016234

GRIZZLY PROJECT

PRE-FEASIBILITY STUDY
DECEMBER 1996

Anvil Range
MINING CORPORATION

1

1. EXECUTIVE SUMMARY

Anvil Range Mining Corporation ("Anvil") owns and operates a lead-zinc mine and concentrator near the town of Faro, Yukon Territory, Canada. The Grum open pit on the Anvil property is now the principal source of ore to the concentrator, but its reserves are expected to be depleted in approximately five years. In order to continue the Faro operation beyond that point, Anvil will need to bring one or more undeveloped deposits into production.

Only the Grizzly deposit and an underground extension of the Grum pit have been identified on the Anvil property as potential economic producers in the near future.

The Grizzly deposit (formerly known as the "Dy" deposit) is a lead-zinc-silver-gold stratiform, syn-sedimentary, pyritic massive sulphide deposit lying approximately 580 to 920 meters below surface and dipping 20° to 35° to the southwest. It is located approximately 20 km from the Faro concentrator in a southeast line from the Faro, Grum and Vangorda deposits.

Table 1.1
Grizzly Project: Summary

Production rate	1,500,000 tonnes/year
Mine life	11.5 years
Development period	61 months
Phase 1 development	Cdn$ 26.4 million
Construction	Cdn$ 51.6 million
Mining equipment	Cdn $17.1 million
Replacement Capital	Cdn$ 11.0 million
Cashflow	
NPV @10% before tax	Cdn$ 44.6 million
Payback	3.5 years
IRR	16.18%
Mining cost/tonne ore	Cdn$
Year 1	$31.87
Year 2	$29.50
Year 3	$21.64

Geological inventory calculations for the Grizzly deposit, including 10% dilution at 0% Pb-Zn grade, indicate mineralized resources in the range of 20 to 40 million tonnes, for cutoff grades of 9% and 6% Pb-Zn respectively. Geological consultants have placed the inventory estimates in the probable (60%) and possible (40%) categories.

The orebody is open in several directions, and perimeter drilling has yet to define its overall outline.

The orebody is open in several directions, and perimeter drilling has yet to define its overall outline.

This pre-feasibility study report recommends the initiation of a 61-month program for the development of the Grizzly Project. The proposed program consists of a 27-month period of access development, reserve evaluation and final feasibility study, followed by an additional 34 months of infrastructure development.

The technical and economic feasibility of the Grizzly Project has been assessed on a preliminary level, based on historical data available and the following concepts:

- Two parallel 1,700 m long declines from the Blind Creek valley for exploration and access;
- Production shaft in the center of orebody;
- Main infrastructure of the mine in ore;
- Haulage of ore with diesel trucks;
- Milling in the existing Faro concentrator.

Within the limitations of the present information base, this report demonstrates that, with the right management and the right approach, the technical and economic factors are favorable for the development of the Grizzly deposit into a profitable mine.

The next stage of underground exploration and development will be necessary to prove orebody continuity, to raise a major portion of inventory estimates to the proven category and to improve confidence in the Grizzly deposit as an economic resource. In the course of this stage, underground mining methods will also be proven for safety and economy, and various conceptual assumptions made during the pre-feasibility study will be confirmed and refined. As additional information is acquired, the underground program will evolve into the final mine plan.

A successful development of the exploration ramps will lead to additional lateral drift development, exploration drilling, refined resource estimates, metallurgical testing, permitting and feasibility study for the full production decision.

2

2. INTRODUCTION

2.1 Purpose of the Study

This pre-feasibility study has been undertaken to evaluate historical and new information about the Grizzly deposit, to establish a preliminary development plan upon which an initial indication of the project's technical and economic feasibility can be derived, and to recommend a course of action.

More specifically, this Study addresses the question of whether further expenditures are warranted in the continuation of the exploration and development of the Grizzly deposit.

2.2 Study Team Participants

This Study has been prepared by Anvil personnel with the written contributions of the following participants who prepared the respective sections of the report:

Piteau Associates N.D.(Nick) Rose	Geology and Mineral Reserve Evaluation
Parwest Mining International D.M. (David) Parkes S.L. (Steve) Szabolcsy N.R. (Neville) Pease	Mining
Access Mining Consultants Ltd. G.A. (Gregg) Jilson	Environmental
H.M. Visagie Consultants H.M. (Rick) Visagie	Milling Transportation Marketing Project economics

In addition, Mssrs. Gregg Jilson and Norman Anderson reviewed data on geology and mineralogical resource estimates.

Mr. Fritz Prugger directed the pre-feasibility study and report preparation. He was assisted by Proton International Engineering Corporation, who compiled and edited the final report document.

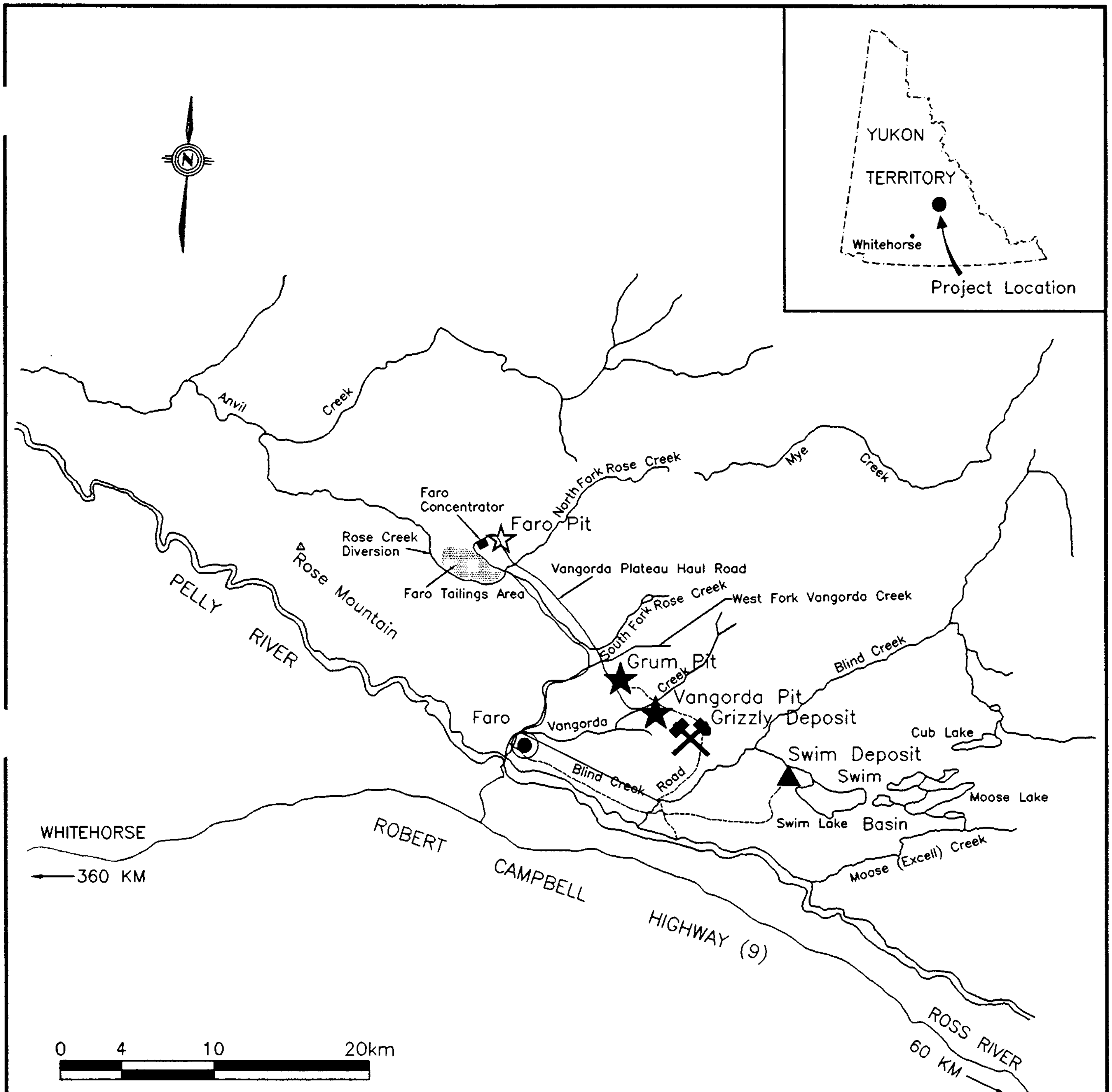
2.3 Project Location and Setting

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

The development of the Grizzly Project is predicated on the continuing use of the existing infrastructure, which 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
- Head office

The Faro concentrator is located about 13 and 20 km north west of the Grum and Grizzly deposits respectively.



GRIZZLY PROJECT

PRE-FEASIBILITY STUDY

Fig. 2.3-01 Location of the Grizzly Deposit

Anvil Range MINING CORPORATION

	APPROVED	
DRAWING: LOC-PLAN.DWG	CHECKED	APPROVED 96/09

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 rates reached 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 (named the Dy deposit in those days), 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.

The Dy deposit (for Dynasty Exploration) was the object of surface exploration drilling between 1976 and 1991.

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.

After the demise of Curragh Inc., the property was held on a care and maintenance basis from 1993 to 1994.

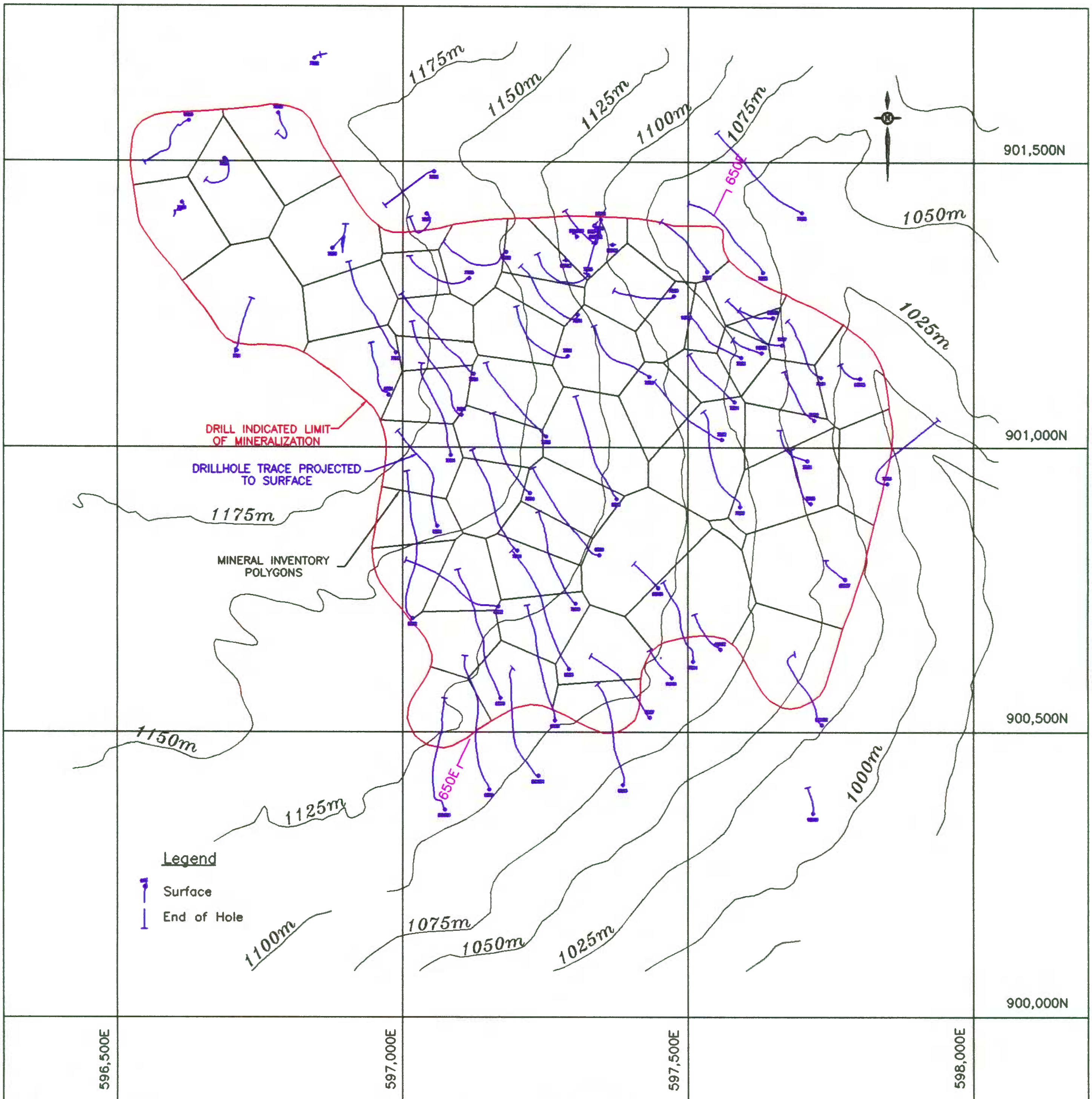
In 1994, Anvil Range Mining Corporation acquired the Faro holdings of Curragh Inc. and mining resumed in the Vangorda and Grum pits. The mined-out Faro pit is now being used for tailings disposal.

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 onto ships for markets in Europe and Asia. Proven open pit mineable ore reserves indicate a mine life of approximately five years.

In July 1996, Anvil changed the name “Dy” deposit to “Grizzly”.

2.5 Sources of Information and Data

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 approximately 1,000 meters deep (see Figure 2.5-01).



GRIZZLY PROJECT

PRE-FEASIBILITY STUDY

Anvil Range MINING CORPORATION

Fig. 2.5-01 Surface Drillhole Plan

SCALE	APPROVED
DRAWING: 004001MD.DWG	CHECKED

DATE 96/11

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

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.

Cost estimates for a substantial portion of mining development are based on various proposals submitted by contract mining companies. 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 $28,000.00. 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 meters 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.

Appendix A lists the sources of data and information used to perform the pre-feasibility study.

3

3 SUMMARY, CONCLUSIONS AND RECOMMENDATIONS

3.1 Project Development Concept

The proposed concept for the Grizzly Project involves the development of an underground mining operation ultimately producing 1.5 million tonnes of ore per year.

This report demonstrates parameters and a development concept under which the Grizzly Project can be brought into profitable operation.

It recommends the initiation of an underground exploration program as the next stage of the Grizzly Project.

3.2 Geology and Ore Reserves

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. Core data have located the deposit at a depth of approximately 580 to 900 metres below surface and ranges in thickness from a few metres up to approximately 28 m on two different interpreted mining horizons (*Lower-G and Upper-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.

