in the centre of the orebody:
    - shaft location must be confirmed through underground exploration
    - calculation of shaft pillar
    - shaft pilot hole

- Shaft west of the orebody:
    - surface drilling must define the edge of the orebody
    - shaft pilot hole

- Shaft north of the orebody in combination with:
    - inclined shaft hoisting of uncrushed ore with a skip travelling on rails
    - conveyor haulage of crushed ore
    - haulage of uncrushed ore with diesel trucks
    - haulage of uncrushed ore with Kiruna Electric trucks
    - surface drilling must define the edge of the orebody
    - shaft pilot hole

- Conveyor ramp from the Vangorda pit:
    - standard conveyor with several flights
    - standard conveyor with booster drives
    - long flights with steel cord belt
    - cable belt conveyor

5.1.2 Hoisting Arrangement

- Double drum hoist

- Koepe hoist ground mounted (earth quakes)
- Koepe hoist tower mounted
  (preferably six rope friction hoist without deflection sheaves which will reduce the height of the headframe and increase the life of head ropes)
- Pumping of ore after crushing and grinding

5.1.3 Headframe

- Concrete or steel

5.1.4 Mining Methods

- Room and pillar mining with pillar robbing (to 6.5 m mining height), without backfill
- Consideration is given to contour mining
- Room and pillar mining with elongated pillars with backfill, using horizontal and vertical benching
- Bulk mining methods in areas where this is possible

5.1.5 Main Underground Haulage

- Conveyor haulage of crushed ore
- Track haulage of uncrushed ore
- Haulage of uncrushed ore with diesel trucks
- Haulage of uncrushed ore with Kiruna Electric truck
- Inclined skip haulage on rails of uncrushed ore
- Pumping of ore after crushing and grinding

5.1.6 Ventilation Raises

5.1.7 Mine Dewatering

5.2 Project Schedule

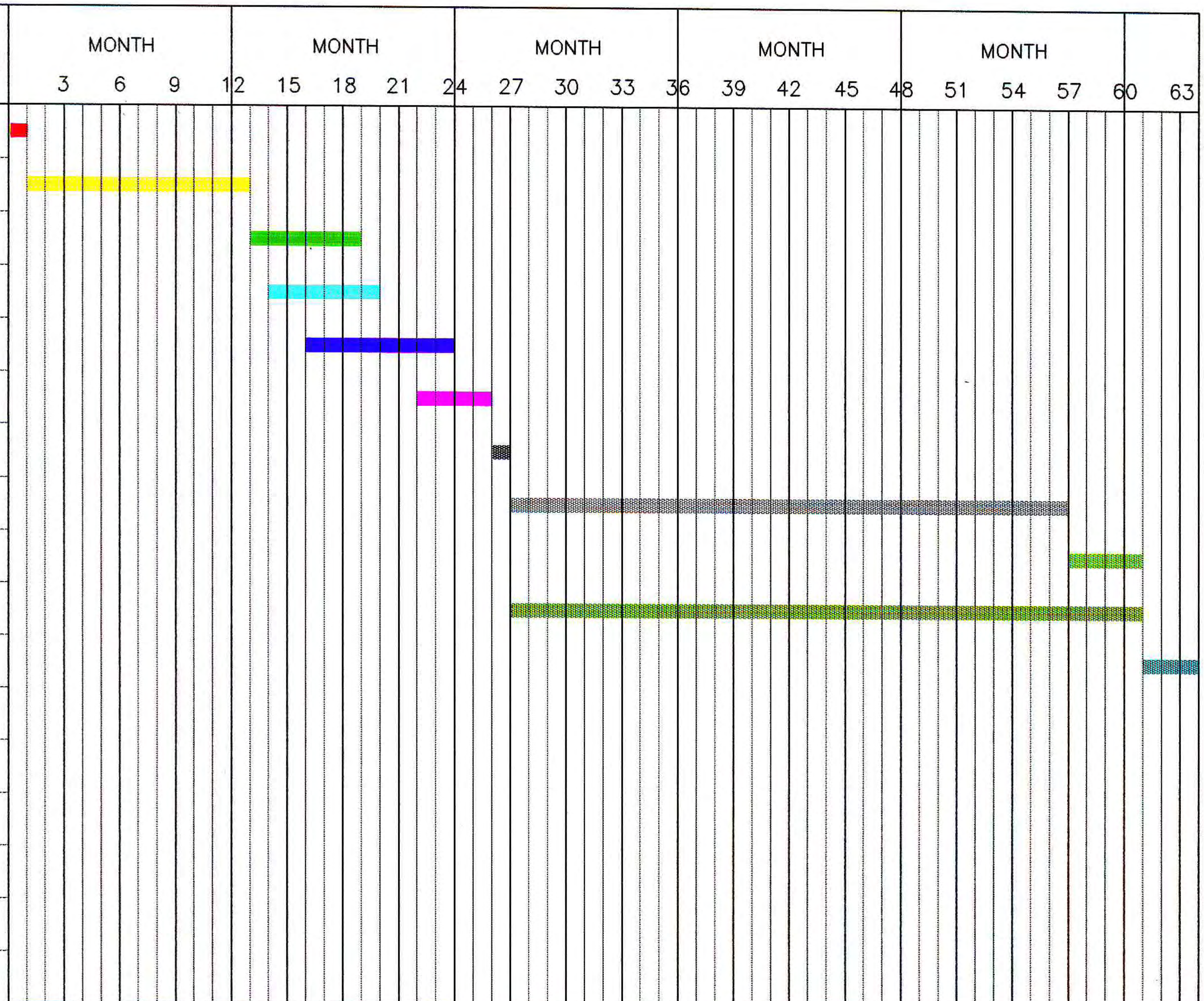
See Figure 5.2-01.

- Ramp Advance
- Underground Exploration
- Evaluation of Data and Feasibility Study
- Review by the Board and Approval for Go-ahead
- Construction of Headframe
- Construction of Raw Ore Storage Bin
- Hoist Installation
- Shaft Sinking
- Shaft Commissioning
- Loading Pockets
- Underground Development
- Underground Crusher
- Main Underground Conveyor
- Surge Bin
- Loadout Conveyor
- Training of Operators
- Production Start-Up

It is assumed that the Board of Directors will approve the expenditure of $22 million for the underground exploration program at the next board meeting in December, 1996. Ramp advance will then start on March 01, 1997. Based on the schedule shown in Figure 5.2-01, board members will receive the bankable feasibility study for review 26 months after start-up of ramp advance.

Production start-up is expected 30 months after the production decision has been made.

No.	DESCRIPTION	DURATION
1	MOBILIZATION AND PORTAL	1 MO.
2	ADVANCE OF TWIN RAMPS	12 MO.
3	LATERAL DRIFTING	6 MO.
4	EXPLORATION DRILLING	6 MO.
5	EVALUATION OF DATA	8 MO.
6	FEASIBILITY	4 MO.
7	BOARD REVIEW	1 MO.
8	CONSTRUCTION OF HEADFRAME AND SHAFT	30 MO.
9	LOADING POCKETS, ROPING UP AND COMMISSIONING	4 MO.
10	UNDERGOUND DEVELOPMENT	34 MO.
11	PRODUCTION START-UP	3 MO.



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GRIZZLY PROJECT

UNDERGROUND EXPLORATION PROGRAM

Fig. 5.2-01 Development Schedule

Anvil Range MINING CORPORATION

5.3 Action Plan

5.3.1 Preparation Work

- Re-logging of the existing core
- Creation of three dimensional computer model (Gemcom GS-32)
- Review experience and knowledge from Faro open pit and underground:

    Geology

    Mining

- Utilise experience and knowledge from Grum:

    Review all existing cross sections

    Review all existing longitudinal sections

    Develop three dimensional computer model (Gemcom GS-32)

    Design surface exploration drilling program for Grum underground

    Schedule Grum underground for production

    Design Grum underground

- Review all information on Grizzly.
- Visit mines with features similar to Grizzly

5.3.2 Access for Underground Exploration

- Obtain data on:

    geology

    geological structures

    hydrology

    geotechnical conditions

5.3.3 Underground Exploration

- Obtain information of the orebody concerning:
  - Thickness
  - Grade
  - Continuity
  - Dip
  - Tectonic
  - Geotechnical data of:

    Ore horizon

    Hanging wall

    Footwall
  - Hydrology
  - Metallurgy

5.3.4 Mine visits to build up expertise

5.3.5 Gathering Information on Mining Equipment and Mining Technology

5.3.6 Evaluation of Data

A considerable portion of evaluation and interpretation of data will be done while underground exploration progresses. However, some activities (core logging, chemical analyses, writing of reports) will continue for about two months after the actual field work is complete.

5.4 Bankable Feasibility Study

The knowledge and information obtained from the underground exploration program will form the basis of the bankable feasibility study. It is estimated that it will take four months to complete this study.

5.5 Presentation to the Board of Directors for Approval of the Project

The Board of Directors will be informed on an ongoing basis. For this reason only one month has been allowed in this action plan for the members of the board to review the data presented to them prior to making the decision for the go-ahead.

6. GEOLOGY AND ORE RESERVES

6.1 Regional Geology

The Anvil District is part of the Selwyn Basin of the Canadian Cordillera which formed part of the ancient North American miogeocline in the early Cretaceous. The district contains five Cambrian to early Ordovician SEDEX (sedimentary exhalitive) type Pb-Zn-Ag (barite) deposits of economic significance that lie in a curvilinear trend on the southwest side of the Anvil Batholith and adjacent to a major orogen scale dextral strike slip fault, the Tintina Fault (Fig. 6.1-01). The deposits are interpreted to have formed in terraced, extensional rift basins similar to other deposits in the Selwyn Basin. The Anvil District has been affected by five deformation events (D_1-D_5) and metamorphosed from greenschist to amphibolite facies during the D_2 event (Brown and McClay, 1994).

6.2 Deposit Geology

The Grizzly deposit is a lead-zinc-silver-gold stratiform, syn-sedimentary, pyritic massive sulphide deposit. The deposit consists of several exhalitive massive sulphide horizons within a series of quartzites, phyllites and schists. One main horizon, termed the AB Zone by Curragh Resources Incorporated (CRI), hosts the majority of sulphide mineralization and forms the most correlatable and continuous sequence defined by surface drilling. Hanging wall and host rocks to Grizzly mineralization consist predominantly of calcareous phyllites of the Vangorda Formation, with a poorly defined transition to the older underlying non-calcareous phyllites of the Mount Mye Formation which occur in contact with or below the Grizzly sulphide horizons.

The orebody lies at a depth of approximately 480 to 920m below surface and dips 20 to 35^{o} to the southwest. Two relatively distinct zones define the orebody in plan view (Fig. 6.2-01), with the southern A Zone (relatively lead-rich) and the northern B Zone (relatively zinc-rich) separated by a central apparent barren massive sulphide zone. This zone is comprised predominantly of disseminated sulphide in quartzite and has recently been termed the "Q Zone".

The B Zone is generally characterized by relatively consistent high grade, pyritic and pyrrhotitic massive sulphide ore with quartz forming the main gangue mineral. The A Zone consists of

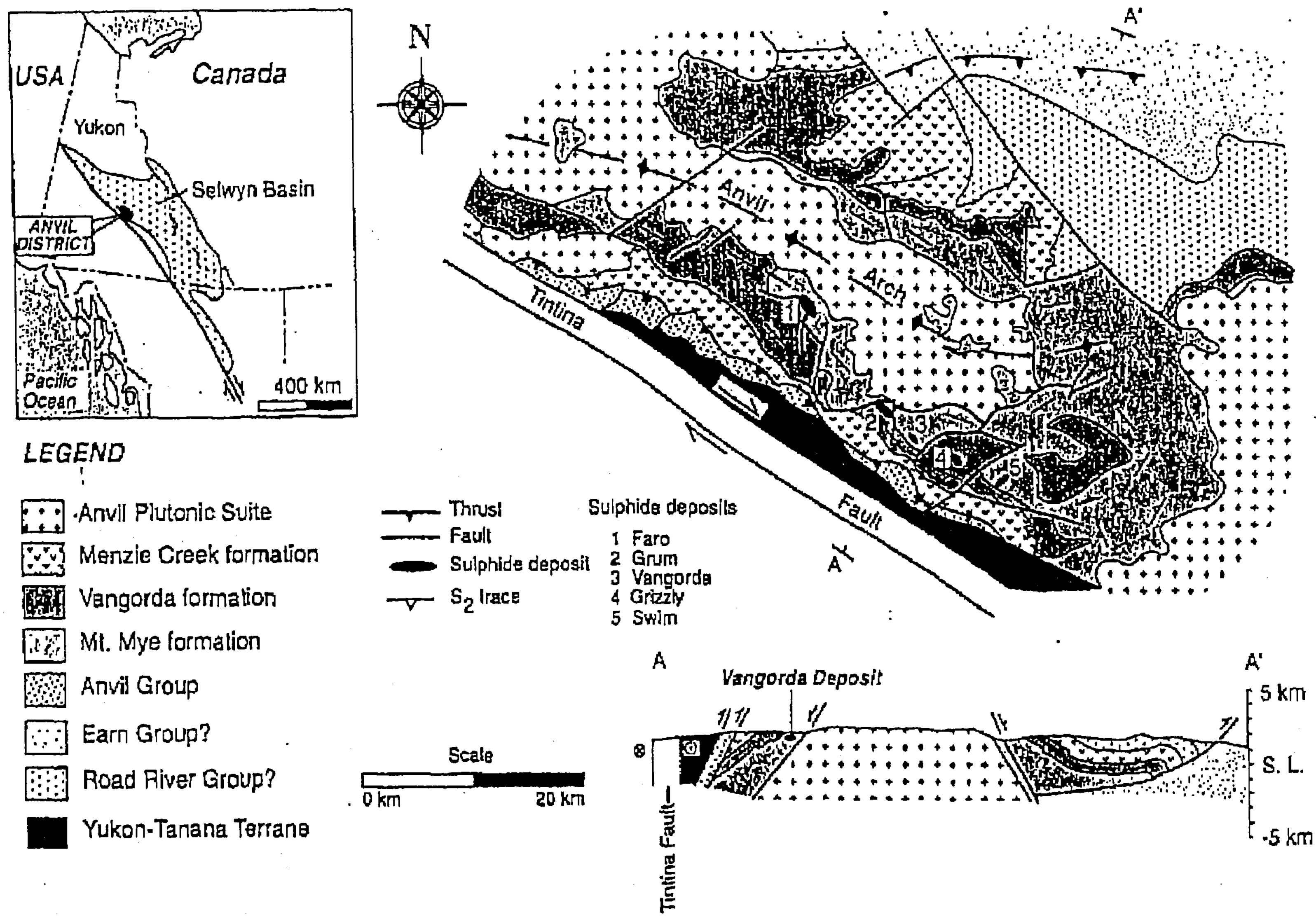


Fig. 1. Generalised geological map and cross-section of the Anvil District. The mineral deposits occur on a curvilinear trend along the southwestern margin of the Anvil Arch (redrafted from Jennings and Jilson, 1986).

FIG. 6.1–01

thick intervals of pyritic massive sulphides, generally of lower grade and greater variation in lead and zinc content. Gangue mineralogy in the A Zone is dominated by barite.

On the east side of the deposit, an approximately 25 to 30m thick quartz diorite dyke with an orientation of approximate strike of 040° and dip of 45 to 60° southeast crosses the orebody on the east side of the deposit (see Fig. 6.2-01).

6.2.1 Structural Geology

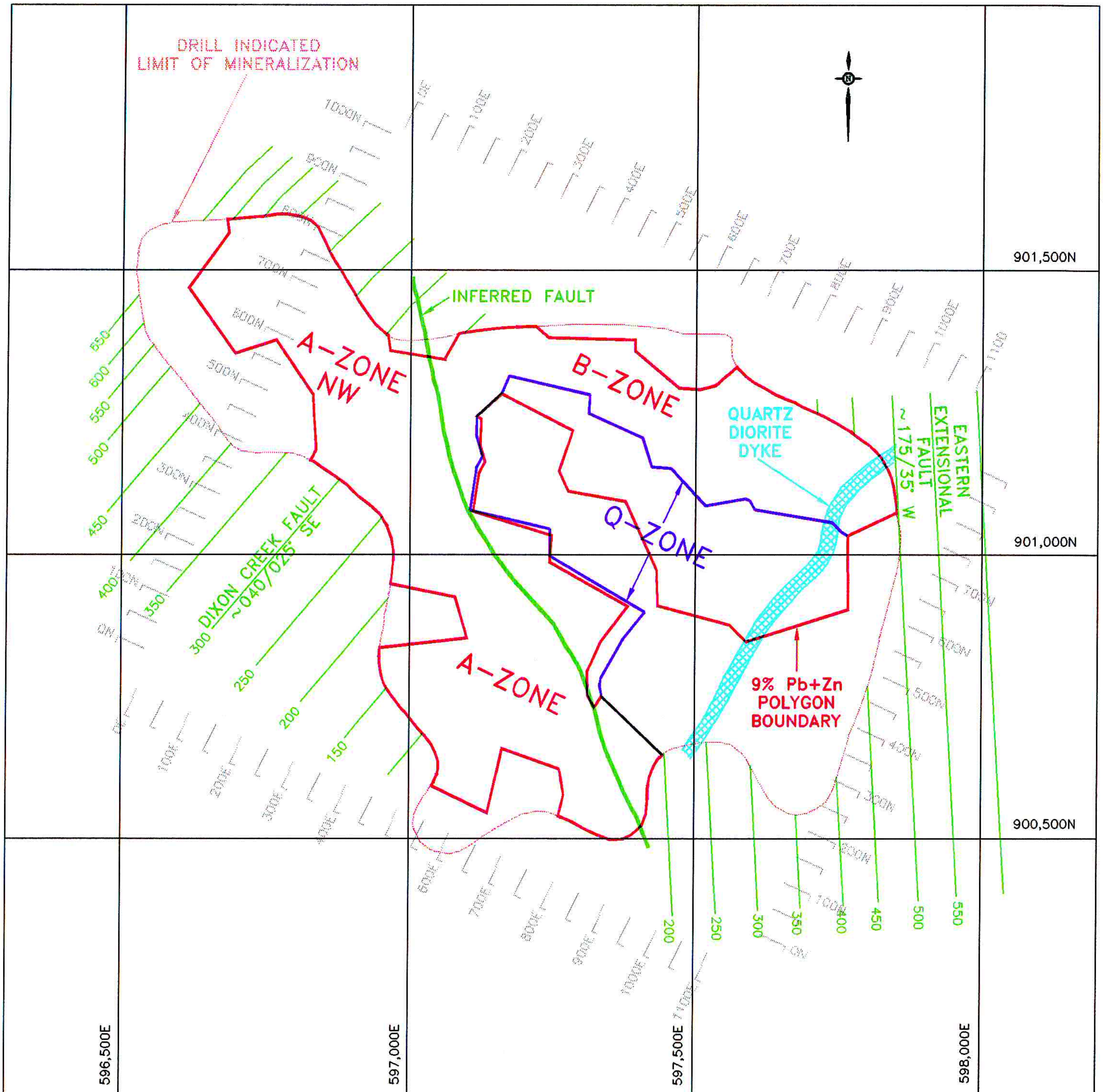
The structural characteristics of Grizzly are poorly understood due to a limited amount of information, though it is reasonable to expect that similarities in structural complexities can be drawn from other deposits within the district (i.e. Vangorda, Grum and Faro). Evidence of at least five phases of deformation occurs on the Vangorda Plateau, the first two of which appear to have the greatest effect on the distribution and nature of the mineralized zones and host rocks.

The first structural event (S_1) defines an early stage fold event which has a significant role in forming the overall geometry and character of the Grum deposit. Typically (S_1) is overprinted by a stronger metamorhpic cleavage (S_2), which is generally subparallel to sulphide layering and defines the most obvious and dominant fabric (foliation) within the phyllitic rocks. The D_3 to D_5 deformation events produced minor folding and steeply dipping crenulation cleavages (S_3 to S_5) that locally overprint S_1 and S_2.

At least two phases of faulting are believed to have occurred during or after the S_2 to S_5 deformational phases.

6.2.2 Faults

Limited information with respect to high angle faults at Grizzly is evident due to bias from near vertical and widely spaced surface drillholes. Drillholes that have encountered steeply dipping faults indicate that faults with significant displacement faults do occur (e.g. a large scale fault encountered in drillhole 90DY04 on the north side of the deposit). This type of faulting is consistent with near vertical displacement faulting encountered at other deposits within the



GRIZZLY PROJECT

UNDERGROUND EXPLORATION PROGRAM

Fig. 6.2–01 General Geology Plan

Anvil Range
MINING CORPORATION

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district. Larger displacement structures of approximately 10 to 20m, or greater, tend to be characterized by thick clay filled gouge and breccia zones which can often transmit water.

Low angle extensional faults at Grizzly have been identified beneath the deposit, and are believed to truncate the orebody on the northwest and east sides. In drill core, the interpreted extensional faults have been logged as lithologic unit "5A*" which consists of relatively intact or healed fault breccia or tectonite. These extensional faults appear, in most cases, to be relatively competent in drill core, but could be expected to have disrupted and have had an adverse effect on the surrounding lithologies in proximity to the low angle structures.

On the northwest, the Dixon Creek Fault has a regional dip of approximately 25° southeast and an approximate strike of 040° (see Fig. 6.2-01). Locally, the fault appears to change in dip between approximately 25 to 45°.

On the east side of the deposit, the Eastern Extensional Fault dips at approximately 35° to the west and appears to have a north-south strike. Drill intercepts on the north end of the deposit indicate a change in strike to the northwest towards the Dixon Creek Fault. Possible interpretation of these structures is that they may have been formed during the same extensional event, forming a down dropped graben which contains the present deposit, or they may have been the same structure at one time but have been offset or disrupted by later stage deformation.

An inferred fault with a northwest-southeast strike and possible near vertical dip to the west is interpreted to occur within the A Zone, as shown on Fig. 6.2-01. Although this fault (or possible fold) has not been verified from drillhole intercepts, a strong roll in structure contours of the ore horizons, as well as an apparent downward displacement of approximately 50m to the west in the quartz diorite dyke, indicates that this fault may be present.

It is important to recognize the importance of steeply dipping faults within the district with respect to understanding the potential complications in mining by underground methods. Experience gained at the Faro underground mine provides valuable insight into the potential structural complexities at Grizzly. At Faro, the occurrence of high angle, 65 to 80° structures, with vertical displacements of 3 to 6m were very common and created difficulties in mining with conventional rubber tired LHD equipment. Displacement faults of 10 to 40m were less common, but were also encountered.

6.2.3 Geology Sections and Structure Contours

A re-interpretation of the Grizzly Deposit geology was carried out in August 1996 in order to define exploration drilling targets and a bulk sampling location from an underground exploration program.

A cross-section grid was created through the deposit on an azimuth of 025° which was considered to best approximate the normal to the average strike of the deposit. This orientation compares to section orientations of 063° used by CRI in 1991, and 019° used by Cyprus Anvil Mining Corporation (CAMC) in 1982.

Using a drillhole database set-up in Gemcom's PCXPLOR software, a series of twenty-three cross-sections and twenty longitudinal sections were generated on a 50m spacing through the deposit as shown on Fig 6.2-01. Assay grades and lithologies were plotted on the sections and colour coded to aid in correlations between drillholes. A sectional influence of 25m either side of section was used to reduce the projectional influence of widely spaced drillholes.

Based on the consideration of a possible 6% Pb+Zn cutoff grade for mining, inspection of preliminary assay composites led to the recognition of two main horizons of economic interest. These correspond to an upper and lower horizon within the AB Zone defined by CRI, and have subsequently been called the UPPER-G and LOWER-G horizons, respectively.

Figures 6.2-02 to 6.2-07, included in Appendix 6, show the geologic interpretations of the UPPER-G and LOWER-G ore horizons on cross-sections 300E, 600E, and 850E, and longitudinal sections 150N, 450N and 750N, respectively. These interpretations are based on wide drillhole spacings and limited information, but at present are considered to define the most correlatable and possibly "mineable" sequences within the AB Zone. In general, the waste interval between the two horizons, consisting mainly of phyllite, varies from approximately 15m on the west side of the deposit up to approximately 60m in the central and eastern sides of the deposit.

Using PCXPLOR, composite intervals representing 6% and 9% Pb+Zn cutoff grades for the UPPER-G and LOWER-G ore horizons were entered into the database and hanging wall and footwall pierce points were generated in plan view. Pierce point locations and elevations were then exported from AutoCAD into SURFER, a computer contouring package, and hanging wall and footwall contours were generated for the two horizons at the two cutoff grades (see Figs. 6.2-08 to 6.2-11).

6.3 Geological Reserve Estimate and Mining Inventory

Previous geological reserve estimates for the Grizzly Deposit (previously known as Dy) have been conducted by B.V. Hall, CAMC, 1981; Rollings, CAMC, 1982; P.C. Coltas, Kilborn Ltd., 1989; CRI (Mineral Inventory), 1991. The CRI 1991 Mineral Inventory provides the details of the previous geological reserve estimates and a detailed description of deposit geology and in-situ resource.

A previous pre-feasibility study involving a mineable reserve estimate and underground mine plan was conducted by N.D. Rose of Fox Geological Consultants (FGC), 1992.

An updated estimate of geological reserves was conducted in August and September, 1996, using Gemcom's GEOMODEL software for the UPPER-G and LOWER-G ore horizons, to form the basis for estimates of a mineable inventory to be used in pre-feasibility investigations of underground mining at Grizzly. It should be noted that all premises and justifications for ore limits in the 1991 CRI Mineral Inventory have been carried over to this investigation. A detailed account of all Probable and Possible (approximately 60% Probable and 40% Possible) mineralization at Grizzly are included in that report.

6.3.1 Calculation Method

A plan view polygonal reserve calculation was conducted with GEOMODEL for the UPPER-G and LOWER-G ore horizons at 6% and 9% Pb+Zn cutoff grades. A detailed account of drillhole composites for the two ore horizons and cutoff grades is included in Tables 6.1 and 6.2 in Appendix 6.

Drillhole composites for the 6% and 9% cutoff grades were calculated in PCXPLOR over a minimum core length of 3.5m. Intersections less than 3.5m in length were diluted to a minimum 3.5m core length using footwall material. Intervals of waste of greater than 3.5m were excluded from weight average composites, whereas intervals of waste less than 3.5m in length were included.

Due to the amount of deviation and flattening in the surface drillholes, a 3.5m core length corresponds to an approximate 3.2 to 3.5m vertical thickness depending on the amount of deviation for each drillhole. This was considered to best estimate an approximate minimum 3m mining height.

Polygons were generated in GEOMODEL by mid-point projections between drillholes (to a maximum of 170m). At the edges of the deposit, the ore zone area of influence was taken from the CRI 1991 Mineral Inventory. This boundary corresponds to a 60m projection beyond the most outboard drillholes containing mineralization.

Polygon volumes were calculated (by GEOMODEL) by multiplying the vertical thickness of the composites by the polygon area. The vertical thickness is derived by correcting for deviation in each drillhole from vertical at the location of each composite centre.

Polygon volumes were converted to tonnage using a density of 3.92 tonnes/cubic metre for all ore types (this value was derived by CRI and is discussed in the 1991 Mineral Inventory Report). Details of polygon areas, tonnages and grades are included in Tables 6.3 to 6.6 in Appendix 6.

6.3.2 Results

The results of the Grizzly mineable inventories for the UPPER-G and LOWER-G ore horizons at 6% and 9% Pb+Zn cutoff grades and 10% dilution are shown in Table 6.7.

Table 6.7

Grizzly Mineable Inventory

10% Dilution

Cutoff Grade	Zone	Pb+Zn (%)	Pb (%)	Zn (%)	Au (g/t)	Ag (g/t)	Tonnage
6%	UPPER-G	8.86	4.03	4.83	58.3	0.66	19,267,173
Pb+Zn	LOWER -G	9.05	3.49	5.56	55.6	0.58	20,001,771
TOTAL		8.95	3.75	5.20	56.9	0.62	39,268,944
9%	UPPER-G	10.85	5.19	5.66	73.1	0.83	11,086,376
Pb+Zn	LOWER -G	11.61	4.45	7.16	69.6	0.68	10,283,155
TOTAL		11.22	4.84	6.38	71.4	0.75	21,369,532

Dilution was added to in-situ values by adding 10% dilution at 0% Pb+Zn grade. This was considered to represent the majority of areas which have a phyllite hanging wall.

For mine planning purposes, drillhole polygons of similar thicknesses and grade within the different zones (A Zone, B Zone and Q Zone) were grouped together to form ore blocks with weighted average grades and thicknesses (see Figs. 6.2-08 to 6.2-11). Details of the drillhole polygons defining each ore block are included in Tables 6.8 to 6.11 in Appendix 6.

Ore blocks consisting of vertical thicknesses of less than 6.5m high were given mining recoveries of 70%. Ore blocks of average thickness greater than 6.5m were assigned mining recoveries of 85% (backfilled areas). The rational for choice of mining recoveries is explained in Section 7.3 under "Selection of a Mining Method".

Approximately 29% of the overall tonnage is defined by thin mining areas (less than 6.5m) and 71% in areas of thick (greater than 6.5m) ore. Therefore, the total recoverable mineable inventory for the 9% Pb+Zn cutoff is shown in Table 6.12.

Table 6.12

Grizzly Recoverable Mining Inventory

10% Dilution

Cutoff Grade	Zone	Pb+Zn (%)	Pb (%)	Zn (%)	Au (g/t)	Ag (g/t)	Tonnage
9%	UPPER-G	10.84	5.23	5.61	73.6	0.83	8,956,019
Pb+Zn	LOWER -G	11.66	4.44	7.22	69.5	0.67	8,284,830
TOTAL		11.24	4.85	6.39	71.6	0.75	17,240,849

6.3.3 Discussion of Results

In consideration of mining at Grizzly, the above compositing criteria is suggested to represent a selective mining situation which would be based strictly on an economic cutoff grade with no defined geologic controls, and may require great detail in drillhole definition.

It should be noted that no lithologic constraints were placed on composite intervals, thus all composites were generated on a grade basis only. Therefore, no differentiation was made between quartzite mineralization and massive sulphide ore, and no restriction was placed on separation of massive sulphide ore and sulphide waste in composite intervals. This was partly due to the recognition that relogging of Grizzly drill core will be required to bring rock codes to a common standard, but also that not enough information exists to adequately define the limits and geological constraints on mineralization at Grizzly.

The pre-feasibility study by N.D. Rose of FGC in 1992 involved rigid geologic parameters in defining a mineable criteria based on experience in underground mining at the Faro Underground Mine. The reader is referred to that study as a comparison with the present results in consideration of mining of massive sulphide ore only. Experience at Faro was that ore grading quartzites were difficult to mine on a visual basis, and that continuity in quartzite grades was extremely variable. More information from underground exploration is required at Grizzly to establish continuities in grade and distinction of ore types.

When considering mining methods, the 3.5m or greater waste exclusions should likely be incorporated as dilution or accounted for in mining and haulage of waste. Also, the recognition of possible mixing or blending of lower grade materials should possibly be addressed in the overall mining inventory and envisioned mining scheme.

7. OREBODY- GEOTECHNICAL CONSIDERATIONS

The Grizzly orebody is considered to be genetically and structurally similar to the Faro, Grum and Vangorda deposits. Ore is hosted by a sequence of quartzites, phyllites, and schists, and is assumed to be variably folded and structurally disrupted by dominantly near vertical faulting.

Very limited geotechnical information exists at Grizzly with respect to the nature and extent of displacement faults, rock competency and structural controls which could affect mining. All present rock strength information is based on empirical estimates from drill core and is not supported by laboratory testing. Collection of geotechnical and hydrogeological data is recommended in further investigations, preferably from underground exploration.

The Grizzly deposit occurs at a depth of approximately 480 to 920m below surface and ranges in thickness from a few metres up to approximately 28m on two different interpreted mining horizons (UPPER-G and LOWER-G). The orebody has an average strike of approximately 115°, typically dipping 20° to 35° southwest; however, steeper dips are anticipated in areas of folds or displacement faults. Large variations in strike (apparent variations of up to 90° locally) are assumed to be related to structural folding or drag due to structural displacements along the Dixon Creek and Eastern Extensional Faults on the northwest and east sides of the deposit, respectively.

Initial underground access for exploration at Grizzly is proposed from the Blind Creek Portal Location (see Fig. 7.1-01).

7.1 PROPOSED DECLINE FOR EXPLORATION AND UNDERGROUND ACCESS

Approximately ten geotechnical diamond drillholes were drilled by CRI in 1990, to test ground conditions along the azimuth of the proposed decline, originating from the Blind Creek Valley (see Fig. 7.1-01). Geomechanical core logging data from the drillholes provides the very limited source of geotechnical information at Grizzly.

