016235

GRIZZLY PROJECT

COPY #1

PRE-FEASIBILITY STUDY
DECEMBER 1996

APPENDICES

Anvil Range
MINING CORPORATION

PROTON
INTERNATIONAL
ENGINEERING CORPORATION

February 6, 1997
File No. 1341-00

Anvil Range Mining Corp.
117 Industrial Road
Whitehorse, YT
Y1A 2T8

Attention: Dick Arndt
Superintendent, Explorations Project

Dear Sir:

RE: GRIZZLY PROJECT PRE-FEASIBILITY STUDY

Further to our discussion, please find attached two copies of the Appendices to the Pre-feasibility Study which was issued to Anvil Range in December 1996.

Please note that copy #1 is the only copy with a full set of color plates (where color plates were available). Copy #2 contains black-and-white plates for many of the same figures.

We are also returning to you a roll of associated drawings for your records.

Very truly yours,
PROTON INTERNATIONAL ENGINEERING CORPORATION

Warren Yau
Manager Director

WY/kr
Enclosure:
cc:

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APPENDICES

APPENDIX A

SOURCES OF DATA
AND
INFORMATION

LIST OF INFORMATION

Wright Engineers Ltd., Cyprus Anvil Mining Corporation, Dy Project - Preliminary Study for Underground Exploration and Mining, May 1978

Cyprus Anvil Mining Corporation (CAMC), Dy Reserve Calculations, Hall, 1981

Cyprus Anvil Mining Corporation (CAMC), Dy Reserve Calculations, Rollings, 1982

Kilborn Ltd., Dy Reserve Calculations, Coltas, 1989

Curragh Resources Incorporate, Dy Deposit, Estimate of Geological Reserves, Chornoby and Reed, June 1991

Curragh Resources Incorporate, Dy Deposit, Mineral Inventory, Jilson, December 1991

Curragh Inc., Summary of the Geology, Mineral Inventory and Reserves of the Dy Deposit, Jilson, May 1993

Canadian Mine Development (CMD) Reports and Proposals for Ramp and Shaft Construction, Underground Exploration and Contract Mining:

- June 1988
- March 1989
- April 1989
- September 1990
- October 1990
- January / February 1991
- December 1992

Curragh Resources Incorporate, Faro Underground, Monthly Reports 1990/92

Kilborn Ltd., Review of Mineral Properties of Curragh Resources Incorporate and Affiliates October, 1989

Rescan Ltd., Advanced Exploration and Development of the Dy Underground Mine, March 1991

Lakefield Research Ltd., Recovery of Lead, Zinc and Silver from Dy Samples, September 1992

Curragh Incorporate, Pit Wall Mining Underground at Curragh's Faro Operation, Leo R. Hwozdyk, October 1992

Fox Geological Consultants, Dy Deposit, Mineable Reserve Estimate and Underground Mine Plan, Nick D. Rose, October 1992

Anvil Range Mining Corporation, Dy Project, Review of available Information and Possibilities for Underground Access, Fritz F. Prugger, April 1996

Anvil Range Mining Corporation, Grizzly Deposit - Underground Exploration, Application for Water Licence, Eric Denholm and Fritz F. Prugger, October 1996

Canadian Mining Journal, Don Mills, Ontario, 1996 Mining Source Book

Proposals and cost estimates for shaft and ramp access and underground exploration from the following mining contractors:

Aurora Mining and Tunnelling Ltd., Aurora, Ontario
Dynatec Ltd., Toronto, Ontario
Main Street Mining Ltd., Whitehorse, Yukon
Procon Mining and Tunnelling Ltd., Burnaby, British Columbia
Redpath Ltd., North Bay, Ontario

Redpath Ltd., schedules and cost estimates for shaft sinking, ramp drivage, underground construction and development, and contract mining

Procon Mining and Tunnelling Ltd., technical details, schedules and cost estimates of twin ramp drivage using conveyor haulage as well as mining methods and mining costs for contract mining

APPENDIX B

DESIGN AND OPERATING PARAMETERS FOR THE BASE CASE

GRIZZLY PROJECT.

DESIGN AND OPERATING PARAMETERS.

1. OVERVIEW.

Production	1.5 Mio tpy at 9% Pb+Zn cutoff plus mining mix
Workforce	Anvil Range Existing bargaining unit Commitment to training
Access	Decline in the Blind Creek valley for exploration Shaft for production/personnel
Development	Mainly in ore
Mining	Two separate horizons in the AB-Zone
Method	Elongated Room and Pillar mining (pillar robbing in retreat) Benching (horizontal or vertical) Bulk Mining Methods where possible Backfill for mining height over 6.5 metres Cut and Fill using elongated rooms Long hole mining
Flow of Ore	Scoop trams Trucks (diesel or electric) Primary crusher Conveyor Surge bin Loadout conveyor Loading pockets Skip hoisting Raw ore storage Truck transport to concentrator
Waste Handling	Decline conveyor to surface (initially) Underground backfill
Planning Concept	Simple Computerised mine planning
Operating Philosophy	Simple High level of automation Use of latest technology

	Minimum number of people Emphasise on safety and training
Ventilation System	Fresh air intake and main air heating at the production shaft Exhaust air through twin declines plus ventilation raises 500,000 cfm fresh air requirement
Pumping	Main Pump station at mining level Booster pump in mid shaft Pump discharge line in the shaft Surface discharge into Vangorda drainage system

2. SURFACE ARRANGEMENS.

Dry and change room	:	At the production shaft
Mine office	:	Existing Grum Facilities for Grizzly and Grum underground
Warehouse	:	Existing facilities at the Faro concentrator Underground warehouse (open) near underground shop
Lamps	:	Shaft collar
Crew check in	:	Shaft collar

3. PRODUCTION SHAFT.

3.1 Location:

West of A-Zone, outside the influence of mining

Coordinates	900880 N	(approximately)
	596720 E	(approximately)

3.2 Depth (preliminary):

Elevations	Collar	1,140 m
	Station	400 m
	Bottom	
	Depth	

3.3 Function

- Ore hoisting
- Transport of people at scheduled times only
- Fresh air supply
- Mine air heating
- Pump line for mine dewatering
- Main electric power cable
- Supply lines for (if allowed by the mine act)
  - Anfo
  - diesel fuel
  - hydraulic oil

3.4 Operation

Hoist will have automatic control for hoisting as well as for transportation of people

- no special hoist operator

Trained hoist operator for shaft inspection and special trips only

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

Central control for:

- hoisting
- pumping
- ventilation
- mine air heating
- underground crushers and conveyors
- surge bin
- reclaim conveyor
- loading pockets
- headframe bins
- electric stench gas warning system
- all communications

Automatic greasing of

- head ropes
- guide ropes and
- rubbing ropes

Manual greasing of tail ropes

3.5 Layout

- concrete or steel headframe
- tower mounted Koepe hoist
- Thyrister control
- multi rope installation (probably six headropes, no deflection sheaves, less height of the headframe)
- cage over skip combination
- rope guides
- rubbing ropes
- cheese weights
- clean-up ramp to shaft bottom
- pumping from shaft bottom to pump station
- fresh air fans
- mine air heating

4. DECLINE.

4.1 Function

Air exhaust

Moving large equipment underground

Transporting bulk materials

Conveyor haulage of development muck

4.2 Operation

Exhaust air

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

No pumping .

4.3 Portal

All temporary facilities will be remove by the contractor

Core sheds will be maintained at the portal site

Screen gate - normally locked

All acid generating material and ore will be removed

Seeding and tree planting

5. UNDERGROUND INFRASTRUCTURE.

Refuge stations
Electric stench gas warning system
Lunch rooms
Shifters office
Sanitary facilities
Underground communications
Power supply
Compressed air supply
- stationary electrical compressors
- mobile diesel and electric compressors

Ventilation system
- auxiliary fans
- ventilation ducting
- bulkheads
- overpasses

Underground shop
Underground warehouse (open)

6. Development Mining

Waste rock from development mining will be used as backfill
Drop lines from surface for supply of backfill material

7. PRODUCTION MINING.

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

Methods
- Elongated room and pillar
- Pillar robbing in retreat
- Benching (vertical or horizontal with or without backfill)
- Bulk mining where applicable
- Elongate Cut and fill mining
- Contour mining
- Long hole mining

Design Parameters depending of geotechnical investigation
- Opening size
- Pillar size
- Extraction rate
- Ground support

8. BACKFILL.

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

9. Environmental Issues
- Permits
- Constraints

10. Public Relations

Ross River Kaska Dena

Training

Contracts

Build and maintain good contact

Governmen Agencies

9. Other Matters.

Safety training

ongoing

safety meetings

mines act

"Incident Recall System"

"Management of Total Loss Control"

First Aid Training

Mine Rescue Training

Development of standard procedures for

operating

maintenance

Development of cost accounting

APPENDIX C

DESIGN AND OPERATING PARAMETERS FOR THE CASE A

Appendix C
SUMMARY OF DESIGN AND OPERATING PARAMETERS FOR CASE A

OVERVIEW

Production	1.2 Mio tpy at	9% Pb+Zn cutoff plus mining mix
Workforce	Anvil Range Existing bargaining unit Commitment to training Mining contractor for the first two years might be considered Camp will be available a.) Contractors equipment	
Access	Decline (Blind Creek valley) plus Shaft (production, south west of orebody)	
Development	In the footwall (waste)	
Mining	Two separate horizons in the AB-Zone	
Method	Room and Pillar with pillar robbing in retreat Benching (horizontal or vertical) Bulk Mining Methods where possible Contour Mining is being considered Backfill for mining height over 6.5 metres	
Flow of Ore	Scoop trams Ore passes Crushers Conveyors Surge bin Loadout conveyor Loading pockets Skip hoisting Raw ore storage Truck transport to concentrator	
Waste Handling	Decline conveyor to surface (initially) Underground backfill	
Planning Concept	Simple Computerised mine planning	
Operating Philosophy	Simple High level of automation	

Use of latest technology
Minimum number of people, well trained

Ventilation System	Fresh air intake and main air heating at the production shaft Exhaust air through decline
Pumping	Main Pump station at mining level Second pump station in mid shaft Pump discharge line in the shaft