The main body extends about 1,000 m from west to east and 850 m north to south and has a general dip of 20° to 35° southeast. There is, in addition, the northwest corner, measuring 350 m from west to east and 230 m from north to south.

Geological inventory calculations for the Grizzly deposit, including 10% dilution at 0% Pb-Zn grade, indicate mineralized resources in the range of 20 to 40 million tonnes, for cutoff grades of 9% and 6% Pb-Zn respectively. Geological consultants have placed the inventory estimates in the probable (60%) and possible (40%) categories.

Anvil's underground work at the Grum and Faro deposits led to increased ore reserves, 25% more in the case of Grum and, at the Faro underground, 50% more than that indicated by surface drilling. Possibilities exist for similar increases in mineral inventory estimates for the Grizzly if it follows the trend of the other deposits.

The Grizzly deposit is open in several directions, ie, its perimeter has not been drilled and the overall outline has not been defined. Continuity of mineralization exists throughout certain areas of the deposit as it is known today.

3.3 Underground Exploration

This report recommends the initiation of an underground exploration program to define the structure of the Grizzly orebody, improve the confidence of reserve estimates, and to prove the practicality and economics of mining methods.

The program would begin with the driving of twin ramps from the Blind Creek valley for a length of 1,737 metres to a point above the so-called the *Barren Zone*. Exploration drifting into the B-Zone and A-

Zone would begin from this point.

Approximately 20% to 25% of the orebody plan would be covered in the program, in the heart of good grade and substantial ore thickness. Due to the grade variations in the orebody, this exploration may be able to represent 50% or more of the metal that is presently estimated in the orebody.

3.4 Production Access

The base case proposed in the report involves the installation of a production shaft in the center of the orebody. Other options identified in the study include conveyor haulage, inclined shaft hoisting, pumping ore as slurry and a vertical shaft in other locations. Detailed investigations of these options are now premature, as the relative merits will only become clear after a better understanding is obtained from the underground exploration phase of the project.

A possibility also exists for the positioning of the production access to serve both the main Grizzly orebody and the potential Grizzly East deposit. Initial drilling for Grizzly East began in November 1996.

3.5 Mining Method

Two mining methods are presented in the base case:

- Room and pillar without backfill in areas where the ore vertical thickness is less than 6.5 metres; and
- Concrete pillar mining with backfill in areas where the ore vertical thickness is greater than 6.5 metres.

The potential of a weak hanging wall requires caution, especially in areas of greater ore thickness. Other mining methods have been identified in the course of the study, with their applicability to be eventually governed by geotechnical considerations and economics. Procon Mining & Tunneling Ltd, in particular, examined the data in detail and proposed a post pillar cut-and-fill method for certain stopes in the orebody.

A significant backfill operation is associated with the base case, for which estimated costs are included in the economic analysis.

3.6 Milling and Metallurgy

Lakefield Research undertook testwork in 1992 on samples from two Grizzly drillholes. They reported a mineralization of massive sulphides in host rocks of carbonaceous quartzite, barite, dolomite, graphite phyllite and magnetite. In comparison with Grum ore, the spalerite is less disseminated in the galena matrix.

The testwork indicated that the ore would require a treatment similar to that for the Grum ore that is now in place in the Faro concentrator, except that one less stage lead regrinding would be required.

The indicated metallurgical recoveries are 87.4% for lead and 90.8% for zinc.

Additional metallurgical testing will be undertaken with a bulk sample to be collected during the underground exploration program.

3.7 Environmental Considerations

With the proper environmental planning and mine development, the Grizzly Project can avoid significant impact to the adjacent Blind

Creek. Blind Creek, a sensitive salmon-bearing stream, is at the forefront of several issues including first nation traditional land use, wildlife impacts and other competing land use. The issues will need to be carefully managed during the underground exploration and development of the project.

Environmental issues effectively preclude the use of the Blind Creek portal for production purposes.

There is little reason to believe that the Grizzly development can not proceed at reasonable cost and with little environmental impact.

3.8 Concentrate Transportation and Marketing

The base case assumes a continuation of the present regime of concentrate transportation and marketing. Anvil's ongoing operations provides ample reliable data in both these subjects for the economic evaluation of the Grizzly Project.

3.9 Capital Cost

Capital costs have been estimated for the base case, with four major breakdowns:

- Phase 1 development, including the underground exploration program (Cdn $26.4);
- Construction (Cdn $ 53.6 million);
- Mining equipment (Cdn $ 17.1 million);
- Replacement capital beginning Year 6 (Cdn $ 11.0 million).

The estimated capital cost for the base case is Cdn $ 106.1 million.

The base case is calculated to incur an operating cost of Cdn$ 21.64 per tonne mined. A total workforce of 178 persons will be supported by a staff numbering 26.

Other significant cost items are backfill, fuel and explosives.

3.11 Economic Analysis

The net present value of future cashflows is calculated to be Cdn $44.6 million for the base case, before tax. The project cashflows pay back the initial investment in 3.5 years, and the internal rate of return is 16.18%.

4

4. PROJECT DEVELOPMENT CONCEPT

The present concept for the Grizzly Project involves the development of an underground mining operation ultimately producing 1.5 million tonnes of ore per year. An initial ramp-up period of three years will be required before achieving the ultimate production rate.

Anvil's long-range plan of 1994 anticipates the cessation of Grum pit surface mining in November, 2000. By that time, a northwest-dipping extension of the Grum deposit is expected to be placed into production as an underground mining operation, capable of supplying approximately 600,000 tonnes or more annually for a combined (Grum and Grizzly) production approaching 2.5 million tonnes per year.

This timetable is predicated on the ability to start a portal to gain underground access at Grum's north pit wall while the open pit is still in operation. The underground development at Grum will be similar in technique to that applied in pit wall underground mining at the Faro pit during the Curragh days.

Ore from Grizzly and the Grum pit underground may be augmented by expected surface ore finds at the Grum and the possibility of finding the extension of the Grizzly deposit that has been cut off by the East Fault ("Grizzly East"). The drilling of Grizzly East started in November 1996 and its orebody is anticipated at a depth of 300 meters or more, east of the known Grizzly deposit. Underground mining would be required to exploit this orebody.

Table 4.1
Grizzly Project Mine Development Parameters
Base Case

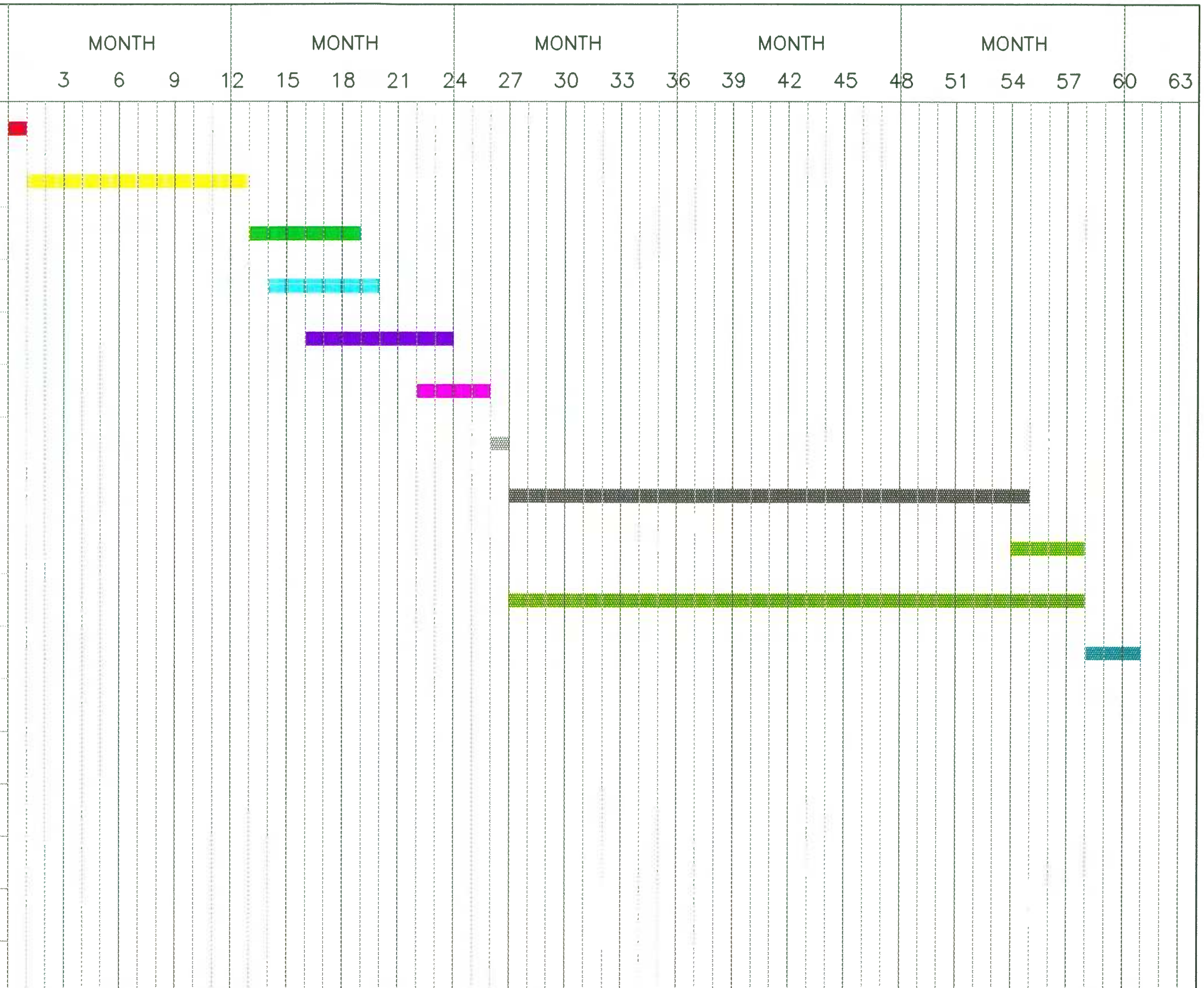
Production rate	1,500,000 tonnes/year
tonnes per day	4,300
Shaft location	Central "Barren Zone"
Shaft diameter	18 feet
Vein haulage	40 t diesel trucks
Shaft depth	750 meters
Main station depth	690 m
Ventilation	500,000 cfm
Fan HP	1,700
Additional vent raises	

A summary of Grizzly mine development parameters is presented in Table 4.1. These parameters form the basis of the capital cost and operating cost estimates that are presented later in this report. Several options, involving various shaft locations, haulage methods and production rates, were considered during the study and are described in Appendices D, E and G. The concept selected for presentation (identified as "Case B" in the Appendices) includes a production shaft in the center of the orebody and diesel trucks for haulage of ore to a centrally located underground crusher. Crushed ore is conveyed underground by belt conveyors to a surge bin and the shaft. The main underground development is in ore.

The final feasibility study proposed in the upcoming development program will establish the production shaft location and the combination of mine infrastructure, mining methods and mining equipment that is best suited for Grizzly's conditions and provide the most favorable economic benefits.

The proposed development schedule for the Grizzly Project is presented in Figure 4-01.

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	28 MO.
9	LOADING POCKETS, ROPING UP AND COMMISSIONING	4 MO.
10	UNDERGOUND DEVELOPMENT	31 MO.
11	PRODUCTION START-UP	3 MO.



SCALE

DRAWING: DEVSCHED.DWG

APPROVED

DATE 96/11

GRIZZLY PROJECT

PRE-FEASIBILITY STUDY

Fig. 4.0-01 Development Schedule

Anvi Range
MINING CORPORATION

5

5 GEOLOGY AND ORE RESERVES

5.1 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).

5.2 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 580 to 920m below surface and dips 20 to 35° to the southwest. Two relatively distinct zones define the orebody in plan view (Fig. 5-01), 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.

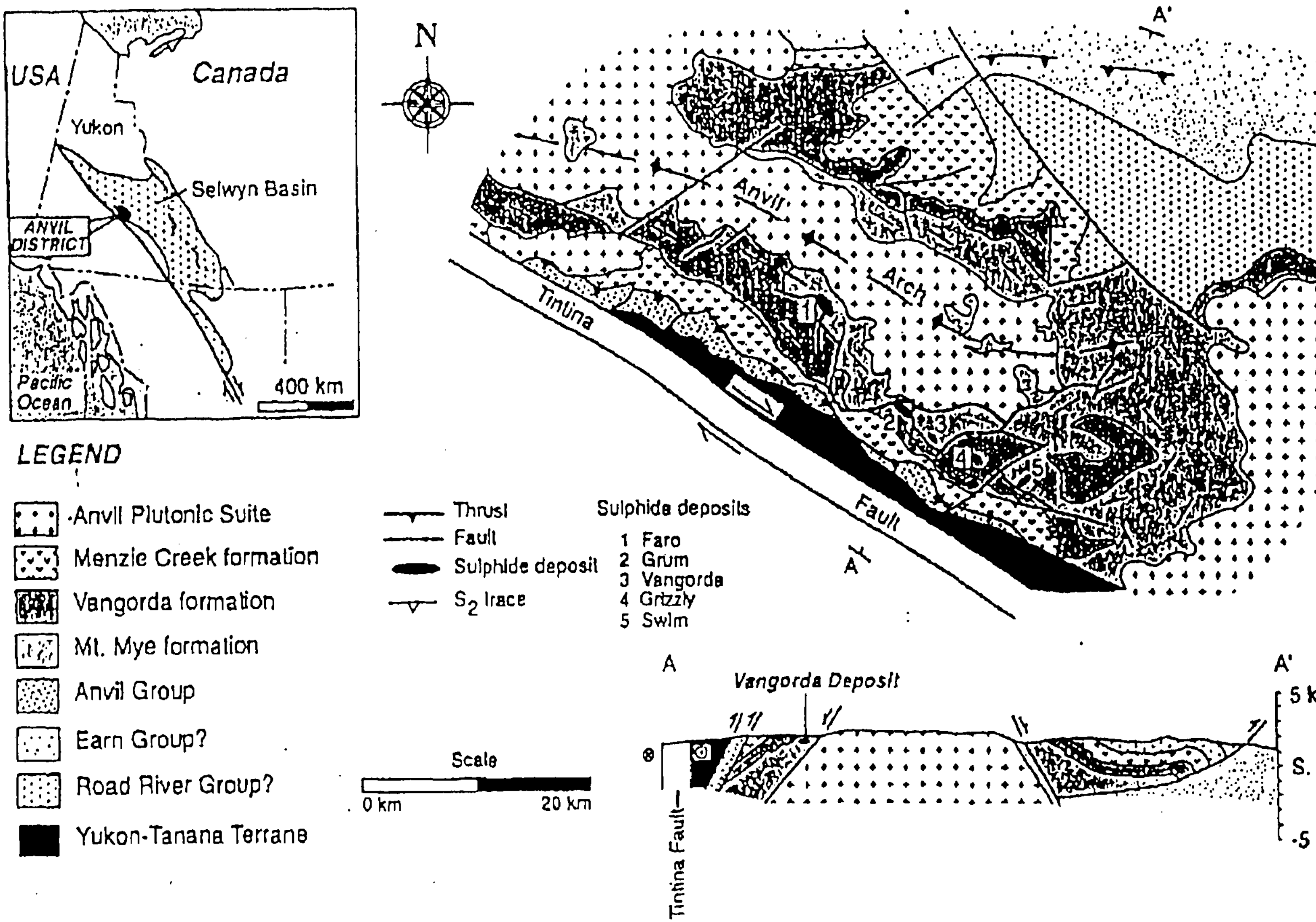


Fig. 1. Generalised geological map and cross-section of the Anvil District. The mineral deposits occur on a curvilinear trend along the southwestern margin of the Anvil Arch (redrafted from Jennings and Jilson, 1986).