7.1.1 Geomechanical Core Logging Data

Geotechnical drillholes for the Grizzly decline were geotechnically logged by CRI personnel in 1990 according to the CSIR geomechanics classification system described by Bieniawski (1976) and the mining rock mass rating (MRMR) system described by Laubsher (1977, 1984), which is a modification of the CSIR system. The CSIR geomechanics system calculates a rock mass rating (RMR) based on estimates of intact rock strength, RQD (rock quality designation), fracture frequency, fracture condition and groundwater. The MRMR system combines fracture condition and groundwater ratings, and involves additional calculations to assess effects of mining conditions on the rock masses.

Due to inconsistencies in CRI logging methods in the initial drillholes, and the later change to the MRMR logging system, only five drillholes contained complete information that could be assessed on a statistical basis. Re-interpretation of core logging results according to the CSIR geomechanics system was conducted by N.D. Rose of Piteau Associates Engineering Ltd. (Piteau) in July 1996. Weighted averages of geomechanical logging data from drillholes along the decline azimuth are summarized in Table 7.1.

Table 7.1

Summary of Weighted Average Geomechanical Core Logging Data From Drillholes Along Decline Azimuth

Rock Type	Total Core Length (m)	RQD (%)	Estimated UCS (MPa)	CSIR		NGI
				RMR	Class	Q
Weathered Calcareous Phyllite	178	25	20	35	IV Poor	0.37
Calcareous Phyllite	724	43	16	41	IV Poor	0.72
Metabasite Dyke	127	64	43	52	III Fair	2.43

As seen in Table 7.1 and Fig. 7.1-02 in Appendix 7, the phyllites, which define the main lithology that will be encountered in the exploration decline and the hanging wall to the Grizzly Deposit, have RMR values in the 30 to 45 range, which characterize a "poor" to "fair" rock mass. A zone of weathered calcareous phyllite was identified to a depth of approximately 50m below surface or approximately the first 150m of proposed decline advance. RMR values for the weathered phyllites range from 25 to 40 (poor). The phyllites exhibit a strong foliation (S_2) which defines weak parting along which separation occurs with only minor displacement. Depending on the orientation of drive with respect to foliation, RMR values are downgraded by 5 to 10 points to account for unfavorable conditions encountered in mining. Estimates of unconfined compressive strength for the phyllites range from approximately 14 to 35 MPa.

7.1.2 Estimation of Support Requirements for Proposed Decline

As a means of estimating ground support requirements for the decline advance, the 'Q' System, an empirical system by developed by Barton et al. (1974) of the Norwegian Geotechnical Institute (NGI), was implemented. This system was developed from 212 tunnel case histories from Scandinavia, and provides a valuable empirical approach to estimating tunnel support requirements as a function of opening size.

Q values (see Table 7.1) can be estimated from the following empirical formula:

$$RMR = 9 \log Q + 44$$

Estimates of support requirements can be derived from the chart shown in Fig. 7.1-03 included in Appendix 7, or can be calculated using a series of empirical calculations outlined in Barton et al. (1974). Tables 7.2 and 7.3, included in Appendix 7, detail support estimates for the proposed decline based on individual intersections down the decline and weighted average results by rock types, respectively. The Q values reported in these tables were derived from specific logging results and may not correspond to the above empirical formula.

7.1.3 Assessment of Kinematically Possible Wedges Based on Surface Mapping

Geotechnical mapping of joint sets at the Blind Creek Portal Cut was conducted by N.D. Rose of Piteau in July 1996. A lower hemisphere equal area projection of mapping data is shown in Fig. 7.1-04 in Appendix 7.

Three main joint sets were defined with joints of set JA1 oriented 79°/069 (dip/dip direction), joint set JB1 oriented 65°/153 and foliation joints FO1 oriented 19°/271. Average joint spacings for the joint sets were 0.9m, 1.5m and 0.09m, respectively.

Stability analyses of wedges were conducted using the computer program UNWEDGE, for an arched drift size of 3.75m wide by 4.6m high, using friction angles (ϕ) of 32° for cross-joints and 21° for foliation joints and zero cohesion (c=0 KPa). These values were derived from shear testing results for phyllites and schists from the Faro Pit as reported to CAMC in a report by Piteau, Gadsby and Macleod Limited (Piteau) entitled "Slope Stability Analysis and Design of the Open Pit Slopes", 1976.

Wedge stability assessments indicate that sidewall wedges could be formed by joint sets JA1/JB1 (up to 6.8 tonnes) with a Factor of Safety (FS) of 0.4 (failure), and sliding along foliation FO1 (up to 3.2 tonnes) at FS = 1.1 (stable) under dry conditions. Assessment of ground support spacings and lengths as detailed in Tables 7.2 and 7.3 in Appendix 7, indicates that sidewall wedges will be controlled with pattern rock anchors providing FS's greater than 2.0. Small wedges of up to 0.2 tonnes are indicated to fall instantaneously from the back, suggesting that some overbreak due to foliation joints should be expected.

7.2 Geotechnical Considerations for Underground Mining

When considering mining of an orebody, physical and geotechnical considerations which affect the selection of a mining method must be considered. A letter report entitled "Conceptual Mining Methods - Dy Deposit" written by Dr. C.H. Page and Dr. J.I. Mathis of Steffen, Robertson Kirsten Incorporated (SRK) in August 1992, outlines general recommendations on mining methods and rock mechanics at Grizzly. This report was included as part of the 1992 CRI pre-feasibility study by N.D. Rose of FGC.

As discussed in the 1992 SRK report, physical parameters which control a mining method are:

- orebody strike and dip
- orebody thickness
- ore uniformity (grade, thickness, strength)
- ore, hanging wall, footwall rock mass strength
- major geological structures as well as rock fabric
- orebody depth and in-situ stresses
- amount of surface disturbance allowed

Only pertinent geotechnical considerations affecting the mining methods which are envisioned to be used at Grizzly will be discussed. The reader is referred to the 1992 SRK report for details of the rationale for selection of the mining methods which are considered in this pre-feasibility study. Recognition is given to the limited present source of information and understanding of the Grizzly deposit.

7.2.1 In-situ Stresses

Little is known concerning in-situ stresses at Grizzly as well at other areas within the Anvil District. It is believed that in-situ horizontal stresses could be as high as twice the vertical stress. Tectonic stresses could also be present at Grizzly.

For depths of 480 to 920m below surface, the vertical overburden stresses within the ore are assumed to range from 13 to 25 MPa.

7.2.2 Rock Competency and Rock Mass Strength

As discussed in Section 7.1.1, the phyllites have RMR values ranging from 25 to 45 (poor to fair), and estimated unconfined compressive strengths from 14 to 35 MPa (values of R2 to R3 on the ISRM hardness scale). These strength estimates derived from core logging suggest that the compressive strengths of the hanging wall phyllites will be in the same range as the in-situ overburden stress (13 to 25 MPa). It is suggested that, based on experience from mining at the Faro underground, leaving a one metre thickness of ore at the back, with added reinforcement (ground support), will help maintain stability in mining spans. This will also help reduce the potential for bearing failure or "punching" with stiff pillars in immediate contact with weak hanging wall or footwall material.

It should be noted that no testing has been done on the phyllites or any of the other rock types at Grizzly. It is recommended that strength index (point load) testing be carried out on drill cores from the underground exploration program at intervals within the hanging wall, ore zone and footwall rocks. Confirmatory laboratory testing (uniaxial compressive tests) should be conducted on some specimens to provide a check on index testing, and to define modulus and deformability parameters which could be applied to numerical analyses of proposed pillar, stope and shaft pillar designs.

Previous point load and uniaxial compressive testing carried out by SRK on geotechnical drill core at the Faro Underground, as reported in a study for CRI entitled "Preliminary Evaluation of Underground Mining at the Faro Pit", 1988, indicates average compressive strength values of approximately 125 MPa for the massive sulphide ore and 110 MPa for the graphitic quartzites perpendicular to foliation. These values are consistent with point load test results of 155 MPa for the massive sulphides at Faro, as reported in the Piteau, 1975 study for CAMC mentioned in Section 7.1.3.

A rock mass strength (RMS) was calculated for the Faro Underground using the following empirical equation after Laubsher (1984):

$$RMS = \sigma_c \times \left(\frac{MRMR - \sigma_c rating}{80}\right) \times 0.8$$

Average MRMR values for the massive sulphide ore and quartzites were 66 and 49, respectively. A design rock mass strength (DRMS) was derived by adding adjustments to the RMS value which relate to the mining environment, such as weathering, joint orientation and blasting effects.

A design rock mass strength for the Faro underground was estimated at 54 MPa and has been used for pre-feasibility assessments of allowable extractions and pillar design at Grizzly. Assessment of core photographs and split drill core indicates that the rock mass strengths of the massive sulphide ore at Grizzly may be similar to Faro.

7.3 SELECTION OF A MINING METHOD

7.3 Selection of a Mining Method

The depth and intermediate dip of the Grizzly deposit ultimately pose the greatest challenge in choosing a suitable mining method. Conventional room and pillar at Grizzly is possible, but extraction will be limited due to its depth. Table 7.3 illustrates the maximum theoretical recoveries for varying safety factors in different portions of the orebody (see SRK report, Appendix 7).

Table 7.3

Maximum Theoretical Recoveries

Standard Room and Pillar

Mining Area	Pillar Stress (MPa)	% Recovery @ SF = 1.5	% Recovery @ SF = 1.3
A Zone	23	36	47
B Zone	17	53	59
Northwest A Zone	21	42	49

As indicated from Table 7.3, theoretical recoveries (based on tributary area theory) indicate that only approximately half of the ore is recoverable using standard room and pillar mining, as was conducted at the Faro underground.

In the above, pillar strength is calculated from the following empirical formula after Stacey and Page (1986):

$$\text{pillar strength (MPa)} = \frac{K\ (W_{eff})^{0.5}}{(H)^{0.7}}$$

where:

K = DRMS (MPa)

W_{eff} = $4 \times \dfrac{\text{(pillar plan area)}}{\text{(pillar perimeter)}}$

H = pillar height

Pillar stress is calculated by:

$$\text{pillar stress} = \frac{\sigma_v \cos^2\alpha + \sigma_h \sin^2\alpha}{1-e}$$

where:

σ_v = vertical stress field

σ_h = horizontal stress field

α = dip of ore (degrees)

e = extraction ratio

The Factor of Safety equals:

$$FS = \frac{\text{pillar strength}}{\text{pillar stress}}$$

In order to increase the extractable reserve, two different mining scenarios were considered.

7.3.1 Proposed Mining of Thin Ore

In areas of thin ore (up to 6.5m thick), a method of caving using post pillars is proposed with recoveries in the range of 70%. The 6.5m height was considered to be a maximum height for single pass mining.

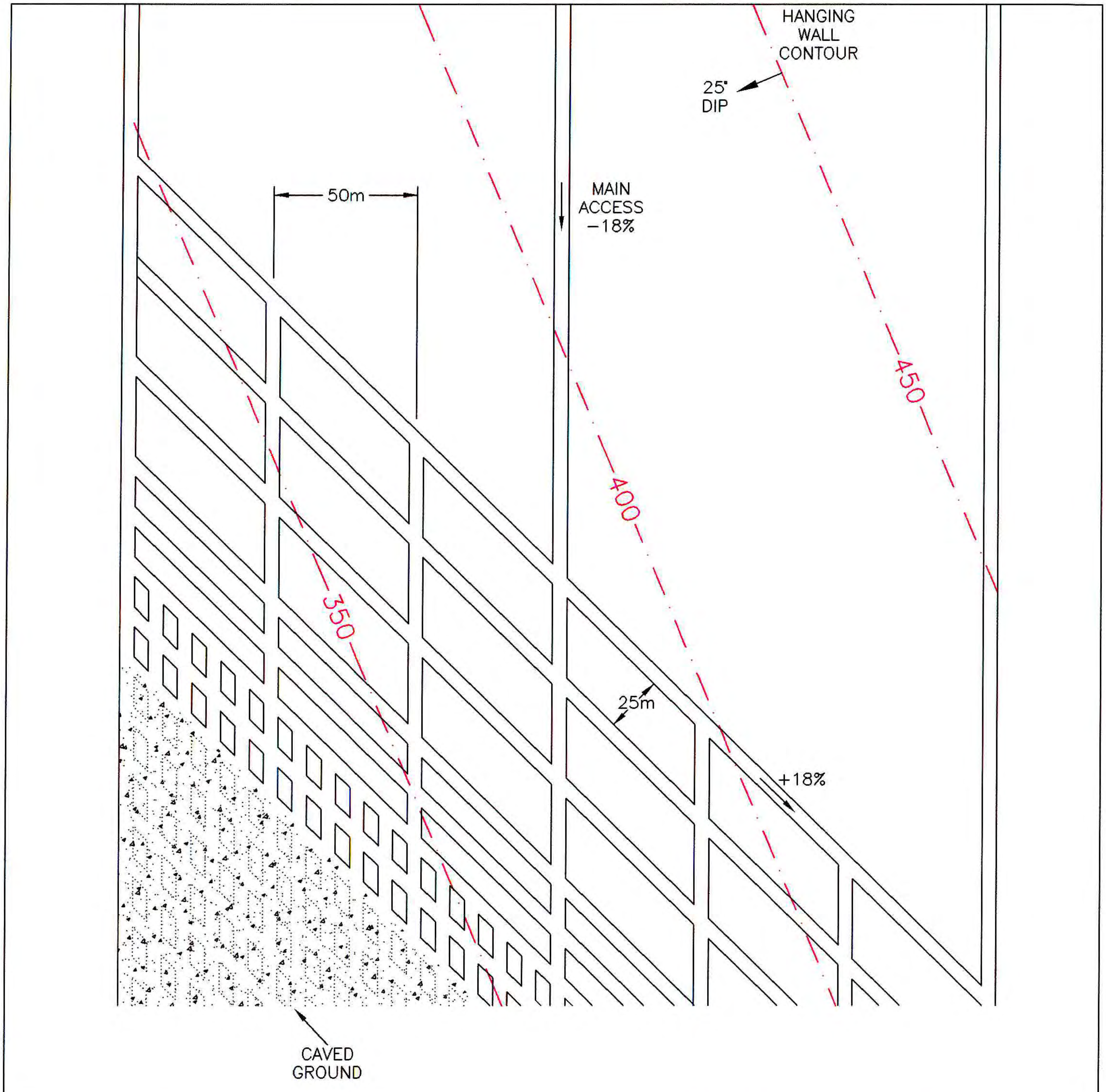
Primary mining would consist of an initial development phase involving 5m wide drifts, driven flat along strike in shallow dipping ore (up to 20° dip), and on components of the dip in steeper ore, with maximum gradients of 18%. Cross-cuts would be driven on 50m centres between the drifts leaving 20m wide by 45m long pillars. Extraction for the initial phase development would be 28%, with FS's indicated to be in the range of 2.0 to 3.4, depending on depth.

Mining spans are suggested to be kept to a minimum (up to 5m) so that full vertical extraction can be considered without leaving a one metre reinforceable sulphide skin at the back. A systematic pattern of 1.8m (6 ft) long rockbolts on an approximate 1.1m square pattern with mesh and straps is considered for preliminary planning purposes. In areas where development may be left open for a long period of time, a resin or cement grouted rebar or corrosion resistant bolt (e.g. galvanized split set) would be used.

The second phase of the primary extraction would consist of 5m wide drifts mined through the centre of the 20m wide pillar, leaving 7.5m wide by 45m long pillars, as shown in Fig. 7.3-01. Extractions of 46%, with factors of safety of approximately 1.1 to 1.7 (depending on depth), are achieved with this phase.

It is recommended that due to the low indicated safety factors, the second phase of the primary extraction be left until retreat in areas where other stoping blocks are accessed from areas of thin ore. This will prevent progressive deterioration of pillars between the time primary extraction ceases and secondary extraction (retreat) begins. This will also reduce the potential for roof failure due to stiff pillars and a weak hanging wall, as pillars take load.

Secondary extraction on retreat is to consist of mining 5m wide breakthroughs on 10m centrelines between the 7.5m spaced drifts, leaving 7.5m long by 5m wide pillars, thus arriving at 70% extraction. These pillars would act as post pillars, with safety factors of 0.4 to 0.7, and



GRIZZLY PROJECT

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Fig. 7.3-01 Conceptual Stoping Method in Areas of Thin Ore (Less Than 6.5m Vertical Thickness)

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would be designed to fail or crush as the mining front retreats. Secondary extraction areas would require mucking by remote controlled machines.

As tributary area theory tends to be conservative and does not take into account shifts in loads from weak pillars to stronger pillars, or the shift of loads to mining abutments, actual stope designs and extraction sequences should be assessed during a full feasibility study, once adequate geotechnical information has been collected from the proposed underground exploration program. An extraction sequence of this type would require confirmation with numerical analyses, trial mining to gain practical experience and a considerable amount of engineering to ensure that an adequate safety barrier from the cave front would be established.

7.3.2 Proposed Mining of Thick Ore

To increase the extractable reserve in thick mining areas (greater than 6.5m thick), a remnant of cut and fill mining, namely 'concrete pillar' or 'panel' mining is contemplated. As illustrated in Figure 7.3-02 in Appendix 7, this method involves mining of primary and secondary panels, with high quality cemented rockfill being placed in primary stopes to provide hanging wall support for extraction of secondary panels. Equal panel widths of 8m were chosen with an optimal panel length of 80m. Alternate hanging wall and footwall accesses allow development, production (vertical benching) and dumping of cemented rockfill from the hanging wall drive; mucking of the ore is to occur from the footwall drive.

Tight placement of high quality fill is critical for the success of concrete pillar mining. The concrete pillars would be designed to prevent large roof displacements, thus preventing roof collapse as well as to carry some minor stress.

In theory, extraction with this method should approach 90% to 100%; however, due to an increased mining span of 8m, it will likely be necessary to leave a one metre thickness of ore to help support mining spans. Based on a reduction in mining thickness, losses due to geological complications and reserves tied up in pillars in roadways, a recovery of 85% is chosen more likely.

Roof support in thick mining areas was estimated using 2.4m (8 ft) long frictional bolts (e.g. split sets) or resin grouted rebar on an adequate pattern (approximate 1.2m square pattern) to support the one metre thickness of massive sulphide and provide support pressure to the back.

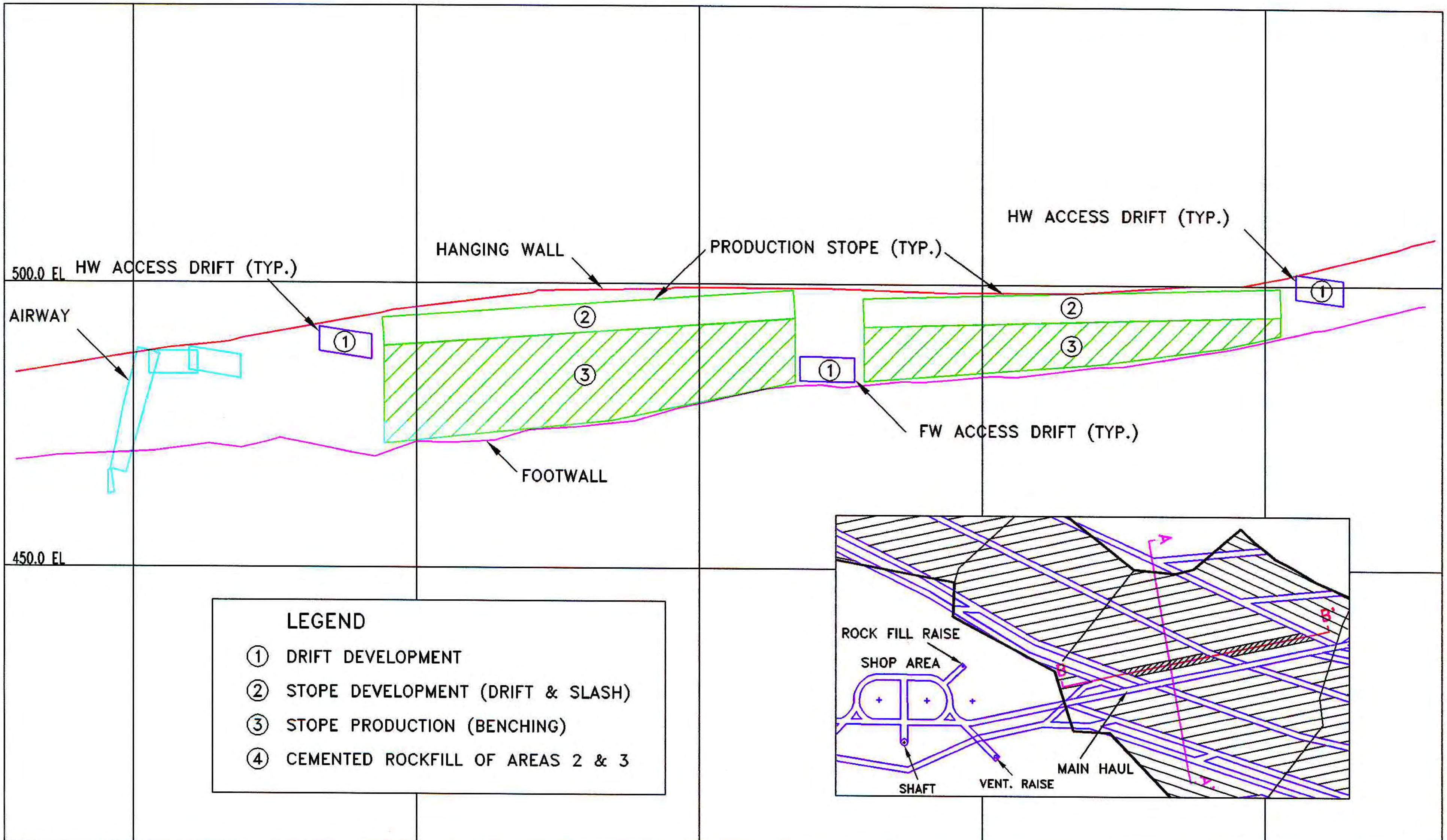
The mining sequence is illustrated in typical B Zone longitudinal and cross sections shown in Figs. 7.3-03 and 7.3-04. Alternate hanging wall (HW) and footwall (FW) drifts allow complete access to the stopes. Development of 8m wide (5m drift plus 3m slash) by 80m long panels would occur from the hanging wall drives, every second stope being developed as part of the primary cycle. Once development on the hanging wall is complete and the ground is secured, longhole production benching would follow with mucking of stopes from the footwall access. Securing of stope walls with 2.4m (or longer) grouted rebar and straps could occur off the muck from the FW access drives. In high stopes, remote control mucking may decrease support requirements and allow increased productivity. Cemented rockfill would then dumped by truck from the HW drive until the stope is full and rock fill is jammed tight to the back. Fill would be allowed to cure for seven days before mining continues.

Consideration to pre-supporting pillars using grouted cable bolts installed from the development level, could be given if joint set combinations define unfavorable orientations that could cause problems with wall slabbing.

Only three active stopes should remain open at any one time. The mining cycle should include development of the first stope, benching of the second, and filling of the third. This will keep the fill cycle current to the active mining and minimize active loads over working areas. Once mining extends to the reserve limits, mining of secondary panels would occur on retreat, with cemented rockfill or waste fill placed in mined out stopes.

7.4 Shaft Access and Shaft Pillar Design

Shaft access for production mining is being considered for pre-feasibility investigations at Grizzly. Two separate shaft locations are considered for the Case A and Case B mining scenarios being considered. These shaft locations are shown on the Case A and Case B mine plans shown on Figs. 13.3-01 to 13.3-04.



SCALE

0m 10m 20m 30m

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GRIZZLY PROJECT

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Fig. 7.3-03 Typical B-Zone Cross-Section

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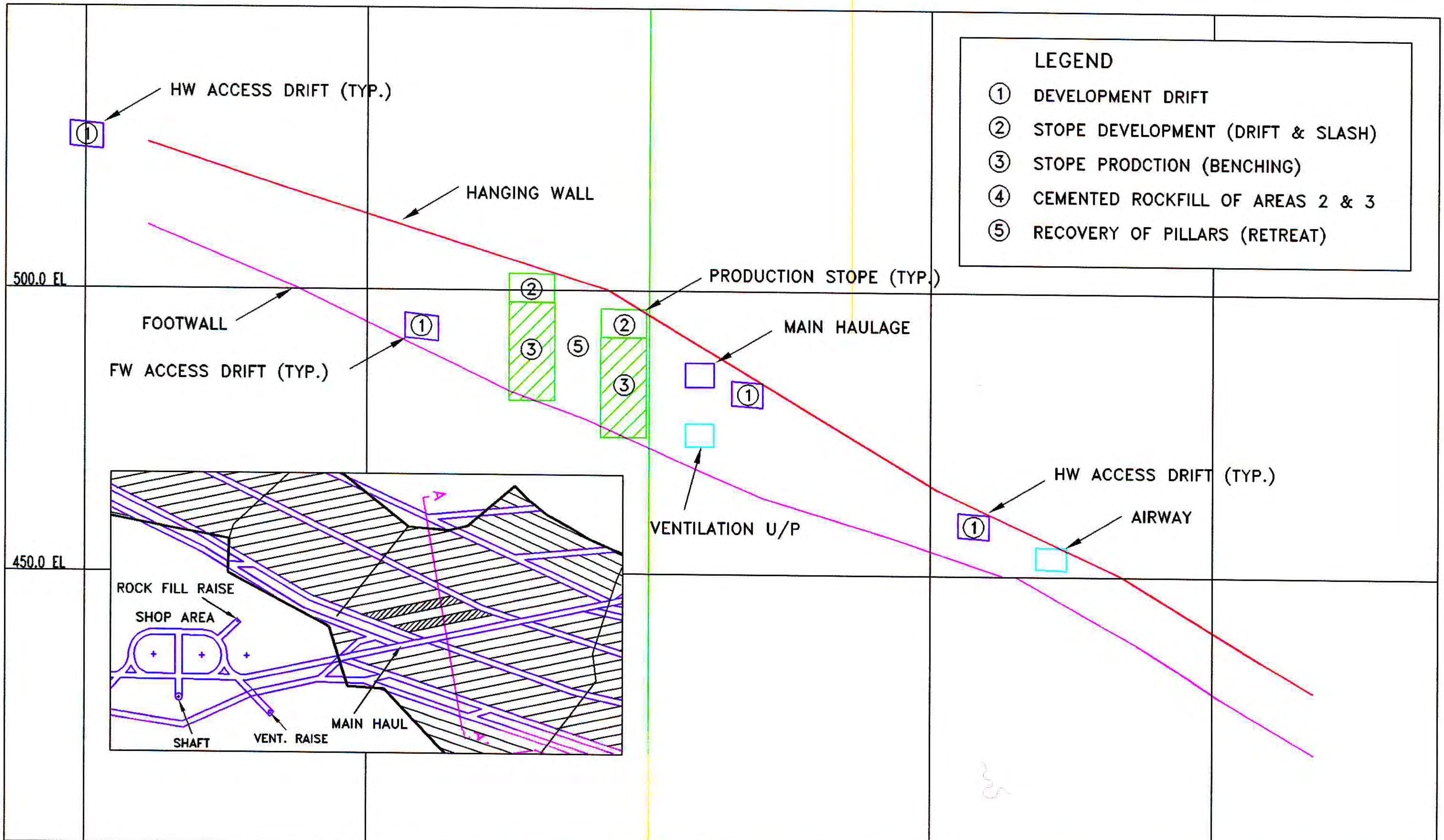

SCALE
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GRIZZLY PROJECT

UNDERGROUND EXPLORATION PROGRAM

Fig. 7.3–04 Typical B–Zone Cross–Section

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7.4.1 Shaft Location and Pillar - Case B Mine Plan

Based on recommendations from Dr. C.H. Page of SRK (personal communications), a shaft pillar with a 10° cone of influence was deemed reasonable for the pre-feasibility mine design conducted by N.D. Rose of FGC in 1992. It was assumed that negligible subsidence would occur with a cemented fill method, resulting in limited potential for divergence in a shaft. The present shaft location (Case B) corresponds to the same shaft location chosen in the 1992 study (see Figs. 13.3-03 and 13.3-04). The previous shaft location was chosen based on the premise that backfill would be placed in all stoping areas whereas present considerations involve backfilling of thick ore and caving in areas of thin ore.

It is recommended that confirmation and design of a shaft pillar be conducted during the stages of a full feasibility study using numerical analysis techniques. This is especially important if a central shaft is considered in conjunction with a caving method, such as the one outlined conceptually in Section 7.3.1. In such a case, a shaft pillar would not only be designed for stress protection, but also for possible movement related to subsidence.

An important geotechnical consideration for the placement of a central shaft is the location of a 25 to 30m thick quartz diorite dyke which crosses the east side of the deposit with an approximate strike of 040° and 45 to 60° dip to the southeast. This dyke would intercept the proposed shaft (Case B) at approximately 100m to 150m below surface. Mining beneath the quartz diorite dyke (especially caving) could lead to subsidence causing possible movement along the dyke contact (acting as a discontinuity) which could cause divergence in the shaft. This should be investigated further during a full feasibility study.

7.4.2 Shaft Location and Pillar - Case A Mine Plan

The Case A Mine Plan involves a shaft location on the west side of the deposit as shown on Figs. 13.3-01 and 13.3-02. This area of the deposit is largely unexplored, with the potential for continuation of reserves up to the Dixon Creek Fault shown in plan on Fig. 6.2-01 and on longitudinal section of Fig. 6.2-05. This extensional fault has a regional dip of approximately 25° (with local variations up to 45°) to the southeast and a strike of 040°. It was assumed that the shaft bottom would be at least 50m above the fault plane. A cone of influence of 15° was considered, taking into account the possible extension of reserves in this area and possible subsidence effects due to caving in areas of thin ore. Consideration was also given to the fact that the majority of ore blocks in the vicinity of the shaft pillar would be in thick ore and that backfilling of these areas would occur.

8. HYDROLOGY

Data on hydrology is hardly existent and needs to be developed, as ramp advance and underground exploration progress. It has been suggested that most of the rock mass will not make water; however, faults can be expected to be significant aquifers, which could discharge large amounts of water into mine openings until pressure is relieved. They will drain and are assumed not to recharge.

A hydrogeological assessment of the deposit area was performed by EBA Engineering Consultants Ltd. in 1990. The field work performed consisted of falling head and constant head permeability testing in diamond drill holes. The constant head tests used double and triple packer equipment.

The general conclusion of the testing was that the observed rock permeabilities were in the range of $5x10^{8}$ to $2x10^{9}$ m/s with lower permeabilities in fault zones (EBA Engineering Consultants, 1990).

The observed data was used to estimate the order of magnitude of potential inflows into a single heading decline for both instantaneous conditions when a fault zone was encountered and for a long term steady state condition. The estimate for the instantaneous condition was approximately 4 litres per second (50 Igpm) for severely fractured zones which do not contain gouge or other joint healing. The estimate for the long term steady state condition was up to 28 litres per second (370 Igpm).

All Grizzly mineralization is below the level of Blind Creek and will not enter the drainage system to the creek by natural flow.