SURFACE ARRANGEMENTS

Dry and change room	:	At the production shaft
Mine office	:	Existing Grum Facilities
Warehouse	:	Existing facilities at the Faro concentrator Open warehouse near underground shop
Underground lamps	:	Shaft collar
Shifters Office	:	Shaft collar
Check in	:	Shaft collar

0.0.1 PRODUCTION SHAFT

Location:
West of A-Zone, outside the influence of mining
Coordinates 900880 N (approximately)
596720 E (approximately)

Depth (preliminary):

Elevations	Collar	1,175 m
	Station	400 m
	Bottom	
	Depth	

Function

- ore hoisting
- transport of people at scheduled times only
- fresh air supply
- mine air heating
- pump line for mine dewatering
- supply of electric energy through main power cable
- supply pipe lines for
  - Anfo
  - diesel fuel
  - hydraulic oil

Operation

Hoist will have automatic control for hoisting as well as for transportation of people

no special hoist operator
Trained hoist operator for shaft inspection and special trips only
Hoist controls are located in the central control room at the shaft collar
Central control for:
hoisting
pumping
ventilation
mine air heating
underground crushers and conveyors
surge bin
reclaim conveyor
loading pockets
headframe bins
electric stench gas warning system
all communications
automatic greasing of
head ropes
guide ropes and
rubbing ropes
manual greasing of tail ropes

Layout

concrete or steel headframe
tower mounted Koepe hoist
Thyrister control
multi rope installation (probably six headropes, no deflection sheave)
cage over skip combination
rope guides
rubbing ropes
cheese weights
clean-up ramp to shaft bottom
pumping from shaft bottom to pump station
fresh air fans
mine air heating

0.0.2 DECLINE

Function
Air exhaust
Moving large equipment underground
Transporting bulk materials
Conveyor haulage for development muck

Operation

Exhaust air
The haulage conveyor will be left in place until all main development work is complete and sufficient room is available underground for placing waste as backfill
No pumping

Portal

All temporary facilities will be remove by the contractor
Core sheds
Screen gate - normally locked
All acid generating material and ore will be removed
Seeding and tree planting

0.0.3 Underground Infrastructure

Refuge stations
Electric stench gas warning system
Lunch rooms
Shifters office
Sanitary facilities
Underground communications
Power supply
Compressed air supply
  stationary electrical compressors
  mobile diesel and electric compressors
Ventilation system
  auxiliary fans
  ventilation ducting
  bulkheads
 overpasses
Underground shop
Underground warehouse (open)

0.0.4 Development Mining

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

0.0.5 Production Mining

Orebody Characteristics
  Depth
  Strike
  Dip
  Number of mineable horizons
  Uniformity
  Thickness
  Grade
  Strength of ore

Strength and shape of hanging wall
Strength and shape of footwall
In-situ stresses

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

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

0.0.6 Backfill

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

0.0.7 Roof Support

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

0.0.8 Environmental Issues

Permits
Constraints
No discharge of water or waste into Blind Creek

0.0.9 Ross River Kaska Dena

Good contact
Training
Contracts

0.0.10 Other Matters

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

APPENDIX D

UNDERGROUND ACCESS OPTIONS

9. UNDERGROUND ACCESS

9.1 Access for Underground Exploration

9.1.1 Introduction

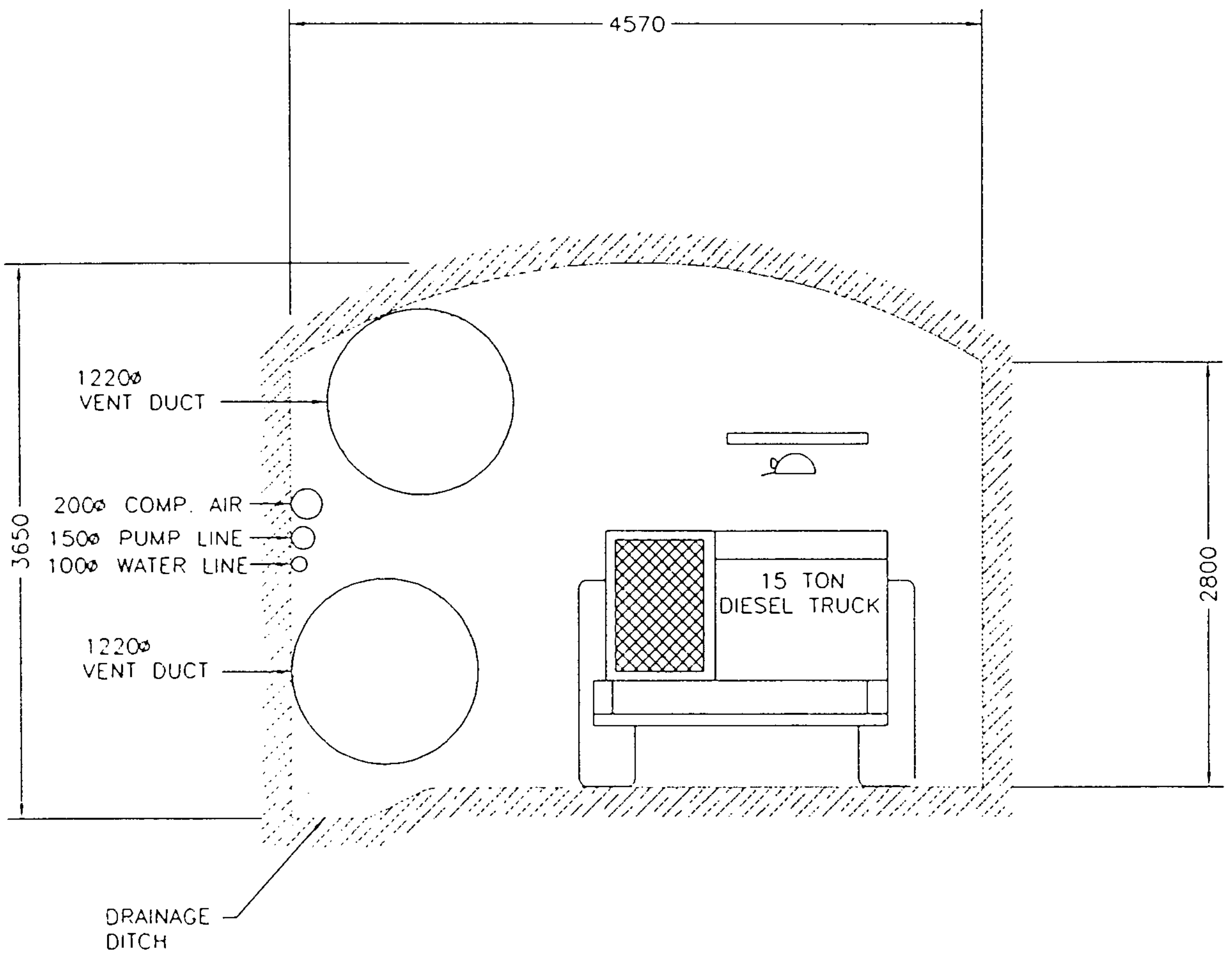
Alternative means of access for underground exploration that were considered include several shaft locations and various configurations of decline (ramp).

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

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

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

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



MAIN RAMP DRIFT SECTION

TYPICAL DURING EXCAVATION PHASE

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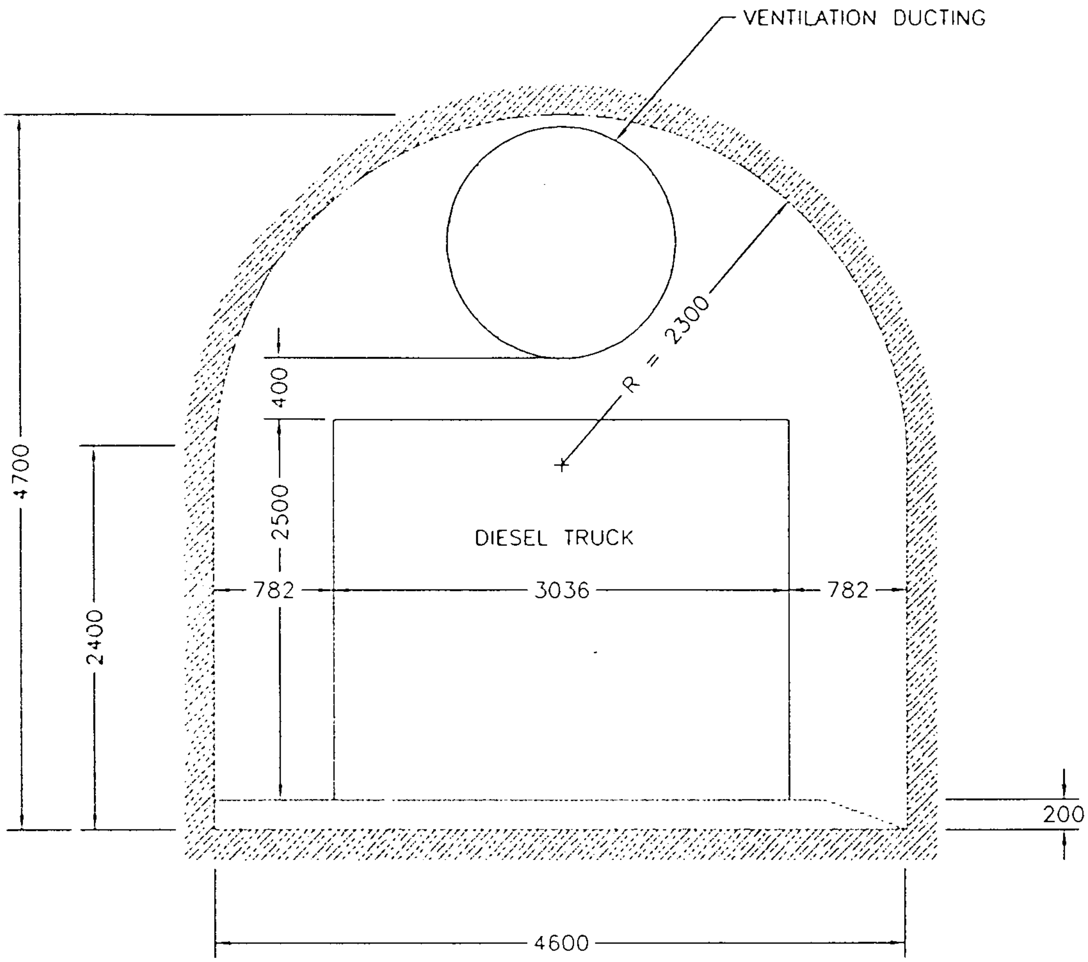
PRE-FEASIBILITY STUDY

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

Anvil Range
MINING CORPORATION

SCALE 1 : 50	APPROVED	
DRAWING: RMP–SMTR.DWG	CHECKED	APPROVED 96/09



AREA 19.34 sq. m.