FIG. 5.1-01

D. Brown, K. McClay / Ore Geology Reviews 9 (1994) 61–78

This zone is composed 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 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 30 m 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. 5-01).

5.3 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 metamorphic 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.

5.4 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 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 district. Larger displacement structures of approximately 10 to 20 m, 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. 5-01). 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 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. 5-01. 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 50 m 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 6 m were very common and created difficulties in mining with conventional rubber tired LHD equipment. Displacement faults of 10 to 40 m were less common, but were also encountered.

5.5 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 50 m spacing through the deposit as shown on Fig 5-01. Assay grades and lithologies were plotted on the sections and color coded to aid in correlations between drillholes. A sectional influence of 25 m 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 K, 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 15 m on the west side of the deposit up to approximately 60 m 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, Appendix K).

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 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.

5.7 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 3.5 m. Intersections less than 3.5 m in length were diluted to a minimum 3.5 m core length using footwall material. Intervals of waste of greater than 3.5 m were excluded from weight average composites, whereas intervals of waste less than 3.5 m in length were included.

excluded from weight average composites, whereas intervals of waste less than 3.5 m in length were included.

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

Polygons were generated in *Geomodel* by mid-point projections between drillholes (to a maximum of 170 m). 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 60 m 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 center.

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.

Table 5.1
Grizzly Mineable Inventory
10% Dilution

Cutoff Grade	Zone	Pb+Zn (%)	Pb (%)	Zn (%)	Au (g/t)	Ag (g/t)	Tonnage
6%	*Upper-C*	8.86	4.03	4.83	58.3	0.66	19,267,173
Pb+Zn	*Lower-C*	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-C*	10.85	5.19	5.66	73.1	0.83	11,086,376
Pb+Zn	*Lower-C*	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

5.8 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 5.1.

Inventories were calculated by adding 10% dilution at 0% Pb+Zn grade to in-situ values. This was considered to represent the majority of areas which have a phyllite hanging wall. Undoubtedly, some blocks will be mined at varying dilutions at varying grades. For the purpose of this Study, however, the described calculation method has been used to provide a reasonable indication of mineable inventory. This method has been used consistently throughout the Study and in past evaluations of the Grizzly deposit.

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 6.5 m high were given mining recoveries of 70%. Ore blocks of average thickness greater than 6.5 m 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 6.5 m) and 71% in areas of

Table 5.12
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

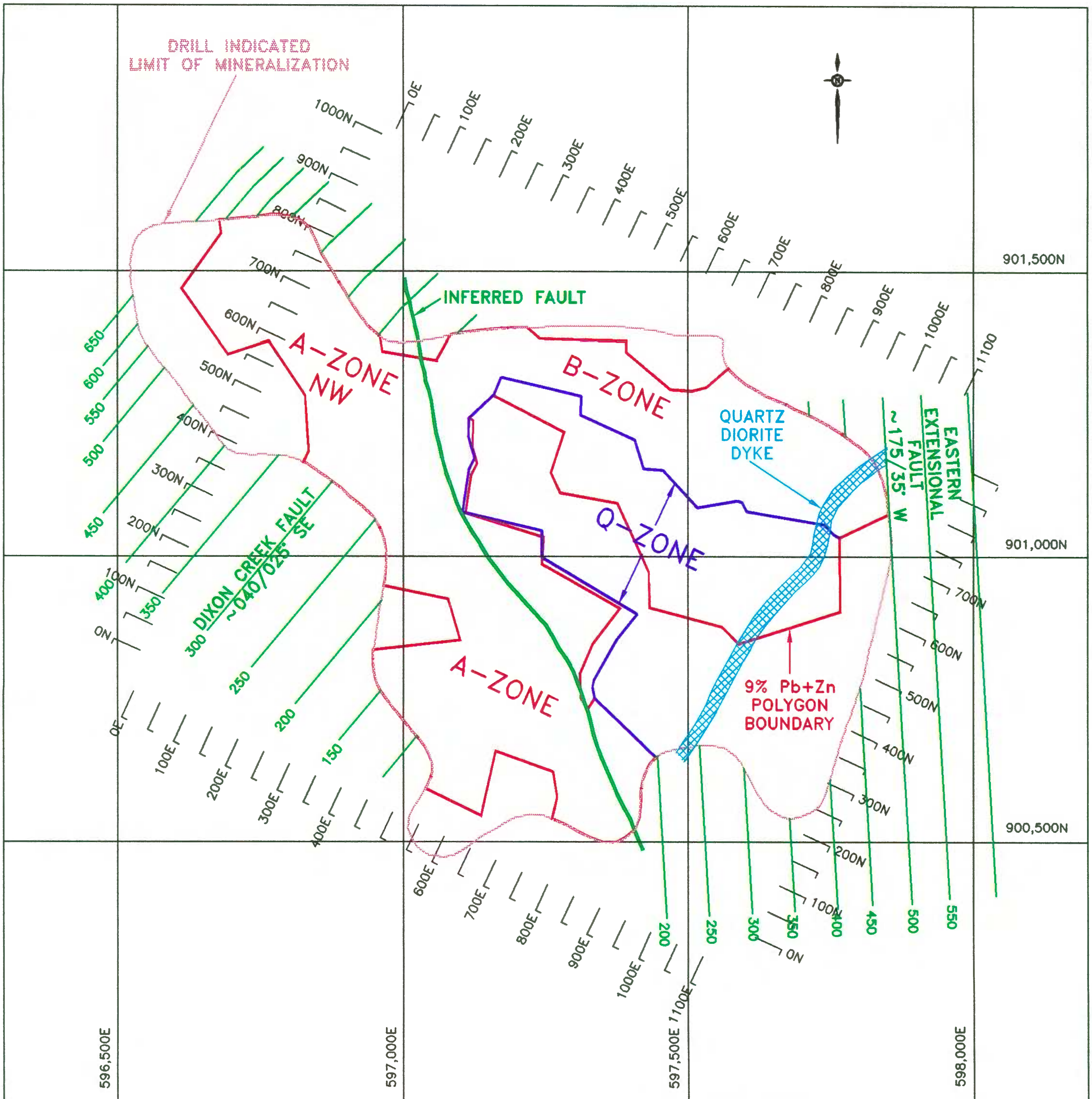
strictly on an economic cutoff grade with no defined geologic controls, and may require great detail in drillhole definition.

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 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 3.5 m 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.

6



GRIZZLY PROJECT

PRE-FEASIBILITY STUDY

Fig. 6.5-01 General Geology
Ore Zones

Anvil Range
MINING CORPORATION

SCALE	APPROVED	
DRAWING: 6201MOD.DWG	CHECKED	DATE 96/11

6 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.

7

7 UNDERGROUND EXPLORATION

7.1 Underground Exploration Rationale

The existing body of knowledge about the Grizzly deposit is based on the data from 86 test holes, surveying of base holes, core logging, chemical analyses, data collection, reserve calculations and studies. Approximately Cdn $15,000,000 has been spent since 1976 on these and related activities.

Additional surface drilling would contribute little or no useful information to improve our understanding of the orebody and how it can be economically mined. This report recommends the initiation of an underground exploration program as the next necessary step for the development of the Grizzly deposit.

The proposed underground exploration program is designed with the following objectives:

- 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" (the central quartzite zone) to allow a decision on the location of the production/personnel shaft.
- to collect data concerning geotechnical conditions.
- to collect data about hydrology

Anvil anticipates that the exploration program can be executed in a cost-effective manner and that its success will confirm the Grizzly to be an economically mineable orebody.

The following paragraphs describe the proposed exploration drifts into A-Zone and B-Zone of the Grizzly orebody. These drifts will be positioned to provide the best coverage of the entire orebody, and to explore in detail, through fan drilling from the hanging wall, the high grade and thick ores of the B-Zone and A-Zone that are first scheduled for mining. While achieving the objectives of the program, Anvil will develop a greater understanding of the diorite dyke, the East Fault and the NW-SE inferred fault in the center of the orebody.

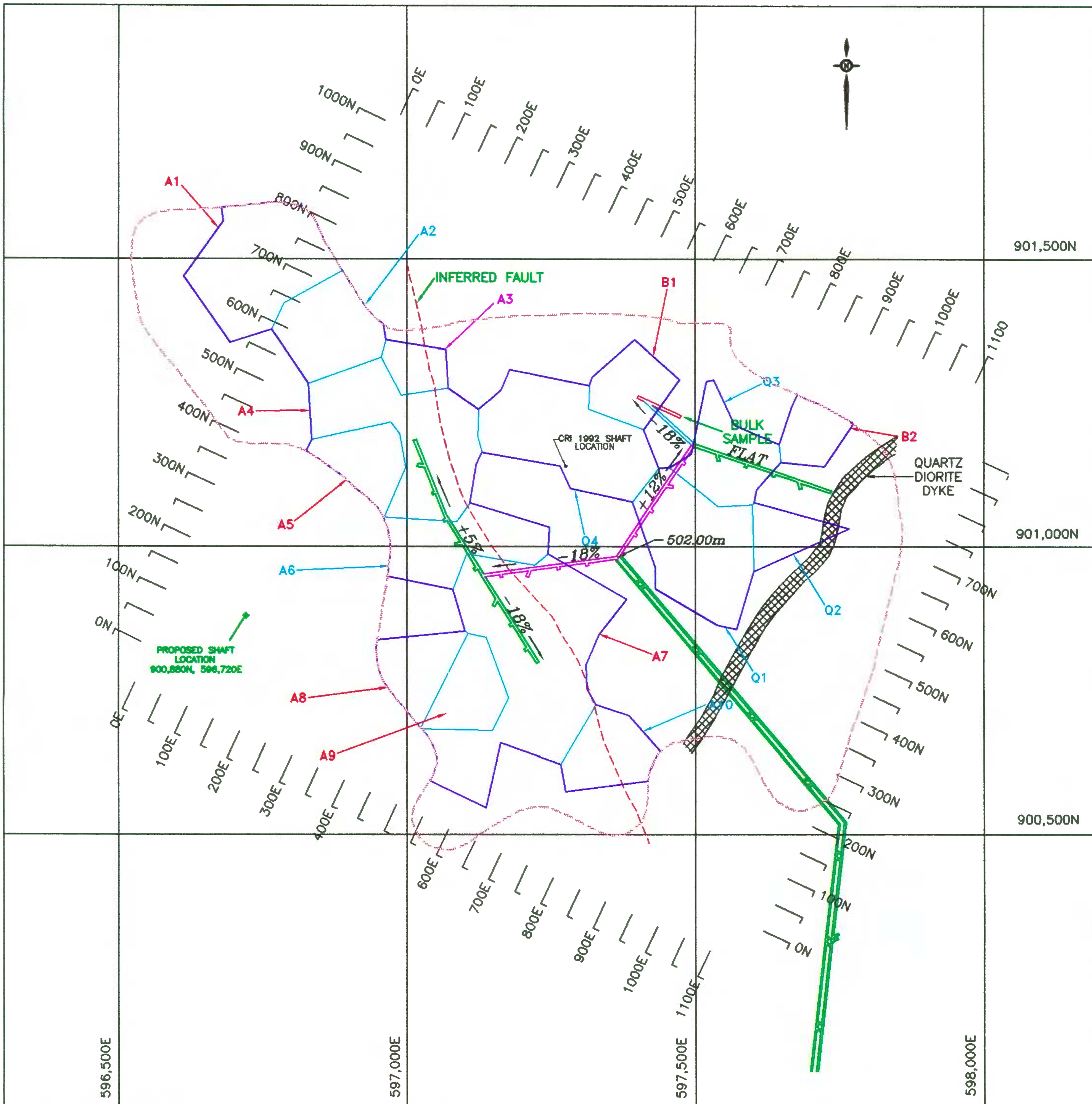
7.2 Underground Exploration Access

The favored site for the ramp portal is located at an elevation of 835 m in the Blind Creek valley, about 550 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. Refer to Figure 10.1-01.

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 center of the orebody.