9. UNDERGROUND ACCESS

9.1 Access for Underground Exploration

9.1.1 Introduction

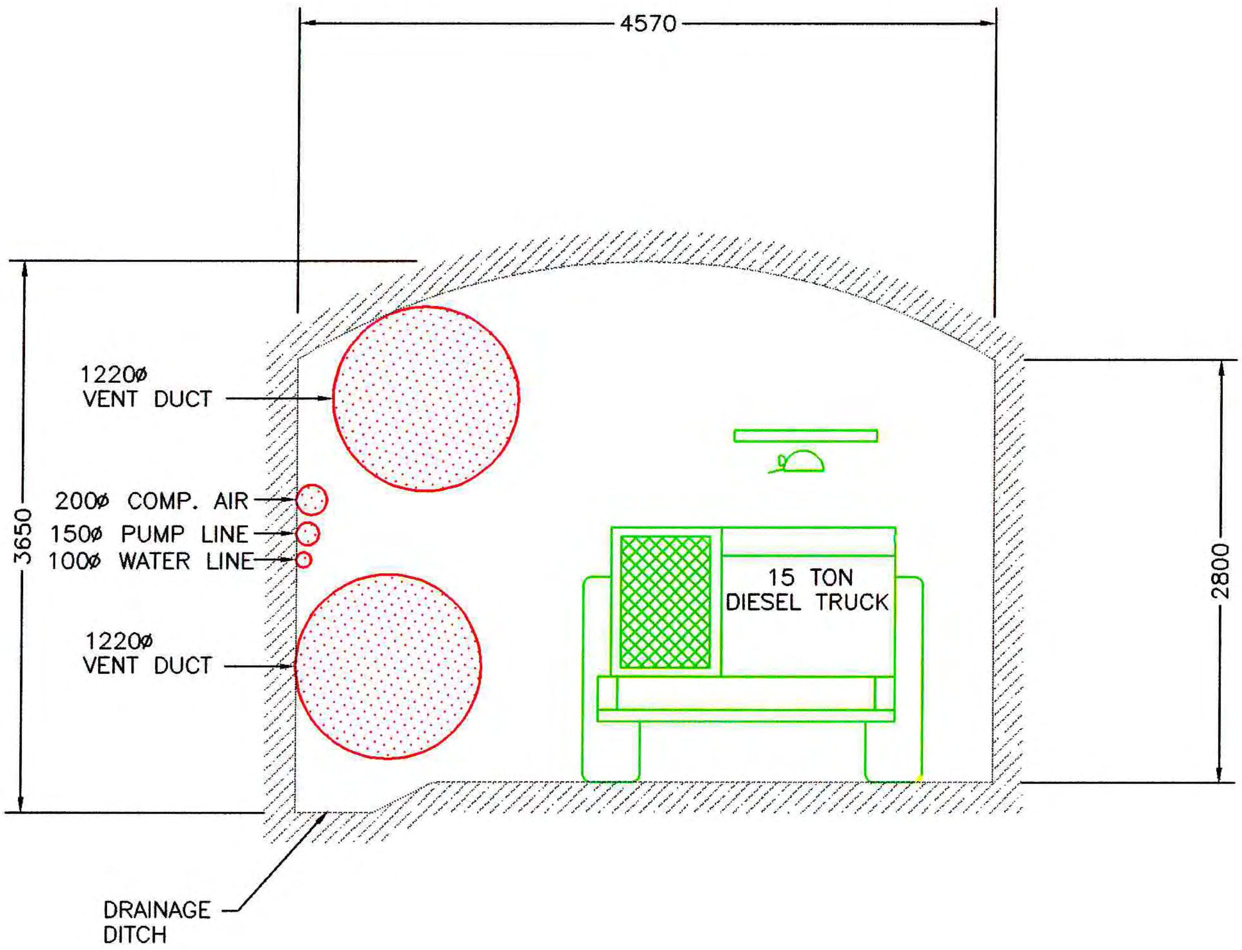
Alternative means of access for underground exploration that were considered include several shaft locations and various configurations of decline (ramp).

Ramp access was chosen because of lower cost than that of an exploration shaft, and the difficulty of finding a location within the area of the Grizzly deposit that would not tie up ore reserves in a safety pillar.

The Blind Creek Valley provides the lowest starting point for a ramp within the reach of the deposit. In 1991 an area some 500 metres from Blind Creek was prepared for a ramp portal. At that time, ten test holes along the proposed ramp route were drilled as well.

Mechanized advance with a full face tunnel bore machine (TBM) or a road header type continuous miner was considered as an alternative to conventional drilling and blasting. Water bearing faults reduce the successful application of a TBM. It would be advantageous to drive the decline in phyllite with a continuous miner. However, the frequent occurrence of quartz veins and some diorite dykes, combined with uncertainty regarding hydrological conditions, render the use of mechanized advance very risky. Conventional drilling and blasting will, therefore, be used to drive the exploration ramp.

The ramp is over 1,700 metres long. Rigid ducting in excess of 60 inches in diameter is required to provide sufficient fresh air for single ramp advance using explosives and diesel loading and haulage equipment. This would require a ramp cross section of about 5.0 m x 5.0 m or more. A large opening will need extra ground support through roof bolting, strapping and possible shotcreting. Shotcrete is always expensive, but due to the transportation costs from Edmonton or Vancouver, the cost of shotcrete at Faro is twice as high as at locations in southern Canada. A number of ramp configurations has been considered. Of these, three are shown in Figures 9.1.1-02 and 9.1.1-04.



MAIN RAMP DRIFT SECTION

TYPICAL DURING EXCAVATION PHASE

GRIZZLY PROJECT

UNDERGROUND EXPLORATION PROGRAM

Fig. 9.1.1–02 Single Ramp – Typical Section Using Small Diesel Truck

Anvil Range MINING CORPORATION

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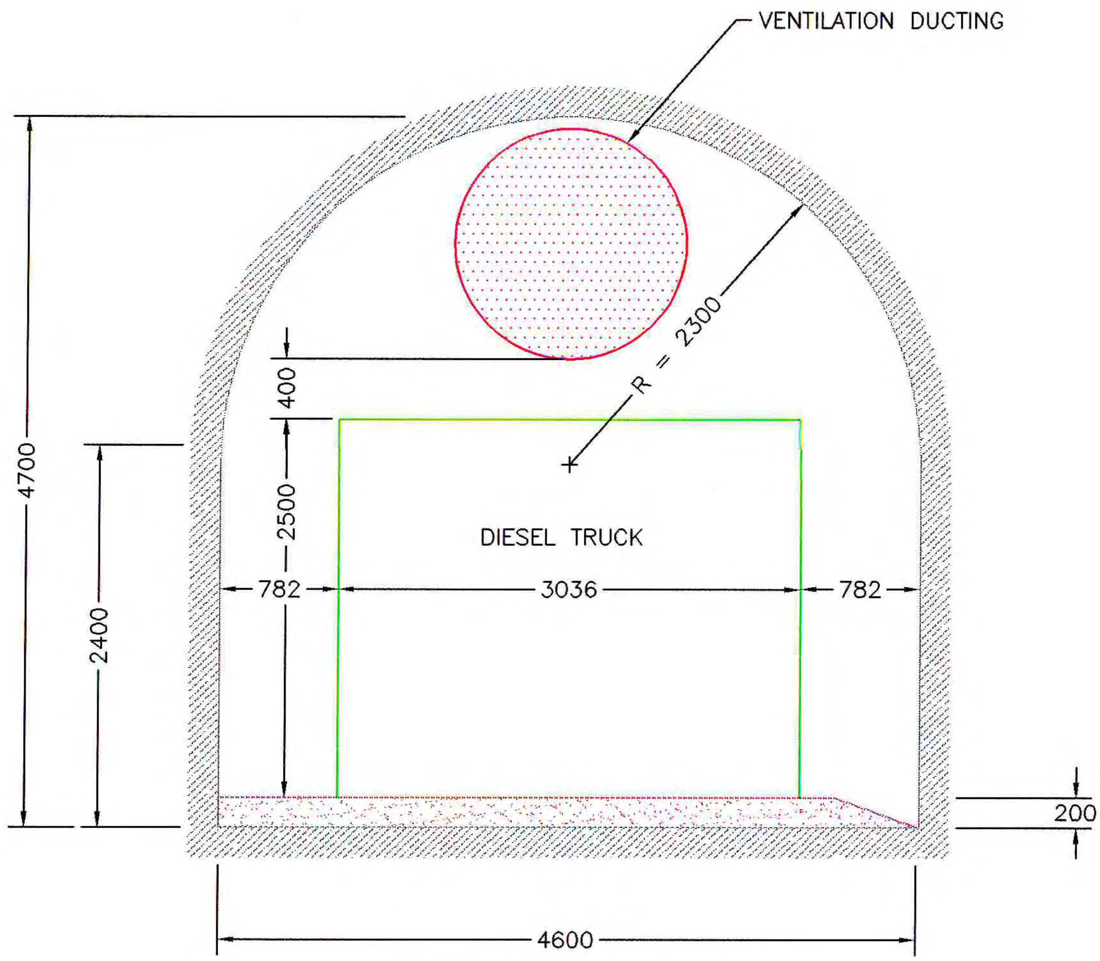

AREA 19.34 sq. m.

GRIZZLY PROJECT

UNDERGROUND EXPLORATION PROGRAM

Fig. 9.1.1–03 Single Ramp – Typical Section Using Large Diesel Truck

Anvil Range MINING CORPORATION

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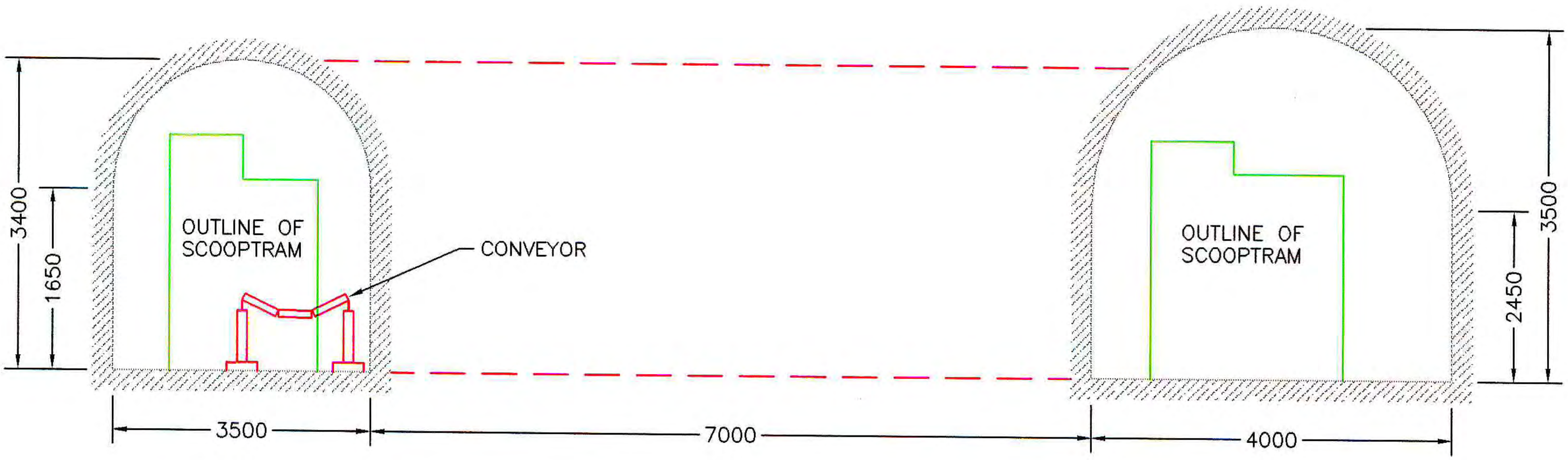

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GRIZZLY PROJECT

UNDERGROUND EXPLORATION PROGRAM

Fig. 9.1.1–04 Twin Ramp Conveyor Haulage

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Two parallel ramps of smaller size have been selected to reduce support requirements, provide better ventilation and increase safety for the workmen. Muck removal is planned to be accomplished with a belt conveyor installed in one of the two ramps. The system of twin ramps is sufficient for carrying out the underground exploration program as well as the pre-production mine development once underground exploration proves Grizzly to be a mineable ore deposit.

9.1.2 Protective Measures for Blind Creek

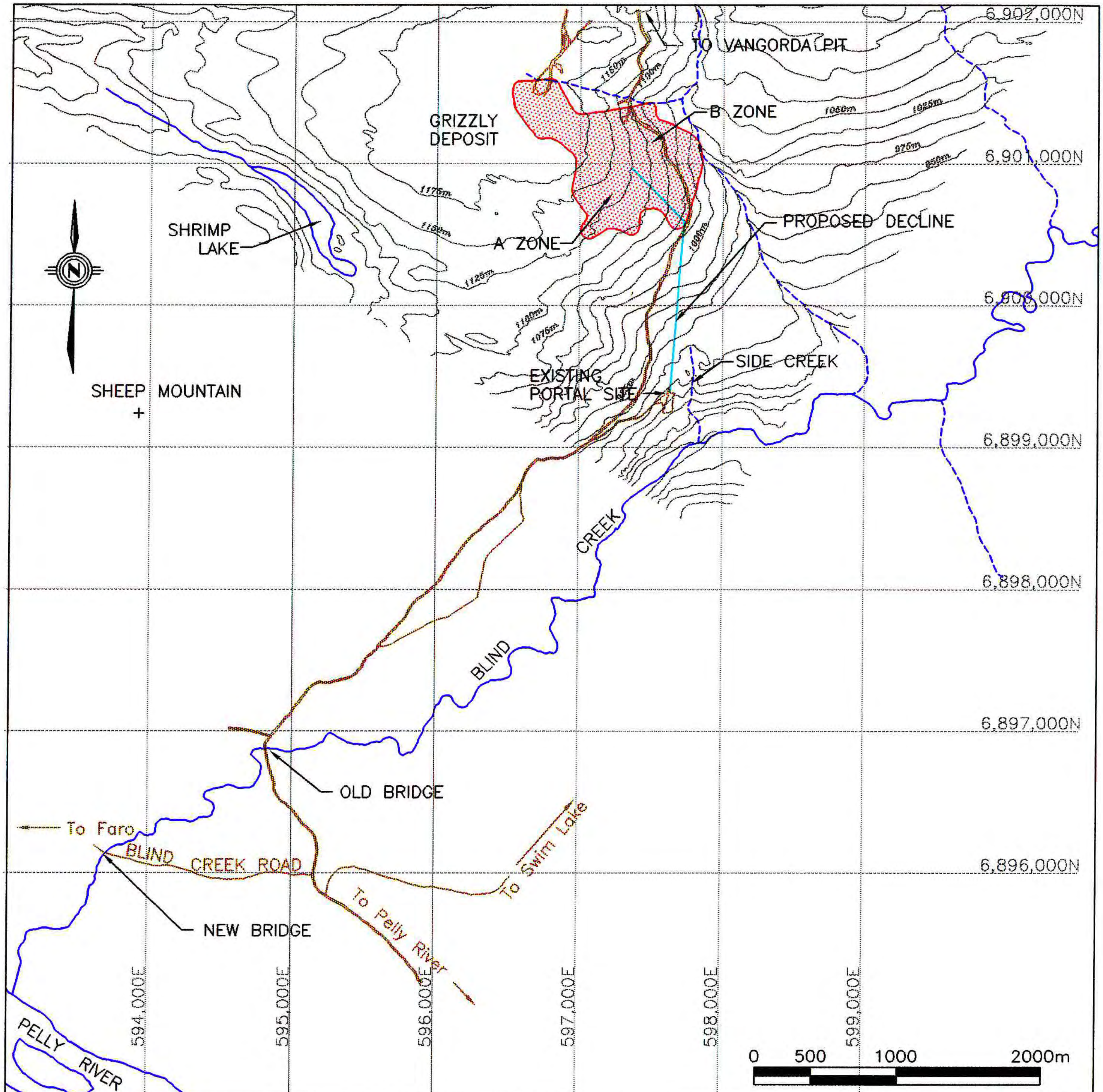
Blind Creek is a salmon spawning river and is of importance to the Kaska Dena of Ross River as a traditional fishing ground. The ramp portal is located some 110 metres above the Creek and is about 800 metres away from it. Discharge water from the ramp will possibly contain suspended solid particles, ammonia and chemicals from contact with acid generating rocks or ore.

The following measures will be taken to protect Blind Creek:

- Settling ponds will allow solids to settle out before discharge into the Side Creek
- Creation of ammonia can be reduced through proper blasting procedures and good house keeping. Once in water, ammonia will dissipate through contact with oxygen. A slow discharge that allows time for contact with air is important. For this reason water discharge is accomplished through a drill hole, and not directly through the ramp portal.
- Acid water will be treated before discharge.
- Acid generating rocks and all ore will be removed from the portal site and hauled to the Vangorda pit.

Figures 9.1.2-02 9.1.2-03 show the arrangement for pumping water through a drill hole to the treatment plant and the polishing pond on surface.

A detailed description of protective measures is available in the "Supporting Information for the Application for a Water Licence" dated October 1996, which was submitted together with the application for water use to the Yukon Territorial Water Board. The Grizzly project was explained to the band council of the Kaska Dena at Ross River in September. The application for the water licence was presented to the same group in October 1996.



GRIZZLY PROJECT

UNDERGROUND EXPLORATION PROGRAM

Fig. 9.1.2-01 Deposit Vicinity Plan

Anvil Range
MINING CORPORATION

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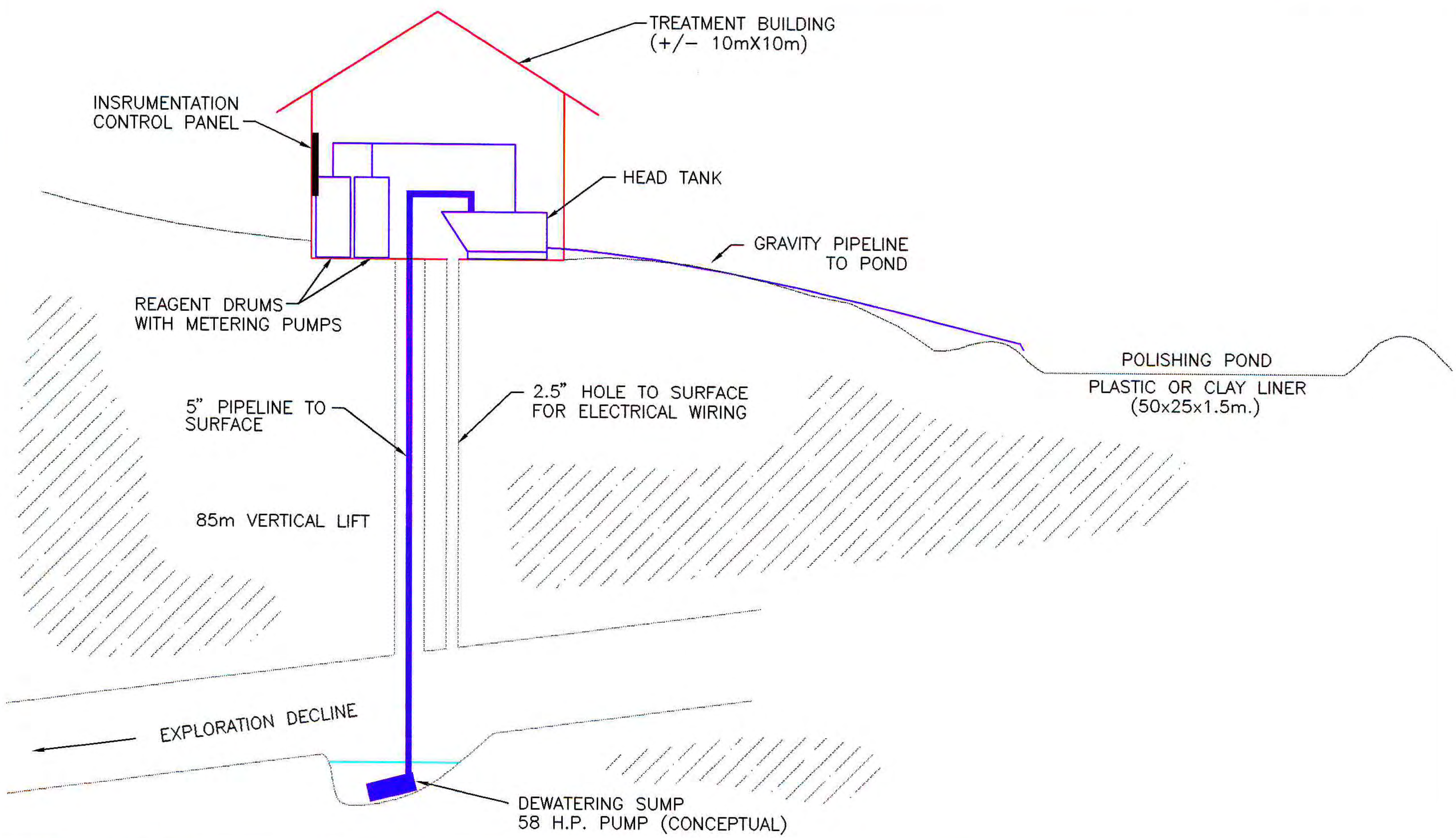
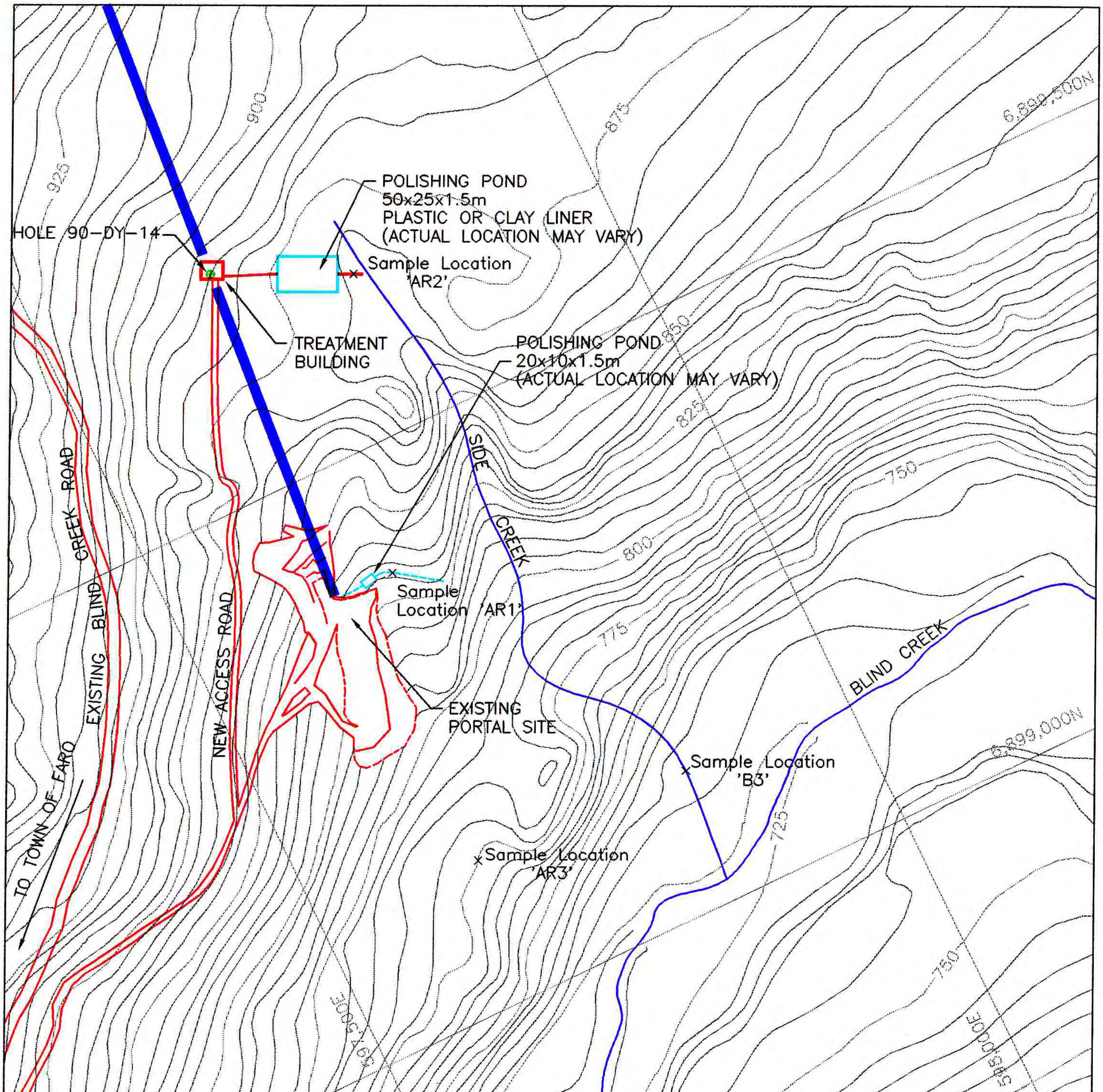


SCALE NOT TO SCALE	**GRIZZLY PROJECT** UNDERGROUND EXPLORATION PROGRAM	Anvil Range MINING CORPORATION
DRAWING: DEWATER.DWG		
APPROVED	Fig. 9.1.2–02 Conceptual Layout of Dewatering and Treatment Facilities	
DATE 96/09		



GRIZZLY PROJECT

UNDERGROUND EXPLORATION PROGRAM

Fig. 9.1.2-03 Layout of Treatment Facilities and Water Discharge

Anvil Range
MINING CORPORATION

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9.1.3 Access Options

Ten different ramp access options for underground exploration access have been considered and costed out. They are illustrated in Figures 9.1-3 to 9.1.3-10 included in Appendix 9.

- Option 01: Single ramp at -15%, exploration drilling from two hanging wall drifts. Switch back ramp extension to reach main ore horizon to obtain bulk sample for metallurgical testing. Only a small portion of the B-Zone is being covered.
- Option 02: Single ramp at -15%, exploration drilling from a hanging wall drift and a footwall drift. Switch back ramp extension to the footwall drift and for bulk sample. Better coverage, but again only a small area of the B-Zone is being drilled off.
- Option 03: Twin ramp at -18%, exploration drilling from one hanging wall drift and one footwall drift, switch back ramp extension for the footwall drift and for bulk sampling. Coverage is the same as in option 02. Better ventilation and improved safety.
- Option 04: Single ramp at -15% over the "Barren Zone" to the centre of the orebody to end up in the footwall for drilling. Too long and too expensive.
- Option 05: Twin ramps at -18%, exploration drilling from two parallel hanging wall drifts. Switch back ramp extension into the lower horizon. Drift in ore of lower horizon along the hanging wall. Good for drill information and ore characteristics, however small coverage only of one area of B-Zone.
- Option 06: Twin ramps at -18%, exploration drilling from two parallel hanging wall drifts, in-line ramp extension with continuation to the west to reach lower ore horizon for drift advance in ore along the hanging wall to obtain ore for the bulk sample. Same as option 05.
- Option 07: Twin ramps at -18%, exploration drilling in the B-Zone from one hanging wall drift, in-line ramp extension to reach the lower horizon and drive a drift in ore to the west to obtain ore for the bulk sample and also to reach the A-Zone. The west drift will reach A-Zone North for exploration drilling. A decline to the south allows exploration drilling in the A-Zone South from the exploration drift in the hanging wall. This option gives coverage of the B and the A Zones, but has constraints on ventilation and is expensive.
- Option 08: Twin ramps at -20%, lateral drift to the west in ore of upper horizon with continuation to the A-Zone for exploration drilling from the hanging wall drift in A-Zone South and in ore and the footwall in A-Zone North. A short ramp into the B-Zone provides access

for bulk sample. This option provides the best coverage, but has some ventilation constraints and is expensive.

- Option 09: Twin ramps at -18% with a turn to the northwest to reach the centre of the deposit from where a shallow ascending drift to the northeast provides access to the B-Zone, and a -18% ramp access to a hanging wall drift in the A-Zone.
- Option 10: Twin ramps at -20% with a turn to the northwest to reach the centre of the deposit. This option is similar to option number 09, provides, however, the same drilling opportunities with less drifting. This option provides best coverage of the entire orebody at the most favourable expenditure. It is further commented on in section 10, Underground Exploration. Figure 9.1.3-10 depicts this option.

Future mining activities in the "Barren Zone" could affect ramp stability. In this case, a new decline can be extended from the turning point to the north.

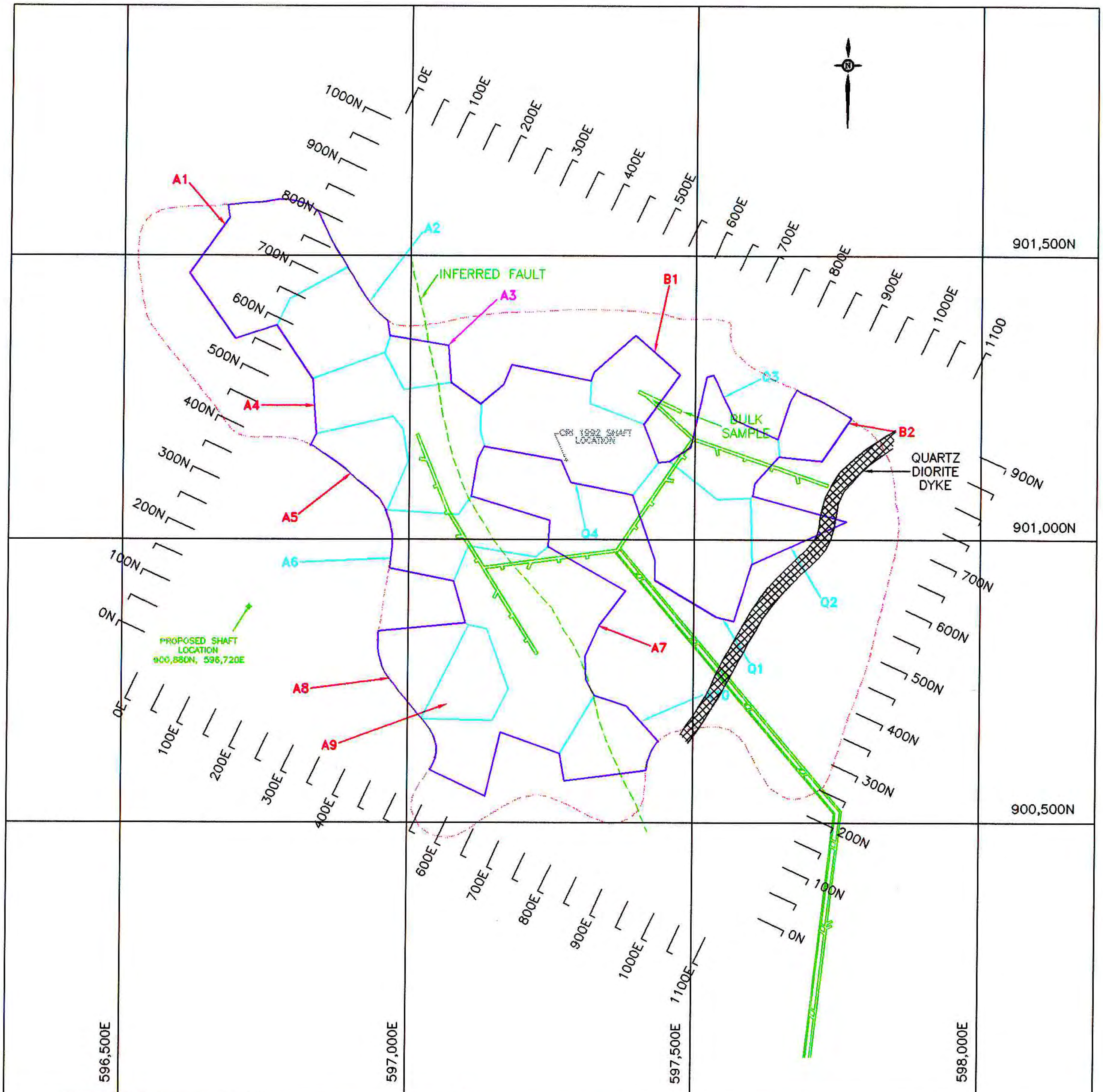
9.1.4 Preparation for Underground Exploration Access

In June/July 1996, contacts were made with seven Canadian mining contractors to establish their interest in, and make them familiar with, the proposed up-coming work on the Grizzly project. These contractors are:

- Aurora Mining and Tunneling, Ontario
- Canadian Mine Development, Ontario
- Dynatec, Ontario
- Main Street Mining Ltd., Yukon
- Procon Mining and Tunneling, British Columbia
- Redpath Ltd., Ontario
- Thyssen Mining and Contracting, Saskatchewan

All seven visited the site for familiarization.

Two (Canadian Mine Development and Thyssen) declined to prepare a bid; the other five submitted bids for ramp drivage and two for the construction of an exploration shaft.



GRIZZLY PROJECT

UNDERGROUND EXPLORATION PROGRAM

Fig. 9.1.3–10 OPTION 10

Anvil Range MINING CORPORATION

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Of the five bidders for the ramp, two (Aurora and Dynatec) submitted proposals that indicated little interest. The three remaining companies have continued interest; two (Procon and Redpath) have put an enormous amount of work into their proposals to accommodate changes made as the various ways of underground exploration changed. Both have also developed mining costs for possible contract mining.

At the same time, drilling contractors have supplied their estimates. Thus, it would be possible to use actual cost of ramp drivage and underground exploration, and be ready in short time, once a decision on the go-ahead is made.

The required application for a Water Licence, together with "Supporting Information", was prepared and submitted to the Yukon Territory Water Board.

In two separate meetings with the Band Council of the Ross River Kaska Dena, the underground exploration project was explained. Several meetings have been conducted with various government agencies to introduce details of the up-coming work.

9.1.5 Summary

Of various options for reaching the deposit, a system of twin ramps from the valley of Blind Creek has been selected as the most economic and practical way of getting underground for further exploration.

The precautions planned will ensure protection of Blind Creek.

The arrangement described as Option 10 in this Chapter and shown on Figure 9.1.3-10, gives the most extensive exploration coverage of both the B and A Zones, and will be chosen for underground access and exploration.

9.2 *Access for Production*

9.2.1 Introduction

Selection of production access will be part of the feasibility study, once underground exploration is completed and the existence of a mineable orebody has been assured.