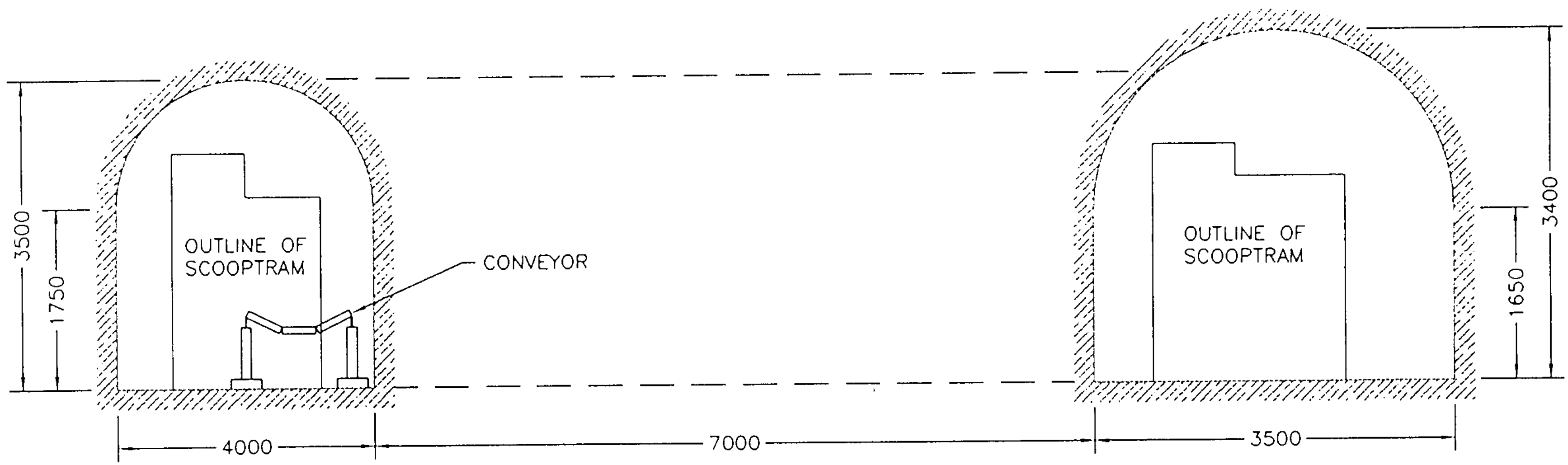
GRIZZLY PROJECT

PRE-FEASIBILITY STUDY

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

Anvil Range
MINING CORPORATION

SCALE 1 : 50	APPROVED	
DRAWING: RMP–LGTR.DWG	CHECKED	DATE 96/09



SCALE NOT TO SCALE	GRIZZLY PROJECT PRE-FEASIBILITY STUDY	Anvil Range MINING CORPORATION
DRAWING: TWINRAMP.DWG		
APPROVED	Fig. 9.1.1-04 Twin Ramp Conveyor Haulage	
DATE 96/11		

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

9.1.2 Protective Measures for Blind Creek

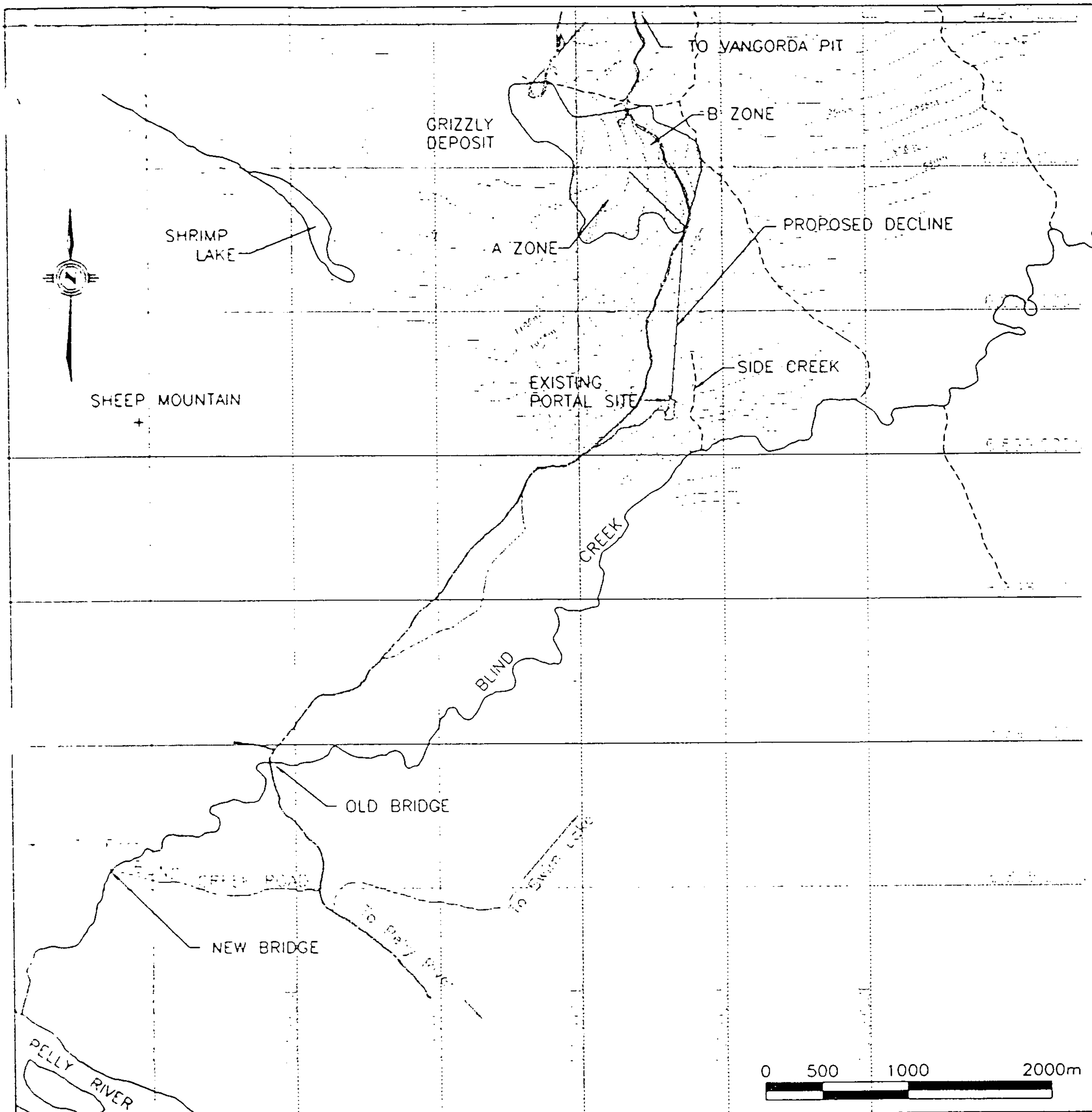
Blind Creek is a salmon spawning river and is of importance to the Kaska Dena of Ross River as a traditional fishing ground. The ramp portal is located some 110 metres above the Creek and is about 800 metres away from it. Discharge water from the ramp will possibly contain suspended solid particles,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

The following measures will be taken to protect Blind Creek:

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

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

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



GRIZZLY PROJECT

PRE-FEASIBILITY STUDY

Fig. ~~9.1.2~~ 13-01 Surface Plan

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MINING CORPORATION

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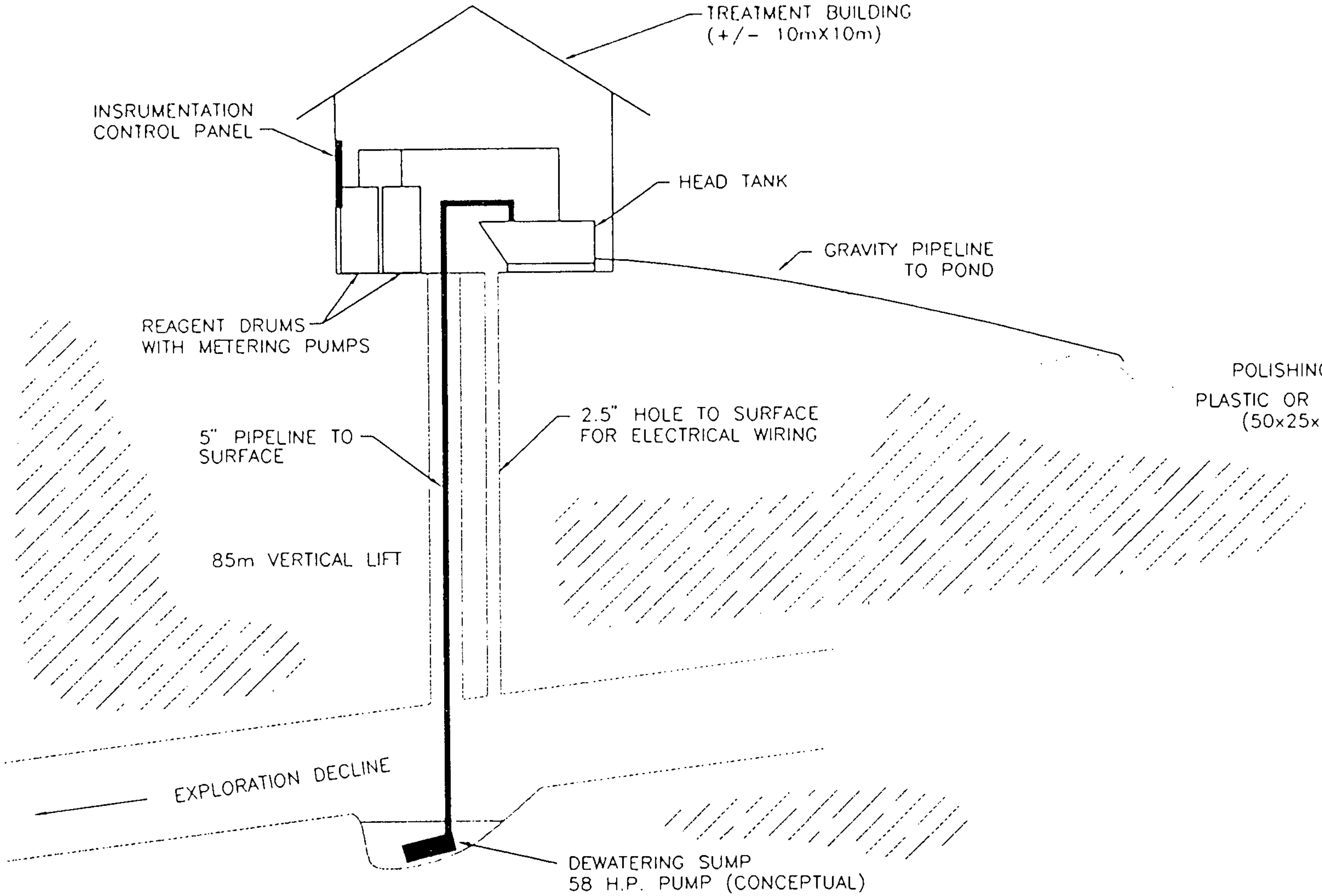