In the summer of 1996, bids were solicited from Canadian mining contractors for ramp drivage, which form the basis of this portion of the capital cost estimate. Five companies, after site familiarization, expressed continued interest in the project and submitted bids for the ramp drivage. Interested drilling contractors have also submitted



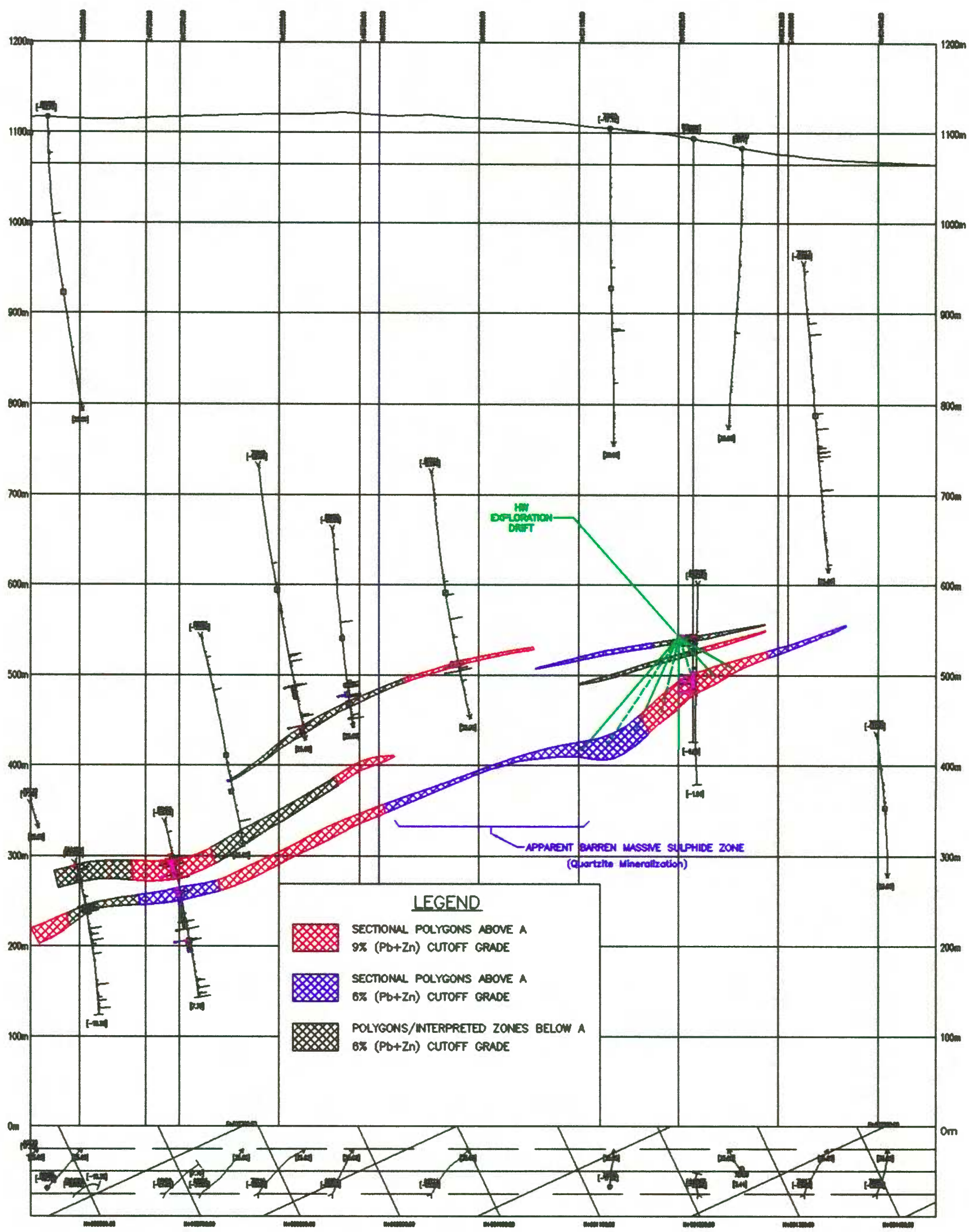
GRIZZLY PROJECT

PRE-FEASIBILITY STUDY

Fig. 7.2–01 Underground Exploration of A–Zone and B–Zone

Anvil Range MINING CORPORATION

SCALE	APPROVED	
DRAWING: EXP10UP.DWG	CHECKED	DATE 96/11



GRIZZLY PROJECT

PRE-FEASIBILITY STUDY

Fig. 7.2-02 Underground Exploration Drilling – Cross Section 600E

Anvil Range MINING CORPORATION

SCALE	APPROVED	
DRAWING: 003001MD.DWG	CHECKED	DATE 96/11

estimates. More details concerning the most interesting mining contractors are contained in Appendix D.
Ten options were considered for accessing the Grizzly deposit. The evaluation of the options and the reasons for selecting the base case are presented in Appendix D.

7.3 Exploration of B-Zone

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

From this point, a drill drift will be driven flat to the east *(Drill Drift East)* for a distance of 250 m. The drill drift will 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.

At the starting point for Drill Drift East (at elevation 530 m), another drift will be driven to the northwest, at a grade of -18% into, and switching back easterly in 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.

7.4 Exploration of A-Zone

At the end of the access ramp, the A-Zone access drift will be driven to the west downwards at -18% into the hanging wall for 238 m. At this point, a drill drift will continue to the southeast downwards at -18% for another 183 m. This will allow detailed fan drilling of block A-7, which contains large tonnage at good grades.

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

8

8 FINAL FEASIBILITY EVALUATION AND PRODUCTION ACCESS

Information from the exploration program will be compiled and continually monitored during the program. When sufficient confidence is achieved with proven mineral inventories, mining methods and other factors affecting project feasibility, a definitive feasibility study will be undertaken to prepare the way for a production decision.

The options listed in Table 8.1 for production access were identified during this Study, and of these options, the skip hoisting of crushed ore through a shaft in the center of the orebody was selected as the basis of the capital cost estimates. The final selection of the production access, however, will be based on information that becomes available during the exploration program and evaluated as part of the feasibility study. Appendix E describes the options in ore removal method and shaft locations that were considered during this Study.

Table 8.1
Options for ore removal to surface

- skip hoisting of crushed ore in vertical shaft
- inclined shaft hoisting of uncrushed ore
- conveyor haulage of crushed ore
- pumping of ore after underground crushing and grinding

9

9 MINING METHODS AND BACKFILLING

9.1 Basis of Mining Concepts

The mining methods presented in this section are largely derived from the August 6, 1992 report by Steffen, Robertson and Kirsten (Canada) Inc. Dr. James Mathis, the author of the report for SRK, drew heavily on his substantial experience in the Faro underground to anticipate the conditions of the Grizzly underground, including his assessment of such factors as rock mass rating (RMR). Although Dr. Mathis had access to only a single split core for his visual inspection, and although no compressive testing was carried out on this and other Grizzly core samples, the geological similarities between the Grizzly and Faro deposits are sufficiently strong to select SRK's proposed methods as part of the base case for this Prefeasibility Study.

As part of the 1996 study, the Anvil study team reviewed the data from past investigations; this review included the visual inspection of approximately 20 cores from the Grizzly, with particular attention paid to the core sections representing the hanging wall. N.D. Rose contributed (as part of the 1996 effort, and in addition to other geological and geotechnical interpretations) a statistically-based evaluation of RMRs representing five drillholes. Please refer to Appendix H for the full text of Piteau's discussion of "Orebody - Geotechnical Considerations".

9.2 Geotechnical Considerations

For mining the Grizzly deposit, SRK has identified the main constraints as the irregular, intermediate dip, the high ore strength with weak hanging wall, moderate ore thickness and relatively high vertical in-situ stresses. Potential tectonic stresses have also been mentioned.

Because of its greater depth, the in-situ stresses at Grizzly will be much higher than those experienced at the Faro underground. SRK estimated

Grizzly's vertical overburden stress within the ore to be two to three times greater than at the Faro underground, with horizontal stresses up to twice the vertical stress.

SRK's report cautions the reader that their proposed conceptual mining method is based in part on limited drillhole data and forced interpretations between drillholes. While their discussion emphasizes the weakness of the hanging wall (RMR 20 to 35), the weak foliation and relative strength of the sulphide ore (RMR 45 to 60), subsequent discussions with Dr. Mathis during the week of November 17, 1996 confirmed the variability of the underground conditions that could be encountered and thus the need to take a conservative approach at the inception of planning and execution.

N.D. Rose calculated RMRs for weathered calcareous phyllite and calcareous phyllite to be 35 and 41 respectively. Although these strengths are higher than those estimated by SRK, the differences may only be an indication of the expected variability and cannot be construed as definitive parameters for mine planning purposes.

Visual inspection of approximately twenty Grizzly cores included observations that the hanging wall appeared competent in many samples and that the phyllite had not turned to mud despite the age of the cores (+15 years old). No attempt was made to relate the apparent competence of the material to its competence when subjected to underground in-situ stresses. Other samples, particularly at fault intersections and with weathered phyllite, exhibited the expected weaknesses.

9.3 Concrete Pillar Method and Other Methods

The concrete pillar and standard benching method with backfilling (CPM) forms this Study's base case for ore zones 6.5m and thicker. As the proposed underground program provides better data and experience concerning the Grizzly deposit, other mining methods may merit stronger consideration. Undercut-and-fill and caving are mentioned as possibilities by SRK, whereas the conventional room-and-pillar method in the thicker zones appears to be problematic (in terms of recovery) if the risk of a weak hanging wall is present.

Procon Mining & Tunneling Ltd. suggested the post pillar cut-and-fill (PPCF) for stopes with an average vertical thickness of 15 m and an average grade of 27°, developed in the footwall of the orebody. The description of their proposed mining method is contained in Appendix G.

For ore zones thinner than 6.5m, this report proposes a modified room-and-pillar method without backfilling.

Various technical issues surrounding mining methods will need to be addressed in the future. Even with the CPM favored by SRK, the practicality and cost-effectiveness of placing high-quality fill tightly in the concrete pillars remains to be proven. SRK identify allowable displacement, support requirements, location of the production shaft and pillar loadings as issues needing resolution before a definitive mining decision.

9.4 Mining Methods

Two mining methods are proposed in this report, depending on the thickness of ore in a given area.

In areas where the ore thickness is less than 6.5 m, room and pillar mining without backfill is anticipated, with an expected recovery of 70%. Three steps would be involved:

- 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 center of the pillar;
- pillar robbing.

Figure 12.4.01 shows the configuration of a stope in an area where the bed is dipping at about 30 degrees.

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 safely 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.

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 per unit.

In areas where the ore thickness is greater than 6.5 m concrete pillar mining with backfill is anticipated with a recovery of 85%. Elongated rooms would be mined, followed by placing cemented fill and pillar recovery in second pass mining. The following steps are involved:

- Development of top cut close to the hanging wall and back support;
- Vertical or horizontal benching;
- Mucking (remote scoop trams);
- Cemented backfill (for stopes);
- Pillar extraction with backfill without cement.

The concept of concrete pillar mining is shown in Figure 12.4.02. Devleopment consists of a footwall drift, a hanging wall drift and access ramps, all in ore. Broken ore is loaded with scooptrams and hauled to a truck loading station, from where it is transported with 40 tonne trucks to the underground crusher.

A top cut is driven near the hanging wall. Eight foot resin rebar bolts are expected to be suitable for back support. Additional use of straps and/or mesh is anticipated.

Vertical or horizontal benching follows. Mucking will be done with scoop trams, remotely operated for vertical benching.

The empty stope is then backfilled with cemented fill. Fill will normally be brought by truck at the hanging wall access and dumped into the stope. Final stowing against the back will be done by use of a small scoop fitted with a blade. Tight packing is important. Cement slurry is added as the fill material is dumped into the stope. Pillars between the backfilled stopes are then mined in second pass mining using uncemented fill.

Production output per unit is derived from the following three steps of operation:

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. Vertical benching using an average bench thickness of 6.5 m and an average daily advance of 6.6 m produces:

   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.

Dry backfill is anticipated for high stopes to optimize overall recovery. Cemented fill will be used in primary stopes. Pillar extraction is by second pass mining with uncemented backfill.

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 is expected to provide a sufficiently strong fill. Testing will be required to confirm these initial assessments.

Cement is delivered from surface as slurry in a 2-inch pipe underground, to be added to the backfill as it is dumped into the stope.

Testing is necessary for the design of the proper mix of backfill materials, but could consist of:

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

Backfill is delivered from the underground silo to the stopes in 26-tonne trucks.

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.

10 MINING EQUIPMENT

10.1 Major 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.

10.2 Ancillary Equipment

10.3 Basis of Equipment Cost Estimate

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.

11 VENTILATION, MINE DEWATERING ELECTRICAL DISTRIBUTION

11.1 Ventilation

Fresh air requirement is based on the Yukon Mines Act, which states the need of 75 cubic metre per minute (cfm) of fresh air per diesel brake horsepower. Additional requirements for people and the ventilation of special locations have no significance for the calculation of air requirements. Based on this a total of 500,000 cfm is required for Grizzly.

The 18 foot production shaft, in the centre of the orebody, supplies fresh air to the mine. The concrete lined circular shaft with rope guides and without steel sets causes minimum friction. Exhaust air is brought to surface via the twin ramps and vertical bored ventilation raises. Using this arrangement requires 1,700 fan horsepower. Fresh air fans and mine air heating will be installed near the shaft collar.

Auxiliary ventilation for development work and special ventilation requirements will be provided by skid mounted 60 and 30 KW portable fans in combination with flexible ducting.

11.2 Mine Dewatering

The capacity of the main pumping system has been designed for 300 imperial gallons per minute (22.73 litres per second). A standby pump with the same capacity has been included for emergency.

Each pumping system consists of a 6 inch, 150 KW stainless steel submersible pump and a booster pump with the same capacity, installed half way up the shaft.

Small 5 KW sump pumps will be used for drift advance, three collection pumps at 60 KW each will be installed at strategic locations throughout the mine.

11.3 Electrical Distribution

A new overland transmission line at 138 KV will supply electric power to a tranformer station. Further power distribution on surface and underground is at 13.8 KV, 6.9 KV and 600 Volt. Figure 12.8-01 shows the electrical schematic, a detailed list of electrical items is included in Table 14.3.1, of Chapter 14, Capital Estimates.

12

12 MILLING AND METALLURGY

12.1 Testwork

The test work on Grizzly is based on work done by Lakefield Research 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 Table 12.1.

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.

12.2 Concentrate Specifications

The preliminary multi-element analysis for the Grizzly concentrates are presented in Table 12.2. The final analysis will be part of the underground exploration program.

Table 12.1
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.*

Table 12.2
Concentrate Specifications

	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%

12.3 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 will require 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. Table 12.3 summarizes the operating costs for the mill.

Table 12.3
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

13

13. ENVIRONMENTAL CONSIDERATIONS

The Grizzly deposit is located in the drainage basin of Blind Creek, southeast of the Vangorda Plateau. No mining activity has been undertaken 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. 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 with regard to 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 4, 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., 110 m. vertically above the creek and approximately 550 m. horizontally from it. This site offers significant advantages over those further up hill and farther from the creek but also increases the risk to the creek and to Anvil from a mishap at the portal.

Dewatering flows from underground will increase as the decline lengthens. It is anticipated that up to 300 USGPM of water may have to be pumped from the decline on a steady basis. 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.

The Study indicates 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.