The production opening will serve the following functions:

- bring ore to surface
- move people into and out of the mine for shift change
- ventilation (fresh air supply)
- emergency exit

- Ore removal to surface can be done by the following methods:
    - skip hoisting of crushed ore in a vertical shaft
    - inclined shaft hoisting of uncrushed ore
    - conveyor haulage of crushed ore
    - pumping of ore after crushing and grinding

- Movement of people in and out of the mine can be fastest accommodated in a vertical shaft.

- Fresh air supply and mine air heating can be best provided by a vertical, circular, naked concrete-lined shaft.

- Emergency exit is provided in the fresh air way. An emergency hoist (winch) combined with a standby diesel generator and an emergency cage form the standard equipment serving this function.

The following options for production access ought to be considered:

- Production shaft in the centre of the orebody
- Production shaft west of the orebody
- Production shaft north of the orebody
- Production shaft east of the orebody

- Production shaft south of the orebody
- Production ramp from the Vangorda pit

A shaft must be located such that it is protected from stresses induced by mining and from ground displacement from a potential subsidence or cave. A circular concrete lined shaft is better able to resist mining induced stresses than an unlined rectangular timber shaft, but the concrete shaft would not have significant resistance to ground movement unless the lining were designed to be flexible. Flexible lining (usually through the use of bitumen) is often used in coal and potash shafts in Germany, but unknown in hard rock shafts. In this case, protection against displacement is best achieved by locating the shaft outside the potential zone of subsidence.

Steeply dipping orebodies in competent ground are typical in hard rock mining. In such cases, the shaft is normally located in the footwall and the shaft safety pillar is of little or no concern. With the shaft in the footwall, subsidence has no effect on the shaft structure or on surface installations at the shaft collar. In soft rock mining (coal, potash, salt), large areas are often affected by subsidence. Design criteria for shaft pillars are well known for specific areas. In hard rock mining subsidence often creates pipes that might continue up to surface.

In the case of Grizzly, the flexibility of phyllite, the existence of major and minor geological structures, vertical and angular faults and joints, and major dykes must be considered in conjunction with the rest of the design parameters when designing the shaft pillar.

9.2.2 Production Shaft in the Centre of the Orebody

The optimum location for the production/man access shaft is in the centre of the orebody. This is to minimize the average tramming distance for ore and the time lost in travel to the workplace by operating personnel.

Part of the proposed underground exploration will evaluate the possibility of placing a production shaft in the centre of the orebody.

9.2.3 Production Shaft West of the Orebody

Without tying up ore reserves in a shaft safety pillar, a location west of the orebody outside the influence of mining is favoured for a production/personnel shaft. Further exploration from surface or underground is required to define the western boundary of the orebody. The shaft bottom would be about 50 mabove the Dixon Creek Fault.

Underground development is greater than in option 9.2.2; so are tramming distances and transportation of people to and from the workplace. Conveyor haulage of crushed ore or train haulage of uncrushed ore are possibilities for transportation of ore to the surge bin.

9.2.4 Production Shaft North of the Orebody

In this case the production/personnel shaft would be placed north of the orebody outside the influence of mining and consist of a vertical shaft and an inclined shaft at about 35 degrees to follow the footwall of the orebody. The vertical shaft would have a depth of about 550 metres and bottom out at about elevation 600 metres. The inclined shaft would be located in the centre of the orebody to follow the ore in the footwall to depth. Uncrushed ore from the various mining levels would be transferred through drop raises to a skip on rails and hoisted to the crusher and surge bin close to the shaft bottom.

Inclined shaft hoisting of uncrushed ore in a skip on rails is an old hoisting method that has been, and still is, successfully used world wide. People would be transported from the shaft station to their various workplaces by man carrier. A major fault north of the orebody must be taken into consideration when studying this option.

9.2.5 Production Shaft East of the Orebody

A production shaft east of the orebody will be considered if the present drilling campaign at Grizzly East proves positive. However, a major fault east of Grizzly is a detriment to this location. A shaft east of Grizzly could be used for hoisting of ore from both deposits.

9.2.6 Production Shaft South of the Orebody

A shaft south of the orebody is being eliminated because it would be deep and would require substantial capital up front.

9.2.7 Vangorda Production Ramp

This option requires a ramp of around 4,000 metres. Conveyor haulage provides the possibility of production increases if it is found that the orebody is capable of this. Ramp access for personnel transport is slower than shaft transport. Friction loss for ventilation is greater in the ramp than in the shaft.

9.2.8 Summary

Location and design of the production/personnel shaft will be decided after completion of underground exploration of Grizzly, and surface exploration drilling of Grizzly East, and will form part of the final feasibility study.

10. UNDERGROUND EXPLORATION

10.1 Introduction

It is estimated that approximately CAD $15,000,000 has been spent on this deposit since 1976.

This includes drilling of 86 test holes, surveying of base holes, core logging, chemical analyses, data collection, reserve calculations and studies.

At this stage, drilling from surface would contribute little or no useful information. Underground exploration will add a new dimension. It will prove the existence of a mineable orebody.

A ramp from the valley of Blind Creek has been chosen as the least expensive means of reaching the orebody and carrying out the planned underground exploration program.

Several options were considered for this program, such as fan drilling from hanging wall drifts, fan drilling from a footwall drift, and drifting in ore.

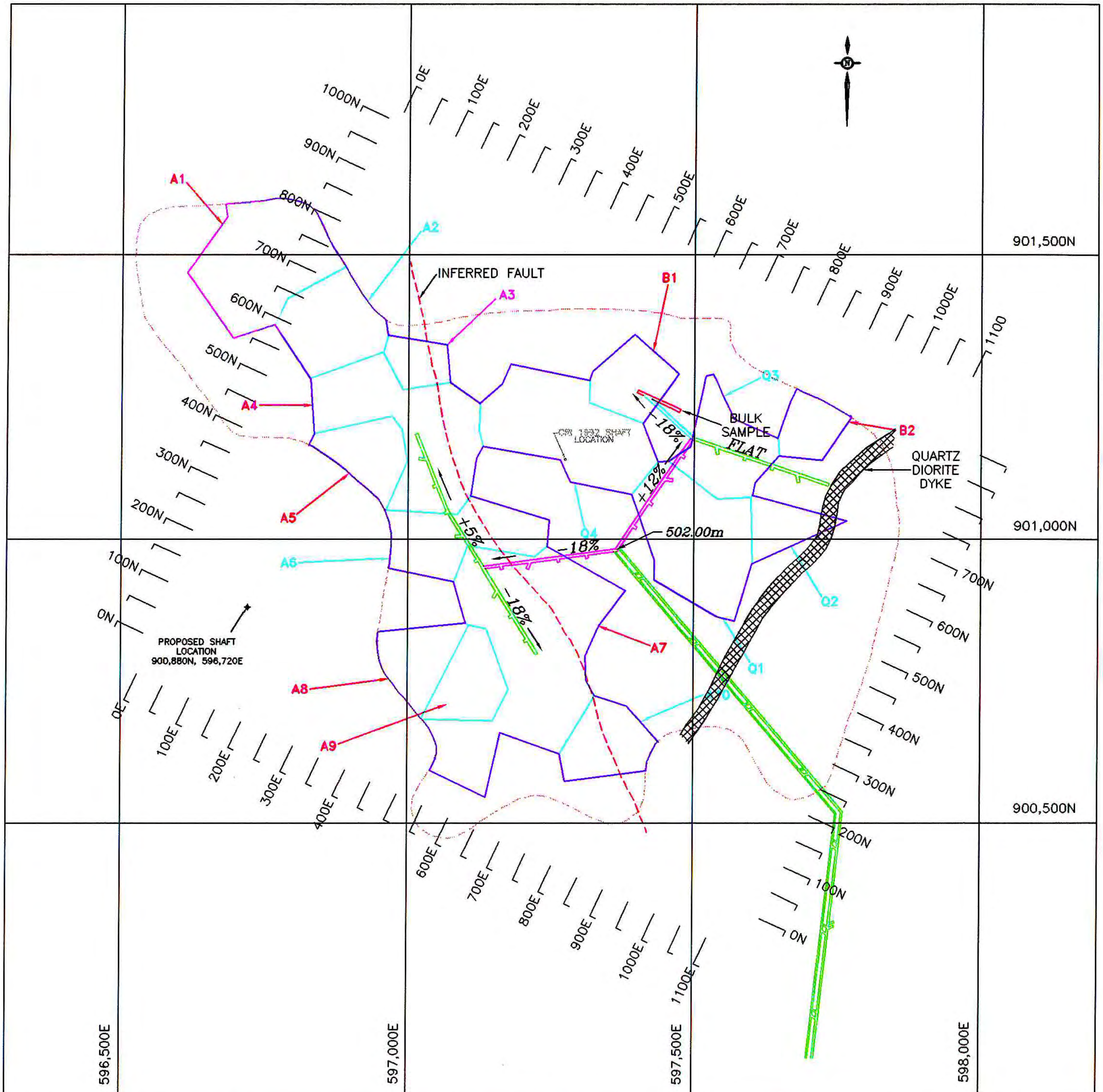
This section details the exploration plan for the B-Zone and the A-Zone respectively, and describes the access option selected (see Figure 10.1-01).

Up to four electro-hydraulic drills will be used for core drilling.

10.2 Purpose and Objective of the Underground Exploration Program

The purpose of the underground exploration program will be:

- to confirm continuity of the ore zones and enhance the certainty of reserve estimates.
- to define the structure of the orebody with greater confidence to add greater certainty to choice of mining method, mining costs and required development.
- to gain familiarity with ground conditions and support requirements.
- to obtain a bulk sample, representative of the ore, for metallurgical testing.
- to investigate the so-called "Barren Zone" to allow a decision on the location of the production/personnel shaft.



GRIZZLY PROJECT

UNDERGROUND EXPLORATION PROGRAM

Fig. 10.1-01 Underground Exploration of A-Zone and B-Zone

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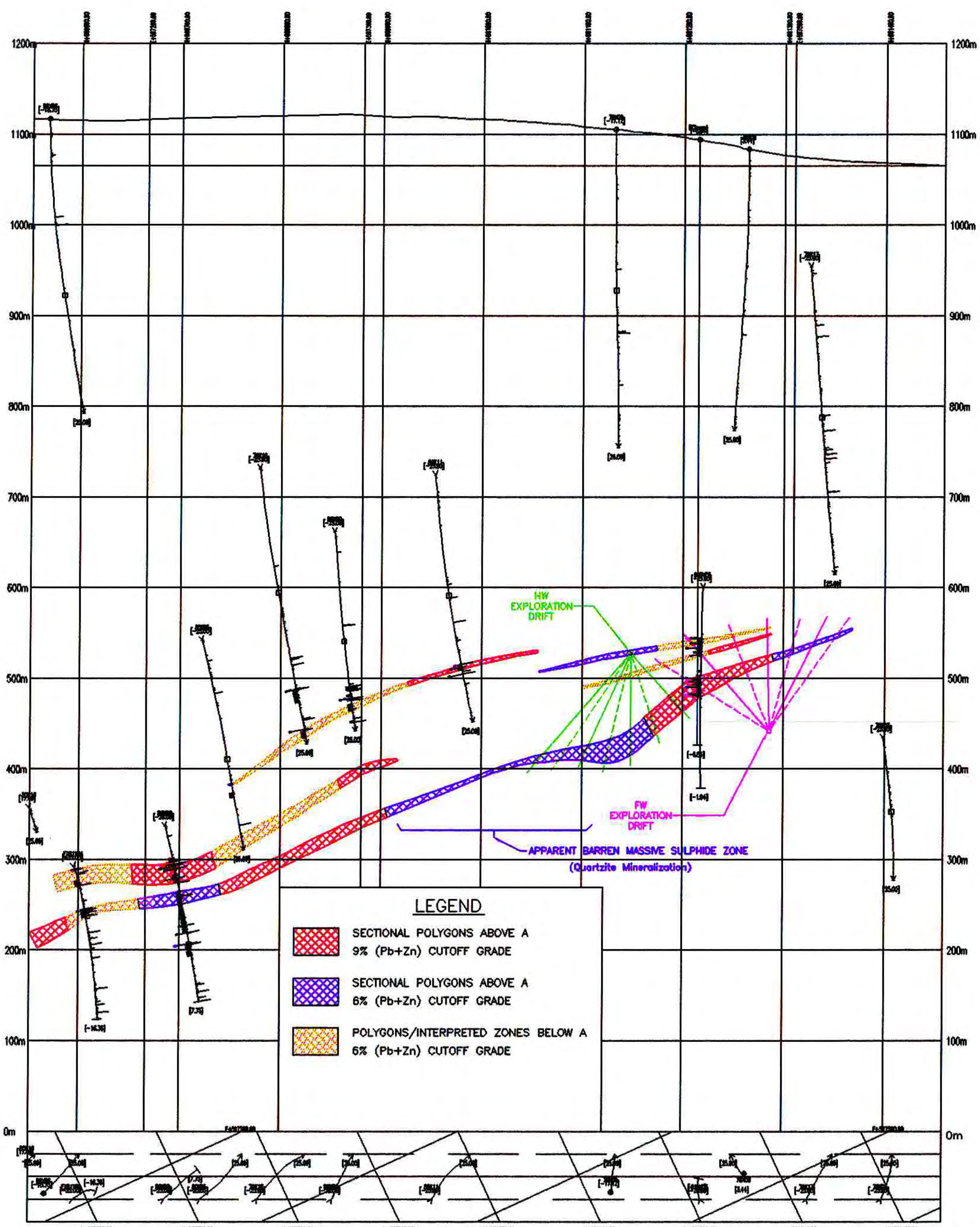

GRIZZLY PROJECT

UNDERGROUND EXPLORATION PROGRAM

Fig. 10–01 Underground Exploration Drilling – Cross Section 600E

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MINING CORPORATION

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- to collect data concerning geotechnical conditions.
- to collect data about hydrology

The objective of this program is to prove, through underground exploration at the lowest cost, the existence of a mineable orebody.

10.3 Access

Ten access options are described in section 9.1 "Access for Underground Exploration". The most favourable site for the ramp portal is located at an elevation of 835 m in the Blind Creek valley; about 800 m from, and about 110 m above, the river.

Twin ramps are designed to reach the starting point for underground exploration. With a gradient of -20% and a length of 1,737 m, they will reach a point above the so-called "Barren Zone" at an elevation of 502 m, from where exploration drifting to the B-Zone and to the A-Zone will begin.

These access drifts will provide an excellent section through the central zone and allow evaluation of the large scale structure of the deposit and the possibility that steeply dipping fold closures of high grade massive sulphides have been missed by vertical surface drilling. It will also show whether or not a low-grade zone exists that would allow for a shaft to be located in the centre of the orebody.

10.4 Exploration of B-Zone

At the end of the ramp the B-Zone access drift is driven upwards at a gradient of +12%, and a total length of 237 m to the northeast, to reach elevation 530 m.

From this the drill drift East is driven flat for a distance of 250 m to stop before the quartz-diorite dyke is reached. Drilling from this drift will give good coverage of the B-Zone and will allow drilling through the dyke to define the East Fault to take place.

At elevation 530 m, a drift of -18% will start from the B-Zone access drift to enter, over a switch-back, the main Lower-G ore horizon. A drift in ore (about 200 m), along the hanging wall will provide ore for a bulk sample and allow testing of geotechnical and mining conditions.

10.5 Exploration of A-Zone

At the end of the ramp the A-Zone access drift starts to the west downwards at -18% into the hanging wall for 238 m. At this point, a drill drift continues to the southeast downwards at -18% for another 183 m. This allows detailed fan drilling of block A-7, the longest block of the A-Zone, to take place.

A second drill drift extends to the northwest upwards at +5% for 267 m. Fan drilling from the hanging wall will define this portion of the A-Zone.

10.6 Summary

The underground exploration program, as outlined above, gives the best coverage of the entire orebody and explores, in detail, through fan drilling from the hanging wall, the high grade and thick ores of the B-Zone and the A-Zone that are first scheduled for mining.

At the same time, it will give a good picture of the central quartzitic zone ("Barren Zone"), the diorite dyke, the East Fault, the NW-SE inferred fault in the centre of the orebody, as well as ore for a metallurgical analysis , and mine openings for geotechnical test work. At the same time, hydrological information will be collected.

11. ENVIRONMENTAL CONSIDERATIONS

The Grizzly deposit is located in the drainage basin of Blind Creek, southeast of the Vangorda Plateau. Currently there has been no mining activity in the Blind Creek basin other than minor non-intensive exploration activity. Blind Creek is a Chinook salmon spawning stream and appears to support one of the larger spawning populations in the Pelly river system. Salmon spawning streams are considered the most sensitive stream types in the Yukon and it has been the practice of the Department of Fisheries and Oceans to demand no impact, and in the case of placer mining, essentially zero discharge to spawning streams. Given this situation the development of the Grizzly deposit poses certain environmental challenges.

Other environmental issues exist in the area such as potential wildlife impacts, competing land uses, and potential impacts first nation enjoyment of traditional lands. These issues must be dealt with but are relatively minor compared to the fisheries related matters noted above. As with all environmental issues, perception commonly seems more important than reality, thus, it will be important for Anvil Range to manage public, first nation and government liaison over the Grizzly exploration and development with considerable care.

Fortunately the deposit has several advantages that place it favorably to deal with the above issues. Most significantly, it is sufficiently deep that it will be possible to develop the mine in a way that will have no significant impact on Blind Creek either during production or after closure. To achieve this it will be necessary to incorporate environmental planning into all stages of the mine development process from the earliest stages of exploration. It will also be necessary for the environmental plan to be followed at all times, particularly during the exploration phase when a mishap could have serious implications for further development simply because of perceptions of the regulators and the public.

As discussed in Section 9, access for the exploration phase is most efficiently gained by decline from a portal site near the bottom of the Blind Creek valley. To minimize the length of the decline a portal site has been selected that is relatively close to Blind Creek. The proposed portal site is at an elevation of 835 m., 115 m. vertically above the creek and approximately 800 m. horizontally from it. This site offers significant advantages over those further up hill and further from the creek but also increases the risk to the creek and to Anvil from a mishap at the portal.

It is anticipated that up to 300 USGPM of water may have to be pumped from the decline on a steady basis, the flow will increase as the decline lengthens. Water from the upper portion of the decline will contain suspended solids and ammonia but no significant dissolved metals. Waters from the deeper workings will, at some time, contain dissolved metals requiring chemical treatment. Waste rock from the upper decline will be benign calcareous phyllite, the deeper workings will, however, encounter acid generating rocks in places.

All of the above not withstanding it is believed, with considerable certainty, that the exploration program can be completed as planned without significantly affecting the creek or its fish. The environmental plan developed for the exploration phase incorporates the following features:

- Minimizing the footprint of operations.
- Drainage control and control of suspended solids through sedimentation ponds.
- Monitoring and, if necessary, chemical treatment of mine water to control dissolved metals.
- Source control of ammonia in mine water through good housekeeping practices.
- Location of water management facilities in a flat area further from Blind Creek than the portal site and draining indirectly to the creek via a small side creek.
- Removal of acid generating waste rock from the Blind Creek drainage basin.
- Use of trucks for ore and waste haulage that allow use of existing roads and require minimal upgrading of roads.
- Location of surface facilities in Faro rather than the portal site where possible.
- Establishment of a rigorous monitoring schedule to track the success of the above measures.

Application for a Type "B" Water License for the exploration program was filed on October 22, 1996. It is anticipated that the licensing process will take from 3 to 6 months assuming that a public hearing is not required. Under current legislation it would be possible to begin the decline without a water license however as soon as water discharge exceeds 300 cu. m. per day a license would be required. Amendments to the mining legislation currently in the Senate may be in effect by early 1997, in which case a land use approval may also be required.

Given the environmental sensitivity of the area, mining of the deposit will probably not be possible from the portal used for the exploration program. This is partly because the site is small and will not easily accommodate the increased scale required for efficient operations. The increase in scale would be so large that some impact on the nearby creek is almost inevitable; this would be particularly true for ore haulage. Another matter that is expected to cause difficulty in obtaining approvals for production would be the discharge of mine water from the exploration portal. It appears, based on statements and correspondence of federal regulators, that obtaining permits for production from this site would be very difficult or that onerous conditions could be attached and long delays expected.

The current plan for production is to hoist ore in a shaft located near the divide between the Blind Creek and Vangorda Creek drainage basins. Production from a shaft location further from the creek will not pose the same risks to the creek and will neutralize the concerns noted above. Waste water would be pumped up this shaft, treated and discharged into the Vangorda Creek basin. Acid generating waste hoisted to the surface would be minimal but the commitment would be made to remove the material to the Vangorda or Grum site for storage with other acid generating wastes unless used for backfill at Grizzly.

Closure of the site should pose no major issues since the deposit is deep and will flood once dewatering ceases. There will be a requirement to block all openings and to reslope and revegetate disturbed areas however these will not be extensive. A bulkhead may be required to limit water flow from the deeper parts of the mine to surface via the decline. There is little reason to expect significant ongoing closure costs such as maintenance or water treatment.

In summary, while the proposed Grizzly Mine is located in a sensitive area and posed several permitting and operational challenges, there is little reason to believe that mine development can not proceed at reasonable cost with little environmental impact.

12. MINING METHODS

12.1 Introduction

Two options, Case A and Case B, have been studied. Details of the two cases are described in Section 3.4, Design and Operating Parameters, and in the respective appendices.

Case A assumes a production rate of 1.2 million tonnes per year, the production shaft west of the orebody, the main infrastructure of the mine in waste rock of the footwall, and conveyor haulage.

Case B assumes a production rate of 1.5 million tonnes per year, the production shaft in the centre of the orebody, the main infrastructure of the mine in ore, and haulage of ore with diesel trucks.

The reserves of 17,240 million tonnes, as currently identified, provide a mine life of 14.4 years at 1.2 million tonnes per year and 11.5 years at 1.5 million tonnes per year. With strong expectations of finding more ore, mine life will increase. Shaft capacity for production increase is available.

12.2 Knowledge Base

A great number of reports have been produced over the last twenty years since the first holes were drilled. Most holes were drilled between 1977 and 1980, with some additions up to 1991.

Although 86 holes have been drilled, only 56 report intercepts in which mineralization is present. The distance between holes is up to 200 metres.

The geology has been described in the chapters dealing with Ore Reserves and Orebody.

N.D. Rose prepared a *Mineable Reserve Estimate and Underground Mine Plan* in October 1992. Information from this report has been incorporated here. While Rose considered mainly massive sulphides, this study includes quartzites where they are of economic grade, here taken at 9% combined lead and zinc cutoff, on an undiluted basis.

12.3 Geotechnical Considerations

The following features are important for the development of a suitable mining method:

- Folding has affected both strike and dip.
- The strike can change through ninety degrees.
- The dip varies from 15 to 30 degrees (actually it is near 40 in places).
- Vertical or sub-vertical faults are expected, but are not apparent in vertical drillholes.
- Horizontal stress could be as high as twice the vertical stress.
- The foliation at Grizzly is extremely weak, and separates with only minor displacement.

Major constraints for a mining method at Grizzly are:

- Irregular dip
- Variable dip from 15 to 30 degree (neither flat nor steep enough for many methods)
- High ore strength and weak hanging wall
- Ore thickness of 3 to 30 metres
- Possible tectonic stresses

For the purposes of this study, the method of mining will be determined by mining height:

1. Under 6.5 m:

   Room and pillar mining without backfill in three steps:

   - first pass mining of two rooms 5 m wide leaving a 20 m wide pillar
   - second pass mining of a 5 m wide room through the centre of the pillar
   - pillar robbing

   Recovery is expected to be 70%.

2. Mining of elongated rooms with backfill:

- development of top cut close to the hanging wall and back support
- vertical or horizontal benching
- mucking (remote scoop trams)
- cemented backfill (for the stopes)
- pillar extraction with backfill without cement
- expected extraction rate is 85%

12.4 Stoping Options

12.4.1 Mining Conditions

Mining conditions will vary considerably:

- thickness variation from 3 m to 20 m or more in less than 100 metres;
- one possible major fault with a displacement of from 20 to 50 metres;
- changeable ore type - massive sulphides which are easy to recognise and can be used as a mining horizon, and quartzites which appear to the eye the same as some non-mineral-bearing rocks;
- experience at Faro Underground led to the decision to leave about one metre of massive sulphide ore in the back to form a competent roof. It was considered too time-consuming and expensive to adequately support the hanging wall rocks if the top layer of ore was extracted;
- gradients of from 15 to 40 degrees in local places.

12.4.2 Stopes in Ore of Under 6.5 m

Figure 12.4.01 shows the configuration of a stope in an area where the bed is dipping at about 30 degrees. This gradient requires that the stope (in plan) be at an acute angle to the main conveyor drift in Case A, if both are to follow the maximum recommended gradient of 18%.

Rooms are planned to be five metres wide. This is to allow good control of the back and to reduce the amount of footwall dilution which would be required if a wider near-horizontal cross pitch was adopted in the rooms. These rooms are planned to be on 25-metre centres advancing on strike ± 18%. Thus, the pillars between them will be 20 metres wide, sufficient to support the vertical stress. With angled crosscuts spaced on 50 metre centres, the percentage extraction during first pass mining would be under 30%, well within the geotechnical guidelines. During pillar splitting, the long pillars can be split into two pillars 7.5 m wide, and further carved into smaller pillars 7.5 x 5 m. The plan view will be a parallelogram. This works out to a 70% recovery at a safety factor of 1.02 to 1.20 (depending on depth), acceptable during pillar robbing over a short time span. The length of the stope would ordinarily be limited to 80 m.

Pillar robbing will be in retreat. It will probably take the form of splitting the pillar along its major axis, leaving a 7.5m pillar on each side, or it can take the form of slashing or slicing the ends of the pillars parallel to the crosscuts. The method used will depend on local conditions. Experience will be gained regarding how much ore can be extracted from the pillars. In particular, a balance will have to be established as to what size, shape and quantity of post pillars will be left to support the back and prevent early caving.

It is anticipated that three stopes will be advanced at one time, over a front of about seventy five metres. With crosscuts for ventilation and equipment movement, four or five faces will be available for drilling, bolting, blasting and loading operations.

12.4.3 Stope Performance Under 6.5 Metres Thick

It is planned to be able to drill one face, bolt another, blast the third and load out the fourth per shift. The thickness to be mined lies between 3.5 and 6.5 metres - use 5.0, and a round of 3.3 metres.

Hence tonnage per round = 3.3 x 5 x 5 x 3.92 = 323 tonnes.
Mining of three faces per day provides 970 tonnes.
Allow for roof problems, breakdowns, section moves, etc. - deduct 20%.
Hence the average daily production should be in the range of 776 to 970 tonnes..

12.4.4 Stopes in Ore Over 6.5 Metres

Figure 12.4.02 shows a typical stope in a bed dipping at about 30 degrees. Cross pitch stopes will mostly be driven up at a maximum of 18% but can also be driven downwards.

The objective is to form a stope 8 metres wide on the primary extraction, leaving a solid pillar also 8 metres wide.

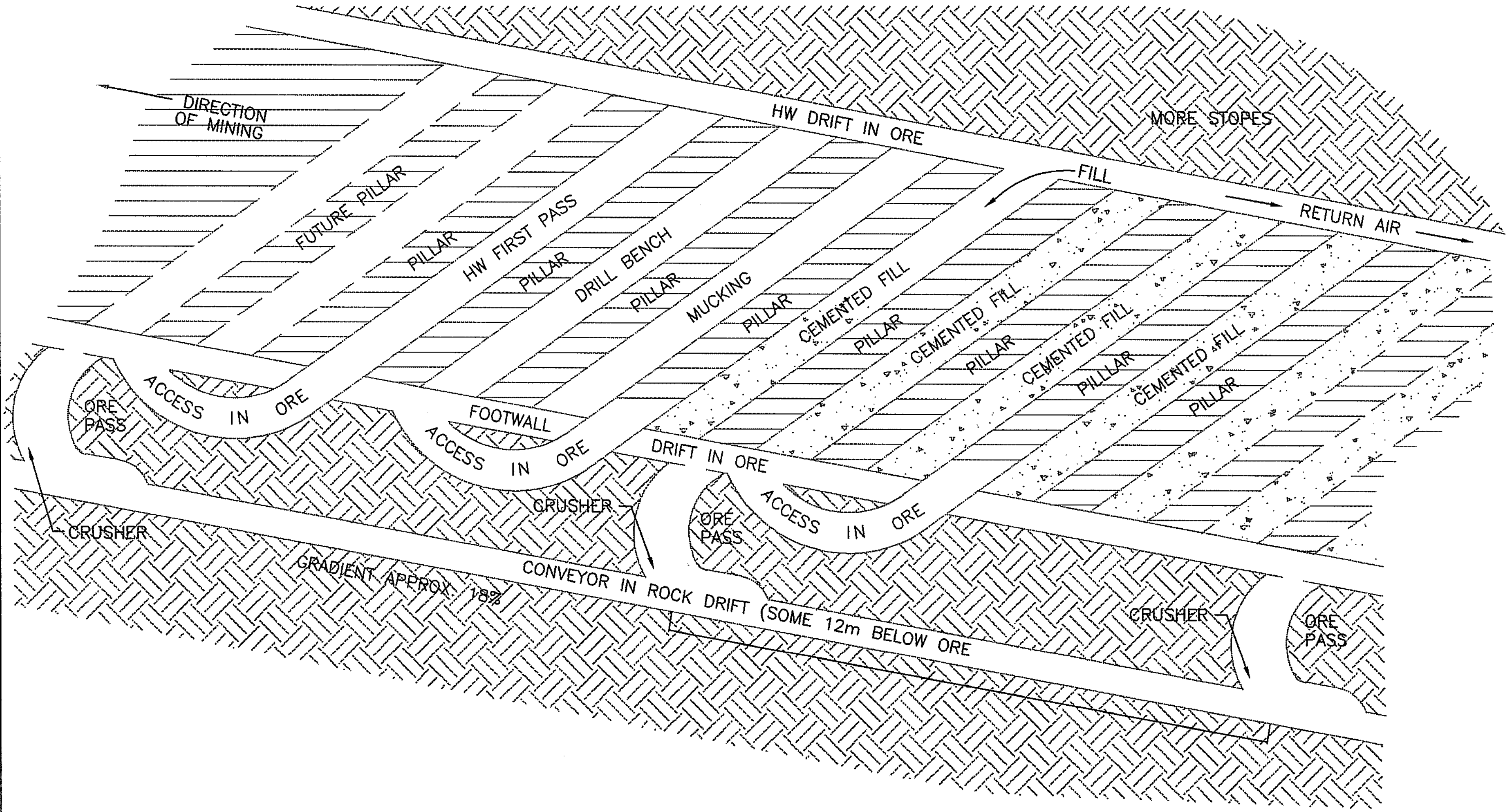
The first operation in the stope is to drive near the hanging wall. Eight foot long rebar bolts and resin are expected to be most suitable for back support. Additional use of straps and/or mesh is anticipated.

The next operation is to drill down holes and blasting. Loading of the blasted ore will be done with remotely operated scoop trams.

The third operation is to stow the empty stope with cemented backfill. Fill will normally be brought by truck at the hanging wall access and dumped into the stope. Final cramming against the back will be done by use of a smaller scoop fitted with a 'snout' which fits into the bucket. Tight packing is very important. Cement slurry is added at the stope. After the stope has been completely backfilled, the next stope can be mined.

In Case A, the development of the stoping area will be done by footwall drift vertically above a conveyor drift located from 10 to 20 metres below the footwall, the actual distance depending on the variation in gradient along the line of the conveyor. At the other end of the stope, a hanging wall drift is required for driving the first lift against the back, an access for the fill trucks, and a return airway. To access the stope to drive the hanging wall first pass, a short ramp, all in ore, will be driven from the footwall main drift up to the entrance to each stope, see inset on Figure 12.4.02

In Case B, the main development will be on footwall, with alternate drifts placed on the hanging wall.



SCALE NOT TO SCALE	**GRIZZLY PROJECT** UNDERGROUND EXPLORATION PROGRAM	Anvil Range MINING CORPORATION
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APPROVED	Fig. 12.4.2 CASE A Stoping Area In Thick Ore	
DATE 96/11		

Pillars left between the backfilled stopes will be extracted by retreat mining. As the orebody becomes better known through underground exploration, it will be possible to allow for bulk mining methods.

12.4.5 Stope Performance Over 6.5 Metres Thick

The three stages of ore extraction are:

1. Driving in ore, 4.5 metres high, 8 metres wide, if possible, an average of 2 rounds per day = 930 tonnes. At 80% efficiency, the tonnage is 745 tonnes per day.

2. Benching to keep pace with HW drivage of 6.6 metres per day. The average thickness of thick ore is about 12 metres, which provides an average bench height of 6.5 metres.

   Ore produced = (6.5) 8 x 6.6 x 3.92 = 1345 t.
   Applying the 80% factor - tonnage/day = 1076 t.