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DRAWING: DEWATER.DWG

APPROVED

DATE: 96/09

GRIZZLY PROJECT

PRE-FEASIBILITY STUDY

Fig. ~~9.1.2~~ 13 -02 Conceptual Layout of Dewatering and Treatment Facilities

Anvil Range
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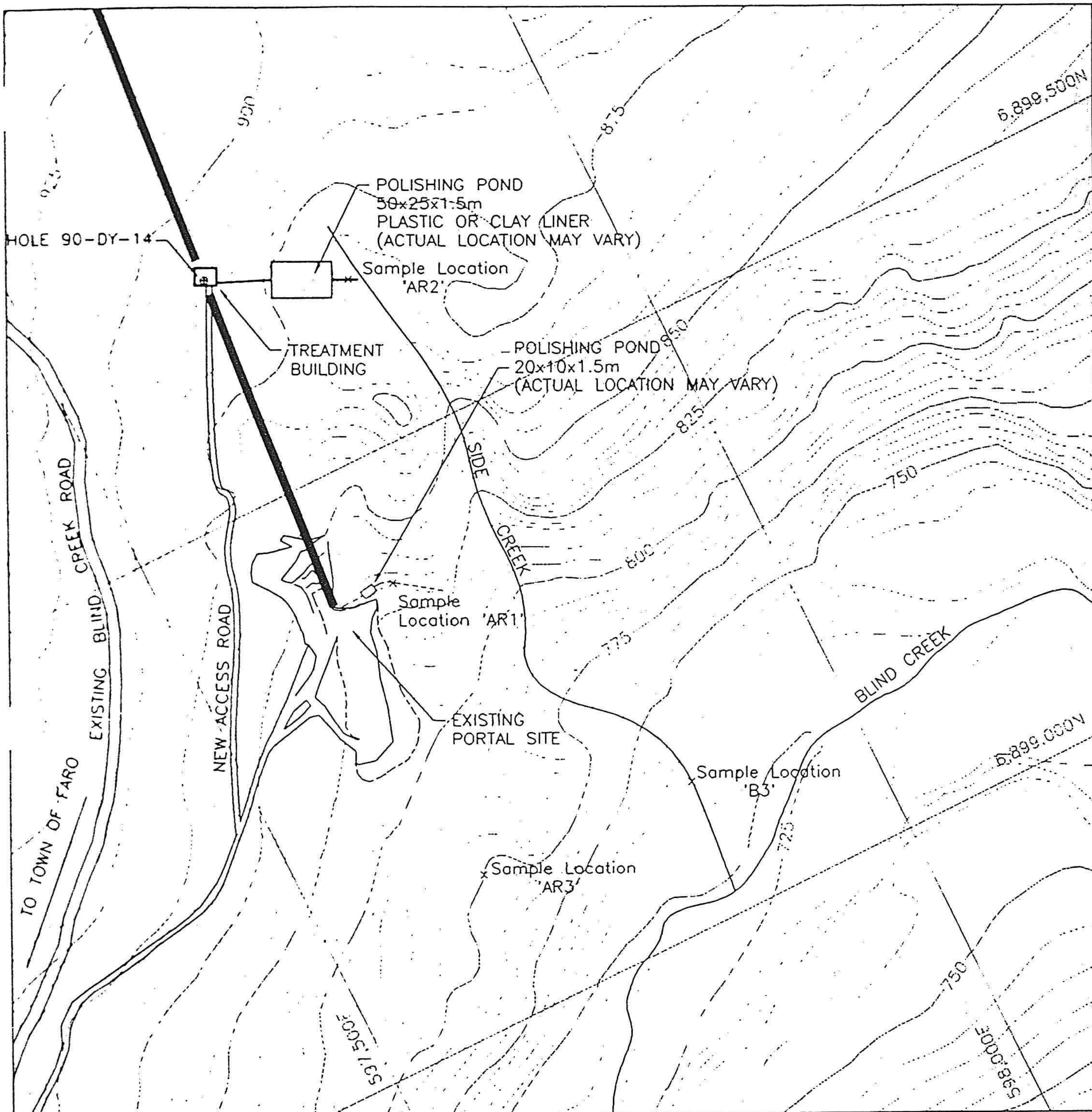

GRIZZLY PROJECT

PRE-FEASIBILITY STUDY

Fig. ~~9.1.2~~ 13-03 Layout of Treatment Facilities and Water Discharge

Anvil Range
MINING CORPORATION

SCALE 1 : 5,000	APPROVED	
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9.1.3 Access Options

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

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

and in ore and the footwall in A-Zone North. A short ramp into the B-Zone provides access for bulk sample. This option provides the best coverage, but has some ventilation constraints and is expensive.

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

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

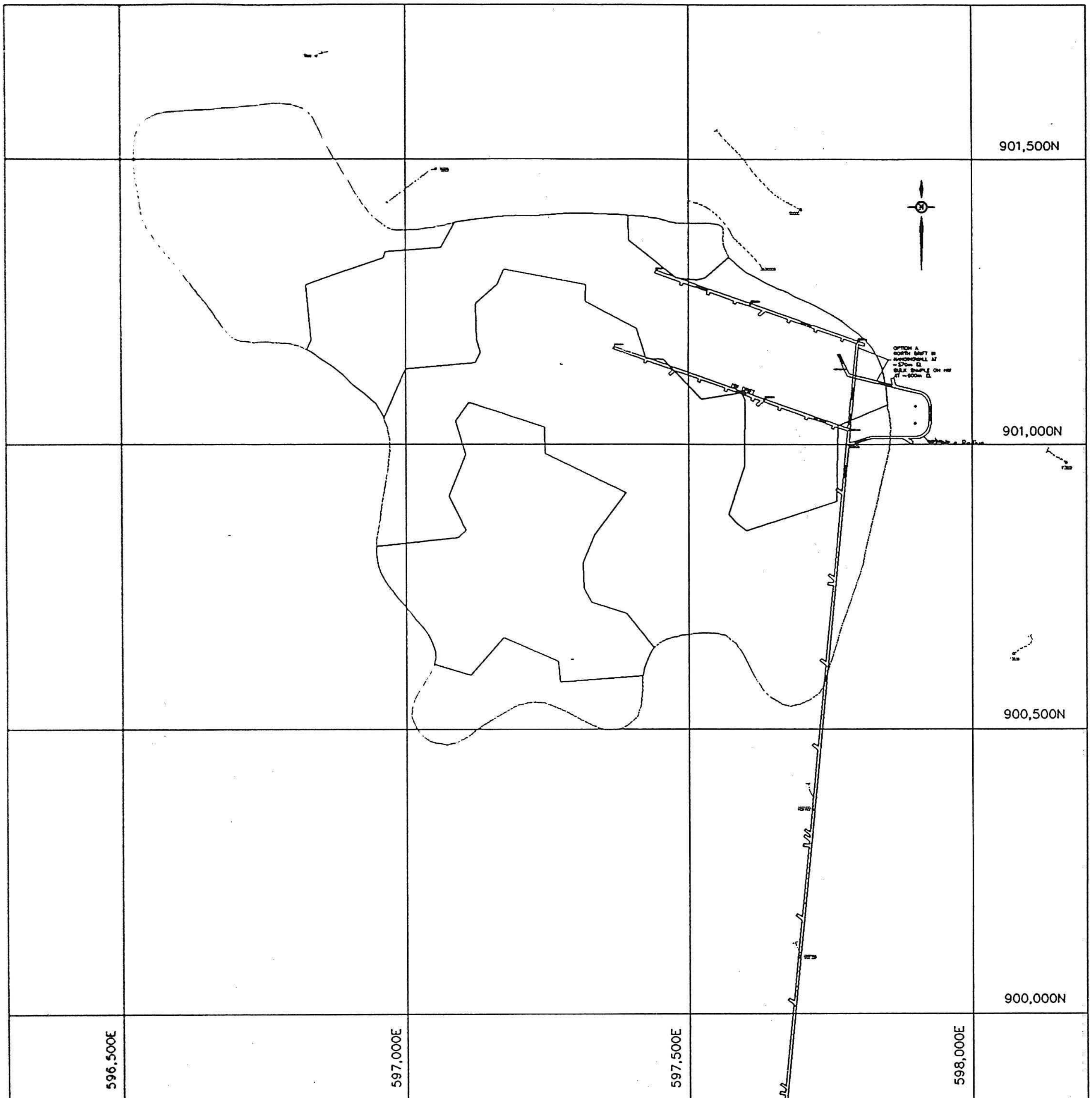
9.1.4 Preparation for Underground Exploration Access

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

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

All seven visited the site for familiarization.