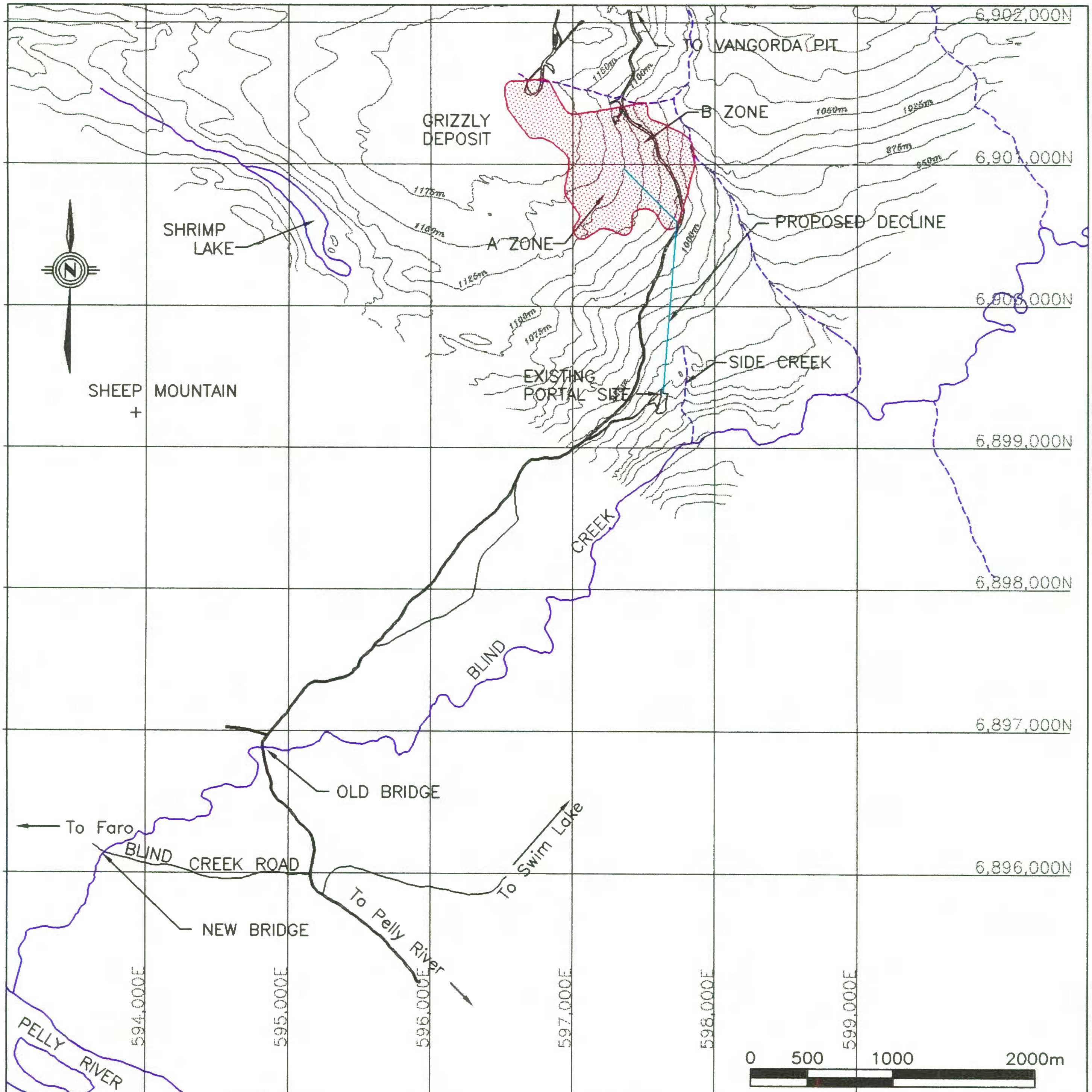
Application for a Type "B" Water License for the exploration program was filed on October 22, 1996. It is anticipated that the licensing process will take from 3 to 6 months assuming that a public hearing is not required. Under current legislation it would be possible to begin the decline without a water 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 poses several permitting and operational challenges, there is little reason to believe that mine development can not proceed at reasonable cost and with little environmental impact.



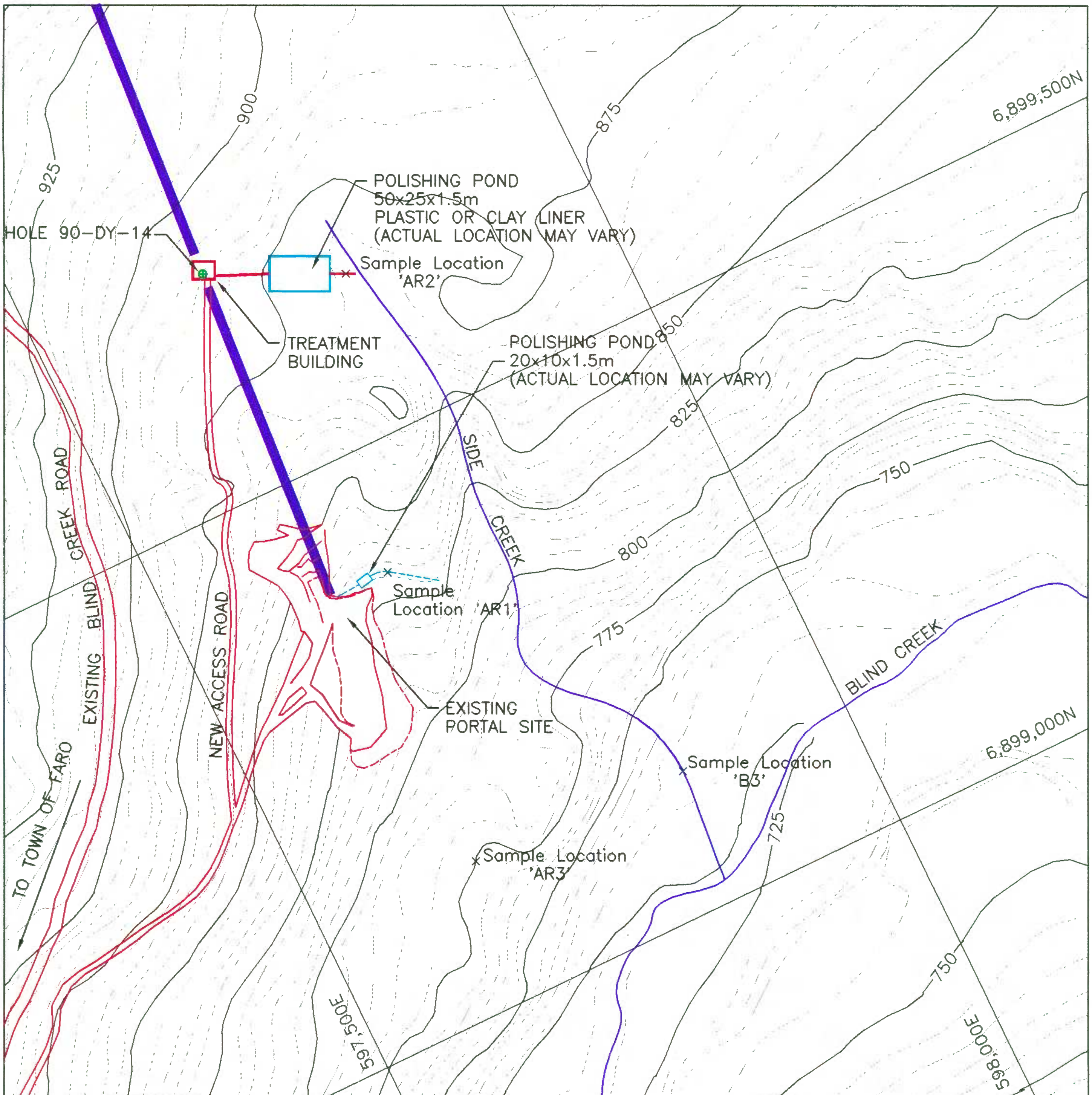
GRIZZLY PROJECT

PRE-FEASIBILITY STUDY

Fig. 13.0-01 Surface Plan

Anvil Range
MINING CORPORATION

SCALE 1 : 40,000	APPROVED	
DRAWING: VICINITY.DWG	CHECKED	APPROVED 96/09



GRIZZLY PROJECT

PRE-FEASIBILITY STUDY

Fig. 13.0-03 Layout of Treatment Facilities and Water Discharge

Anvil Range
MINING CORPORATION

SCALE 1 : 5,000	APPROVED	
DRAWING: TF-SITE.DWG	CHECKED	APPROVED 96/10

14

14. TRANSPORTATION

At the minesite, the lead and zinc concentrates are loaded onto 10-axle B-Train units that carry a 50 wet tonne payload of either lead or zinc concentrates. The concentrates are trucked 552 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.

Unloading trucks and loading ships at Skagway are done by a contractor, with Anvil supplying most of the equipment.

Table 14.1
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**

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 Table 14.1.

14.1 Port Costs and American Road Tax

Currently Anvil is the only user of the Skagway port. As a result, Anvil pays 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 Table 14.2, the road tax is set at US $0.40 per wmt.

Table 14.2

American Port Costs - US Dolllars

	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**

14.2 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. Cost details are shown in Table 14.3.

Table 14.3
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**

15

15. MARKETING

15.1 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.

Table 15.1 sets out the amount of zinc and lead concentrates shipped from Skagway, Alaska, to Europe, Asia, and other destinations for the periods indicated.

Table 15.1

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		

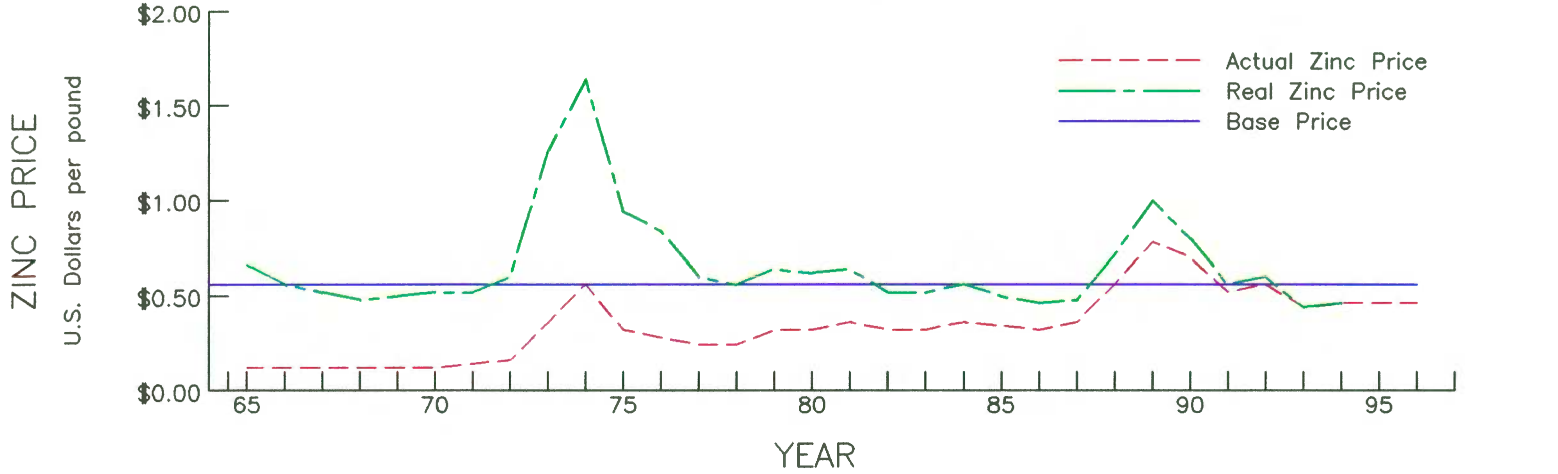
15.2 Pricing

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

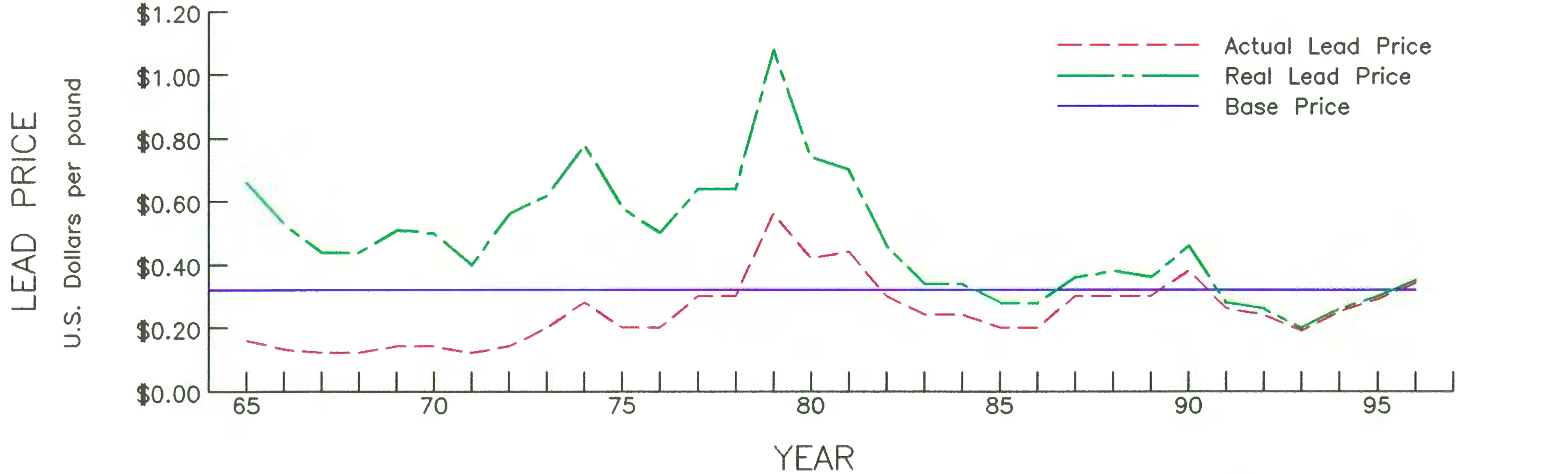
15.3 Concentrate Terms

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.



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

DRAWING: PRICES.DWG

APPROVED

DATE 96/11

GRIZZLY PROJECT

PRE-FEASIBILITY STUDY

Fig. 15.1 LME Lead/Zinc Prices

Anvil Range CORPORATION

16

16. CAPITAL COST

16.1 Summary

Table 16.1 summarizes the capital cost estimate for the base case. The Phase 1 development includes the cost of underground exploration and the expenses to bring the project to the production decision.

Table 16.1
Capital Cost Estimate Summary for the base case

	Cdn $
Phase 1 development	26.4
Construction	51.6
Mining equipment	17.1
Replacement Capital	11.0
Total	106.1

Until the proposed underground exploration program provides sufficient geological information and allows physical access to the ore horizons, it will not been possible to definitively lay out the mine plan or assess mining performance in designs which will be used when the deposit is mined. The present cost estimate is therefore limited in an accuracy: the normally acceptable accuracy for a study of this nature is ± 20%. An additional contingency has been included where appropriate.

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.

Table 16.2 presents the capital cost estimate for mine construction and mine infrastructure. Additional details, including estimated cash flows, are presented in Appendix F.