3. Backfill to keep pace must replace 6.6 m x 8 x 11 per day at 60% of the ore tonnage (see calculation in Section 15.3.2) = 1366 tonnes of fill.
   Use 80% = 1093 tonnes/day.

On average, the total ore extracted is from 2276 to 1821 tonnes per day per stoping unit.

12.5 Backfilling Operations

12.5.1 Introduction

Backfilling of high stopes and good grades is required to optimize recovery. The extracted ore is replaced by cemented fill. Backfill is packed to the back as tightly as possible. Pillar extraction is by second pass mining with uncemented backfill.

12.5.2 General Arrangement

Figure 12.5.01 shows the diagrammatic arrangement of the proposed system.

Backfill material is brought underground through an 8 inch drop line from surface to be discharged into an underground silo.

A mixture of classified tailings(to reduce cement usage) and gravel screened to a top size of 40 mm is anticipated for backfill.

A 4 percent admixture of cement will provide a sufficiently strong fill. Testing will be required to confirm these initial assessments.

The cement will be delivered to a silo on surface and delivered as slurry in a 2-inch pipe, to be added to the backfill as it is dumped into the stope.

The proportion of different types of material as planned is:

gravel 3 - 40 mm	85%
classified tailings or sand 0 - 4 mm	11%
cement	4%
water cement ratio	1:1

A retardent might be considered.

12.5.3 Placement

The placement of the fill has been envisaged done by 26 tonne trucks. Tamping to the roof would be done by small LHD fitted with a rammer. Further investigation may prove that at least the top 2 metres will have to be placed by pneumatic or mechanical slinger, the latter placed in the stope or fixed to the rear of a truck. Figure 12.5.02 shows these alternatives.

Rates of placement required could be up to 2500 tonnes per day, with the average at 1496 over the long term, 50% cemented for primary stopes and the remaining 746 tonnes per day of uncemented fill for secondary stopes.

Considerably more test work is necessary in the next phase of investigation leading up to the final feasibility study.

12.6 Mining Equipment

12.6.1 Introduction

Equipment specifications and prices were obtained from suppliers. At this Pre-Feasibility level, no spreadsheets were set up, and an average price, if more than one supplier offered similar machines, was used.

12.6.2 Major Items of Equipment

Drills

A twin boom electrohydraulically powered jumbo is selected for drifting in development, in thin bed production and for the first lift of the thick ore stopes.

Bench Drill

An electrohydraulic single boom unit is proposed for drilling 3 or 3.5 inch (76-89 mm) diameter holes for a length of 30 metres. It should be maneuverable and accurate.

Roof Bolter

A single boom electrohydraulic unit is proposed,

Loader

Well proven, large machines are on the market. A selection can be made during the final feasibility study.

Conveyors are only used in Case A.

The planned system of ore haulage is by 42 inch conveyors. They will be placed in drifts from 10 to 20 metres below the footwall and accessed by small ore passes at 120 metre intervals. The conveyor belting would be 5/8 inch thick to withstand the hard lumps of high specific gravity ore.

Three methods of size reduction have been examined: jaw crushers, feeder breakers and grizzly with rock breakers. Standard conveyor drive units of 150 KW are planned with a double drive unit for the long conveyor. Structures would be floor mounted because of the necessity to have headroom for the crusher under the orepass at 120 metre intervals.

12.6.3 Ancillary Equipment

- Mancarrier, capable of holding 8 men (2 face crews), four-wheel drive, diesel powered
- Anfo loader
- Scissor lift
- Lubrication and service truck
- Supervisors' vehicles: small 30 HP units with air-cooled diesel engines

12.7 Ventilation

12.7.1 Aspects of Ventilation for Case A

The major requirement is to provide sufficient air for the mobile diesel vehicles planned for underground operation. With the legal requirement of 75 cfm per installed BHP (brake horsepower), this component far exceeds any other requirement.

Case A assumes conveyor haulage. Still fresh air requirement is 400,000 cfm.

For reduction of expenditures during the exploration phase, a twin ramp has been selected, each opening being small enough to reduce roof support costs, and to provide an intake and return passage for air during ramp drivage and underground exploration drilling. In addition, the conveyor in one of the twin ramps will assist in the drivage of development drifts in waste while shaft sinking is in progress.

The differences from Case A which affect the ventilation requirements are: shaft location in the centre and use of large trucks instead of conveyors to transport ore.

The main production shaft, 18 feet (5.50 metres) diameter, is planned for downcast. The main fan and air heater will be located at the shaft collar.

Underground, the air splits into two major airways towards the North East. Each drift is planned to be 4.5 metres wide and 5 metres high. The resistance is therefore not high. As the mine expands, the number of splits will increase.

Rather than increase the size of the twin ramps for ventilation reasons, it is proposed to construct two ventilation raises as part of production development. This will reduce overall air resistance as shown in the ventilation calculations in Appendices 12.7-01 and 12.7-02. The final feasibility study will deal with this issue. Fan capacity for this arrangement will be 1,700 hp.

12.7.2 Aspects of Ventilation for Case B

The production shaft for Case B (18 foot diameter) is assumed to be located in the centre of the so-called "Barren Zone". It will be 690 metres deep to the main shaft station, compared to 778 metres in Case A, eighty eight metres less.

Three 40 tonne diesel trucks will be required for ore transport. Each truck has a 450 BHP engine requiring 33,750 cfm of fresh air. With other equipment being similar to Case A, a total of 500,000 cfm is required.

Two main return airways will be used: the twin ramps moving, on average, about one third of the total and two 480 metre vent/raises of 3 metres diameter located at the Eastern end of the orebody, each passing one third. By using parallel airways above the orebeds in most areas, the return air can be brought from each or all of the three production areas (NE, NW or SW) to the two return air raises. One raise is scheduled to be constructed in year 2 and the second early in year 3. Two raises will require 2,300 HP fans. Three similar raises would require 1700 HP.

12.7.3 Auxiliary Ventilation (Cases A and B)

It is proposed to use 60 KW fans to ventilate development drifts and 30 KW units for stope ventilation. Both will use standard 36" forced air vent tube suspended from bolts in the back. They will be mounted on skids for easy towing to new locations.

Table 12.A7.1

Ventilation Calculations
Scheme #1: Circulation through shaft, drifts, stopes and decline

Fresh Air Quantity Requirements

Units	Quantity	CFM/Unit	Total CFM		Hp	
Men	100	50	5000		0.67	
Scoops	5	22500	112500		1500	
Trucks 36 T	2	28125	56250		750	
Jeep	10	2250	22500		300	
ANFO Charger	1	7500	7500		100	
Explosives Carrier	1	7500	7500		100	
Utility Vehicles	2	7500	15000		200	
Future Unspecified			100000			
Total			**326250**		**2950.67**	
Say			**400000**	**or**	**188.7772**	**m^3/sec**
Pressure Losses		**hp**	**Pa**	**PL (m)**	**k (factor)**	**A (m^2)**
In 18'D shaft:		183.77	720.37	13354.57	0.02	23.64
Shaft to Decline Drift #1		278.87	1093.17	26160.00	0.03	18.56
Shaft to Decline Drift #2		278.87	1093.17	26160.00	0.03	18.56
Split flow		557.75	2186.34			37.13
Stope Development Headings		16.64	65.23	6240.00	0.04	40.00
Stopes		11.52	45.16	8640.00	0.04	80.00
Waste Development Headings		12.38	48.54	3700.00	0.04	21.25
Raises		5.69	22.30	141.30	0.04	1.77
Split flow		46.23	181.23			143.02
Decline Conveyor way 3.0x3.5		1308.16	5127.93	19541.50	0.03	9.18
Decline Drift 3.4x4.0		1150.62	4510.40	22236.00	0.03	11.88
Split flow		2548.78	9638.33			21.06
Total		**3246.53**	**12726.28**			

Table 12.A7.2

Air Calculations Decline(s) and its effect on the whole mine

Scheme	Section		Area	Air Speed		Air HP	Air HP	Cost Per Year
	Con. A	Access	m^2	m/sec	ft/min	combined drifts	whole mine	$000 @ 7.4¢/kwh
1	2.5 x 3.0 3.0 x 3.5	3.4 x 3.5 3.4 x 4.0	7.50 + 11.90 = 19.40 9.18 + 11.88 = 21.06	9.68 8.96	1906 1764	3535 2458	4323 3246	1777 1334
2	3.0 x 3.5	4.25 x 4.5	9.18 + 18.07 = 27.26	6.92	1362	1491	2279	937
3	4.25 x 5.0	3.4 x 4.0	18.56 + 11.88 = 30.44	6.20	1220	991	1779	731
4	4.25 x 5.0	4.25 x 5	18.56 + 18.56 = 37.13	5.08	1000	592	1380	567
5	4.0 x 4.0	4.0 x 4.0	14.28 + 14.26 = 28.56	6.61	1301	1401	2189	900

12.8 Electrical System

The power supply for Grizzly will come from the substation at Grum over a new overland transmission line at 13.8 kilovolts into an incoming transformer, capacity 6 MVA (Mega Volt Ampere).

A distribution centre at 6.9 KV will distribute power to the main hoist 2000 HP (1500 KW), main fan, 1700 HP (1200 KW), underground through a shaft cable, and to a second transformer 6.9 KV to 600 volts for ancillary surface compressor, ore bin, lighting, propane heater, etc. Underground power will pass the main underground substation with transformer 6.9 KV to 600 volts, capacity 1.5 MVA. It will be used for the shops, pumps, skip loading arrangements, conveyor (in Case A) and ore storage bin.

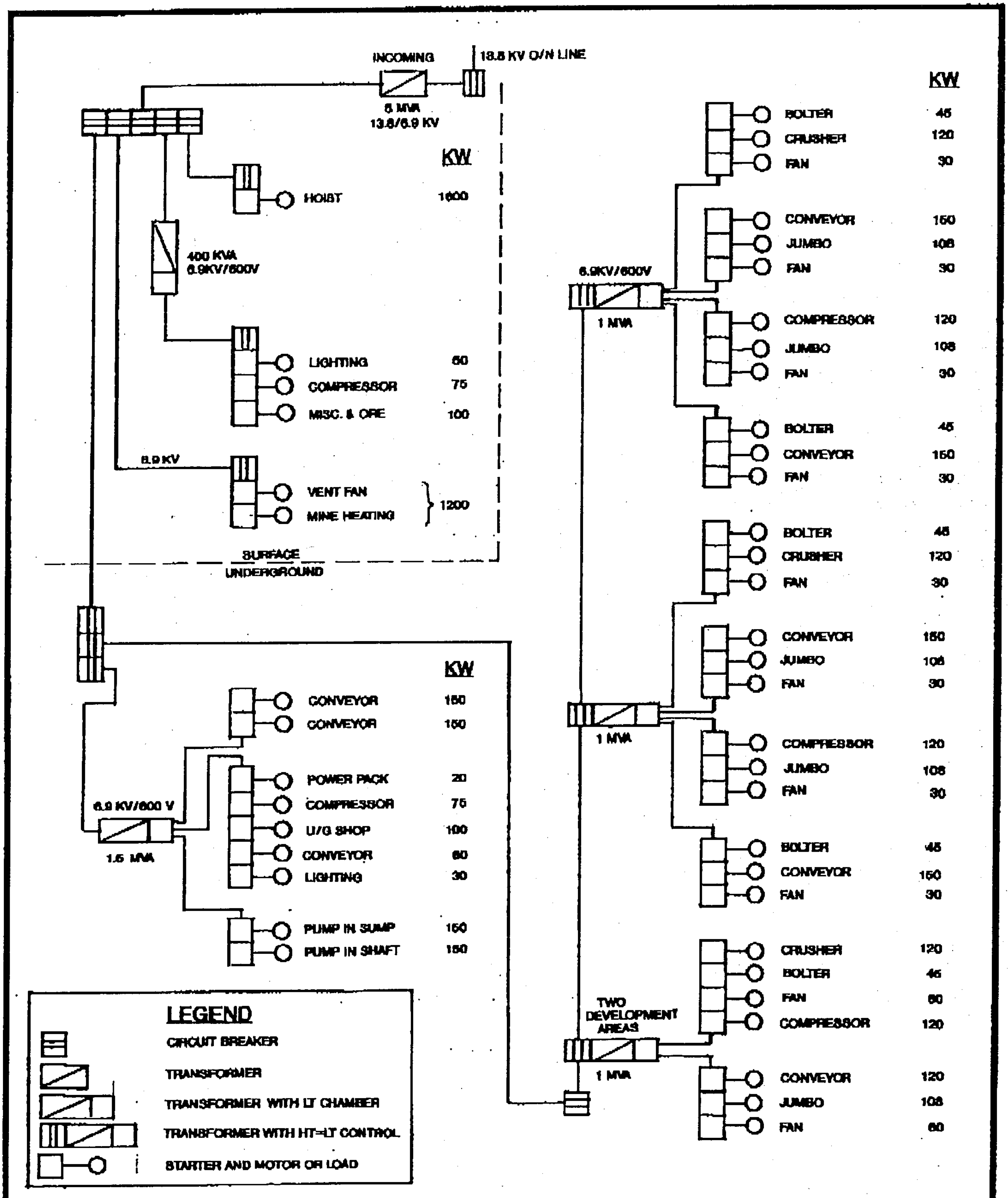
In the mining sections, a single one MVA transformer (6.9 KV to 600 volts) will feed two sections of mining equipment. A second transformer will feed another two sections, and a third similar unit will supply the development drivages.

Armoured cables will be used in semi-permanent positions, and flexible trailing cables will be used for drills, fans, etc.

The electrical distribution is shown as Figure 12.8-01. A list of major items, with cost, appears in the Capital Cost Estimates in Chapter 14, subsection 14.3-03.

There are two differences between Case A and Case B

- Case A will need power for the conveyor system, whereas in Case B, diesel trucks will be used.
- The main fan for Case B will require 2300 HP, compared to 1700 in Case A, unless there is a third ventilation raise available.



ANVIL RANGE MINING CORPORATION
GRIZZLY PROJECT
UNDERGROUND MINING STUDY

PROPOSED ELECTRICAL SCEMATIC FOR CASE A MINE PLAN

BY: NP/BL	DATE: OCT.96
APPROVED:	FIG: 12.A8-01

The estimated power costs have been calculated for each case, and are included in the Chapter on Operating Costs, section 15.3.10. The average cost of power is based on four parameters:

A	Fixed Charge	$10,724/month
B	Peak Power Demand Charge	$107,880 per KVA
C	Charge per KWH	
D	Rider and Revenue Shortfall	

The average works out to 7.4 cents per KWH.

In Case A, the sump and main pumps are placed 20 metres vertically below the main shaft station and 200 metres from it. In Case B, the main sump will be constructed at an elevation of about 430 metres, also twenty metres below the shaft station.

12.9 Mine Dewatering

Water inflow is expected to be low for the size of the area of the mine. When water inflow does increase, it will tend to come from faulted areas where the impervious phyllites are fractured.

Therefore in both Cases A and B, a pumping capacity of 300 igpm (22.73 litres per second) is planned to be installed.

At first assessment, it was planned to back up this pump with a large sump holding 48 hours' inflow at 300 igpm. The cost of excavation became so high that a second complete pumping system is recommended, for a lower price.

The pumping system proposed comprises two 6" stainless steel submersible pumps, each capable of pumping 300 igpm halfway up the shaft with a 150 KW motor. The second unit will be placed halfway up the shaft as a booster pump without the need for a pump station.

Three small electric pumps (5 KW) for use at development headings, and three intermediate size pumps of 60 KW will be available, mainly for use in declines.

13. PRODUCTION PLAN AND SCHEDULES

13.1 Introduction

Examination of drill hole data and the development of new sections perpendicular to the assumed strike led to a new interpretation of the orebody that indicates the existence of two mineable ore horizons from 15 to 60 m apart. The production plan is based on this concept. Folding in the Anvil area will, undoubtedly, have influenced the Grizzly deposit.

While it is expected that underground drilling will give a different picture of the orebody, the present interpretation is being used to develop the production plans presented here.

Assumptions and planning is sufficiently detailed to come up with realistic estimates for production forecasting.

A number of ore reserve calculations have been carried out over the years. They have always been based on economic cutoff grades of 6, 8, and 9%, combined Pb and Zm. Dilution factors and mining recovery have been estimated and added to come up with realistic numbers.

Under close scrutiny, and with the perception of production mining, it becomes apparent that not economic, but mining cutoffs will have to be used to allow more accurate grade estimates. This will be better understood once underground drilling is in progress.

The deposit has always been considered as the B-Zone to the northeast and the A-Zone to the northwest and southwest.

Mining is expected to start in the less deep, but high grade and fairly thick, B-Zone.

13.2 Resource Valuation

Calculations of the Net Smelter Return (NSR) have been carried out for each polygon using a cutoff of 9% and a 6% combined PbZn.

The calculation involves using the vertical thickness of the observed interval in the core and its assay value of the four economic metals, lead, zinc, silver and gold. The total metal credits were calculated in U.S. dollars using the following values:

Lead	U.S. $0.30 per pound
Zinc	U.S. $0.55 per pound
Silver	U.S. $5.50 per ounce
Gold	U.S. $400.00 per ounce

The smelting and transportation costs from the port of Skagway were deducted. Smelter losses or penalties were also deducted. The numbers used were actuals experienced by the existing operations of Anvil Range Mining Corporation. These costs were then converted into Canadian dollars at the rate of 0.73.

Multiplying the tonnage in the polygon by the metal values gives the gross revenues per tonne. Similar treatment of the costs provides the smelting costs, subtracting one from the other provides the “net smelter return”, normally in U.S. dollars.

This number was deducted from the NSR to assess whether each polygon and block would contribute to the profit of the operation. The summary is shown in Tables 13.1 to 13.5.

Table 13.2.01

Resource Valuation - 9% Cutoff

9% Cutoff	Tonnes millions Recoverables	Pb%	Zn%	Ag g/t	Au g/t	NSR/t	Net Revenue $ millions
Upper Horizon	6.667	5.57	5.85	77.07	0.90	85.35	141.917
Lower Horizon	8.284	4.47	7.28	70.04	0.67	92.89	202.100
Total	17.241	4.98	6.61	73.38	0.78	88.45	344.017

TABLE 13.1

9% Pb+Zn CUTOFF GRADE BLOCKS
UPPER HORIZON - NET REVENUES @ $Cdn 68.50 OP. COST/TONNE

	BLOCK	DHOLE	TONNES MILLED	DILUTED GRADES Pb%	Zn%	Ag g/t	Au g/t	NSR $Cdn/t	TOTAL NET REVENUE $Cdn
A ZONE NORTH	A1	77x01	203850	4.21	6.86	71.8	0.86	90.47	4,477,678
		78x08	226212	4.01	8.70	74.1	0.82	105.00	8,257,275
		SUBTTL	430062	4.10	7.83	73.0	0.84	98.11	12,734,952
	A2	77x03	246553	5.52	4.55	73.6	0.56	74.98	1,596,844
	A3	77x05	205688	4.79	6.96	98.5	1.23	99.82	6,442,465
	A4	79x06	771432	8.40	5.41	108.8	1.07	102.38	26,139,828
		79x11	364540	5.21	5.15	68.5	0.72	78.99	3,823,467
		79x12	580876	4.57	4.85	65.1	0.61	72.60	2,378,868
		79x14	476201	4.71	4.00	61.7	1.20	69.26	364,042
		SUBTTL	2193049	6.05	4.91	80.3	0.92	83.41	32,706,206
	A5	80x01	190814	5.42	5.19	72.9	1.05	82.81	2,730,662
	TOTAL NORTH		3266166	5.64	5.41	79.6	0.91	85.71	56,211,129
A ZONE SOUTH	A6	79x18	119402	2.69	6.28	55.4	0.85	76.81	992,448
		80x04	136995	4.73	5.95	74.1	1.40	88.59	2,752,816
		SUBTTL	256396	3.78	6.10	65.4	1.14	83.11	3,745,264
	A7	79x13	726143	5.97	6.30	79.3	0.86	94.17	18,636,880
		79x16	484679	4.09	4.61	59.6	0.58	67.66	(404,983)
		80x05	650160	5.59	7.21	82.8	0.83	100.45	20,775,213
		80x08	1167469	3.83	4.77	60.1	0.63	68.28	(255,241)
		80x09	712017	7.48	4.08	86.5	0.75	82.31	9,832,541
		80x10	635627	4.93	6.36	80.9	1.33	93.45	15,861,095
		SUBTTL	4376094	5.23	5.49	73.9	0.81	83.23	64,445,506
	A8	80x02	266589	8.05	9.02	92.5	1.10	128.81	16,078,232
	A9	80x06	189011	4.24	4.56	60.5	0.18	65.54	(559,199)
	TOTAL SOUTH		5088090	5.27	5.67	74.0	0.82	84.95	83,709,802
NORTH/SOUTH			8354256	5.41	5.57	76.2	0.86	85.25	139,920,932
B ZONE	B1	78x09	248134	2.45	6.05	39.8	0.54	69.74	306,681
	B2	78x04	104476	3.07	6.96	52.5	0.42	81.04	1,310,160
	Q1	77x09	249154	2.81	6.14	29.0	0.48	70.02	379,095
	TOTAL B ZONE		601764	2.70	6.25	37.5	0.49	71.82	1,995,936
	UPPER HORIZON		8956020	5.23	5.62	73.6	0.83	84.35	141,916,868

TABLE 13.2

9% Pb+Zn CUTOFF GRADE BLOCKS
LOWER HORIZON - NET REVENUES @ $Cdn 68.50 OP. COST/TONNE

	BLOCK	DHOLE	TONNES MILLED	DILUTED GRADES				NSR/TON $Cdn	TOTAL NET REVENUE $Cdn
				Pb%	Zn%	Ag g/t	Au g/t		
A ZONE NORTH	A18	79x11	429564	5.55	5.03	82.0	0.92	82.51	6,017,492
	TOTAL NORTH		429564	5.55	5.03	82.0	0.92	82.51	6,017,492
A ZONE SOUTH	A11	80x07	182724	3.45	5.68	44.1	0.73	72.57	744,490
		80x13	163062	3.61	5.19	47.8	0.64	69.14	104,221
		SUBTTL	345786	3.52	5.45	45.8	0.68	70.95	848,711
	A13	80x06	165826	5.19	4.63	72.1	0.81	75.59	1,175,702
	A14	80x02	527590	4.15	8.23	72.5	1.23	103.91	18,681,995
	A16	80x09	149476	8.00	5.32	127.5	0.85	101.33	4,906,620
	TOTAL SOUTH		1188679	4.59	6.55	71.6	0.96	90.05	25,613,029
NORTH/SOUTH			1618243	4.85	6.15	74.3	0.95	88.05	31,630,521

TABLE 13.3

9% Pb+Zn CUTOFF GRADE BLOCKS
LOWER HORIZON - NET REVENUES @ $Cdn 68.50 OP. COST/TONNE

	BLOCK	DHOLE	TONNES MILLED	DILUTED GRADES Pb%	Zn%	Ag g/t	Au g/t	NSR/TON $Cdn	TOTAL NET REVENUE $Cdn
B ZONE	B3	90D05	1023585	5.35	9.17	72.5	0.41	111.77	44,290,262
	B4	78x04	154153	8.63	11.71	137.3	1.04	159.78	14,071,382
		79x02	101816	4.39	7.11	62.2	0.33	88.56	2,042,663
		SUBTTL	255970	6.94	9.88	107.4	0.75	131.45	16,114,045
	B5	77x06	651336	5.41	10.01	96.3	0.52	123.10	35,563,670
		78x11	246798	4.29	6.89	70.3	0.66	89.58	5,201,329
		79x07	83162	4.35	8.35	60.8	0.63	100.38	2,650,844
		90D09	311055	3.72	8.09	61.0	0.35	93.96	7,918,309
		91D03	338999	4.15	5.95	60.2	0.66	79.66	3,782,860
		91D05	792643	3.59	7.84	62.6	0.47	92.27	18,841,910
		SUBTTL	2423992	4.27	8.11	71.8	0.52	99.01	73,958,921
	B6	78x09	204434	3.87	5.98	68.5	1.02	82.19	2,799,152
	B7	78x05	473616	4.52	8.46	70.8	0.95	105.45	17,501,779
		90D04	236720	3.52	6.60	43.9	0.48	79.00	2,484,402
		SUBTTL	710335	4.18	7.84	61.9	0.79	96.64	19,986,181
	B8	79x08	141541	3.95	5.13	84.2	0.99	77.50	1,274,535
	Q5	77x09	262313	4.55	6.45	71.5	0.89	88.54	5,256,697
		79x04	146807	3.35	7.09	58.1	0.57	84.99	2,421,041
		79x05	249020	3.65	5.85	56.5	0.25	73.69	1,291,539
		SUBTTL	658140	3.94	6.36	62.8	0.58	82.13	8,969,278
	Q6	78x02	842137	3.65	5.18	58.7	0.64	70.83	1,966,346
	Q7	78x01	200311	3.23	5.99	49.3	0.72	74.94	1,290,580
		79x09	206142	3.50	4.94	44.4	0.89	67.63	(179,065)
		SUBTTL	406453	3.37	5.46	46.8	0.81	71.23	1,111,514
	TOTAL B ZONE		6666587	4.35	7.48	68.3	0.60	94.07	170,470,233
	LOWER HORIZON		8284830	4.44	7.22	69.5	0.67	92.89	202,100,754
	TOTAL 9%		17240850	4.85	6.39	71.6	0.75	88.45	344,017,622

TABLE 13.4

6% Pb+Zn CUTOFF GRADE POLYGONS
UPPER HORIZON - NET REVENUES @ $Cdn 68.50 OP. COST/TONNE

	DHOLE	TONNES MILLED	Pb%	Zn%	Ag g/t	Au g/t	NSR/TON $Cdn	TOTAL NET REVENUE $Cdn
	77x01	203823	4.21	6.86	71.8	0.86	90.47	4,477,086
	77x05	207031	4.79	6.96	98.5	1.23	99.82	6,484,519
	78x04	104476	3.07	6.96	52.5	0.42	81.04	1,310,160
	78x08	367352	3.36	6.66	62.6	0.65	82.61	5,183,662
	78x09	248134	2.45	6.05	39.8	0.54	69.74	306,681
	79x06	1266860	6.45	4.68	84.5	0.99	84.18	19,864,305
	79x11	402443	5.29	4.64	64.3	0.83	75.06	2,639,403
	79x13	724399	5.97	6.30	79.3	0.86	94.17	18,592,125
	79x18	282910	2.61	5.20	65.5	1.04	70.10	453,570
	80x04	138222	4.73	5.95	74.1	1.40	88.59	2,777,479
	80x05	649437	5.59	7.21	82.8	0.83	100.45	20,752,118
	80x09	799049	7.09	3.86	83.0	0.85	78.97	8,365,439
	80x10	807613	4.42	5.93	73.7	1.28	86.38	14,438,848
UPPER HORIZON		6201748	5.30	5.60	76.1	0.94	85.53	105,645,395

TABLE 13.5

6% Pb+Zn CUTOFF GRADE POLYGONS
LOWER HORIZON - NET REVENUES @ $Cdn 68.50 OP. COST/TONNE

	DHOLE	TONNES MILLED	Pb%	Zn%	Ag g/t	Au g/t	NSR/TON $Cdn	TOTAL NET REVENUE $Cdn
	77x06	718023	5.10	9.41	90.7	0.49	115.84	33,991,772
	77x09	481237	3.66	5.33	61.3	0.57	72.06	1,714,616
	78x04	153772	8.63	11.71	137.3	1.04	159.78	14,036,590
	78x05	623403	4.03	7.52	63.3	0.79	93.48	15,573,973
	78x09	204243	3.87	5.98	68.5	1.02	82.19	2,796,537
	78x11	247336	4.29	6.89	70.3	0.66	89.58	5,212,659
	79x02	101410	4.39	7.11	62.2	0.33	88.56	2,034,507
	79x04	146641	3.35	7.09	58.1	0.57	84.99	2,418,311
	79x05	248514	3.65	5.85	56.5	0.25	73.69	1,288,911
	79x07	122217	3.63	7.31	52.0	0.55	86.98	2,259,168
	79x11	429564	5.55	5.03	82.0	0.92	82.51	6,017,492
	80x02	778681	3.53	6.69	61.5	0.89	84.87	12,749,021
	80x05	195090	3.44	4.63	54.5	1.48	69.90	272,473
	80x06	511313	3.54	4.46	72.5	1.09	69.16	338,324
	80x07	212268	3.40	5.58	44.5	0.72	71.55	647,067
	80x09	149476	8.00	5.32	127.5	0.85	101.33	4,906,620
	90D05	1023585	5.35	9.17	72.5	0.41	111.77	44,290,262
	90D09	306508	3.72	8.09	61.0	0.35	93.96	7,802,562
	91D03	459840	3.60	5.37	51.5	0.56	70.71	1,016,444
	91D05	792900	3.59	7.84	62.6	0.47	92.27	18,848,022
LOWER HORIZON		7906021	4.31	7.07	69.2	0.67	91.04	178,215,331
TOTAL 6%		14107769	4.74	6.43	72.3	0.79	88.62	283,860,727

A similar calculation was made for those polygons with a 6% combined cutoff. After eliminating thirty six polygons because of their negative net revenue, the results can be expressed in a similar small table, Table 13.2.02.

Table 13.2.02

Resource Valuation - 6% Cutoff

6% Cutoff	Tonnes millions Recoverables	Pb%	Zn%	Ag g/t	Au g/t	NSR/t	Net Revenue $ millions
Upper Horizon	6.202	5.30	5.60	76.1	0.94	85.53	105.645
Lower Horizon	7.906	4.31	7.07	69.2	0.67	91.04	178.215
Total	14.108	4.74	6.43	72.3	0.79	88.62	283.861

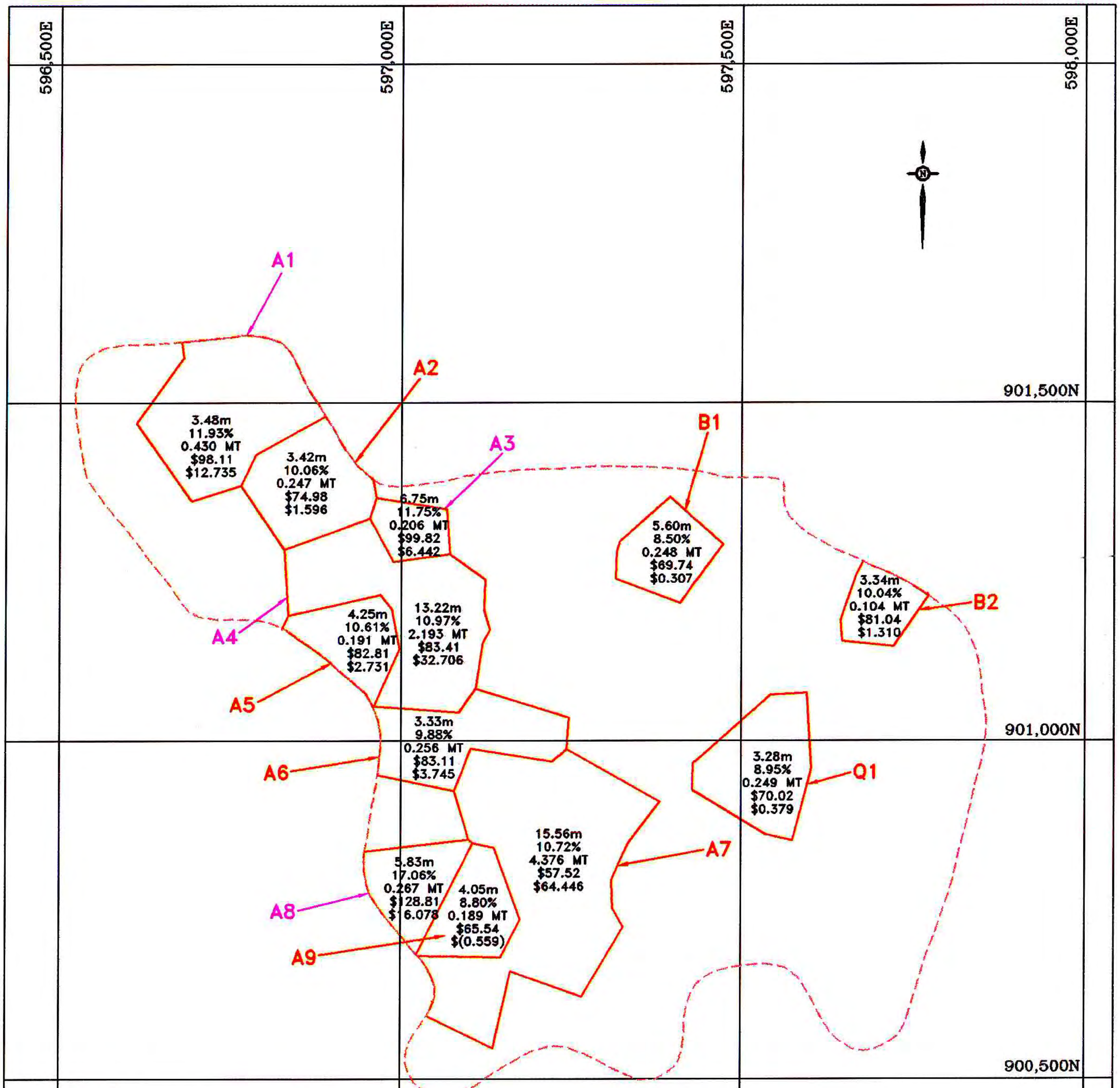
A comparison of Tables 13.2.1 and 13.2.2 indicates that, at the lower cutoff grade, the recoverable tonnage is 3.1 million tonnes less, the NSR is 17 cents higher, and the net revenue is $60 million less.