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



GRIZZLY PROJECT

UNDERGROUND EXPLORATION PROGRAM

Anvil Range
MINING CORPORATION

Fig. 9.1.3-01 OPTION 1

SCALE	APPROVED	
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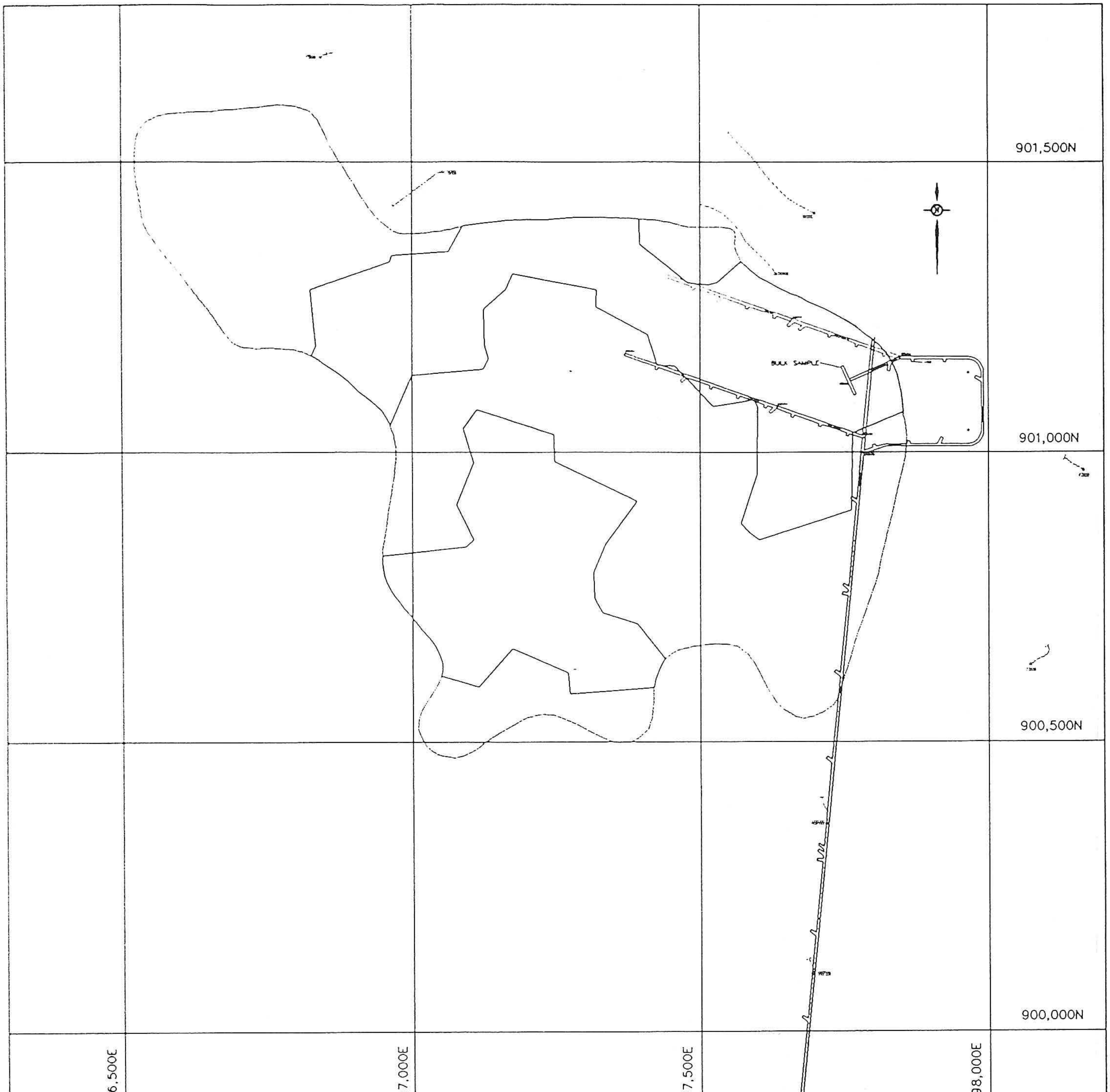

GRIZZLY PROJECT

UNDERGROUND EXPLORATION PROGRAM

Fig. 9.1.3–02 OPTION 2

Anvil Range
MINING CORPORATION

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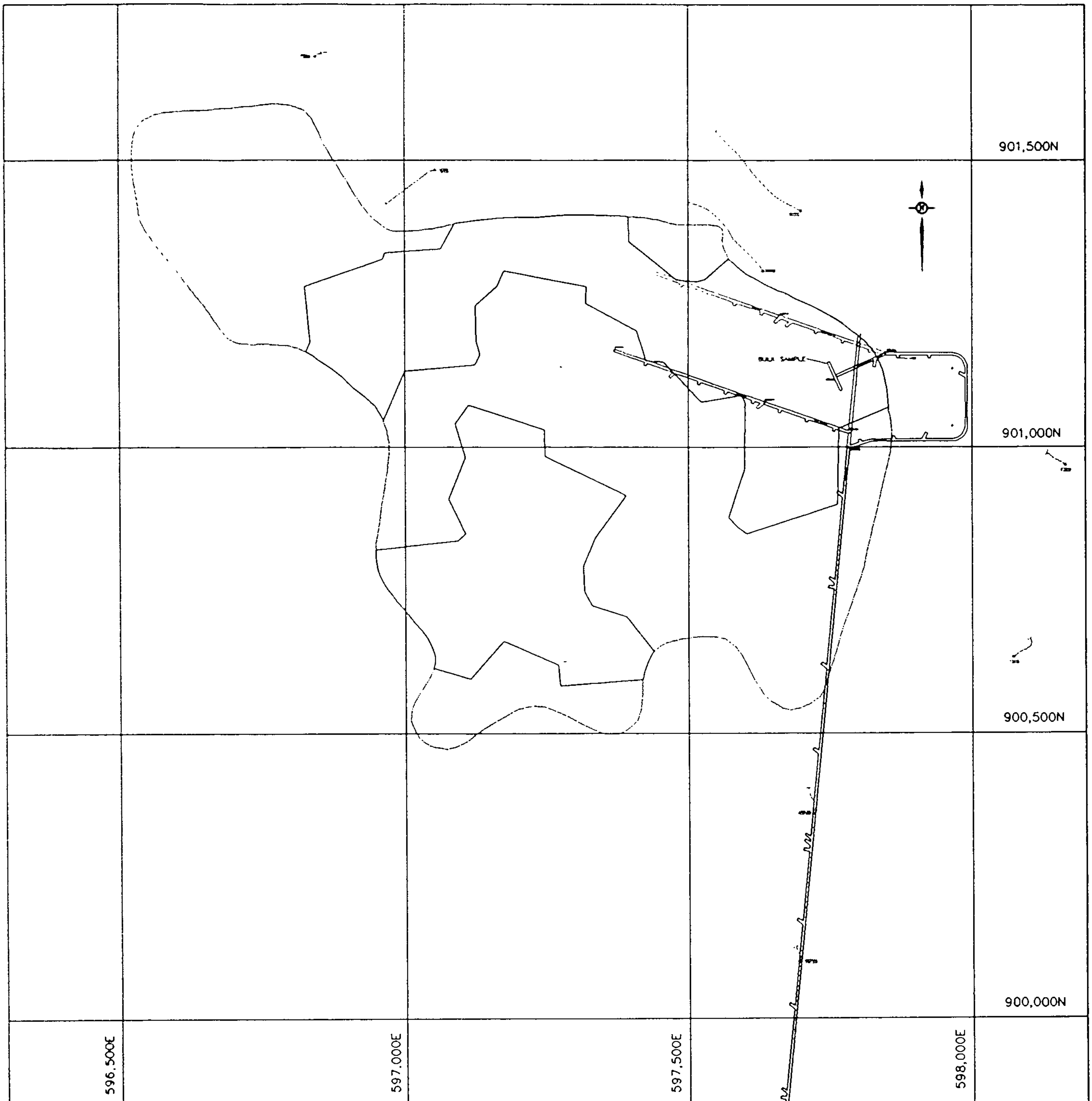