Table 16.2
Capital Cost Estimate
Mine Construction and Mine Infrastructure

	Cdn$
Shaft pilot hole	0.2
Shaft, headframe, hoist, ropes etc(1)	25.0
Surface bin	1.3
Surface haul road	0.7
Underground development	5.6
Ramp to shaft bottom	0.7
Surge bin	2.6
Loadout conveyor	0.2
Crusher	1.5
Main conveyor	1.3
Ventilation raise	1.6
Pumping & ventilation	2.0
Electrical installations	2.8
Backfill installation	2.0
Underground shop and warehouse	0.9
Miscellaneous items	2.1
Total	50.5
Contingency	1.1
Total with contingency	51.6

Contractors' quotations have been used for ramp drivage, underground exploration, shaft sinking, as well as headframe, supply of hoist and silo construction. In particular, the price of the shaft is based on an order-of-magnitude estimate from J.S. Redpath Ltd. of October 11, 1996. This estimate (see Appendix F) is for a shaft of 18 feet diameter and 2,800 feet (856 metres) deep . The Redpath price has not been adjusted for the base case shaft which is 340 feet shorter than estimated by Redpath. The savings in shaft sinking of Cdn$ 1.1 million plus an additional 10% of the cost for underground development ($5.6), ramp to the shaft bottom ($0.7), surge bin ($2.6) and crusher ($1.5) have been applied as contingency in *Capital for Mine Construction and Mine Infrastructure*.

16.2 Exploration Phase

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

Underground exploration cost estimates up to the Decision Day are based on target prices from Procon Mining and Tunneling Ltd. A contingency of 10% has been applied to ramp drivage and lateral drifting to allow for unforeseen conditions that might require additional shotcrete, or extra pumping.

Exploration drilling costs have been estimated by combining a typical drilling

Table 16.3
Phase 1 Capital Cost

Underground Exploration up to Decision Day Cdn$ (millions)		
Twin ramps	8.7	
Exploration drifts	4.0	
Contingency @ 10%	1.3	
Subtotal Ramps & drifting		14.0
Muck removal		6.0
Exploration drilling		3.3
Testing	0.9	
Feasibility study	1.6	
Permitting	0.6	
Subtotal Testing etc		3.1
Total Phase 1		26.4

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.

16.3 Capital for Mining Equipment

The capital cost estimate for mining equipment is shown in Table 16.4.

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.

Table 16.4
Mining Equipment Capital Cost

	No. of Units	Unit Price Cdn$('000)	Total Cost Cdn$('000)
Twin boom jumbo	4	600	2,400
Bench drills	2	520	1,040
Bolting jumbo	4	520	2,080
Core drill	1	120	120
Scoop tram LHD 8yd	4	630	2,520
Scoop tram LHD 6yd	2	500	1,000
Scoop tram LHD 3.5 yd	2	350	700
Scoop tram LHD 2yd	2	225	450
Trucks 40 tonne	3	640	1,920
Trucks 26 tonne	2	430	860
ANFO loader	2	360	720
Shotcrete unit	1	150	150
Forklift	1	175	175
Scissor lift	1	190	190
Road grader	1	350	350
Portable compressor	2	60	120
Stopers and jacklegs	lot	250	250
Man carrier	2	100	200
Lubrication truck	1	230	230
Service truck	1	150	150
Supervisor's vehicle	3	50	150
Emergency vehicle	1	75	75
Surveyor's vehicle	1	50	50
Geology vehicle	1	50	50
Mechanics' vehicle	1	50	50
Electricians' vehicle	1	50	50
Subtotal			16,050
Freight (6%)			960
Total			**17,010**

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

16.4 Capital for Replacements and Additions

Proper training of personnel in the operation of equipment is necessary to avoid abuse of machinery, instil safety consciousness and obtain good productivity. Furthermore, proper daily service of equipment, preventive maintenance programs, scheduled major overhauls and rebuilding of key equipment are assumed. Funds have been allotted for these activities in the operating costs.

Some new equipment will be purchased as replacement after seven years of operation. An additional haulage truck will be required to bring ore from the lower areas of the A-Zone to the crusher. A second ventilation raise will be constructed. Table 16.5 presents the replacement schedule after Year 7. Cost are in 1996 Canadian dollars.

Table 16.5
Replacement and Addition Schedule

	No. of Units	Unit Price Cdn$('000)	Total Cost Cdn$('000)
Twin boom jumbo	2	600	1,200
Bench drills	1	520	520
Bolting jumbo	1	520	520
Scoop tram LHD 8yd	3	630	1,890
Scoop tram LHD 6yd	1	500	500
Scoop tram LHD 3.5 yd	1	350	350
Scoop tram LHD 2yd	1	225	225
Trucks 40 tonne	3	640	1,920
Trucks 26 tonne	1	430	430
Scissor lift	1	190	190
Portable compressor	1	60	60
Stopers and jacklegs	lot	125	125
Man carrier	2	100	200
Lubrication truck	1	230	230
Service truck	1	150	150
Supervisor's vehicle	3	50	150
Surveyor's vehicle	1	50	50
Geology vehicle	1	50	50
Mechanics' vehicle	1	50	50
Electricians' vehicle	1	50	50
Subtotal			8,860
Freight (6%)			530
Ventilation raise			1,600
Total			**10,990**

17

17 OPERATING COST

17.1 Operating Cost per Tonne of Ore

The summary of estimated operating costs is presented in Table 17.1. Details of the cost derivation are presented in this section.

Table 17.1
Summary of Estimated Operating Costs per Tonne Mined

	ITEM	Cdn$
1	Labor and staff	8.28
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 and major overhauls	2.26
9	Shaft operations	0.29
10	Conveyors	.12
11	Power	1.62
12	Sub-contractors road maintenance, cleanup	.32
13	Miscellaneous costs, cables, pipes, safety, core drill, etc.	.60
14	Shutdown allowance	0.25
	Total Estimate per tonne	**21.64**

17.2 Operating Cost - Early Years

The production is planned as follows:

Year 1	500,000 tonnes
Year 2	1,200,000
Year 3	1,500,000

As illustrated in the Schedule, it is only planned to produce 500,000 tonnes in the first (partial) year of operation, following completion of construction that year. All new employees will require some familiarization, including safety, operating or maintenance training. Staff will be on site before production begins, to prepare their departments, set up programs and participate in the induction of new employees.

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 1,200,000 tonnes per year. Costs other than labor 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.

17.3 Basis of Operating Cost Estimate

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 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 itemized categories have been isolated for examination in considerable detail. A further miscellaneous category (13) has been added to cover smaller cost items. Shutdown allowance is shown as category (14).

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 *Mining Methods*, 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 three production crews and one development crew.

Table 17.2
Manpower Requirements
8-hour shifts, 7 days/week

Jobs	Per Shift	Per Day	Extra Shift	Total
Production Miners	12	36	12	48
Truck Haulage	3	9	3	12
Mechanics	4	12	4	16
Electricians	2	6	2	8
Construction Crew	10	30	10	40
Development				
Backfill				
Ground Control				
Ventilation				
Pumping				
Road Maintenance				
Control Centre	1	3	1	4
Crusher	1	3	1	4
Shaft & Loadout	1	3	1	4
Surface	2	6	2	8
Backfill				
Loadout				
Utility	2	6	2	8
Training	2	6	2	8
Diamond Drilling	2	6	-	6
Shop Mechanics	8	8	-	8
Shop Electricians	4	4	-	4
Total				178

17.4 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:

Table 17.3
Staff Requirements for Grizzly and shared positions for Grum underground

	Loaded Cost per position	Grizzly portion		Grum UG portion (1)	
	Cdn$	%/ persons	Cdn $	%	Cdn $
Mine Manager	130,000	75%	97,500	25	32,500
Mine Superintendent	110,000	100%	110,000		
Chief Mine Engineer	100,000	75%	75,000	25	25,000
Chief Geologist	100,000	75%	75,000	25	25,000
Maintenance Super	100,000	75%	75,000	25	25,000
Planning Engineers	60,000	2	120,000		
Geologists	60,000	2	120,000		
Shift Engineer & Geol	60,000	4	240,000		
Maintenance Engineer	60,000	1	60,000		
Safety Supervisor	80,000	75%	60,000	25	20,000
Training Supervisor	80,000	75%	60,000	25	20,000
Maintenance Clerk	50,000	1	50,000		
Office Clerk	50,000	1	50,000		
Shaft Foreman	70,000	1	70,000		
Shift Captains	320,000	4	320,000		
Construction Foremen	70,000	1	70,000		
Shop Foremen	80,000	1	80,000		
UG Warehouse	60,000	1	60,000		
Total	2,060,000		1,912,500		

Notes (1) Grum UG portion indicates only those positions shared with Grizzly.
(2) The Shift Captain is the only staff position on shift. Small production, construction and other operating units are managed by lead hands.
(3) Total staff for Grizzly is 26.

- 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. Average annual cost per hourly-paid personnel = 233 x $32 x 8 = $59,648

Table 17.4
Labor & Staff Costs per tonne mined

	Annual cost Cdn$	Cdn$/ tonne
Hourly personnel (1)	10,617,344	7.08
Staff (2)	1,792,500	1.20
Total Hourly + staff	**12,409,844**	**8.28**

(1) 178 persons @ average annual cost $59,648.
(2) Based on 1,500,000 tonnes per year.

An 8-hour shift schedule conforming to the Yukon Occupational Health and Safety Regulations has been selected. Special dispensation can be obtained for 10-hour shifts, which would permit operations with fewer people.

Adopting the 10-hour continuous work schedule, as shown on Table 17.5, reduces the total mine crew from 178 to 155 people, a reduction of $0.92 per tonne from $7.08 to $6.16 per tonne for all people on hourly pay. However, no adjustment is warranted before a dispensation is granted. Additional equipment required by this modified schedule amounts to one drill jumbo, one 8-yd LHD, one 40 tonne truck and one man carrier.

The continuous schedule of 10-hour shifts engages 123 people, which include production, development, construction, exploration drilling and shift mechanics and shift electricians. Two people are included under "training" to allow for operator and maintenance training as well as training for mine rescue, first aid and allowance for vacation and unscheduled days off.

The continuous schedule of 8-hour shifts engages 24 people, including a mechanic and an electrician on each shift. This crew looks after the operation and maintenance of the ore haulage and hoisting systems from the crusher to the surface bin, which includes the maintenance of the shaft and the hoist.

Eight tradesmen are on a 5 and 2 schedule to look after shop repairs, and, together with the mechanics and electricians on shift, carry out preventive maintenance. Major overhauls will be done with the assistance of contractors.

Some operating staff will share their responsibilities and costs between the Grizzly and Grum underground operations, generally in

Table 17.5
Manpower Requirements
Combination of
10-hour shifts, 7 days/week
8-hour shifts, 7 days/week
8-hour shifts, 5 days/week

Jobs	Per Shift	Per Day	Extra Shift	Sub-Total	Total
Continuous schedule, 10-hour shift:					**123**
Production Miners	16	32	16	48	
Truck Haulage	4	8	4	12	
Mechanics	2	4	2	6	
Electricians	2	4	2	6	
Construction Crew	10	20	10	30	
Development					
Backfill					
Ground Control					
Ventilation					
Pumping					
Road Maintenance					
Crusher	1	2	1	3	
Utility	2	4	2	6	
Training	2	4	2	6	
Diamond drilling	2	4	2	6	
Continuous Schedule, 8-hour shifts:					**24**
Control center	1	3	1	4	
Shaft & Loadout	1	3	1	4	
Electrician	1	3	1	4	
Mechanic	1	3	1	4	
Surface	2	6	2	8	
Backfill					
Loadout					
5+2 Schedule, 8-hour shifts:					**8**
Shop Mechanics	6	6	-	6	
Shop Electricians	2	2	-	2	
Total					**155**

a 75% Grizzly: 25% Grum ratio. The assumed distributions are described in Table 17.3.

Positions such as General Manager, Director of Accounting, Director of Human Resources and others are included in the costs of general overhead.

17.5 Material Cost Estimate

17.5.1 Roof Support

The estimate is 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) at a cost of $116 per linear metre; and in high stopes, 8 foot rebar with resin will be used on similar centres at a cost $120 per linear metre.

The cost per tonne is $0.739 for room and pillar and $0.319 for high stopes, at an average cost (weighted 29% room and pillar, 71% high) of $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.

Difficult roof conditions were experienced from time to time in the Faro underground mine and similar or worse conditions are expected at Grizzly coupled with indicated faulting. For this reason, 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, and some steel sets or shotcrete.

17.5.2 Backfilling Costs

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

Table 17.6 presents the expected source of backfill material and the estimated costs.

Table 17.6
Backfill Material Sources and Estimated Costs

	%	Cdn $/tonne	Cdn $/t (fill)
Screened local gravel*	85		1.00
Tailings from concentrator	11	5.00	
Cement	4	200.00	
	Average		9.40
Chemicals			1.00
Haul & place			1.25
Total cemented backfill			11.65

*screened local gravel from within 3 km, cost at pipe to UG

Cement and chemicals are added underground. The chemicals are estimated to cost $1 per tonne 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 are estimated to be 75 cents and 50 cents per tonne, respectively.

Bulk cement prices are based on current Vancouver quotations at $125 per tonne, and an assumed landed price of $200 per tonne.

For larger tonnages, the price of backfill per tonne of ore remains the same.

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

Table 17.7
Backfilling Assumptions

Ore SG:	3.92
Backfill solids SG:	2.9
Volume per 100t/ore:	25.51 m^3
Voids in backfill:	20%
Apparent SG with voids:	2.35
Tonnes backfill/ 100t ore:	60
Average daily backfill @71% (3500 t/d):	2485 t
Daily cemented backfill:	746 t
Daily uncemented backfill:	
from mine waste:	400t
from surface gravel:	346t
Daily cost of backfill	
- cemented @$11.65/t:	$8,691
- uncemented	$1,650
- Total daily cost of backfill	$10,341
Backfill cost / tonne ore	$2.95

17.5.3 Crushing/rock breaking

A centrally-located jaw crusher will reduce ore to 6 inches (150 mm) for hoisting to surface. The cost for crushing is estimated to be 15 cents per tonne of ore.

Table 17.8
Fuel and Lubricant Cost

	Cdn$/tonne ore
Propane	0.663
Lubrication	0.360
Misc	0.100
Diesel fuel	0.630
Total	1.753

Includes GST and all taxes. ($0.141 for diesel)

17.5.4 Fuel and Lubricants

Table 17.8 summarizes the estimated costs for 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.