On this basis, the expected greater net revenue from the inclusion of some lower grade ore did not materialise. The figures do indicate some marked grouping of values, which justify further study during the underground exploration program, probably doing calculation at 7% and 8% cutoffs.

For this study, only the mineral inventory as defined by the 9% cutoff figure will be included in further mine planning.

Two plans were prepared to show the location of the polygons, using the 9% combined lead/zinc cutoff:

Figure 13.2-01 Upper Horizon
Figure 13.2-02 Lower Horizon



LEGEND

ORE BLOCK	A4
AVERAGE VERTICAL THICKNESS	13.22m
GRADE Pb+Zn %	10.97%
RECOVERABLE TONNES (MILLIONS)	2.193 MT
NSR – VALUE PER RECOVERABLE TONNE	$83.41
NSR NET REVENUE PER POLYGON (MILLIONS)	$32.706

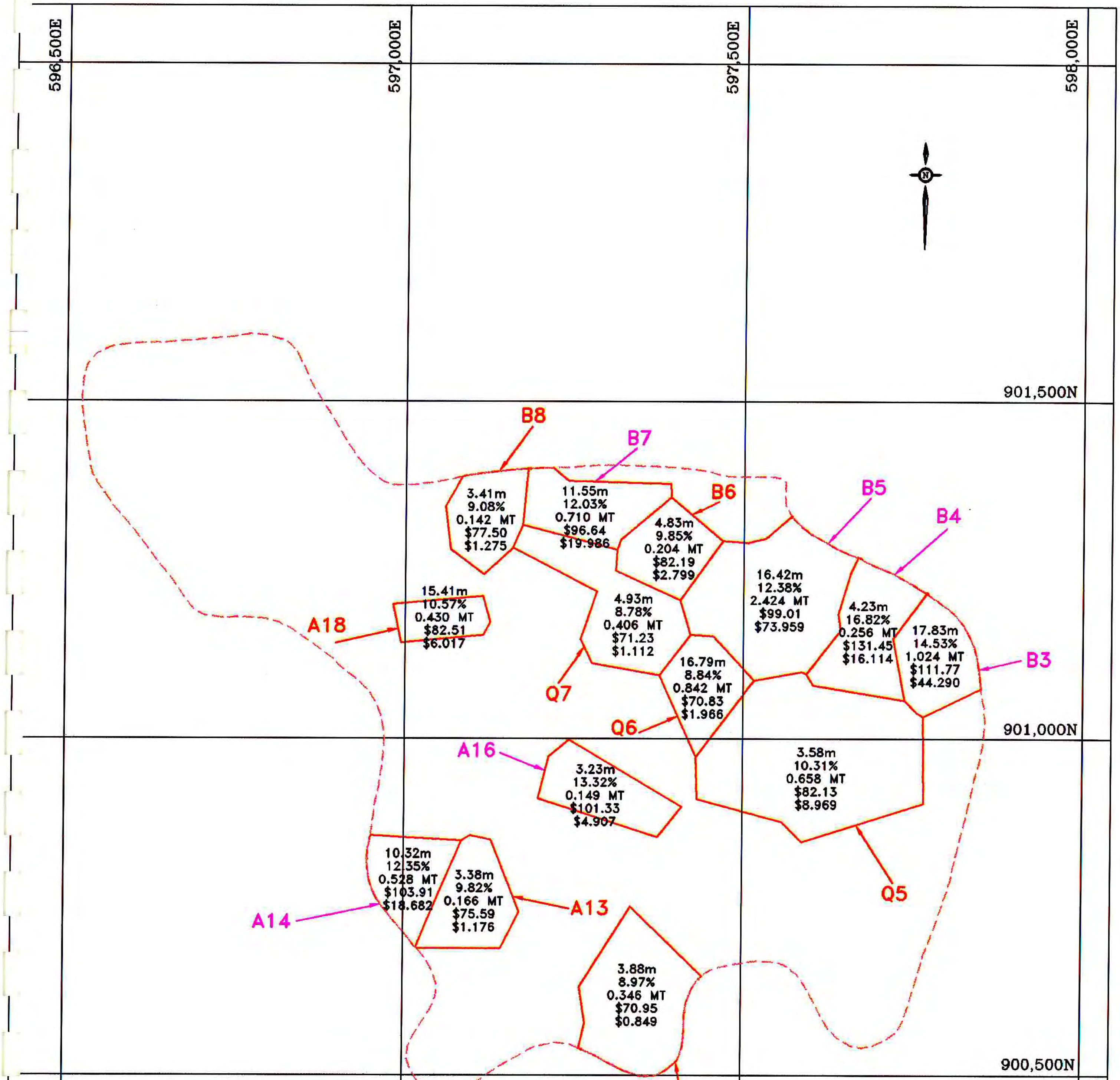
NOTES

1. ALL NSR VALUES AND REVENUES ARE IN CANADIAN DOLLARS.
2. ABOVE BASED ON OPERATING COST OF $68.50/TONNE.

FIG. 13.2–01

Anvil Range Mining Corporation

ANVIL RANGE MINING CORPORATION	DRAWN BY: NDR	DATE: 07/11/96	SCALE: 1:2500	GRIZZLY DEPOSIT UPPER–G HORIZON 9% PB+ZN CUTOFF GRADE ORE BLOCKS WITH NSR AND NR VALUES
	REVISED BY:	DATE:		
	FILE: NSRGRIC9.DWG			



LEGEND

ORE BLOCK	B5
AVERAGE VERTICAL THICKNESS	16.42m
GRADE Pb+Zn %	12.38%
RECOVERABLE TONNES (MILLIONS)	2.424 MT
NSR - VALUE PER RECOVERABLE TONNE	$99.01
NSR NET REVENUE PER POLYGON (MILLIONS)	$73.959

NOTES

1. ALL NSR VALUES AND REVENUES ARE IN CANADIAN DOLLARS.
2. ABOVE BASED ON OPERATING COST OF $68.50/TONNE.

FIG. 13.2-02

Anvil Range MINING CORPORATION

ANVIL RANGE MINING CORPORATION	DRAWN BY: NDR	DATE: 07/11/96	SCALE: 1:2500	GRIZZLY DEPOSIT LOWER-G HORIZON 9% PB+ZN CUTOFF GRADE ORE BLOCKS WITH NSR AND NR VALUES
	REVISED BY:	DATE:		
	FILE: NSRGR2C9.DWG			

Both have five values placed in, showing:

Average vertical thickness m
Combined lead & zinc grade %
Millions of recoverable tonnes
NSR - value per recoverable tonne (s) $CAD
NSR - net revenue per polygon (millions) $CAD

As expected, the B Zone shows a net revenue of $160 million, while the Q Zone (quartzite ore) located in the centre of the deposit produces a net revenue of only $12.4 million.

Net revenue for the four main mining areas is shown in the table below.

Area	Recoverable Tonnes	Grade Diluted					Can $	Net Revenue millions
	Millions	Pb+Zn %	Pb%	Zn%	Ag g/t	Au g/t	NSR	Can $
NE-B	5.112	12.55	4.47	8.08	70.6	.58	99.80	160.039
NE-Q	2.150	9.29	3.59	5.70	54.3	.63	74.26	12.426
NW	3.695	11.00	5.63	5.37	79.8	.91	85.34	62.229
SW	6.277	10.98	5.14	5.84	73.5	.85	85.92	109.323
Total	17.240	11.24	4.85	6.39	71.6	.76	88.45	344.017

However, when it comes to mining, recognition of what is ore at 15%, ore at 10%, ore at 5% or waste, is a very important issue to solve and shows the importance of good grade control.

13.3 Mine Plan

Mining will start in the B-Zone for the following reasons:

- The area contains 5.1 million tonnes of recoverable ore (30% of the total) at an average combined lead/zinc grade of 12.55%, well above the average for the deposit of 11.24%, and has the capability to support full production levels.
- The infrastructure is reasonably accessible, especially to a central shaft and East ventilation raises.
- The beds to be mined lie at a higher elevation.
- Early development can take place through the proposed twin ramps.
- The area has the largest tonnage, and can provide 4 to 5 years of production before development of another area has to be undertaken.

Two different mining cases have been described in Chapter 12. The layout for both mining cases is generally similar, except for the major differences of annual tonnage, shaft location and depth, and the use of conveyors versus truck haulage for ore transportation.

The Figures listed below show mine plans for the upper and lower horizons:

13.3.01	Case A, Upper Horizon (Upper G)
13.3.02	Case A, Lower Horizon (Lower G)
13.3.03	Case B, Upper Horizon (Upper G)
13.3.04	Case B, Lower Horizon (Lower G)

13.3.1 Production Estimates

Throughout the orebody and for the life of the mine, the average thickness of ore is 70% above 6.5 m and 30% below 6.5 m.

Production requirements are:

- 3,500 tpd for 1.2 million tpy, and

Therefore, the thin upper blocks of B2 and B1 will be mined first, which will allow complete mining of the mostly thick rich bed underneath.

When considering the sequence of mining the blocks of ore which vary in thickness from 4-5 metres to 16-18 metres over a distance of as little as 60 metres, a large area must be considered.

For example, in Blocks B3 to B7, the thickness alternates from thick to thin to thick to thin, and back to thick - all in a distance of 550 metres. The sequence of mining is to advance in this same order. In this case, the primary extraction will start in the thick B3 Block, taking about 50% of the ore. The secondary pillar extraction will not take place until later.

Block 4, a thin one to be worked by a caving system would not be mined, just access drifts made through it. In Block 5, a thick one, the primary extraction would next be made. In this case, a large enough area will have been opened up, and the direction of mining will reverse, starting with complete extraction of Block 5, followed by caving of Block 4, and the secondary extraction and backfilling of Block 3.

A start would then be made to mine Northwestwards starting by driving through Block 6 (thin), primary extraction in Block 7 (thick), and caving in Block 8 (thin). The remaining ore would be extracted as the direction of mining reverses, as before.

In addition, Tables 13.A6 and 13.B6 show the following details of production for each year:

- Block being mined- tonnes produced from that block in tonnes.
- Tonnes produced from that block in that year.
- The grade of ore from each block - by each metal.
- The NSR per tonne.
- The net revenue of a particular block that year, and for all blocks in that same year.

Table 13.A8
Schedule and Cost - Years 2 to 5
Case A

	Year 2				Year 3				Year 4				Year 5			
	A	B	C	D	A	B	C	D	A	B	C	D	A	B	C	D
Ore Tonnage (000)	-	-	-	-	-	50	100	150	200	200	250	250	300	300	300	300
Based on straight line access down twin ramps on the Eastern Deposit boundary																
Ramp bottom - main conv.	350m															
Intake B2 - thin					100m											
Conv. to B3 (2)					350m	350m										
Main conv. decline - shaft		570m	300m													
Main access			270m													
Conv. B2				200m												
Return B2																
Backfill drifts					210m											
Drifts B5									160m							
U. Horizon to vent raises				120m												
L. Horizon to vent raises				150m												
Shaft sump ramp					150m	150m										
Shaft bottom bin						120m										
Main sump				50m												
Note: rock raises extra.																
Dev. ore (metres - tonnes included in stope figure below)																
B2						140m										
B3							710m									
B5								600m	600m	470m						
B4														250m	250m	
Stopes (000 tonnes)																
B2 Upper thin						50t	55t									
B3 Lower thick							45t	150	200	116					272	240
B5 Lower thick										84	250	250	300	300	38	
B4 Lower thick																60
Cost $000	1132	1500	1500	1500	2917	3000	2000	1984	Costs as Table 15.A5							
Contractors only - rest on Tables 14.A6.1 and Table 15.A5.																

Table 13.B8
Schedule and Cost - Years 2 to 5
Case B

	Year 2				Year 3				Year 4				Year 5			
	A	B	C	D	A	B	C	D	A	B	C	D	A	B	C	D
Tonnes (000)	-	-	-	-	75	100	150	175	200	250	375	375	375	375	375	375
Rock NE explor. - E shaft	150m															
Rock E shaft to L. Horiz B3	80m	70m														
Rock W Access - main shaft		180m														
Bin ramp shaft bottom			120m													
HW Access B3						150m										
Shaft sump ramp			300m													
HW Access B5							150m									
HW Access B7															250m	250m
Rock B2 haul drift						50m										
Rock, pump sump			50m													
Vent Raise - haul waste - est $10/t				$100 k	$100 k											
Rock raises extra.																
Ore - 000 tonnes																
HW to East 400m - 5m/day					37.5 t											
FW to East 450m - 5m/day					37.5 t											
Connect Return via Sample Drift (complete vent in NE) & access to B2						15 t										
B5 stope (thick)						10 t										
FW Dev SE to B3 (thick)						37.5 t	20 t						199	205	205	273
HW Dev SE to B2						37.5 t										
Access B2 - stope (thin)							60 t	89 t								
Stope B3 - thick							70 t	120	140	170	170	170	170	170	24	-
FW Dev B5 - B8									50		50					
HW Dev B5 - B8										50		50				
Stope B4 (thin)									10	30	155	155	15			
Dev B1 & Stope (thin)															146	102
Men at work each day	20	30	36	48	64	76	100	130	160	206	206	206				
Costs	Labour x 2 plus $1 mill staff				As Table 15.B5				Full Costs For Year				Full Costs 1.5 mill tonnes			

TABLE 13.A.6

MINING SCHEDULE BY YEARS - CASE A - 1.2 MILLION TONNES PER YEAR
DILUTED TONNES AND GRADES
OPERATING COST @ 68.50 Cdn$/TONNE

YEAR	BLOCK	TONNES	Pb+Zn	Pb	Zn	Ag	Au	N.S.R. Cdn$/T	NET REVENUE Cdn$
	B2 NE	104,476	10.03	3.07	6.96	52.55	0.42	81.04	1,310,160
1	B3 NE	195,524	14.52	5.35	9.17	72.45	0.41	111.77	8,460,274
	TOTAL	300,000	12.96	4.56	8.40	65.52	0.41	101.07	9,770,434
	B3 NE	316,269	14.52	5.35	9.17	72.45	0.41	111.77	13,684,857
2	B5 NE	583,732	12.38	4.27	8.11	71.84	0.52	99.01	17,810,352
	TOTAL	900,000	13.13	4.65	8.48	72.05	0.48	103.49	31,495,210
	B3 NE	511,793	14.52	5.35	9.17	72.45	0.41	111.77	22,145,131
	B5 NE	628,265	12.38	4.27	8.11	71.84	0.52	99.01	19,169,108
3	B4 NE	59,943	16.82	6.94	9.88	107.40	0.75	131.45	3,773,584
	TOTAL	1,200,000	13.51	4.86	8.65	73.88	0.48	106.07	45,087,823
	B5 NE	600,000	12.38	4.27	8.11	71.84	0.52	99.01	18,306,724
	B4 NE	196,027	16.82	6.94	9.88	107.40	0.75	131.45	12,340,461
4	B1 NE	248,134	8.50	2.45	6.05	39.82	0.54	69.74	306,681
	B7 NE	155,839	12.02	4.18	7.84	61.85	0.79	96.64	4,384,729
	TOTAL	1,200,000	12.26	4.32	7.94	69.73	0.60	97.95	35,338,595
	B7 NE	199,329	12.02	4.18	7.84	61.85	0.79	96.64	5,608,362
	B5 NE	611,996	12.38	4.27	8.11	71.84	0.52	99.01	18,672,736
	B6 NE	204,434	9.85	3.87	5.98	68.55	1.02	82.19	2,799,152
5	B8 NE	141,541	9.08	3.95	5.13	84.18	0.99	77.50	1,274,535
	A4 WNW	42,701	10.96	6.05	4.91	80.32	0.92	83.41	636,817
	TOTAL	1,200,000	11.45	4.21	7.24	71.38	0.72	92.66	28,991,602
	B7 NE	355,168	12.02	4.18	7.84	61.85	0.79	96.64	9,993,091
	A4 WNW	600,000	10.96	6.05	4.91	80.32	0.92	83.41	8,948,146
6	A7 SW	244,833	10.72	5.23	5.49	73.93	0.81	83.23	3,605,579
	TOTAL	1,200,000	11.22	5.33	5.90	73.55	0.86	87.29	22,546,816
	A4 WNW	179,693	10.96	6.05	4.91	80.32	0.92	83.41	2,679,865
	A7 SW	564,708	10.72	5.23	5.49	73.93	0.81	83.23	8,316,296
7	A8 S	266,589	17.07	8.05	9.02	92.45	1.10	128.81	16,078,232
	A9 S	189,010	8.80	4.24	4.56	60.45	0.18	65.54	-559,197
	TOTAL	1,200,000	11.86	5.82	6.04	76.88	0.79	90.60	26,515,196
	A4 WNW	206,584	10.96	6.05	4.91	80.32	0.92	83.41	3,080,906
	A7 SW	300,000	10.72	5.23	5.49	73.93	0.81	83.23	4,418,016
8	A13 S	165,826	9.82	5.19	4.63	72.09	0.81	75.59	1,175,702
	A14 S	527,590	12.38	4.15	8.23	72.45	1.23	103.91	18,681,995
	TOTAL	1,200,000	11.37	4.89	6.48	74.13	1.01	91.30	27,356,619

TABLE 13.A.6 (Continued)

MINING SCHEDULE BY YEARS - CASE A - 1.2 MILLION TONNES PER YEAR
DILUTED TONNES AND GRADES
OPERATING COST @ 68.50 Cdn$/TONNE

YEAR	BLOCK	TONNES	Pb+Zn	Pb	Zn	Ag	Au	N.S.R. Cdn$/T	NET REVENUE Cdn$
	A4 WNW	304,085	10.96	6.05	4.91	80.32	0.92	83.41	4,534,988
9	A7 SW	465,854	10.72	5.23	5.49	73.93	0.81	83.23	6,860,494
	A1 NW	430,062	11.93	4.10	7.83	73.01	0.84	98.11	12,734,952
	TOTAL	1,200,000	11.21	5.03	6.18	75.22	0.85	88.61	24,130,433
	A2 NW	246,553	10.07	5.52	4.55	73.64	0.56	74.98	1,596,844
	A3 WNW	123,413	11.75	4.79	6.96	98.45	1.23	99.82	3,865,485
10	A4 WNW	311,725	10.96	6.05	4.91	80.32	0.92	83.41	4,648,935
	A7 SW	518,309	10.72	5.23	5.49	73.93	0.81	83.23	7,632,991
	TOTAL	1,200,000	10.75	5.46	5.30	78.05	0.83	83.29	16,147,411
	A7 SW	565,110	10.72	5.23	5.49	73.93	0.81	83.23	8,322,216
	A16 S	149,476	13.32	8.00	5.32	127.45	0.85	101.33	4,906,620
11	A18 WNW	236,260	10.58	5.55	5.03	82.00	0.92	82.51	3,309,618
	Q1 NE	249,154	8.95	2.81	6.14	29.00	0.48	70.02	379,095
	TOTAL	1,200,000	10.65	5.14	5.51	72.86	0.77	82.60	16,917,549
	A7 SW	491,975	10.72	5.23	5.49	73.93	0.81	83.23	7,245,178
	A3 ENW	82,275	11.75	4.79	6.96	98.45	1.23	99.82	2,576,980
12	A4 ENW	548,262	10.96	6.05	4.91	80.32	0.92	83.41	8,176,548
	A7 SE	77,488	10.72	5.23	5.49	73.93	0.81	83.23	1,141,144
	TOTAL	1,200,000	10.90	5.57	5.33	78.53	0.89	84.45	19,139,849
	A7 SE	380,646	10.72	5.23	5.49	73.93	0.81	83.23	5,605,659
	Q6 NE	421,069	8.83	3.65	5.18	58.73	0.64	70.83	983,173
13	A11 SW	207,472	8.97	3.52	5.45	45.85	0.68	70.95	509,228
	A5 NW	190,814	10.61	5.42	5.19	72.91	1.05	82.81	2,730,662
	TOTAL	1,200,000	9.74	4.41	5.33	63.58	0.77	76.69	9,828,722
	A7 SE	461,929	10.72	5.23	5.49	73.93	0.81	83.23	6,802,699
	Q7 NE	406,453	8.83	3.37	5.46	46.78	0.81	71.23	1,111,514
14	A18 ENW	193,304	10.58	5.55	5.03	82.00	0.92	82.51	2,707,874
	A11 SE	138,314	8.97	3.52	5.45	45.85	0.68	70.95	339,483
	TOTAL	1,200,000	9.86	4.45	5.40	62.80	0.81	77.63	10,961,570
	A7 SE	305,244	10.72	5.23	5.49	73.93	0.81	83.23	4,495,235
	Q6 NE	421,069	8.83	3.65	5.18	58.73	0.64	70.83	983,173
15	Q5 NE	386,513	10.30	3.94	6.36	62.83	0.58	82.13	5,267,485
	A6 SE	87,175	9.88	3.78	6.10	65.37	1.14	83.11	1,273,395
	TOTAL	1,200,000	9.86	4.15	5.71	64.40	0.70	78.52	12,019,288
	Q5 NE	271,627	10.30	3.94	6.36	62.83	0.58	82.13	3,701,793
16	A6 SW	169,221	9.88	3.78	6.10	65.37	1.14	83.11	2,471,869
	TOTAL	440,848	10.14	3.88	6.26	63.80	0.79	82.51	6,173,662

TABLE 13.A.7

BLOCK CHARACTERISTICS - CASE A - 1.2 MILLION TONNES PER YEAR

MINING YEAR	HOR.	BLOCK	AREA	THIN	THICK	Pb+Zn	TONNES TO MILL
4	U	B1	NE	X		8.50	248,100
1	U	B2	NE	X		10.03	104,500
1-3	L	B3	NE		X	14.52	1,023,600
3-4	L	B4	NE	X		16.82	256,000
2-5	L	B5	NE		X	12.38	2,424,000
5	L	B6	NE	X		9.85	204,400
4-6	L	B7	NE		X	12.02	710,300
5	L	B8	NE	X		9.08	141,500
	SUBTOTAL					12.55	5,112,400
11	U	Q1	NE	X		8.95	249,200
15-16	L	Q5	NE	X		10.30	658,100
15	L	Q6	NE		X	8.83	842,100
14	L	Q7	NE	X		8.83	406,400
	SUBTOTAL					9.29	2,155,800
9	U	A1	NW	X		11.93	430,100
10	U	A2	NW	X		10.07	246,600
10	U	A3	WNW	X		11.75	123,400
12			ENW	X		11.75	82,300
5-10	U	A4	WNW		X	10.96	1,644,800
12			ENW		X	10.96	548,300
13	U	A5	NW	X		10.61	190,800
11	L	A18	WNW		X	10.58	236,300
14			ENW		X	10.58	193,300
	SUBTOTAL					11.00	3,695,900
16	U	A6	SW	X		9.88	169,200
15	U		SE	X		9.88	87,200
6-12	U	A7	SW		X	10.72	3,150,800
13-15			SE		X	10.72	1,225,300
7	U	A8	S	X		17.07	266,600
7	U	A9	S	X		8.80	189,000
13	L	A11	SW	X		8.97	207,500
14			SE	X		8.97	138,300
8	L	A13	S	X		9.82	165,800
8	L	A14	S		X	12.38	527,600
11	L	A16	S	X		13.32	149,500
	SUBTOTAL					10.98	6,276,800
	TOTAL					11.24	17,240,900

TABLE 13.B.6

MINING SCHEDULE BY YEARS - CASE B - 1.5 MILLION TONNES PER YEAR
DILUTED TONNES AND GRADES
OPERATING COST @ 68.50 Cdn$/TONNE

YEAR	BLOCK	TONNES	Pb+Zn	Pb	Zn	Ag	Au	Cdn$/T	Cdn$
1	B2 NE	104,476	10.03	3.07	6.96	52.55	0.42	81.04	1,310,160
	B3 NE	395,524	14.52	5.35	9.17	72.45	0.41	111.77	17,114,223
	TOTAL	500,000	13.58	4.87	8.71	68.29	0.41	105.35	18,424,383
2	B3 NE	116,269	14.52	5.35	9.17	72.45	0.41	111.77	5,030,908
	B5 NE	579,628	12.38	4.27	8.11	71.84	0.52	99.01	17,685,134
	B4 NE	255,970	16.82	6.94	9.88	107.40	0.75	131.45	16,114,045
	B1 NE	248,134	8.50	2.45	6.05	39.82	0.54	69.74	306,681
	TOTAL	1,200,000	12.73	4.57	8.16	72.86	0.56	101.11	39,136,769
3	B3 NE	511,793	14.52	5.35	9.17	72.45	0.41	111.77	22,145,131
	B5 NE	632,369	12.38	4.27	8.11	71.84	0.52	99.01	19,294,326
	B7 NE	355,168	12.02	4.18	7.84	61.85	0.79	96.64	9,993,091
	B8 NE	672	9.08	3.95	5.13	84.18	0.99	77.50	6,047
	TOTAL	1,500,000	13.02	4.62	8.41	69.69	0.55	102.79	51,438,594
4	B5 NE	600,000	12.38	4.27	8.11	71.84	0.52	99.01	18,306,724
	B6 NE	204,434	9.85	3.87	5.98	68.55	1.02	82.19	2,799,152
	B8 NE	140,870	9.08	3.95	5.13	84.18	0.99	77.50	1,274,535
	B7 NE	355,168	12.02	4.18	7.84	61.85	0.79	96.64	9,993,091
	A4WNW	199,529	10.96	6.05	4.91	80.32	0.92	83.41	2,975,691
	TOTAL	1,500,000	11.45	4.40	7.05	71.31	0.75	92.06	35,349,193
5	A7 SW	355,167	10.72	5.23	5.49	73.93	0.81	83.23	5,230,445
	B5 NE	611,996	12.38	4.27	8.11	71.84	0.52	99.01	18,672,736
	A4 WNW	532,837	10.96	6.05	4.91	80.32	0.92	83.41	7,946,506
	TOTAL	1,500,000	11.48	5.13	6.35	75.35	0.73	89.73	31,849,687
6	A4 WNW	90,028	10.96	6.05	4.91	80.32	0.92	83.41	1,342,632
	A7 SW	954,374	10.72	5.23	5.49	73.93	0.81	83.23	14,054,790
	A8 S	266,589	17.07	8.05	9.02	92.45	1.10	128.81	16,078,232
	A9 S	189,010	8.80	4.24	4.56	60.45	0.18	65.54	-559,197
	TOTAL	1,500,000	11.62	5.66	5.97	75.91	0.79	89.11	30,916,457
7	A4 WNW	804,526	10.96	6.05	4.91	80.32	0.92	83.41	11,998,353
	A7 SW	265,854	10.72	5.23	5.49	73.93	0.81	83.23	3,915,150
	A13 S	165,826	9.82	5.19	4.63	72.09	0.81	75.59	1,175,702
	A14 S	263,795	12.38	4.15	8.23	72.45	1.23	103.91	9,340,998
	TOTAL	1,500,000	11.04	5.48	5.57	76.89	0.94	86.12	26,430,203
8	A4 WNW	17,868	10.96	6.05	4.91	80.32	0.92	83.41	266,476
	A7 SW	541,722	10.72	5.23	5.49	73.93	0.81	83.23	7,977,788
	A14 S	263,795	12.38	4.15	8.23	72.45	1.23	103.91	9,340,998
	A1 NW	430,062	11.93	4.10	7.83	73.01	0.84	98.11	12,734,952
	A2 NW	246,553	10.07	5.52	4.55	73.64	0.56	74.98	1,596,844
	TOTAL	1,500,000	11.25	4.77	6.48	73.43	0.85	89.78	31,917,057

TABLE 13.B.6 (Continued)

MINING SCHEDULE BY YEARS - CASE B - 1.5 MILLION TONNES PER YEAR
DILUTED TONNES AND GRADES
OPERATING COST @ 68.50 Cdn$/TONNE

YEAR	BLOCK	TONNES	Pb+Zn	Pb	Zn	Ag	Au	N.S.R. Cdn$/T	NET REVENUE Cdn$
9	A4 ENW	548,262	10.96	6.05	4.91	80.32	0.92	83.41	8,176,548
	A7 SW	678,849	10.72	5.23	5.49	73.93	0.81	83.23	9,997,218
	A16 S	149,476	10.07	8.00	5.32	73.64	0.56	74.98	968,108
	A3 WNW	123,413	11.75	4.79	6.96	98.45	1.23	99.82	3,865,485
	TOTAL	1,500,000	10.83	5.77	5.38	78.25	0.86	83.84	23,007,359
10	A3 ENW	82,275	11.75	4.79	6.96	98.45	1.23	99.82	2,576,980
	A18 WNW	236,260	13.32	5.55	5.03	127.45	0.85	101.33	4,906,620
	Q6 NE	577,488	8.83	3.65	5.18	58.73	0.64	70.83	983,173
	A7 SW	354,823	10.72	5.23	5.49	73.93	0.81	83.23	5,225,379
	Q1 NE	249,154	8.95	2.81	6.14	29.00	0.48	70.02	379,095
	TOTAL	1,500,000	10.16	4.25	5.49	70.39	0.72	80.02	14,071,246
11	A7 SE	194,352	10.72	5.23	5.49	73.93	0.81	83.23	2,862,167
	Q6 NE	264,649	13.32	3.65	5.18	127.45	0.85	101.33	4,906,620
	Q5 NE	236,260	10.58	3.94	6.36	82.00	0.92	82.51	3,309,618
	A11 SW	207,472	8.97	3.52	5.45	45.85	0.68	70.95	509,228
	A5 NW	190,814	10.61	5.42	5.19	72.91	1.05	82.81	2,730,662
	Q7 NE	406,453	8.83	3.37	5.46	46.78	0.81	71.23	1,111,514
	TOTAL	1,500,000	10.39	4.03	5.52	73.27	0.85	81.31	15,429,809
12	A11 SE	138,314	8.97	3.52	5.45	45.85	0.68	70.95	339,483
	A7 SE	746,502	10.72	5.23	5.49	73.93	0.81	83.23	10,993,525
	Q5 NE	421,880	10.30	3.94	6.36	62.83	0.58	82.13	8,969,278
	A18 ENW	193,304	10.58	5.55	5.03	82.00	0.92	82.51	2,707,874
	TOTAL	1,500,000	10.42	4.75	5.67	69.26	0.75	81.70	23,010,161
13	A7 SE	284,452	10.72	5.23	5.49	73.93	0.81	83.23	4,189,045
	A6 SW	169,221	8.97	3.78	6.10	45.85	0.68	70.95	415,343
	A6 SE	87,175	10.61	3.78	6.10	72.91	1.05	82.81	2,730,662
	TOTAL	540,848	10.15	4.54	5.78	64.98	0.81	79.32	7,335,049

TABLE 13.B.7

BLOCK CHARACTERISTICS - CASE B - 1.5 MILLION TONNES PER YEAR

MINING YEAR	HOR.	BLOCK	AREA	THIN	THICK	Pb+Zn	TONNES TO MILL
2	U	B1	NE	X		8.50	248,100
1	U	B2	NE	X		10.03	104,500
1-3	L	B3	NE		X	14.52	1,023,600
2	L	B4	NE	X		16.82	256,000
2-5	L	B5	NE		X	12.38	2,424,000
4	L	B6	NE	X		9.85	204,400
3-4	L	B7	NE		X	12.02	710,300
3-4	L	B8	NE	X		9.08	141,500
	SUBTOTAL					12.55	5,112,400
10	U	Q1	NE	X		8.95	249,200
11-12	L	Q5	NE	X		10.30	658,100
10-11	L	Q6	NE		X	8.83	842,100
11	L	Q7	NE	X		8.83	406,400
	SUBTOTAL					9.29	2,155,800
8	U	A1	NW	X		11.93	430,100
8	U	A2	NW	X		10.07	246,600
9	U	A3	WNW	X		11.75	123,400
10			ENW	X		11.75	82,300
6-8	U	A4	WNW		X	10.96	1,644,800
9			ENW		X	10.96	548,300
11	U	A5	NW	X		10.61	190,800
10	L	A18	WNW		X	10.58	236,300
12			ENW		X	10.58	193,300
	SUBTOTAL					11.00	3,695,900
13	U	A6	SW	X		9.88	169,200
13	U		SE	X		9.88	87,200
5-10	U	A7	SW		X	10.72	3,150,800
11-13			SE		X	10.72	1,225,300
6	U	A8	S	X		17.07	266,600
6	U	A9	S	X		8.80	189,000
11	L	A11	SW	X		8.97	207,500
12			SE	X		8.97	138,300
7	L	A13	S	X		9.82	165,800
7-8	L	A14	S		X	12.38	527,600
9	L	A16	S	X		13.32	149,500
	SUBTOTAL					10.98	6,276,800
	TOTAL					11.24	17,240,900

Two sets of figures are worthy of special mention: the first 5 production years (actually seven years from The Decision).