GRIZZLY PROJECT

UNDERGROUND EXPLORATION PROGRAM

Fig. 9.1.3-02 OPTION 2

Anvil Range
MINING CORPORATION

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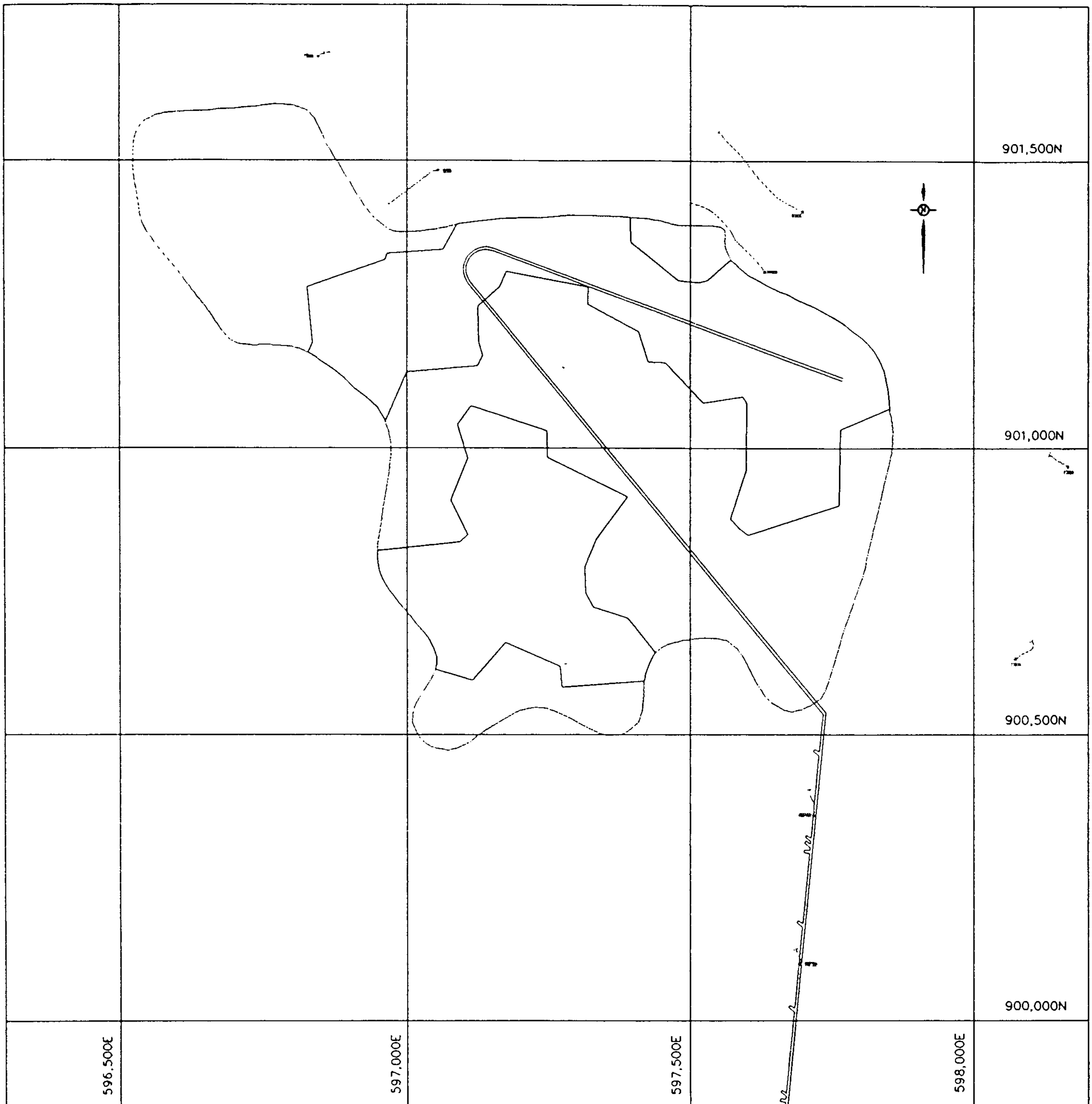

GRIZZLY PROJECT

UNDERGROUND EXPLORATION PROGRAM

Fig. 9.1.3-04 OPTION 4

Anvil Range
MINING CORPORATION

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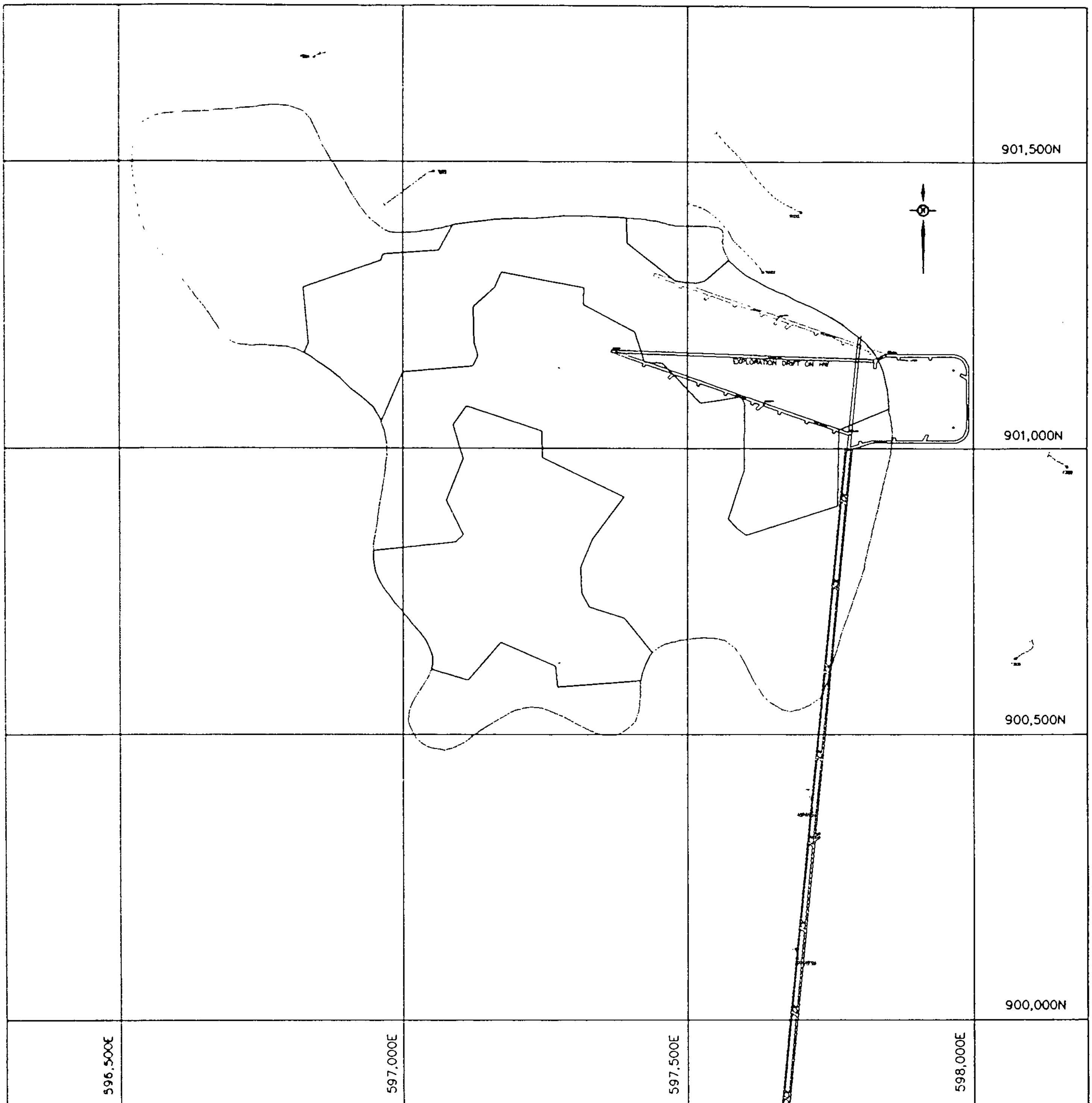

GRIZZLY PROJECT

UNDERGROUND EXPLORATION PROGRAM

Fig. 9.1.3–05 OPTION 5

Anvil Range
MINING CORPORATION

SCALE	APPROVED	
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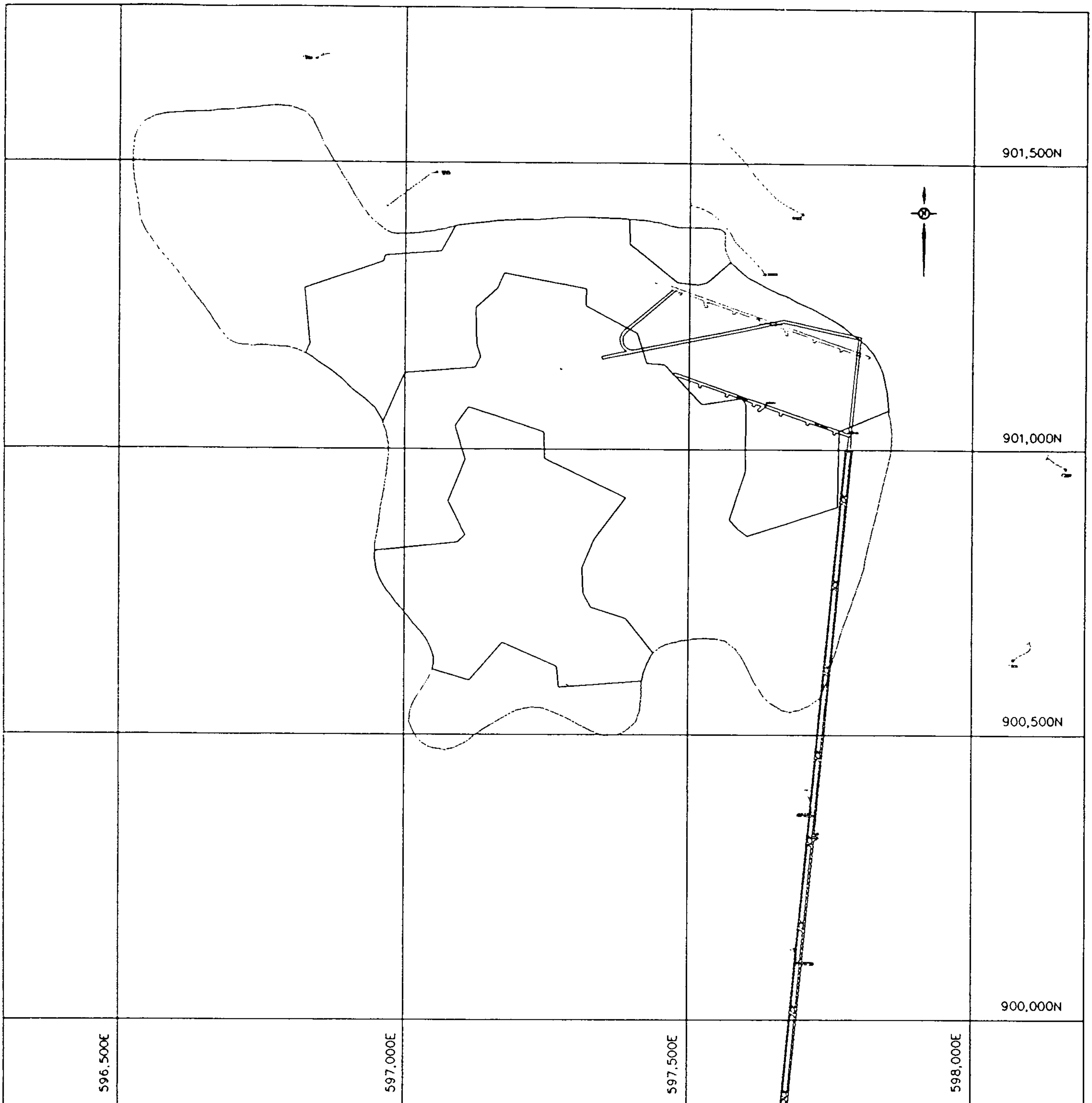