Table 17.9
Diesel Fuel Cost Estimate @ $0.44/litre

Equipment	Fuel l/hour	hours/ day	fuel/day litres	cost/day Cdn$	cost/t Cdn$
3 trucks 450 h.p.	45	18	2430		
4 scoops 300 h.p.	24	12	1800		
2 scoops 185 h.p.	20	15	600		
1 small scoop	6	3	450		
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	919	
Total cost			6180	2719	.630

Propane costs are based on CMD's 1992 estimate of $0.47 per tonne, escalated by Consumer Price Index of 1.08016 to arrive at $0.566 per tonne of ore. By comparison, records indicate that propane costs in October 1992 during the Faro underground operations were $0.20 per tonne.

Lubrication costs are based on historical Faro underground costs of $0.318 per tonne, escalated and adjusted for the base case. By comparison, Caterpillar's handbook indicates $0.290 per tonne for twelve similar machines.

17.5.5 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.

The cost estimate of $1.80 per tonne mined includes an average for development drivage in waste during a normal year.

17.5.6 Drill Rods and Bits

These items were selected as a cost centre and estimated at $0.45 per tonne of production.

17.5.7 Parts for Mining Equipment and Major Overhauls

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.

The capital cost of initial equipment is estimated at $17 million. Ten percent of this number divided by the annual tonnage works out at $1.13 per tonne.

The estimate for major overhauls is assumed to be the same as for spare parts, namely $1.13 per tonne for a total (spares + overhauls) of $2.26 per tonne.

17.5.8 Shaft Operations

Estimates by CMD and Redpath indicated that about $0.29 per tonne would be realistic for maintenance and replacement in this important area.

17.5.9 Power

The calculations of power consumption are shown in Table 17.10.

Average calculated from present mining operations is $0.088 per kilowatt hour.

Table 17.10

Power Costs $000 at 8.8 cents/KWH combined 4 tariffs

		loading	hours/ year	kwh/yr 000	cost/ year $000	cost/ year $000
main fan surface	1700 kw	90%	8400	12852	1,131	
misc. orebin, conveyor	120 kw	50%	8760	526	46	
					1,177	1,177
UG shop, etc.	285	50%	8760	1248	110	
Pump	2x180	65%	8760	2650	233	
					343	343
NE first unit						
2 jumbos/bench	216	50%	4500	486	43	
2 bolter	90	50%	4000	90	8	
2 crushers	240	25%	6000	360	32	
1 conv.	nil					
4 fans 30kw	120	60%	7000	504	44	
					127	127
NE second unit (same as first)						127
1 jumbo	108	50%	4500	243	21	
1 bolter	45	50%	4500	101	9	
1 crusher	120	25%	6000	180	16	
1 compressor	60	60%	7000	252	22	
2 fans 60 kw	120	60%	7000	504	44	
2 pumps	120	50%	6000	36	3	
3 small pumps	45	50%	7000	157	14	
Sub-Total					130	130
Hoist 1500 kw	1500	70%	7500	6825	601	583
Total						2,486
Cost per tonne of ore						**$1.62**

The average cost over the mine life is estimated at $1.62 per tonne of ore.

17.5.10 Sub-Contractors

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

- Haulroad repair or snow removal;
- Extra diamond drilling;
- Major equipment overhauls;
- Consultants of various disciplines;
- Extra rock excavation;
- Shaft and hoist maintenance.

An estimate of 32 cents per tonne is used.

17.5.11 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.

18

18 PRODUCTION PLAN AND SCHEDULES

18.1 Basis of Production Plan

Piteau Associates have prepared estimates of mineable inventory in Section 5 of this report, based on a plan view polygonal reserve calculation at 6% and 9% (Zn + Pb) cutoff grades.

Dilution and mining recovery factors, as discussed by Piteau and in Section 9 on mining methods, will determine the actual economic cutoff grades once production begins. The practicality of achieving, say, a 6% cutoff grade will be better understood with information from the underground exploration program.

Sections 5 and 7 of this report describe the general configuration of the Grizzly deposit and the initial exploration and drifting program into Zones A and B. Production mining is expected to begin in the high-grade Zone B, which is fairly thick and located less deeply than Zone A.

The mine plan and mining sequence presented in this section are intended to maximize early mining of the higher grade ore, with correspondingly earlier operating profits.

Table 18.1
Production Parameters

Annual mine production:	1,500,000 tonnes
Daily mine production:	4,300 tonnes
Year 1 production:	500,000 tonnes
Year 2 production:	1,200,000 tonnes
Year 3 production:	1,500,000 tonnes
% of ore in thickness:	
▸ exceeding 6.5 m:	70%
▸ below 6.5 m:	30%

18.2 Net Smelter Return Calculation

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 18.1 to 18.5 at the end of this section. A total operating cost of Canadian $68.50 was assumed for this exercise. This figure is slightly more conservative than the operating cost estimates developed in the study, but the difference is not considered to be significant for the purposes of this study.

Table 18.2.1
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

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 Table 18.2.2.

Table 18.2.2
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 18.2.1 and 18.2.2 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.

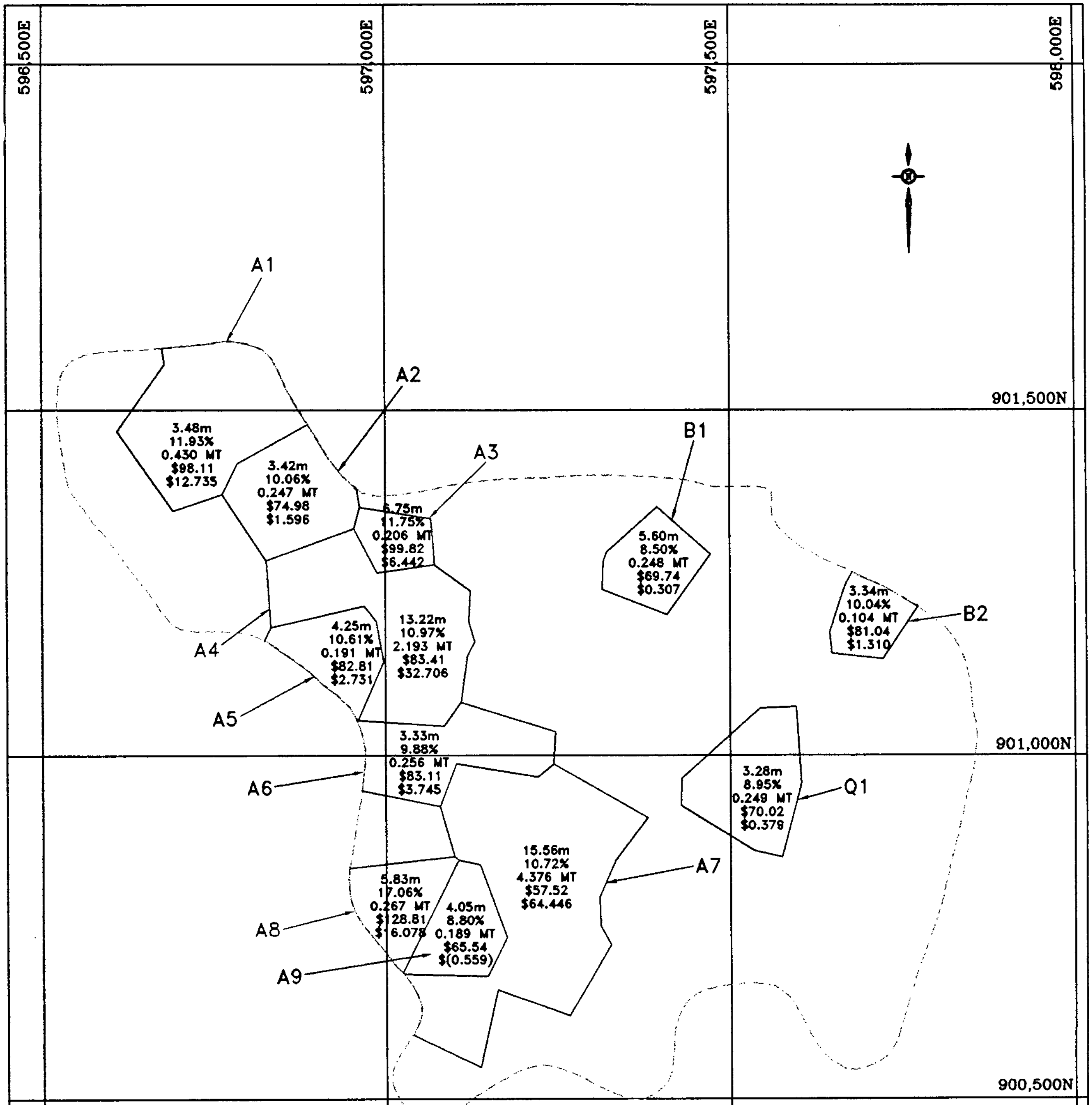
For this study, only the mineral inventory as defined by the 9% cutoff figure will be included in further mine planning.

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

Figure 18.2-01	Upper Horizon
Figure 18.2-02	Lower Horizon

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



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

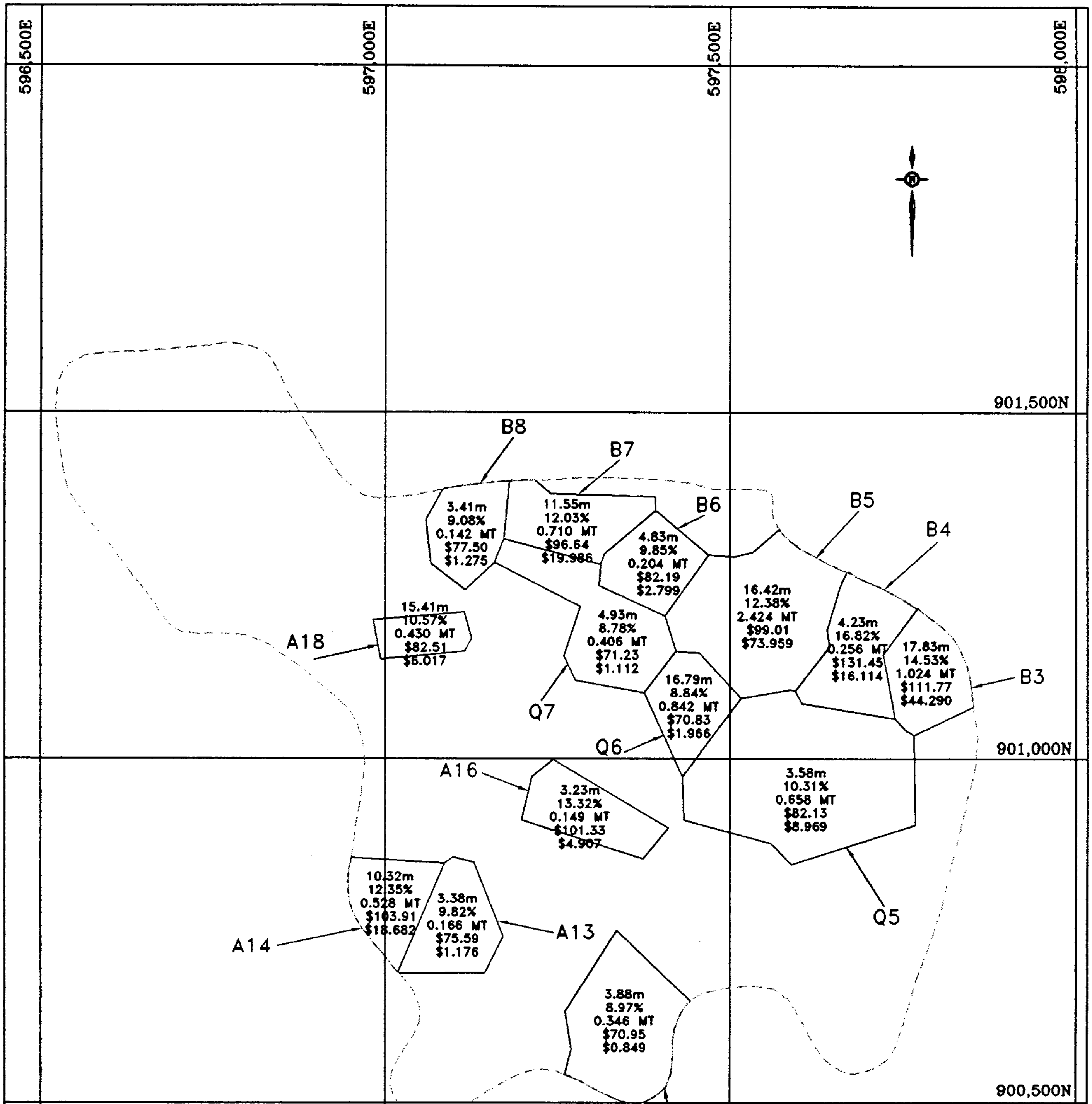
NOTES

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

FIG. 18.2-01

Anvil Range MINING CORPORATION

ANVIL RANGE MINING CORPORATION	DRAWN BY: NDR REVISED BY: FILE: NSRGR1C8.DWG	DATE: 07/11/96 DATE:	SCALE 1:2500	GRIZZLY DEPOSIT UPPER-G HORIZON 9% PB+ZN CUTOFF GRADE ORE BLOCKS WITH NSR AND NR VALUES



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. 18.2-02

Anvil Range Mining Corporation

ANVIL RANGE MINING CORPORATION	DRAWN BY: KDR APPROVED BY: FILE: NSRGR2C9.DWG	DATE: 07/11/98 DATE:	SCALE: 1:2500	GRIZZLY DEPOSIT LOWER-G HORIZON 9% PB+ZN CUTOFF GRADE ORE BLOCKS WITH NSR AND NR VALUES

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 this table.

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.

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

18.3 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.

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

7.4-01 Upper Horizon (Upper G)
7.4-02 Lower Horizon (Lower G)

These are full-sized drawings contained in pockes at the back of this volume.

18.4 Sequence of Mining

Lower G of the B-Zone is the prime target for early mining.

There are three small blocks in Upper G which lie above the extensive Lower Horizon orebody. It is advisable to mine them out prior to working an orebed of moderate thickness from 20 to 70 metres beneath. The first block is B2, overlying B4, and contains only 104,500 recoverable tonnes. The second block, B1, contains only 248,000 tonnes. However, both are accessible from the planned exploration drifts in or near the Upper Horizon, to be used as a return airway for the production period.

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 in the northwest direction 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, Table 18.B6 shows 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.

18.5 Production Equipment

Four mining units are planned to permit simultaneous development and production work in both thick and thin zones.