	Tonnes Years 1-5 Millions	Grade Pb + Zn%	Net Revenue Can $ millions
Case A	4.800	12.58%	150.689
Case B	6.200	12.48%	176.198

13.5 Summary

Based on present knowledge of the deposit, the proposed plans, methods of working and subsequent results are considered to be appropriate.

The sequence of mining recommended maximises early mining of the higher grade ore, with corresponding earlier operating profits.

The basis for this forecast is predicated on good mine management and adequate training of personnel for what to many may be a new experience.

Of the two Cases studied, Case B looks more attractive for the following reasons:

- The location of the shaft in the middle of the deposit reduces the ore and backfill haul distances.
- The extra 300,000 tonnes of output per year greatly helps the annual revenue, particularly in the early years.
- The drivage of most drifts in ore reduces the amount and cost of mining in waste.
- Although requiring more ventilation, the use of trucks for ore (and waste) haulage, is more flexible and probably better suited for this type of ore for the moderate haulage distances.

14. CAPITAL COSTS

14.1 Introduction - Basis - Sources of Information, Accuracy, Contingency

Costs for Cases A and B have been developed along similar lines and comprise four major constituents:

1. Exploration Phase, up to "The Decision Day" - the decision to put the mine into production.
2. Construction Capital Requirement, for shaft sinking, underground development, power supply, etc.
3. Equipment Capital Requirements - almost all for underground use.
4. Total Capital Requirements, which is a summation of the previous three, with a 10% contingency added at the bottom. Outside the three earlier tables, a bond for closure is added as a special item, spread over years one and two.

14.1.1 Basis For the Report

Contractor's quotations have been used for ramp drivage, underground exploration, shaft sinking, as well as headframe and silo construction.

Equipment prices and specifications have been received from a number of companies which supply such equipment to the mining industry.

The provision for spare parts has not been included in these numbers. This cost category is being handled under a separate inventory account.

An average freight price is added to equipment to cover transportation from distribution point to the mine. A figure of 6% of purchase value is used.

No provision is made for GST, Yukon taxes or other burdens placed on mine operators.

With limited geological data from widely spaced drill holes, and no physical access to the ore horizons yet, it has not been possible to layout the mine plan or assess mining performance in designs which will be used when the deposit is mined. The normally acceptable accuracy for a study of this nature is ± 20%. The contingency figure of 10% is not part of this 20% figure, it is mainly put in to cover the small items which are required to augment the major items for which written estimates were not obtained.

The years have been referred to by reference to "The Decision Day". Minus values are years prior to this "Day" and positive ones come after.

14.2 Exploration Phase

This phase consists of the construction of the twin ramps from a portal in the Blind Creek Valley and underground drifting. Numbers have been supplied by Redpath and Procon, and other mining contractors.

Exploration drilling costs have been estimated by combining a typical drilling contractor price with the necessary geological, analytical and on-site services to support this work.

10% contingency is added. All the expenditures occur in years -3 to -1. The total is estimated at 24.480 million.

Two additional items are placed in this Pre-Decision Day period:

- Full Feasibility Cost estimate $1.8 million
- Mining Permit Application $1.0 million

The second item is difficult to estimate due to the many new processes and standards now being set by Provincial/Yukon and Federal jurisdictions.

14.3 Production Phase

14.3.1 Shaft

The shaft represents the start of major expenditures, and is planned for action as soon as possible after "Decision Day". Considerable data was supplied by Redpath to construct the shaft and headframe and install hoist, skips, etc. It was based on an 18-ft (4.5 metre) diameter shaft, 778 metres deep to shaft station, as proposed under Case A.

For estimates on Case B, sufficient details regarding cost items were available to calculate the difference in both money and time for a shaft 690 metres to the station (88 metres less). The price has been reduced from $25 million to $23.5 million, and the time from 26 months to 24 months.

14.3.2 Mine Development

Mine development has been outlined in Chapter 13. Case A entails considerable rock drivage from the twin access ramps to the shaft located outside the orebody to the Southwest. A number of conveyors are required. The costing is based on contract mining, with estimates based on proposals from the two main potential contractors for similar work.

In Case B, the central shaft location greatly reduces the amount of drivage. The use of some mine labour provides a good opportunity for the men to be trained, and also reduces costs. All the work can be done in year 2 for an estimated $5 million.

14.3.3 Other Areas

- Electrical installations
- Ventilation
- Mine Dewatering
- Backfill
- Underground shop
- Feasibility study
- Permits

14.4 Replacement Equipment

This is estimated for major items, with planned mine production of 14 or 11.5 years, the period has been divided in half for replacement of most heavy wear items like underground loaders, trucks, drills, etc.

14.5 Summary

The following table comprises a juxtaposition of the two Cases.

CAD $ Millions	Production mills t/year	Pre-Decision	Initial Capital Years 1-5	Replacement Capital	Total
Case A	1.2	27.28	85.16	18.61	131.05
Case B	1.5	27.28	74.89	20.24	122.41

Case B requires less capital than Case A, is quicker to bring into production, trains the local work force earlier, and provides a better return on investment.

Table 14.5-01 below shows this comparison by major areas of expense, for the Pre-Production period, initial capital up to year +5, and equipment replacement costs. The extra initial capital required in Case A, of almost $10 million, is mostly explained by the cost of more initial development drivage using contractors and installation of conveyors and crushers. The extra replacement costs in Case B are mainly due to truck replacement compared to very low conveyor replacement.

Table 14.5-01 Summary of Capital Expenditures

$ millions	Pre-Decision		Initial Capital (years 1-5)		Replacement Capital Year 6 to End	
	Case A	Case B	Case A	Case B	Case A	Case B
Ramp	14.0	14.0				
Explor. Drifts	5.5	5.5				
Drilling	4.7	4.7				
Shaft + Pilot Hole			26.1	24.5		
Surface bin			1.3	1.3		
Underground surge bin			2.6	2.6		
Underground Dev.			16.0	5.7		
Underground Conv.			4.1	.45	2.20	
Underground Equip.			21.6	20.4	11.93	15.70
Pumps/Fans			2.1	2.0		
Electrical			3.0	2.8	.37	.37
Backfill			2.0	2.0	.10	.10
Underground shop, etc.			.90	.90		
Bond on Closure			4.40	4.40		
Small Items			1.06	7.84	4.01	4.07
Feasibility/Permit	3.1	3.1				
Total	27.28	27.28	85.16	74.89	18.61	20.24

Table 14.A6.1
Grizzly Development Drivage
Case A

R = rock

Grizzly Decision
⇓

Year	-3 Drivage	-2	-1	+1	+2	+3 $000	Shaft	+4
000s dollars	metres					Ramp Used	in use	
Ramp conveyorway	350(½R)					1260		
Intake B2 (thin..........)	100 ¼R						360	
Raise B2	30R					109		
B3-1 conv. (2)	700R				1260	1260		
B3-2 FW(2)	700						2520	
B3-3 (HW(3)	1070						3852	
Ramps to HW	20						72	
Main conveyor	870(R)				3132			
Main access	350(R)				1260			
Rock removal from airshaft								
AW Raise B3	50m						180	
Backfill Drift	80(R)					288		
								Total
Cost Can$000					5652	2917	6984	15,553

Average Drivage
7m per day

R = Rock

Approx.
2 crews
2 years

			810	1940	Total
Metres	4,320	15%	2750		4,320 metres

Basis - no ore hauled up ramp
All contract mining

Cost Basis for drifting and raising
$3600/m, includes remuck, camp, etc.

Table 14.A6.2
Construction Capital Requirements
Case A

Decision ⇓

Can $000

Year	-3	-2	-1	+1	+2	+3	+4	+5	+6	+7	+8	+9	+10	+11	+12	+13	+14	+15	+16	+17	+18	Total
Production 000		30				300	900	1200	1200	1200	1200	1200	1200	1200	1200	1200	1200	1200	1200	1200	400	17200
Feasibility Study, etc.			1,800															Total After Decision				-
Mine Permit		500	500																			-
Ramp and UG explore	8,000	14,000																				-
Shaft + orebin, etc.				10,000	12,000	3,000																25,000
Fan + Fan drift House				80		260																340
Truck Load					500																	500
Shaft Drill Hole				200																		200
Storage Shaft Bottom					300	2,000																2,300
East (p) Return Airshaft, incl. rock removal					3,200	3,200																6,400
Power from Grum (installed)				500																		500
P. Equip. Surface (installed)					400	102																502
P. Equip. UG (installed					542	1,000																1,542
Upgrade Haul Rd						1,200																1,200
Backfill Plant (Surf.)						800																800
Backfill Lined Holes					400	600																1,000
Sump UG					100	100																200
Main Pump + Range					300	336																636
Ramp & Shaft Sump (210m)						780																780
Air Heating System					200	50																250
UG Workshop, etc.						400	200															600
UG Warehouse						100																100
Conveyor, Ramp to UG Bin					200	600																800
Sub-Total	8,000	14,500	2,300	10,780	18,142	14,528	200															43,650
Use 3% for freight/camp (½ normal)-			-	323	543	440	6															1,310
TOTAL	8,000	14,500	2,300	11,103	18,685	14,968	206															**44,960**
	Total Till Decision 24,800			Total After Decision 44,960																		

Table 14.A6.3
Equipment Capital Requirements
Case A

Decision ⇓ (at year -1)

Can $000

Year	-3	-2	-1	+1	+2	+3	+4	+5	+6	+7	+8	+9	+10	+11	+12	+13	+14	+15	+16	+17	+18	Total
Production 000						300	900	1200	1200	1200	1200	1200	1200	1200	1200	1200	1200	1200	1200	1200	400	17200
Twin drill jumbos (4)					600	1,200	600					600	600	1,200								4,800
Prod. loaders (5)					630	1,890	630					630	630	630								5,040
Bolter, Elec. (4)					520	520	1,040					520	520	1,040								4,160
Bench Drill (2)						520	520						520	520								2,080
26t Truck for fill (2)						430	430						430	430								1,720
Small scoop 3x (3)						640	320						640	320								1,920
Conveyor 42"						1,900	400	650	200	200	410		-	475	200	350						4,785
Anfo loader (2)						360	360						200	200	reb'ld							1,120
Man carrier (2)						200							100	100								400
Scissor lift (2)						380																380
Mech. late truck (1)						230								230								460
Supervisors vehicles (6)						150	150						150	150								600
Crusher/rock braker (5)						600	400						300	200	reb'ld							1,500
Road grader (1)						350								200	reb'ld							550
UG core drill (2)					120	120						120	120									480
Stopers, jackl.., etc.-1 lot						250																250
Compressor, portable (2)						120							100	100	reb'ld							320
Mechanics, truck (2)						300							60		reb'ld							360
FF pipe - valve, etc.						250																250
Cable replacement/ext.						-	120	50	50	50	50	50	50	50	50							520
Fans & vent. tube (10)						60	70								vent tube							130
Face pumps & pipe (4)						50	80								as op. cost							130
Explosives, Mag (1)						30																30
Backfill UG silo, etc.						300								100								400
Misc.												250	250	250	250	250	250	250	250	100		2,100
Sub-Total					1,870	10,850	5,120	700	250	250	460	2,170	4,670	6,195	500	600	250	250	250	100		34,485
Freight 6%					112	651	307	42	15	15	28	130	280	371	30	36	15	15	15	6		2,088
TOTAL					1,982	11,501	5,427	742	265	265	508	2,300	4,950	6,566	530	636	265	265	265	106		36,573

Table 14.A6.4
Total Capital Requirements
Case A
Can $000

Decision ⇓ (between year -1 and +1)

Year	-3	-2	-1	+1	+2	+3	+4	+5	+6	+7	+8	+9	+10	+11	+12	+13	+14	+15	+16	+17	+18	Total
Production 000		30				300	900	1200	1200	1200	1200	1200	1200	1200	1200	1200	1200	1200	1200	1200	400	17200
																			Total After Decision			
Page 1(Table 14.A6.1)					5,652	9,901																15,553
Page 2 (Table 14.A6.2)	8,000	14,500	2,300		11,103	15,339	11,567	206														38,215
Page 3 (Table 14.A6.3)					1,982	11,501	5,427	742	265	265	508	2,300	4,950	6,566	530	636	265	265	265	106	-	36,573
Sub-Total	8,000	14,500	2,300		18,737	36,741	16,994	948	265	265	508	2,300	4,950	6,566	530	636	265	265	265	106		90,341
Bond for Closure				2,000	2,000																	4,000
Contingency Small items 10%	800	1,450	230	200	2,074	3,674	1,699	95	26	26	51	230	495	657	53	64	26	26	26	11		9,433
TOTAL	8,800	15,950	2,530	2,200	22,811	40,415	18,693	1,043	291	291	559	2,530	5,445	7,223	583	700	291	291	291	117		103,774
	Total Till Decision 27,280			INITIAL CAPITAL 85,162																		

Table 14.B6.1
Grizzly Development Drivage
Case B

R = rock

Grizzly Decision ⇓

Year	-3	-2	-1	+1	+2	+3	+4	+5	+6
000s dollars									
Tasks to do									
(from Schedule on Table 13-4)									
Rock Drift NE exploration to East raises			150m						
Rock Drift East raises to Lower lovel of B3			150m						
West Access to new shaft			180m						
Shaft sump ramp			330m						
Shaft bottom bin conv. ramp			120m						
Sump for main pump									
Hourly paid at work each day (ave)					33.5				
Hourly paid cost x 2					4.00				
Staff Cost					1.00				
Cost Can$000					5.00				

Capitalised

Average Drivage
5m per day in first 1.5 years

Basis - no ore hauled up ramp
All employees ARMC

Cost Basis
Year 2 Wage cost x 2 by men at work each quarter + $1 million for staff

Table 14.B6.2
Construction Capital Requirements
Case B

Decision ⇓ (between -1 and +1)

Can $000

Year	-3	-2	-1	+1	+2	+3	+4	+5	+6	+7	+8	+9	+10
Production 000		30				300	900	1200	1200	1200	1200	1200	1200
Feasibility Study, etc.			1,800										
Mine Permit		500	500										
Ramp and UG explore	8,000	14,000											
Shaft + orebin, etc.				11,500	12,000								
Fan + Fan drift House				80	260								
Truck Load					500								
Shaft Drill Hole				200									
Storage Shaft Bottom					2,300								
East (p) Return Airshaft, rock removal in mine shaft cost					3,000	3,000							
Power from Grum (installed)				500									
P. Equip. Surface (installed)					400	102							
P. Equip. UG (installed					542	1,000							
Upgrade Haul Rd						1,200							
Backfill Plant (Surf.)						800							
Backfill Lined Holes					400	600							
Sump UG (in Table 14.B6.1)													
Main Pump + Range					300	336							
Ramp & Shaft Sump (210m) 330m. incl. in ramp to SW.						660							
Air Heating System					200	50							
UG Workshop, etc.						400	200						
UG Warehouse						100							
Conveyor, Ramp to UG Bin					200	600							
Sub-Total	8,000	14,500	2,300	12,280	19,902	5,188	200			Total After decision			
Use 3% for freight/camp (½ normal)-			-	368	597	155	6			38,696			
TOTAL	8,000	14,500	2,300	12,648	20,499	5,343	206						

Total Till Decision
24,800

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Table 14.B6.3
Equipment Capital Requirements
Case B

Decision ⇓ (at +1)

Can $000

Year	-3	-2	-1	+1	+2	+3	+4	+5	+6	+7	+8	+9	+10	+11	+12	+13	+14	+15	Total
Production 000						500	1200	1500	1500	1500	1500	1500	1500	1500	1500	1500	1500	540	17200
Twin drill jumbos (4)				600	600	600	600				600	600	600	600					4,800
Prod. loaders (5)				630	630	1,260	630			630	630	630	630	630					6,300
Bolter, Elec. (4)				520	520	1,040				520	520	520	520						4,160
Bench Drill (2)				nil	520	520						520	520						2,080
26t Truck for fill (2)				430	430							430	430						1,720
Small scoop 3x (3)				320		320	320						640	320					1,920
Ore Trucks 40t (4)					640	1,280	640			640		640	1,280						5,120
Anfo loader (2)						360	360					200	200		reb'ld				1,120
Man carrier (2)						200						100	100						400
Scissor lift (2)						380						190	190						760
Mech. late truck (1)						230								230					460
Supervisors vehicles (6)						150	150					150	150						600
Crusher/rock braker (1) large installed					1,500							200	300		reb'ld				2,000
Road grader (1)						350							200		reb'ld				550
UG core drill (2)					120	120					120	120							480
Stopers, jackl.., etc.-1 lot						250													250
Compressor, portable (2)						120						100			reb'ld				220
Mechanics, truck (2)						300							60		reb'ld				360
FF pipe - valve, etc.						250													250
Cable replacement/ext.							120	50	50	50	50	50	50	50	50				520
Fans & vent. tube (10)						60	70								vent tube				130
Face pumps & pipe (4)						50	80								as op. cost				130
Explosives, Mag (1)						30													30
Backfill UG silo, etc.						300								100					400
Misc.				300	300							250	250	250	250	150	100		1,850
Sub-Total				2,800	5,260	8,170	2,950	50	50	1,840	1,920	4,700	6,120	2,180	300	150	100		36,590
Freight 6%				168	316	490	177	3	3	110	115	282	367	131	18	9	6		2,195
TOTAL				2,968	5,576	8,660	3,127	53	53	1,950	2,035	4,982	6,487	2,311	318	159	106		38,785

Table 14.B6.4
Total Capital Requirements
Case B

Decision ⇓ (between -1 and +1)

Can $000

Year	-3	-2	-1	+1	+2	+3	+4	+5	+6	+7	+8	+9	+10	+11	+12	+13	+14	+15	Total
Production 000		30				500	1200	1500	1500	1500	1500	1500	1500	1500	1500	1500	1500	540	17200
																Total After Decision			
Page 1 (Table 14.B6.1)					5,000														5,000
Page 2 (Table 14.B6.2)	8,000	14,500	2,300	12,648	20,499	5,343	206												38,696
Page 3 (Table 14.B6.3)				2,968	5,576	8,660	3,127	53	53	1,950	2,035	4,982	6,487	2,311	318	159	106		38,785
Sub-Total	8,000	14,500	2,300	15,616	31,075	14,003	3,333	53	53	1,950	2,035	4,982	6,487	2,311	318	159	106		82,481
Bond for Closure				2,000	2,000														4,000
Contingency Small items 10%	800	1,450	230	1,762	3,307	1,400	333	5	5	195	203	498	649	231	32	16	10		8,646
TOTAL	8,800	15,950	2,530	19,378	36,382	15,403	3,666	58	58	2,145	2,238	5,480	7,136	2,542	350	175	116		95,127
	Total Till Decision 27,280			INITIAL CAPITAL 74,887															

Table 14.01
Grizzly Development Drivage
Case A

R = rock

Grizzly Decision
⇓

Year	-3 Drivage	-2	-1	+1	+2	+3 $000	Shaft	+4
000s dollars	metres				Ramp Used		in use	
Ramp conveyorway	350(½R)					1260		
Intake B2 (thin..........)	100 ¼R						360	
Raise B2	30R					109		
B3-1 conv. (2)	700R				1260	1260		
B3-2 FW(2)	700						2520	
B3-3 (HW(3)	1070						3852	
Ramps to HW	20						72	
Main conveyor	870(R)				3132			
Main access	350(R)				1260			
Rock removal from airshaft								
AW Raise B3	50m						180	
Backfill Drift	80(R)					288		
								Total
Cost Can$000					5652	2917	6984	15,553

Average Drivage
7m per day

R = Rock

Approx.
2 crews
2 years

			810	1940	Total
Metres	4,320	15%	2750		4,320 metres

Basis - no ore hauled up ramp
All contract mining

Cost Basis for drifting and raising
$3600/m, includes remuck, camp, etc.

Table 14.02
Construction Capital Requirements
Case B

Decision ⇓ (between -1 and +1)

Can $000

Year	-3	-2	-1	+1	+2	+3	+4	+5	+6	+7	+8	+9	+10
Production 000		30				300	900	1200	1200	1200	1200	1200	1200
Feasibility Study, etc.			1,800										
Mine Permit		500	500										
Ramp and UG explore	8,000	14,000											
Shaft + orebin, etc.				11,500	12,000								
Fan + Fan drift House				80	260								
Truck Load					500								
Shaft Drill Hole				200									
Storage Shaft Bottom					2,300								
East (p) Return Airshaft, rock removal in mine shaft cost					3,000	3,000							
Power from Grum (installed)				500									
P. Equip. Surface (installed)					400	102							
P. Equip. UG (installed					542	1,000							
Upgrade Haul Rd						1,200							
Backfill Plant (Surf.)						800							
Backfill Lined Holes					400	600							
Sump UG (in Table 14.B6.1)													
Main Pump + Range					300	336							
Ramp & Shaft Sump (210m) 330m. incl. in ramp to SW.						660							
Air Heating System					200	50							
UG Workshop, etc.						400	200						
UG Warehouse						100							
Conveyor, Ramp to UG Bin					200	600							
Sub-Total	8,000	14,500	2,300	12,280	19,902	5,188	200			Total After decision			
Use 3% for freight/camp (½ normal)-			-	368	597	155	6			38,696			
TOTAL	8,000	14,500	2,300	12,648	20,499	5,343	206						

Total Till Decision
24,800

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Table 14.03
Grizzly Development Drivage
Case B

R = rock

Grizzly Decision ⇓

Year	-3	-2	-1	+1	+2	+3	+4	+5	+6
000s dollars Tasks to do (from Schedule on Table 13-4) Rock Drift NE exploration to East raises 150m Rock Drift East raises to Lower lovel of B3 150m West Access to new shaft 180m Shaft sump ramp 330m Shaft bottom bin conv. ramp 120m Sump for main pump									
Hourly paid at work each day (ave)					33.5				
Hourly paid cost x 2					4.00				
Staff Cost					1.00				
Cost Can$000					5.00				

Capitalised

Average Drivage
5m per day in first 1.5 years

Basis - no ore hauled up ramp
All employees ARMC

Cost Basis
Year 2 Wage cost x 2 by men at work each quarter + $1 million for staff

15. OPERATING COSTS

15.1 Introduction - Basis - Sources; Degree of Optimization

The major expenditure in a highly mechanised operation is skilled labour. It is addressed in detail in the next sub-section.

The source of information to produce the other costs has been a combination of preliminary engineering, equipment suppliers, commodity suppliers, various reports, mining contractors such as Procon Mining and Tunnelling Ltd., and Redpath Ltd.

All costs have been based on third quarter 1996 prices.

When using earlier reports (Curragh Resources for Faro underground, Canadian Mine Development) the Consumer Price Index (CPI) for Whitehorse was used to escalate the numbers, mostly from 1989 and 1992 (1.202885 and 1.080162 respectively).

For items such as electricity and diesel fuel, the current tariffs and bulk delivery prices for the Grum mine have been used.

The expected operating conditions have formed the basis for the costs associated with these activities. As explained in Chapter 12, Mining Methods, the orebody is variable in thickness and roof condition. The average has been used when calculating the quantities of such items as explosives and roof supports.

Twelve itemised categories have been isolated for examination in considerable detail. A further miscellaneous category (13) has been added to cover smaller cost items.

Operating costs have been calculated for the "average" situation in the mine after full production has been achieved, such as in Year 5 when the NE area is being depleted and major development is in progress to prepare the South West Area to replace half the production.

As described in Chapter 13, it is hoped to "average" one mining unit in thin ore, and one in thick. This will not always be possible. Even when mining the thinner stopes, some production will be from development drivage, and some from robbing pillars where minimum support costs will be incurred. The budget is for 3.5 production crews and one development crew. 8 extra men have been added in Case A to cover the extra development in waste required for the conveyor drifts in the steep areas.

15.2 Labour Cost Estimate

In this estimate a typical situation is used to plan for the labour required. This translates into one thick and one thin stoping unit being at work. In addition, a development crew and backfill crew will be at work. One extra production unit is required to bring the total daily to near the average required daily production..

A wage cost of $2 to $3 above the present cost at Faro has been assumed. For the following reasons, a round figure of $32 per hour is adopted for all underground workers:

- Union negotiations at the rest of the Anvil Range operations are currently under way.
- Contract mining may be used in part or in full for a few years.
- Some people in the present work force will have to be trained in underground work.
- Miners will have to be recruited.
- Bonus payments may or may not be introduced (while mining contractors generally work with a bonus system, it is not recommended for a highly mechanized mining operation).

The $32 per hour includes the following:

- Standard eight-hour day, averaging 5 days per week, but spread over the seven days.
- Any overtime which may be worked (at least 1 shift per 28-day cycle).
- Benefits provided by the company for health insurance, etc.
- Statutory holiday payments (12 per year) plus an average of 15 days paid vacation.
- Employment premiums, such as for underground work and shift differential.

On this basis during the 52-week year, 12 days will be deducted for statutory holidays and 15 for annual leave, leaving 233 days to be paid.

Cost per man year = 233 x $32 x 8 = $59,648

The tables of manpower (Table 15.1 and 15.2) are divided into two columns - those who work 5 days per week and those who work over the seven days, mostly hourly paid people. In addition, a list of staff with expected cost to the company has been prepared as Table 15.3.

In both cases, by coincidence, the total complement of jobs is 206. In case A, 8 extra men are used for the extra conveyor drifts in waste rock. In Case B, the saving in rock drivage, small crushers and conveyor maintenance are replaced by 12 truck drivers and 2 more on shop maintenance.

The total estimated labour costs are:

Production	206 men @ $59,648 =	$12,287,488 per year
Staff	as Table 15.3	$ 1,850,000
Total Cost per year		$14,137,488
Cost per tonne of ore		$11.78

Table 15.1 Manpower Requirements - Case A

Jobs	Crew	5-day week	7-day week men per day	Work Force
Production miners	3½ x 4		42	56
Development miners	1½ x 4		18	24
Mechanics	5		15	20
Electricians	2		6	8
Grade control	1		3	4
Crusher attendant	2		6	8
Supply men/grader	2		6	8
Construction	2		6	8
Backfill - surface	1		3	4
- underground	2		6	8
Control centre	1		3	4
Ventilation		2		2
Pipes - pumps		2		2
Drift maintenance		2		2
UG diamond drill	2		6	8
Geologist	1		3	4
Shift Captain	1		3	4
Section Foreman	2		6	8
Conveyor maintenance	2			2
Shaft	1		3	4
UG shop foreman		1		1
UG shop mechanics	10			10
UG shop electricians	6			6
Surface		1		1
		8	135	206 jobs

Table 15.2 Manpower Requirements - Case B

Jobs	Crew	5-day week	7-day week men per day	Work Force
Production miners	3½ x 4		42	56
Development miners	1 x 4		12	16
Mechanics	5		15	20
Electricians	2		6	8
Grade control	1		3	4
Truck drivers	3		9	12
Supply men/grader	2		6	8
Construction	2		6	8
Backfill - surface	1		3	4
- underground	2		6	8
Control centre	1		3	4
Ventilation		2		2
Pipes - pumps		2		2
Drift repair		2		2
UG diamond drill	2		6	8
Geologist	1		3	4
Foremen	3		9	12
Crusher attendant	1		3	4
Shaft	1		3	4
UG shop foreman		1		1
UG shop mechanics	12			12
UG shop electricians	6			6
Surface		1		1
		8	135	206 jobs

Table 15.3

Staff Requirements for Cases A and B

	Salary and Benefits
Mine Manager	130,000
Mine Superintendent	110,000
Mechanical Engineer	100,000
Electrical Engineer	100,000
Chief Mining Engineer	100,000
Chief Geologist	100,000
Environmental Engineer	100,000
Senior Mining Engineer	90,000
Junior Mining Engineer (3)	180,000
Safety Engineer	90,000
Geologist & Technician	150,000
Accountant & Assistant	150,000
Purchasing & Assistant	150,000
Warehouse (2)	120,000
Transportation Foreman	80,000
Secretary & Clerical	100,000
	1,850,000

Table 15.4

Staff Requirements

	Salary and Benefits
Mine Manager	130,000
Mine Superintendent	110,000
Mechanical Engineer	100,000
Electrical Engineer	100,000
Chief Mining Engineer	100,000
Chief Geologist	100,000
Environmental Engineer	100,000
Senior Mining Engineer	90,000
Junior Mining Engineer (3)	180,000
Safety Engineer	90,000
Geologist & Technician	150,000
Accountant & Assistant	150,000
Purchasing & Assistant	150,000
Warehouse (2)	120,000
Transportation Foreman	80,000
Secretary & Clerical	100,000
	1,850,000

15.3 Materials Cost Estimate

15.3.1 Roof Support

The estimates are based on the use of six foot long splitset bolts with mesh in room and pillar stopes, equivalent to 6 bolts per linear metre for rooms 8 metres wide (4x4 feet spacing); cost: $116 per linear metre; and in high stopes, 8 foot rebar with resin will be used on similar centres; cost $120 per linear metre.

The cost per tonne is \$0.739 for room and pillar and \$0.319 for high stopes.
Average (weighted 29% room and pillar, 71% high) = \$0.441 per tonne.

For pillar robbing, fewer bolts are expected to be required, but this will balance out any extra which may be required for dealing with rib bolting, especially in the thick stopes. The average cost in development drifts is another twenty cents per tonne of full production.

In both cases, due to expected difficult roof conditions experienced from time to time in the Faro underground mine and indicated faulting at Grizzly, an additional 16 cents per tonne are added, to bring the total support cost estimate to 80 cents per tonne. This extra provision is included to cover costs of cable bolting, additional straps, even some steel or shotcrete.

15.3.2 Backfilling

The cost estimate is based on the system described in section 12.4 and from a conversation with the chief mining engineer of Barrick's Bousquet mine.

For both cases, the source of material is expected to be:

screened local gravel (within 3 kms)	85%, cost at pipe to UG	1.00/t fill
tailings from the concentrator	11%	5.00/t
cement	4%	330.00/t
	average	14.60/t fill

Underground, cement and chemicals are added, the chemicals estimated to cost \$1 per ton of fill.

Hauling and placing of fill will be done by truck with ram packing by a medium sized scoop equipped with a ram attachment in the bucket. The cost of these two operations, 75 cents and 50 cents per tonne, brings the total cost of cemented backfill to \$15.85 per tonne of fill.

Assume ore has a specific gravity of 3.92; for 100 tonnes of ore, volume =25.51 m^3.
The specific gravity of the solids in the backfill is 2.9.
With an estimate of 20% voids, the material has a specific gravity of 2.35.
Tonnes required per 100 tonnes of ore is 25.51 x 2.35 = 60 t of fill.

In Case A, the average daily backfilled amount of replaced ore = 3500 x .71 = 2485 tonnes
50% requires cemented backfill, or production of 1243 tonnes per day.

Total cemented backfill required $= 1243 \times \frac{60}{100} = 746$ tonnes

The remaining backfill requirement is for filling the opening left after the final pillar extraction. No cement is required in this operation, just plain waste from within the mine, estimated at 400 tonnes per day, plus 346 tonnes of gravel from the surface.

Cost of uncemented backfill is estimated at	$1.00/t	surface (labour extra)
+	1.75	u/g (labour extra)
	$2.75	

Assume mine waste provides 400 tonnes/day at a cost of $1.75/t (materials) or $700/day, cost/day for 346 tonnes of waste from surface is 346 times $2.75 per tonne or $951/day. Total of $1651 per day.

Therefore cost of cemented fill is 746 x 15.85 per day	=	$11,824
and the cost of uncemented backfill is		$ 1,651/day
Total cost per day:		$13,475/day

The cost of cemented fill is $3.85 per tonne of ore for Case A.

In Case B, one of the parameters which changes is the cost of cement landed at the mine. Starting from a price of $125 per tonne for bulk cement at Vancouver, the landed cost of $200 has been chosen.

Starting from the source of material of $14.60 per tonne of fill, the $200 per tonne cost of cement lowers this figure to $9.40 per tonne; adding the haul costs, etc. the cost rises to $11.65/tonne of fill.