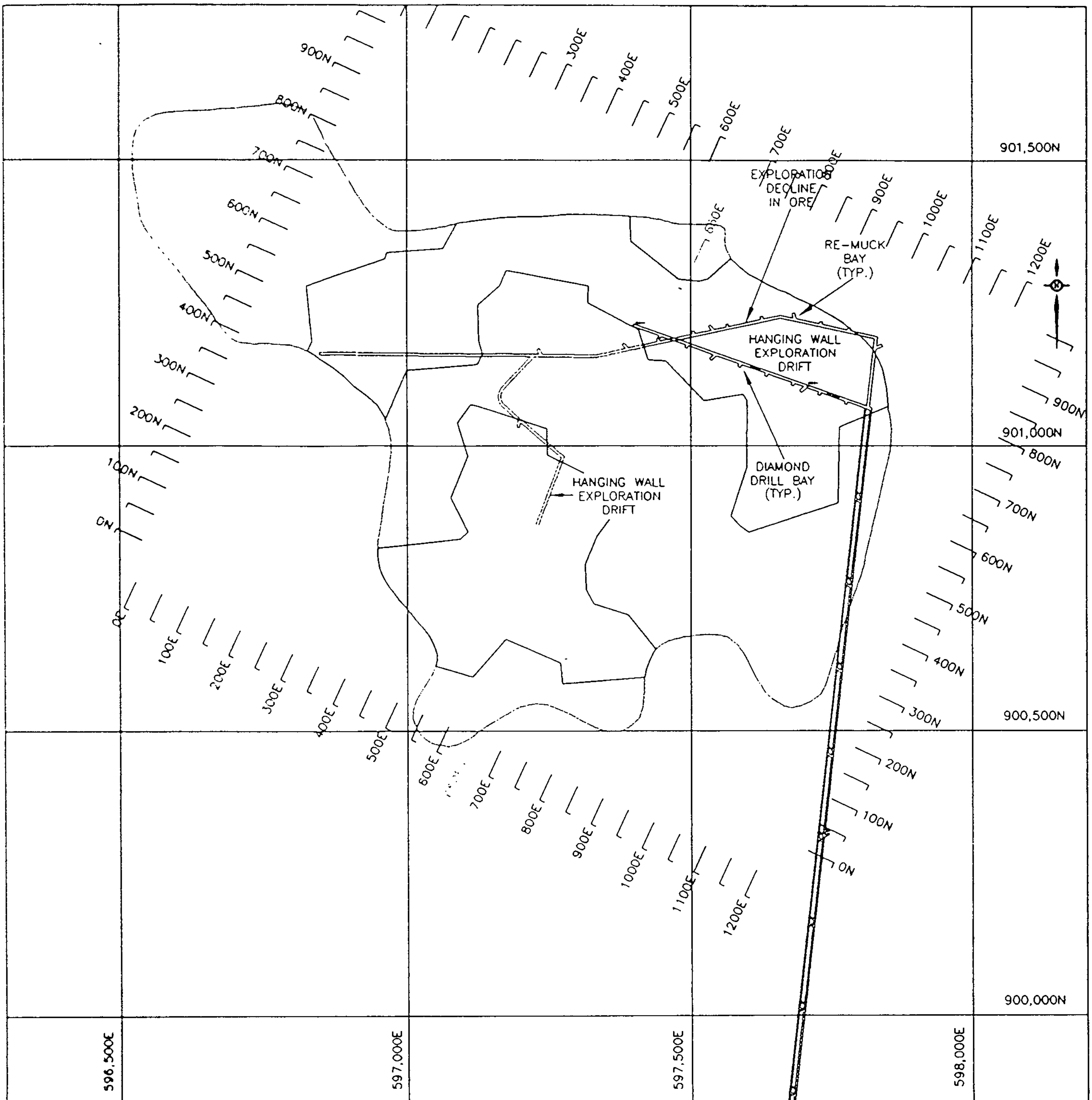
GRIZZLY PROJECT

UNDERGROUND EXPLORATION PROGRAM

Fig. 9.1.3–06 OPTION 6

Anvil Range
MINING CORPORATION

SCALE	APPROVED	
DRAWING: EXP06MOD.DWG	CHECKED	DATE 96/11



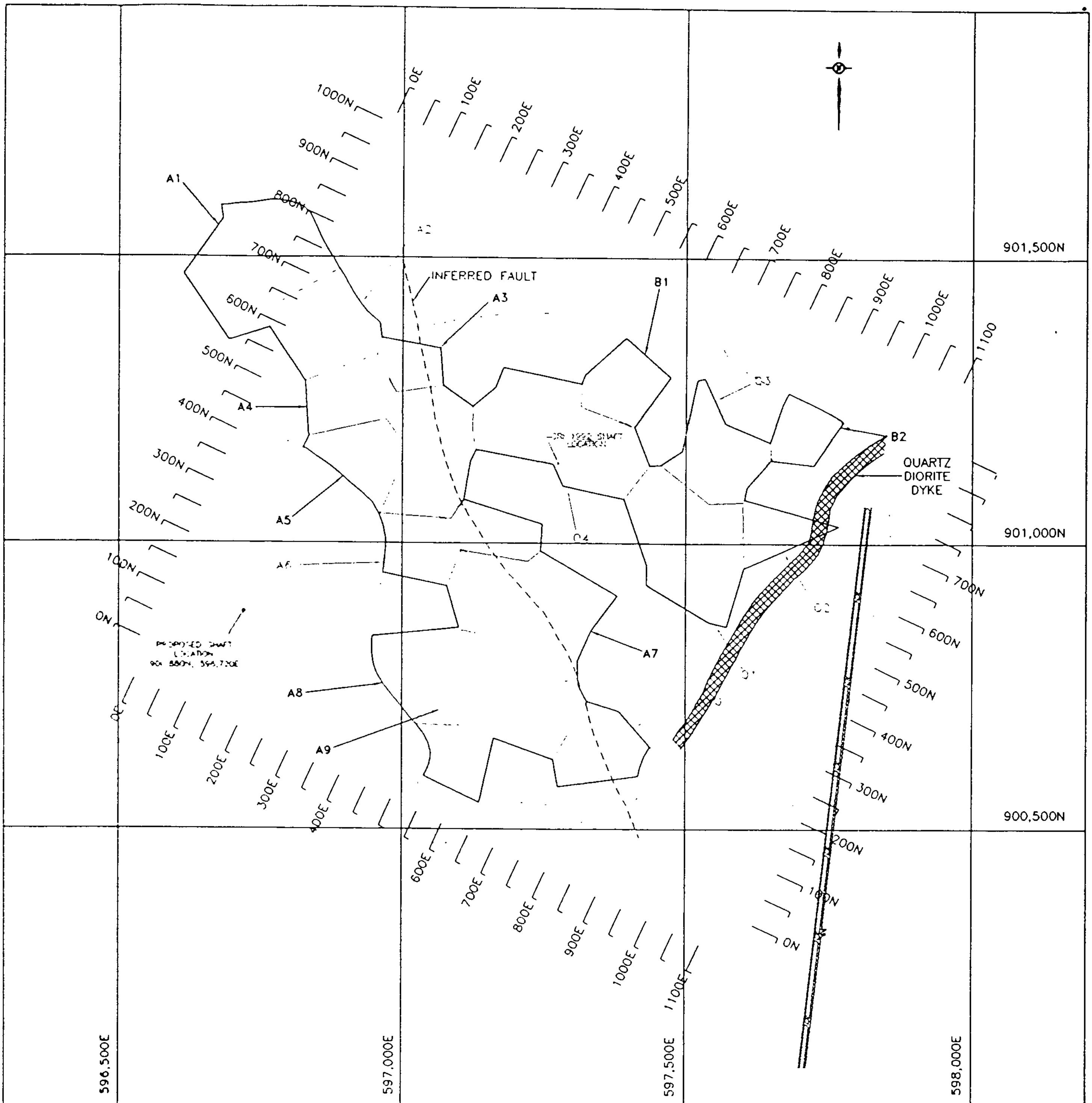
GRIZZLY PROJECT

UNDERGROUND EXPLORATION PROGRAM

Anvil Range
MINING CORPORATION

Fig. 9.1.3–07 OPTION 7

SCALE	APPROVED
DRAWING: EXP07MOD.DWG	CHECKED
DATE 96/11	



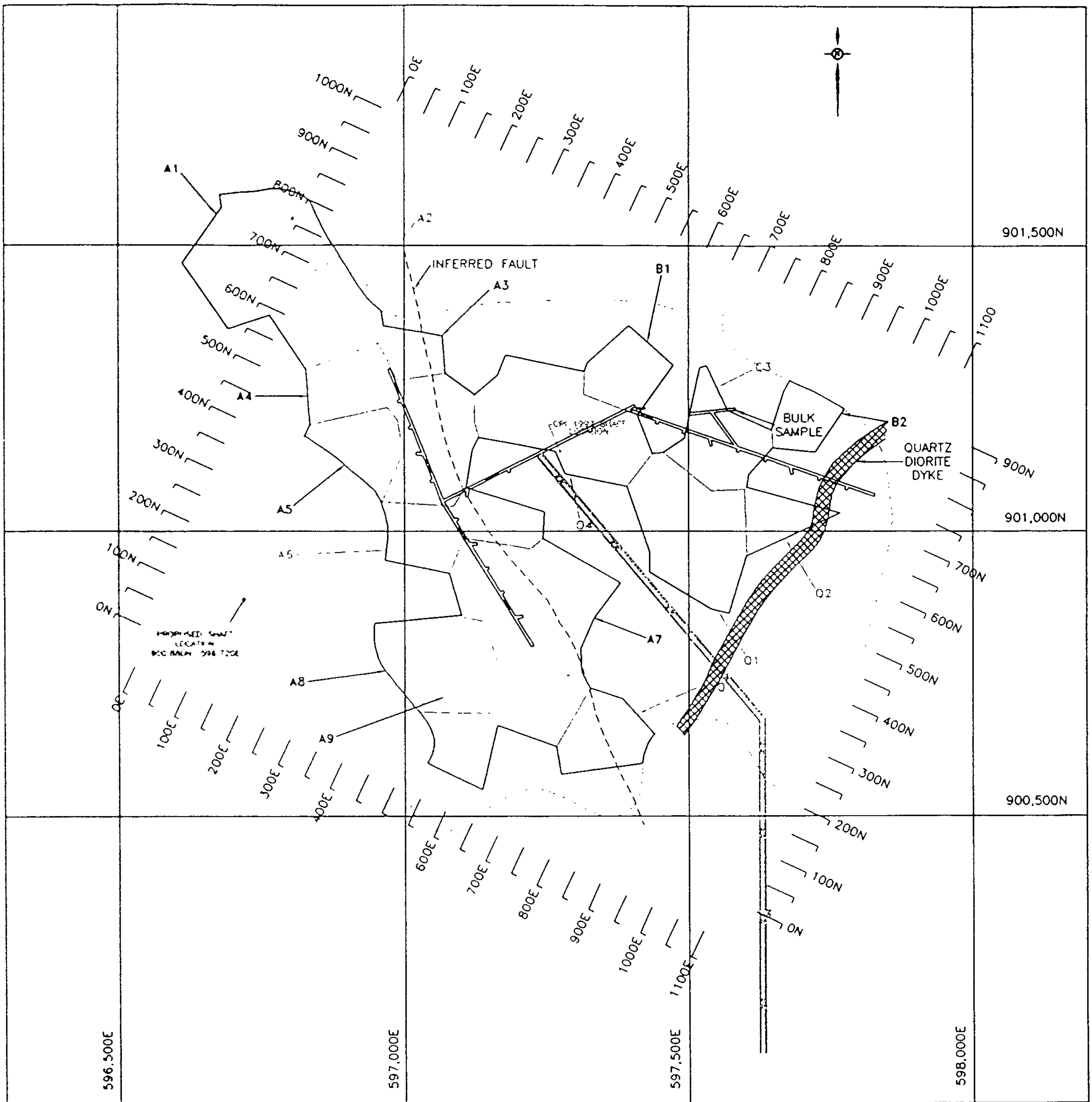
GRIZZLY PROJECT

UNDERGROUND EXPLORATION PROGRAM

Fig. 9.1.3-08 OPTION 8

Anvil Range
MINING CORPORATION

SCALE	APPROVED	
DRAWING EXP08MOD.DWG	CHECKED	DATE 96/11



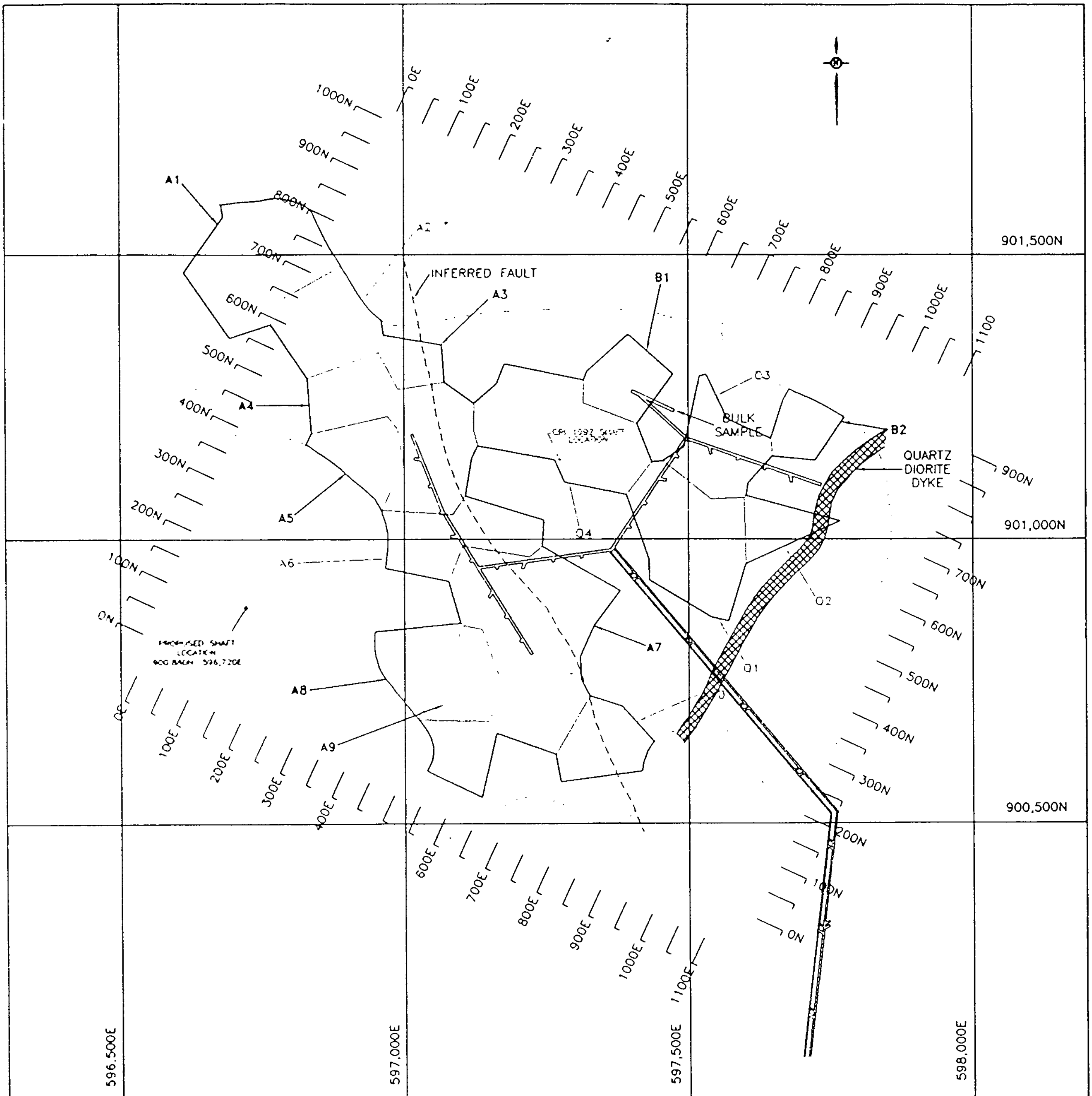
GRIZZLY PROJECT

UNDERGROUND EXPLORATION PROGRAM

Anvil Range MINING CORPORATION

Fig. 9.1.3–09 OPTION 9

SCALE	APPROVED	
DRAWING: EXP09MOD.DWG	CHECKED	DATE 96/11



GRIZZLY PROJECT

UNDERGROUND EXPLORATION PROGRAM

Fig. 9.1.3-10 OPTION 10

Anvil Range
MINING CORPORATION

SCALE	APPROVED	
DRAWING: EXP10MOD.DWG	CHECKED	DATE 96/11

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

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

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

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

9.1.5 Summary

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

The precautions planned will ensure protection of Blind Creek.

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

APPENDIX E

PRODUCTION ACCESS OPTIONS

9.2 Access for Production

9.2.1 Introduction

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

The production opening will serve the following functions:

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

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

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

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

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

The following options for production access ought to be considered:

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

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

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

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

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

9.2.2 Production Shaft in the Centre of the Orebody

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

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

9.2.3 Production Shaft West of the Orebody

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

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

9.2.4 Production Shaft North of the Orebody

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

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

9.2.5 Production Shaft East of the Orebody