Major production equipment consists of (See *Capital Cost* for list of mining equipment):

Twin boom jumbos	Thin stope	1
	Thick stope	1
	Development	1
	Spare stope	1
Total		4
Bench drill	(one spare)	2
Roof bolter		4
Loaders (LHD 8yd)		4
Trucks (for backfill)	26 t	2
Trucks 40		3

TABLE 18.1

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 18.2

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
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 18.3

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 HORIZO		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 18.4

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 18.5

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

TABLE 18.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$/T	Cdn$
	B2 NE	104,476	10.03	3.07	6.96	52.55	0.42	81.04	1,310,160
1	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
	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
2	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
	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
3	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
	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
4	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
	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
5	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
	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
6	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
	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
7	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
	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
8	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 18.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 18.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

Table 18.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			

19

19 ECONOMIC ANALYSIS

19.1 Scope of the Economic Analysis

This section describes the economics of the Grizzly Project according to the technical concepts and costs for the "base case" developed during the pre-feasibility study. As is typical in studies of this type, major efforts have been expended in evaluating the technical feasibility and costs of various options in project execution, such as mining methods. Important options for the Grizzly Project stem from the geology of the deposit, manifested in the geotechnical constraints which will govern mining methods and extraction recoveries. The present uncertainties are well documented in the technical sections of the report and these uncertainties will be addressed during the underground program by progressive refinement of the mining plan, mining methods and other factors such as shaft location. Such refinements to the base case concept presented in this report will ultimately affect the capital and operating costs of the Grizzly Project, and must then be considered in the final feasibility leading to the production decision of the Grizzly deposit.

A computer spreadsheet analysis of the economics is presented in its detailed form in Appendix I.

19.2 Basis of Economic Analysis

The following assumptions were used in the economic analysis:

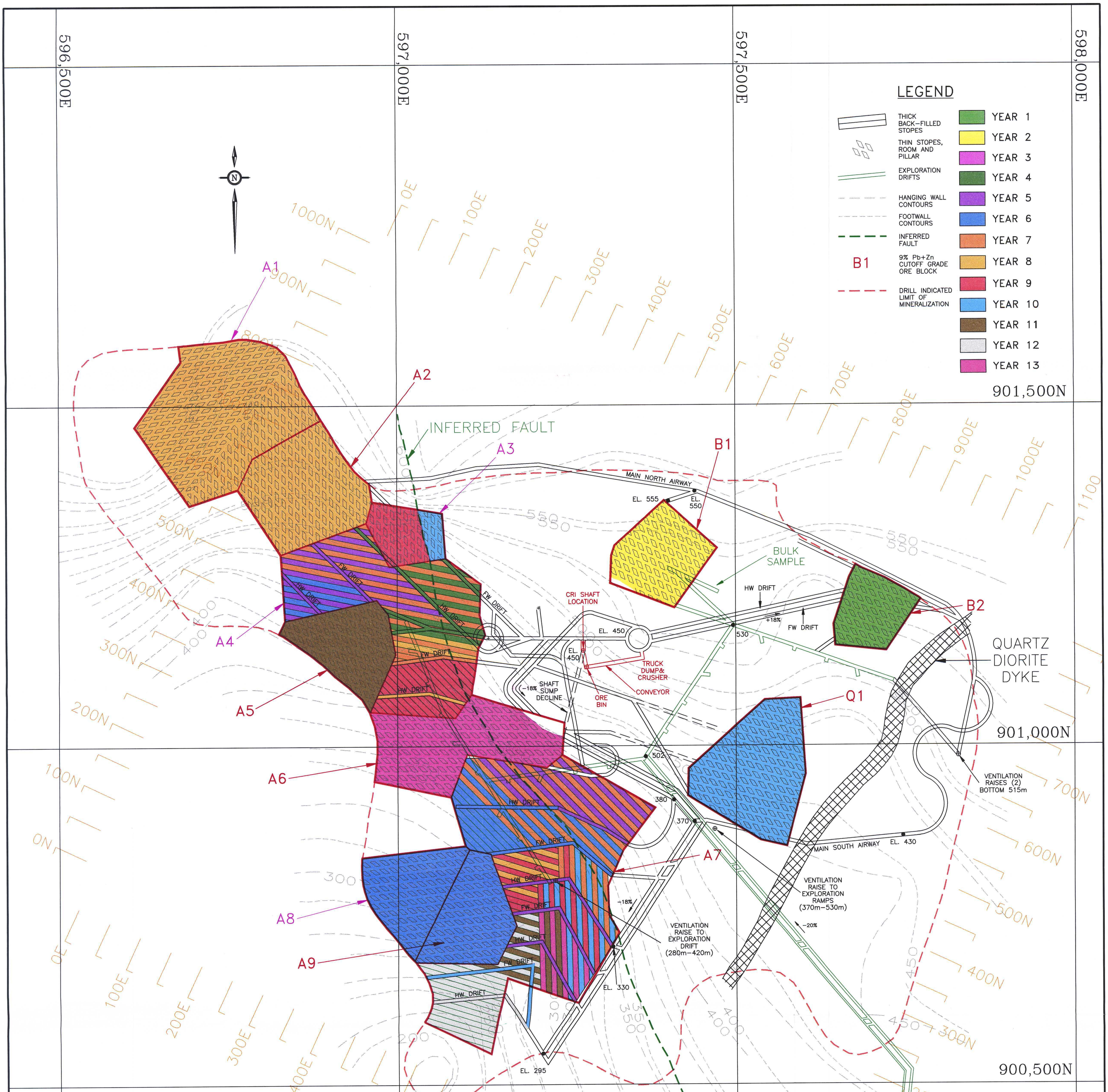
- Mine production, capital expenditures and mining costs are summarized in the technical and cost sections of this report.
- The cost of trucking the ore from Grizzly to the mill site is $3.50 per tonne.

- Mill costs are summarized in Section 12. 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 Section 14.
- Annual head office and marketing costs total $3,500,000 annually.
- Concentrate terms and prices are summarized in Section 15. The base case assumes zinc at US$ 0.55 per pound, lead at US$ 0.32 per pound, silver at US$ 5.50 per ounce, and gold at U.S. $ 400.00 per ounce.

19.3 Economic Summary

Calculated from mine startup and on an after tax basis, the project cashflows pay back the initial investment in 3.5 years. The project's internal rate of return is 16.18%. 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	230,984	44,616	172,643	27,988
Sensitivity Analysis				
Zinc + $0.01/lb	16,619	5,636	12,126	4,208
Lead + $0.01/lb	15,655	5,076	11,424	3,760
Silver + $0.50/oz	18,017	5,842	13,113	4,303
Zinc Rec + 1%	6,945	2,357	5,062	1,744
Zinc Con + 1%	9,851	3,336	7,167	2,471
Exchange Rate + .01	(14,590)	(4,823)	(10,820)	(3,670)
Site Costs + 1%	(6,763)	(2,251)	4,943	(1,694)
Zinc TC + $10	2,032	(689)	1,475	(516)
Lead TC + $1-	1,424	(462)	1,034	(343)



MINING BLOCK	AVG. VERT. THICK. (m)	IN-SITU TONNES	MINING RECOVERY (%)	RECOVERED TONNES	Pb+Zn%	% Pb	% Zn	Ag (g/t)	Au (g/t)
A1	3.48	614,375	70	430,063	11.93	4.10	7.83	73.0	0.84
A2	3.42	352,219	70	246,553	10.06	5.52	4.55	73.6	0.56
A3	6.75	241,986	85	205,688	11.75	4.79	6.96	98.5	1.23
A4	13.22	2,580,058	85	2,193,049	10.97	6.05	4.91	80.3	0.92
A5	4.25	272,591	70	190,814	10.61	5.42	5.19	72.9	1.05
A6	3.33	366,280	70	256,396	9.88	3.78	6.10	65.4	1.14
A7	15.56	5,148,346	85	4,376,094	10.72	5.23	5.49	73.9	0.81
A8	5.83	380,842	70	266,590	17.06	8.05	9.02	92.5	1.10
A9	4.05	270,016	70	189,011	8.80	4.24	4.56	60.5	0.18
B1	5.60	354,477	70	248,134	8.50	2.45	6.05	39.8	0.54
B2	3.34	149,251	70	104,476	10.04	3.07	6.96	52.5	0.42
Q1	3.28	355,934	70	249,154	8.95	2.81	6.14	29.0	0.48
TOTAL	8.13	11,086,376	80.8	8,956,022	10.85	5.19	5.66	73.1	0.83

Table: MINING INVENTORY AT 9% Pb+Zn CUTOFF AND 10% DILUTION

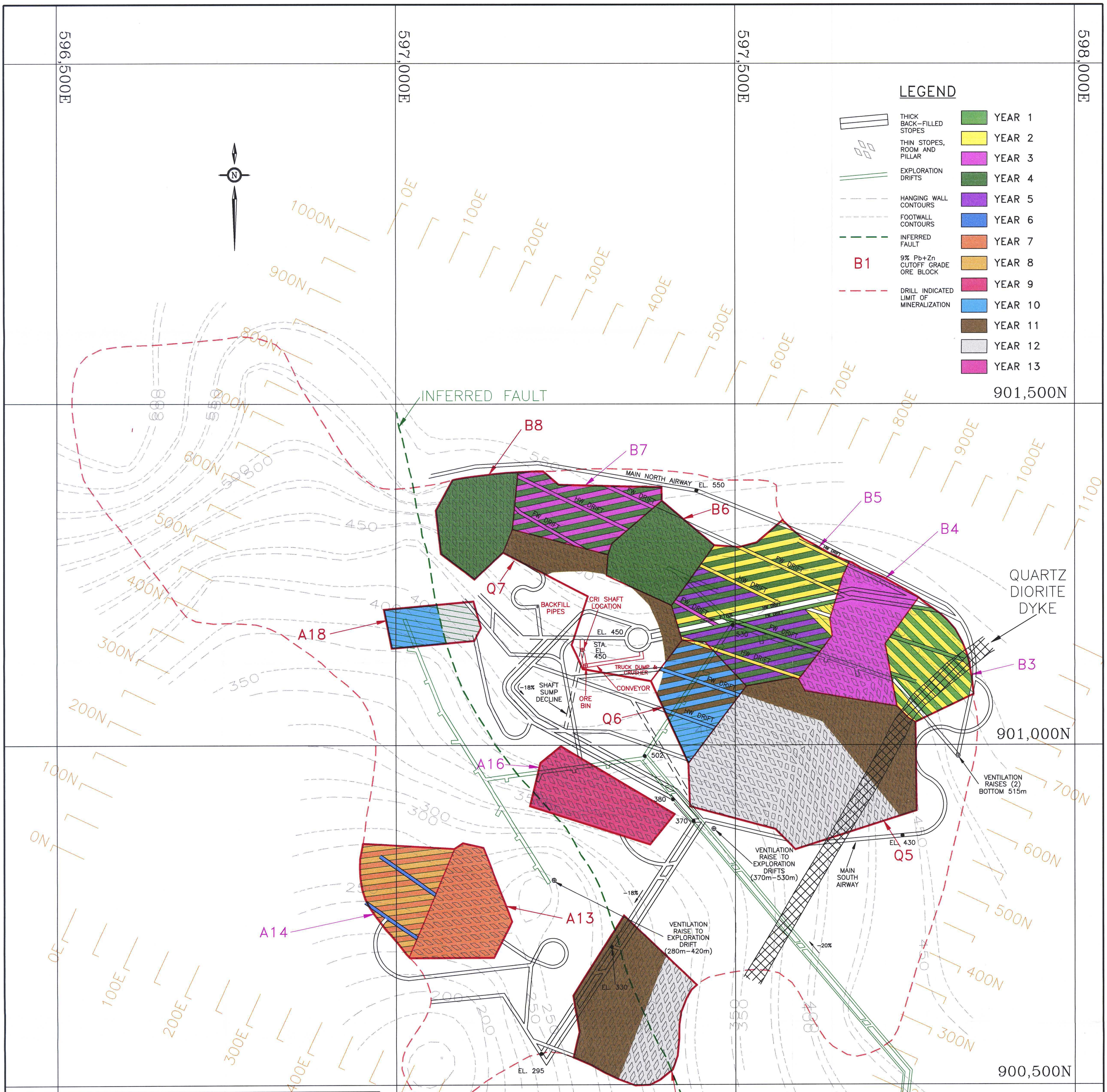
FIG. 7.4-01

Anvil Range MINING CORPORATION

ANVIL RANGE MINING CORPORATION

DRAWN BY: DP/AB
REVISED BY:
FILE: GRUC9MPB.DWG
DATE: 21/11/96
DATE:
SCALE: 1:2500

GRIZZLY PROJECT
PRE-FEASIBILITY STUDY MINE PLAN
UPPER-G HORIZON 9% Pb+Zn CUTOFF



MINING INVENTORY AT 9% Pb+Zn CUTOFF AND 10% DILUTION

MINING BLOCK	AVG. VERT. THICK. (m)	IN–SITU TONNES	MINING RECOVERY (%)	RECOVERED TONNES	Pb+Zn%	% Pb	% Zn	Ag (g/t)	Au (g/t)
A11	3.88	493,381	70	345,787	8.97	3.52	5.45	45.8	0.68
A13	3.38	236,895	70	165,826	9.82	5.19	4.63	72.1	0.81
A14	10.32	620,694	85	527,590	12.35	4.14	8.22	72.4	1.23
A16	3.23	213,537	70	149,476	13.32	8.00	5.32	127.5	0.85
A18	15.41	505,370	85	429,565	10.57	5.55	5.03	82.0	0.92
B3	17.83	1,204,217	85	1,023,584	14.53	5.35	9.17	72.5	0.41
B4	4.23	365,670	70	255,969	16.82	6.94	9.88	107.4	0.75
B5	16.42	2,851,756	85	2,423,992	12.38	4.27	8.11	71.8	0.52
B6	4.83	292,049	70	204,434	9.85	3.87	5.98	68.5	1.02
B7	11.55	835,689	85	710,336	12.03	4.18	7.84	61.9	0.79
B8	3.41	202,202	70	141,542	9.08	3.95	5.13	84.2	0.99
Q5	3.58	940,199	70	658,139	10.31	3.94	6.36	62.8	0.58
Q6	16.79	990,749	85	842,136	8.84	3.65	5.19	58.7	0.64
Q7	4.93	530,149	70	371,104	8.78	3.38	5.41	46.5	0.81
TOTAL	7.88	10,283,155	80.2	8,249,480	11.61	4.45	7.16	69.6	0.68

FIG. 7.4–02

Anvil Range MINING CORPORATION

ANVIL RANGE MINING CORPORATION	DRAWN BY: DP/AKB	DATE: 21/11/96	SCALE: 1:2500	GRIZZLY PROJECT PRE–FEASIBILITY STUDY MINE PLAN LOWER–G HORIZON 9% Pb+Zn CUTOFF
	REVISED BY:	DATE:		
	FILE: GRLC9MPB.DWG			