Using the same proportions material, the 746 tonnes cost $ 8,690.90 per day

and mine waste	$ 1,650.00 per day
Total	$10,341.90 per day

on the same 3,500 tonnes per day, the cost of cemented backfill is $2.95 per tonne of ore. On a larger tonnage, the price per tonne of ore remains the same.

A comparison of other mines in Canada in 1995 revealed a cost of $0.53 to $5.62 per tonne of ore.

15.3.3 Crushing/Rock Breaking

In Case A, jaw crushers at the bottom of ore passes, or rock breakers above,will be used to reduce to 6 inches (150 mm) the size of lumps loaded onto the conveyor.

In Case B, a central jaw crusher will do the same job.

In both cases the estimated cost is 15 cents per tonne.

15.3.4 Fuel and Lubricants

The three main areas of consumption are:

- Diesel fuel for underground scoop trams, haul trucks, and service vehicles
- Propane for air heating
- Greases for all machinery maintenance

Fuel Costs - Estimate for Case A

			hours/day	fuel/day litres	cost/day	cost/t
Basis	5 scoops 300 h.p.	30 l/hr	15	2250		
	2 scoops 185 h.p.	20 l/hr	15	600		
	2 trucks 180 h.p.	28	12	432		
	2 scissors 82 h.p.	8	6	96		
	2 man vehicles 82 h.p.	8	6	96		
	1 grader	15	5	75		
	4 supervisors 30 h.p.	2	3	24		
	1 Anfo loader.	8	9	72		
	1 lube truck	8	12	72		
	1 diesel generator	4	10	40	918.9	
	Fuel at 44 cents/litre			3750	1650	.471
				(Faro	261 x 1.08 = .283)	

Fuel Costs - Estimate for Case B

			hours/day	fuel/day litres	cost/da y	cost/t
Basis	3 trucks 450 h.p.	45 l/hr	18	2430		
	5 scoops 300 h.p.	30 l/hr	15	2250		
	2 scoops 185 h.p.	20 l/hr	15	600		
	2 trucks 180 h.p.	28	12	432		
	2 scissors 82 h.p.	8	6	96		
	2 man vehicles 82 h.p.	8	6	96		
	1 grader	15	5	75		
	4 supervisors 30 h.p.	2	3	24		
	1 Anfo loader.	8	9	72		
	1 lube truck	8	12	72		
	1 diesel generator	4	10	40	918.9	
	Fuel at 44 cents/litre			6180	2719	.630

		Scheme A cost/t *	Scheme B cost/t *
Propane- CMD estimate of .47 cents per tonne, - escalated by Consumer Price Index (CPI) 1.08016		.507	.566
- Faro U/G mine in October 1992 was $0.20/t			
- Faro U/G mine in October 1992 was $0.20/t			
- Redpath		1.01	
- For this estimate a figure of .507 is used			
Lubrication - Faro 318 escalated		.344	.360
Caterpillar Handbook of 12 similar machines		.290	
Here the estimate is		.346	
Total Fuels and Lubricants: includes GST + other Taxes			
	Diesel	.471	.630
(Diesel total taxes = 14.1 cents/litre)	Propane	.507	.663
	Lube	.346	.360
	Misc.	.100	.100
		1.424	1.753

*** Cost per tonne**

15.3.5 Explosives

No statistical information was available from the underground Faro operation. Therefore, an assessment was made based on other similar types of ore and mining methods.

In Scheme A, the extra rock drivage lifted the cost per tonne of ore to $2.00. In Scheme B, the estimate is $1.80 per tonne mined, including an average for development drivage in waste during a normal year.

15.3.6 Drill Rods and Bits

For both Schemes, these items were selected as a cost centre and estimated at forty five cents per tonne of production.

15.3.7 Parts for Mining Equipment

Ten percent of the capital cost spent on equipment was expected to be required to purchase spare parts each year. This figure may be less in early years and more as the machinery ages.

In Scheme A, the capital cost of initial equipment is estimated at $21,700. Ten percent of this number divided by the annual tonnage works out at $1.81 per tonne.

In Scheme B, a smaller capital of $20,400 spread over a larger tonnage, reduces the cost per tonne to $1.36.

15.3.8 Shaft Operations

Estimates by CMD and Redpath indicated that about 29 cents per tonne would be realistic for maintenance and replacement in this important area. The cost is estimated to be the same for both levels of production.

15.3.9 Conveyors

This charge only applies to Scheme A. Much of the cost of conveyors is in replacement belting. If the equipment is well designed and installed, there should be a minimum amount of replacement belting which has to be purchased.

An average $360,000 per year is budgeted for this area, or thirty cents per tonne.

15.3.10 Power

The calculations of power consumption are shown in Table 15.4 and 15.5 for Case A and Case B respectively. The major changes are in main fans, conveyors and the hoist. The loads and duration of service are estimated for a full year of production.

The cost of power has been based on a four component tariff:

A. Fixed Charge $10,724/month $18.60

B. Peak Power Demand Charge $107,880 per KVA

C. Charger per KW Hour $0.04/KWH

D. Rider and Revenue Shortfall

Average calculated from present mining operations is $0.074 per kilowatt hour.

The average cost over the mine life in Case A is estimated at $1.68 per tonne of ore and in Case B at 1.35 per tonne of ore.

Table 15.4

Power Costs

$000

at 7.4 cents/KWH combined 4 tariffs

Case A

		loading	hours/year	kwh/yr 000	cost/year $000	cost/year $000
main fan surface	1200 kw	90%	8400	9072	671.0	
misc. orebin, etc. and compressor	120 kw	50%	8760	526	38.9	
					709.9	709.9
Shaft bottom						
Conveyor	3x150	60%	7000	1260	139.8	
UG shop, etc.	285	50%	8760	1248	92.3	
Pump	2x180	65%	8760	2650	151.7	
					383.8	383.8
NE first unit						
2 jumbos/bench	216	50%	4500	486	35.9	
2 bolter	90	50%	4000	90	6.6	
2 crushers	240	25%	6000	360	26.6	
1 conv.	150	50%	6000	450	33.3	
4 fans 30kw	120	60%	7000	504	37.3	
					139.7	139.7
NE second unit - ditto						139.7
Dev. drift						
1 jumbo	108	50%	4500	243	18.0	
1 bolter	45	50%	4500	101	7.5	

		loading	hours/year	kwh/yr 000	cost/year $000	cost/year $000
1 crusher	120	25%	6000	180	13.3	
1 compressor	60	60%	7000	252	18.7	
2 fans 60 kw	120	60%	7000	504	37.3	
2 pumps	120	50%	6000	36	26.6	
3 small pumps	45	50%	7000	157	11.7	
Sub-Total					133.1	133.1
Hoist 1500 kw	1500	70%	6500	6825	505.0	505.0
Total						2,011.2
Cost per tonne of ore						$1.68

Table 15.5

Power Costs

$000

at 7.4 cents/KWH combined 4 tariffs

Case B

		loading	hours/year	kwh/yr 000	cost/year $000	cost/year $000
main fan surface	1700 kw	90%	8400	12852	951.0	
misc. orebin, etc. and compressor	120 kw	50%	8760	526	38.9	
					989.9	989.9
Shaft bottom						
Conveyor	nil					
UG shop, etc.	285	50%	8760	1248	92.3	
Pump	2x180	65%	8760	2650	151.7	
					244.0	244.0
NE first unit						
2 jumbos/bench	216	50%	4500	486	35.9	
2 bolter	90	50%	4000	90	6.6	
2 crushers	240	25%	6000	360	26.6	
1 conv.	nil					
4 fans 30kw	120	60%	7000	504	37.3	
					106.4	106.4
NE second unit - ditto						106.4
Dev. drift						
1 jumbo	108	50%	4500	243	18.0	
1 bolter	45	50%	4500	101	7.5	
1 crusher	120	25%	6000	180	13.3	

		loading	hours/year	kwh/yr 000	cost/year $000	cost/year $000
1 compressor	60	60%	7000	252	18.7	
2 fans 60 kw	120	60%	7000	504	37.3	
2 pumps	120	50%	6000	36	26.6	
3 small pumps	45	50%	7000	157	11.7	
Sub-Total					133.1	133.1
Hoist 1500 kw	1500	70%	7500	6825	582.7	582.7
Total						2,029.4
Cost per tonne of ore						$1.35

15.3.11 Sub-Contractors

Work to be performed by contractors is expected to fall in the following areas:

- Haulroad repair or snow removal
- Extra diamond drilling
- Crane service (possibly)
- Vulcanising conveyor belts
- Consultants of various disciplines
- Extra rock excavation
- Special shaft and hoist advice, maintenance or installation

An estimate of 32 cents per tonne is used in both Schemes.

15.3.12 Miscellaneous Costs

This category covers all expenditures outside the classifications mentioned above. It would include safety items, safety competitions, underground exploration, staff travel, stationery, seminars, pipes, cables, office equipment, cost of hiring staff, etc.

The cost is estimated at sixty cents per tonne for both Schemes.

15.4 Total Operating Costs

15.4.1 Total Operating Costs - Case A

In summary, a table has been prepared (Table 15.6) which shows the main components which comprise the total estimated operating costs of $25.45 per tonne of ore at full production.

Table 15.4.1

Operating Cost Estimates

1.2 million tonnes per year

Case A

	ITEM	Cost per tonne mined
1	Labour: 206 daily paid, 23 staff	$11.78
2	Roof support	.80
3	Backfill	3.85
4	Crusher/rock breaking	.15
5	Fuel and lubricants (incl. propane)	1.42
6	Explosives	2.80
7	Drill rods and bits	.45
8	Parts for mining equipment	1.81
9	Shaft operations	.29
10	Conveyors	.30
11	Power	1.68
12	Sub-contractors diamond drilling, road work, cleaning	.32
13	Miscellaneous costs, cables, pipes, safety, core drill, etc.	.60
	Total Estimate per tonne	**$25.45**

15.4.2 Total Operating Costs - Case B

In summary, a table has been prepared (Table 15.7) which shows the main components which comprise the total estimated operating costs of $23.60 per tonne of ore at full production.

Table 15.4.2

Operating Cost Estimates

1.5 million tonnes per year

Case B

	ITEM	Cost per tonne mined
1	Labour: 206 daily paid, 23 staff	$11.78
2	Roof support	.80
3	Backfill	2.95
4	Crusher/rock breaking	.15
5	Fuel and lubricants (incl. propane)	1.75
6	Explosives	1.80
7	Drill rods and bits	.45
8	Parts for mining equipment	1.36
9	Shaft operations	.29
10	Conveyors	nil
11	Power	1.35
12	Sub-contractors diamond drilling, road work, cleaning	.32
13	Miscellaneous costs, cables, pipes, safety, core drill, etc.	.60
	Total Estimate per tonne	**$23.60**

15.5 Operating Cost by Year

15.5.1 Operating Cost by Year - Case A

As illustrated in the Schedule, it is only planned to produce 300,000 tonnes in the first year of operation, which is due to construction taking an estimated four months of that year. As in most mines, time is allowed to bring on many new employees all of whom will require some familiarisation, and many safety, operating or maintenance training. In addition, many staff will be on site before production begins, to prepare their departments, set up programs and participate in joint discussions on the new operations.

In the second year, it is assumed that the manning, on average will be 90% of the full complement. In this year, it is expected to produce 75% of the rated full production of 1,200,000 tonnes per year. Costs other than labour are expected to be at the normal full production level.

Thereafter, at this level of study, it is assumed that operating costs per tonne will remain constant. To achieve this, it will be necessary for the overall efficiency of the operation to increase as the equipment ages and requires more maintenance which leads to the use of more spare parts.

Table 15.5.1 shows the operating cost estimates for the first three years.

15.5.2 Operating Cost By Year - Case B

Due to the shorter shaft and larger final annual output of 1.5 million, the production is planned as follows:

as follows:	Production Year 1	500,000 tonnes
	Production Year 2	1,200,000
	Production Year 3	1,500,000

As in Case A, the same comments apply for the reasons for the build up in production. However, the percentages of the full costs are somewhat different from those of Case A because of the different percentage of tonnage in each of the build-up years.

Table 15.5.1

Early Operating Costs

1.2 million tonnes per year

Case A

Year (Project)	+3	+4	+5
Year of operation	1	2	3
Production (000) t	5	1,200	1,500
Production - % of full	33%	80%	100%
Labour - % of full	40%	100%	100%
Supplies -% of full	50%	100%	100%
Labour $ millions	5,654	12,722	14,136
Supplies $ millions	8,202	16,404	16,404
Total $ millions	13,856	29,126	30,540
Cost per tonne of ore	$46.19	$32.36	$25.45

Table 15.5.2

Early Operating Costs

1.5 million tonnes per year

Case B

Year (Project)		+3	+4	+5
Year of operation		1	2	3
Production (000) t		5	1,200	1,500
Production	- % of full	33%	80%	100%
Labour	- % of full	40%	100%	100%
Supplies	-% of full	50%	100%	100%
Labour	$ millions	7,068	17,670	17,670
Supplies	$ millions	8,865	17,730	17,730
Total	$ millions	15,933	35,400	35,400
Cost per tonne of ore		$31.87	$29.50	$23.60

Table 15.5.3

Surface Electrical Costs

Quantity	Description	Cost - $CAD
1	13.-8 kv incoming circuit breaker	60,000
1	Unit of five distribution circuit breakers	125,000
1	Hoist incoming circuit breaker	25,000
1	Mine ventilation and heating circuit breaker	25,000
1	Fan and heating starters	50,000
1	600 v distribution board	32,000
1	6 MVA main transformer 13.8 kv/6.9 kv	120,000
1	400 kva power centre	62,000
4 km	Overhead line 13.8 kv	197,000
50 mts	Cable. 6.9 kv 400 mgm	3,750
50 mts	Cable. 6.9 kv 4/0	3,550
100 mts	Cable. 600 v No. 2	1,700
50 mts	Cable. 600 v No. 6	1,085
1	System of surface lighting	15,000
1	System of surface communications and exchange	60,000

Underground

Quantity	Description	Cost - $ CAD
1	Unit of 3 6.9 kv distribution circuit breakers	75,000
1	6.9 kv distribution circuit breaker	25,000
25	Size 5 starter/contactors	190,000
20	Size 4 starter/contactors	100,000
1	1.5 mva power centre 6.9 kv/600 v	85,000
3	1.0 mva power centre 6.9 kv/600 v	210,000
9	Conveyor lighting and signalling transformers	22,500
9	Conveyor signalling systems	67,500
1	Batch of areas, lighting systems	50,000
1	System of underground communications	80,000
850 mts	Cable. 250 mcm 6.9 kv	63,600
1000 mts	Cable. 4.0 6.9 kv	71,200
1800 mts	Cable. 250 mcm 600 v	117,000
700 mts	Cable. 4/0 600 v	42,500
1000 mts	Cable. No. 2 600 v	27,280
600 mts	Cable. No. 6 600 v	12,990
1000 mts	Cable. 1/0 600 v	35,000
Total		**2,055,655**

Additional Considerations

Installation 15%

Freight 6%

Federal and Provincial Taxes

G.S.T

16. MILLING AND METALLURGY

16.1 Test Work

The test work on Grizzly is based on work done in 1992 on drill holes 91 DY 05 and 91 DY 07. The mineralization was massive sulfides in host rocks of carbonaceous quartzite, barite, dolomite, graphite phyllite and magnetite.

The predicted metallurgy from the lock-cycle test is summarized in the Table 16.1.

Table 16.1 - Predicted Metallurgy

	Assays		Recoveries	
	Lead	Zinc	Lead	Zinc
Lead Concentrate*	60.10%	10.90%	87.40%	7.00%
Zinc Concentrate	1.10%	57.20%	4.00%	90.80%

* *Lead concentrate grade 1077 grams silver with 81.1% silver recovery and grade 5.2 grams gold.*

In comparison with Grum ore, the sphalerite is less disseminated in the galena matrix. The Grizzly flow sheet is similar to the Grum flow sheet except one less stage of lead regrinding is required. Galena liberation requires a grind of 80% passing 21 microns, and the sphalerite liberation requires a grind of 80% passing 19 microns.

16.2 Concentrate Specifications

The multi-element analysis for the Grizzly Concentrates are found in Table 16.2.

Table 16.2 - Grizzly Concentrates

	Zinc Concentrate	Lead Concentrate
Zn	56.00%	61.00%
Cd	0.06%	0.01%
Fe	4.95%	3.80%
Cu	0.13%	0.30%
SiO2	1.00%	2.31%
Mg	0.31%	0.11%
Hg	380 ppm	40 ppm
Sn	0.02%	<.005
Sb	0.01%	0.29%
As	0.02%	0.10%
S	28.00%	18.60%
Cr	0.02%	0.02%
Pb	1.17%	10.00%
Mn	0.05%	0.01%
CaO	0.74%	0.25%
A12O3	0.25%	0.56%
Insol	1.36%	3.10%
Ag	42 g/t	1065 g/t
Au	.45 g/t	5.2 g/t
C	0.45%	1.15%
Ba	0.13%	0.12%
F	0.03%	0.01%
Bi	<100 ppm	<100 ppm
Te	<200 ppm	<.02%
Ni	0.01%	0.01%
Co	0.00%	0.00%

16.3 Mill Operating Cost

The existing Faro mill, which is located 20 km from the proposed shaft, will be used to process the ore. The Faro mill has two circuits: circuit 1 with a capacity of 5,200 tonnes per day, and circuit 2 with a capacity of 7,800 tonnes per day. The combined mill capacity is 13,000 tonnes per day. At 1,500,000 tonnes per year, the Grizzly deposit requires an average mill capacity of 4,300 tonnes per day. The plan would be to run one circuit; which one will depend on whether other ore will be processed through the plant with Grizzly, and how the mill will be operated.

In 1996 the mill costs were approximately CAD $50,000,000 to process 4,500,000 tonnes of ore. In order to estimate the milling cost for the Grizzly ore, it is assumed that 10% of the 1996 costs were fixed and the remaining costs are variable. The low fixed cost is based on the fact that the mill schedule can be adjusted to accommodate lower production levels. The following Table 6.3 summarizes the operating costs for the mill.

Table 16.3 - Mill Operating Costs

	Annual Tonnage		
	1,500,000	3,000,000	4,500,000
Total Cost	**$20,000,000**	**$35,000,000**	**$50,000,000**
Fixed	$5,000,000	$5,000,000	$5,000,000
Variable	$15,000,000	$30,000,000	$45,000,000
Cost Per Unit	**$13.33**	**$11.66**	**$11.11**
Fixed	$3.33	$1.66	$1.11
Variable	$10.00	$10.00	$10.00

17. TRANSPORTATION

At the minesite, the lead and zinc concentrates are loaded onto the large 10 axle B-Train units that carry a 50 wet tonne payload of either lead or zinc concentrates. The concentrates are trucked 500 km to Skagway, Alaska, where the concentrates are off-loaded, warehoused, and then loaded onto ocean-going ships.

Anvil Range Mining Corporation ("Anvil") hires a contractor to load and haul the concentrate. Anvil owns all of the equipment used in loading the concentrate onto the trucks, and the loading contractor is essentially a supplier of operating labour. To facilitate the truck haul, Anvil provides the truck contractor with a truck servicing terminal in Whitehorse, and with all the trailers and ore containers which are required to do the haul. Anvil's Whitehorse office works together with the truck contractor to ensure a low-cost, smooth running operating.

In addition to the contractor costs and Anvil's administrative costs, Anvil is charged a Yukon Territorial Government road tax on the hauled concentrates of $1.42 per wmt. The annual fixed cost of the truck haul are $2,125,000, and the variable costs are $33.96 per wmt. The transportation costs are summarized in the following Table 17.1.

Table 17.1 - Canadian Transportation Costs

	Rate - 250,000 wmt/y *		Rate - 350,000 wmt/y	
	$/wmt	Total	$/wmt	Total
Site Loading Cost	$1.84	$460,000	$1.63	$584,000
Fixed Costs	$0.60	$150,000	$0.43	$150,000
Variable Costs	$1.24	$310,000	$1.24	$434,000
Trucking Costs	$37.30	$9,325,000	$35.59	$12,455,000
Fixed Costs	$6.00	$1,500,000	$4.28	$1,500,000
Variable Costs	$31.30	$7,825,000	$31.30	$10,955,000
Bulk Haul Taxation	$1.42	$355,000	$1.42	$497,000
Truck Administration	$1.90	$475,000	$1.36	$475,000
Total	**$42.46**	**$10,615,000**	**$40.03**	**$14,011,000**

** To translate wmt to dmt, a moisture content for the concentrates is estimated at 7.5% for both lead and zinc.*

17.1 Port Costs and American Road Tax

Currently Anvil is the only user of the Skagway port. As a result, Anvil is on the hook to pay all of the fixed costs including the terminal usage for the terminal operation. These fixed costs total US $2,990,000. The variable costs of the port operation total US $3.10 per wmt. In addition, Alaska levies a road tax on the heavy trucks. The road tax is US $0.91 per wmt and recent discussion with the Alaskan government indicates that the road tax is going to be reduced. For the purposes of the valuation in the following Table 17.2, the road tax is set at US $0.40 per wmt.

Table 17.2 - American Port Costs - US Dollars

	Rate - 250,000 wmt/y		Rate - 350,000 wmt/y	
	$/wmt	Total	$/wmt	Total
AIDEA Costs	$10.64	$2,660,000	$7.74	$2,710,000
Fixed Costs	$10.14	$2,535,000	$7.24	$2,535,000
Variable Costs	$0.50	$125,000	$0.50	$175,000
MSI Costs	$3.15	$787,500	$2.81	$982,500
Fixed Costs	$1.20	$300,000	$0.86	$300,000
Variable Costs	$1.95	$487,500	$1.95	$682,500
Electrical Costs	$0.45	$112,500	$0.36	$127,500
Fixed Costs	$0.30	$75,000	$0.21	$75,000
Variable Costs	$0.15	$37,500	$0.15	$52,500
Taxes - Fixed Cost	$0.32	$80,000	$0.23	$80,000
Other Costs	$0.50	$125,000	$0.50	$175,000
Road Taxation	$040	$100,000	$0.40	$140,000
Total - US Dollars	**$15.46**	**$3,865,000**	**$11.64**	**$4,075,000**
Exchange Rate	0.74	0.74	0.74	0.74
Total - CAD Dollars	**$20.89**	**$5,223,000**	**$15.72**	**$5,507,000**

17.2 Ocean Freight

The concentrate is shipped to ports in Japan, Korea and China and various European ports. The composite freight rate in 1996 was US $26.50 per wmt, and this was the highest shipping rate in the last 5 years. Shipping rates are somewhat dependent on the lot size and, presumably, the smaller the annual production, the smaller the lot size.

Table 17.3 - Ocean Freight

	Annual Shipping Rate in WMT			
	250,000	350,000	450,000	550,000
Average Shipment Size (wmt)	17,500	20,000	22,500	25,000
Ships (no.)	14.3	17.5	20	22
Shipping Cost	$6,680,000	$9,100,000	$11,450,000	$13,700,000
Fixed Costs	$1,430,000	$1,750,000	$2,000,000	$2,200,000
Variable Costs	$5,250,000	$7,350,000	$9,450,000	$11,500,000
Inspection, Assaying, etc.	$750,000	$1,050,000	$1,350,000	$1,650,000
Total Shipping Cost	**$37,430,000**	**$10,150,000**	**$12,800,000**	**$15,350,000**
Cost per WMT				
Shipping Cost	$26.72	$26.00	$25.44	$25.00
Fixed Costs	$5.72	$5.00	$4.44	$4.00
Variable Costs	$21.00	$21.00	$21.00	$21.00
Inspection, Assaying, etc.	$3.00	$3.00	$3.00	$3.00
Total Shipping Cost	**$29.73**	**$29.00**	**$28.55**	**$28.00**

18. MARKETING

18.1 Customers

Currently, Anvil sells its zinc and lead concentrates to smelting companies located in Japan, South Korea, China, Australia, Italy, Spain and other European countries. Substantially, all of the production will be sold to satisfy delivery requirements under long term contracts and marketing arrangements.

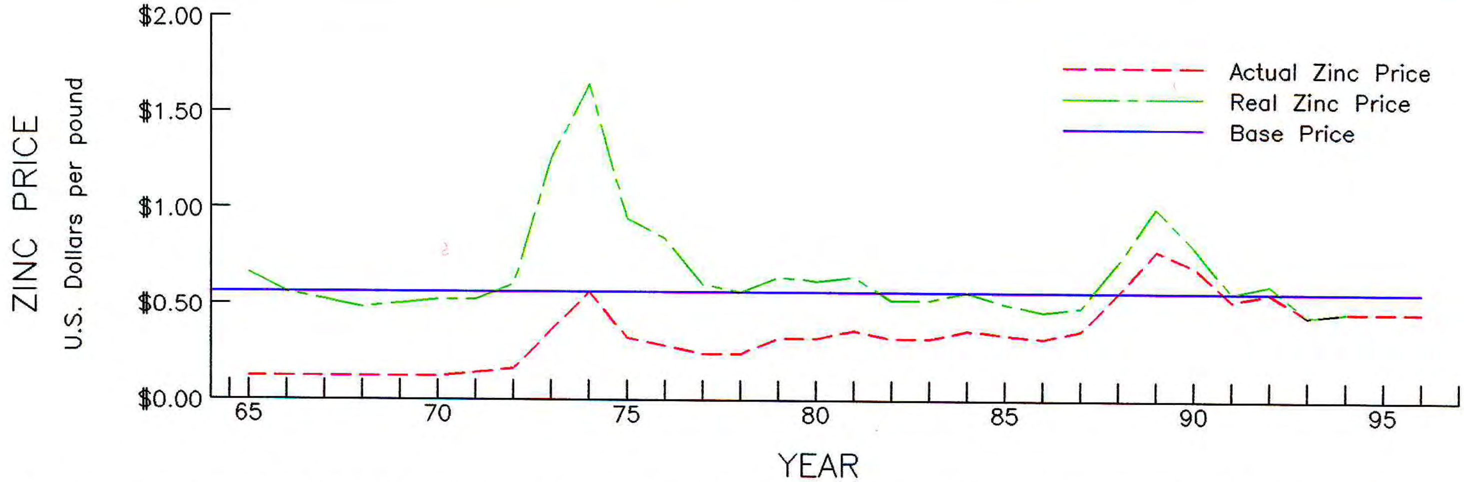
The following Table 18.1 sets out the amount of zinc and lead concentrates shipped from Skagway, Alaska, to Europe, Asia, and other destinations for the periods indicated.

Table 18.1 - Geographical Sales Distribution of Faro Concentrates

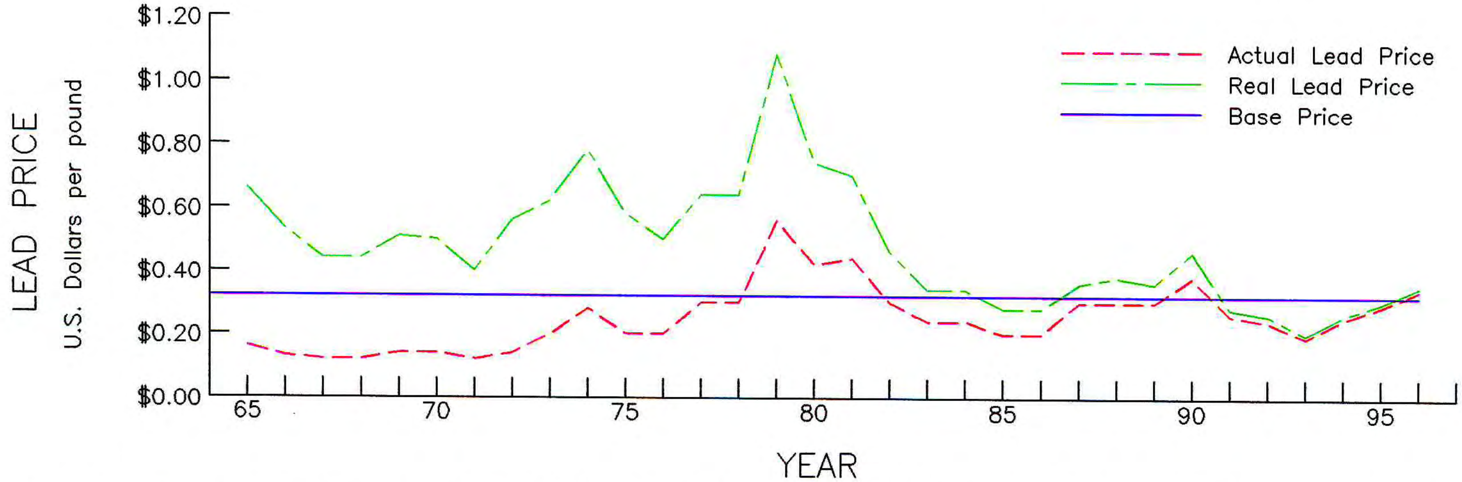
	1990	1991	1992	1995	1996
Zinc Concentrates					
Europe	140,134	148,084	213,686		
Asia	188,016	130,201	198,498		
Other	0	0	0		
Lead Concentrates					
Europe	95,631	65,838	61,267		
Asia	32,075	71,449	123,809		
Other	40,447	16,024	53,376		

18.2 Pricing

Payment from the smelter companies is based on metal prices quoted on the London Metal Exchange ("LME"), less smelter treatment charges and metal deductions, according to the terms negotiated annually with each smelter. The smelters pay an amount equal to the payable metal contained in the concentrate, multiplied by the average price during a quotational period, which is, typically, one to three months following the month in which the vessel carrying the concentrate arrives at the receiving port. In general, smelter charges include a treatment charge set at a base metal price, and a smelter participation component which increases or decreases as the price of the metal changes, relative to the base price. In addition, the miner is



The zinc price used in the analysis was $0.55 per lb. This compares to a 31 year average of $0.64 per pound in real terms.



The lead price used in the analysis was $0.32 per lb. This compares to a 31 year average of $0.462 per pound in real terms.

SCALE

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GRIZZLY PROJECT

UNDERGROUND EXPLORATION PROGRAM

Fig. 18.4 LME Lead/Zinc Prices

Anvil Range
MINING CORPORATION

paid for lead and silver produced from the lead concentrates after smelter deductions for minimum metal levels.

18.3 Concentrate Terms

The treatment charges vary from year to year and are set through negotiations. The price escalator or de-escalator does not accurately predict the treatment charges at different price levels past the year of service. A regression formula based on the last 16 years treatment charges has been developed for both lead and zinc concentrates. This formula has been adjusted upwards to closer reflect the current market.

The adjusted zinc regression formula used for forecasting the long-term treatment charge is $81.09 plus $0.104 per dollar of zinc price. The zinc formula includes an adjustment of $20.00 per tonne to update to the current market. The adjusted lead regression formula is $126.72 and $0.044 per dollar of lead price. The lead formula includes an adjustment of $10.00 per tonne.

19. ECONOMIC ANALYSES

Economic analysis forms an important part of this study and is attached as a separate document as Appendix 19.

19.1 Assumptions

The following assumptions were used in the economic analysis:

- Mine production, capital expenditures and mining costs are summarized in Sections 14 and 15.
- The cost of trucking the ore from Grizzly to the mill site is $3.50 per tonne.
- Environmental abandonment is covered by an initial bond of $2,000,000 and then $2,000,000 amortized over the life of the mine.
- Mill costs are summarized in 16.3. The operating costs include $0.50 per tonne for mill capital replacement.
- Initial recruiting and training costs are $1,000,000. Annual site G&A costs total $6,624,000.
- Environmental costs are $0.25 per tonne.
- Land transportation, port and shipping costs are summarized in 17.1, 17.2 and 17.3.
- Annual head office and marketing costs total $3,500,000 annually.
- Concentrate terms are summarized in 18.3 and prices are summarized in 18.4.

19.2 Economics

The base case economics and the sensitivities to change are summarized in the following table:

Base Case Economics (000)	Before Tax		After Tax	
	Cashflow	NPV@10%	Cashflow	NPV@10%
Base Case	$178,369	$23,855	$134,086	$11,646
Sensitivity Analysis				
Zinc + $0.01/lb	$16,886	$5,726	$12,290	$4,310
Lead + $0.01/lb	$15,912	$5,159	$11,583	$3,848
Silver + $0.50/oz	$18,328	$5,946	$13,383	$4,424
Zinc Rec + 1%	$7,047	$2,393	$5,139	$1,798
Zinc Con + 1%	$11,670	$3,955	$8,513	$2,984
Exchange Rate + .01	($15,492)	($5,121)	($11,521)	($3,921)
Site Costs + 1%	($7,155)	($2,387)	($5,213)	($1,801)
Zinc TC + $10	($2,049)	($695)	($1,495)	($521)
Lead TC + $1-	($1,438)	($466)	($1,052)	($347)