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

9.2.6 Production Shaft South of the Orebody

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

9.2.7 Vangorda Production Ramp

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

9.2.8 Summary

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

SHAFT POSITIONING IN A CENTRAL LOCATION

An earlier study carried out by Fox Geological Consultants Ltd. in October, 1992, suggested positioning of the production shaft at a central location, between the mineralized zones, with a 10 degree cone of influence. More precisely, at coordinates 901,140 N and 597,275 E with a collar elevation at 1140m.

In view that in the apparently barren massive sulphide zone the cone of influence would have intersected only two polygons of mineable reserves, namely 78x01 and 79x09, this scheme merited reconsideration.

Having recently calculated the net smelter return and gross net values of the polygons, it was found that 78x01 has no positive net value in the upper horizon, and only marginal value in the lower horizon, constituting by itself a block, Q7, being in a quarzite zone. The other polygon, 79x09, has no positive net value in either horizon.

The 10 degree cone was repositioned in such a manner, as to avoid any payable ore intersection in the upper horizon, and a minimal one in the lower horizon, that is about 60% of the polygon 78x01.
The new shaft position is about 30m NW from the former, at 901,170 N and 597,250 E, at elevation 1150m.

A 15 degree cone was localized with the same considerations in mind. It was found, that the optimal shaft location would be at an entirely different place, about 110m due south from the other.

More specifically, at 901,061 N and at 597,225 E, also at elevation 1150m. This cone of influence would intersect several polygons, namely: 79x11 and 79x14 in Block A4, 79x18 and 80x04 in Block A6, and 79x13, 79x16 and 80x09 in Block A7. Of these, 79x16 has no positive value, 79x14 and 79x18 have almost no value, and the rest only marginal ones. Only polygon 73x13 has good values, where the grades are projected to an average of 70m in all directions.

On the lower horizon, the following polygons are intersected: 78x01 in Block Q7, 78x02 in Q6, 80x09 in A16 and 79x11 in A18. Of these, in polygons 78x01 and 78x02, the net gross is less than Cdn$10/tonne, about 1/3 of polygon 79x11 would be affected at Cdn$ 14/tonne, and polygon 80x09 could probably be extracted, being 100m below the shaft station and 40m below the shaft bottom, between 10 to 15 degrees from the shaft collar.

In conclusion: a shaft with a 10 degree cone of influence is the better choice, but a 15 degree cone is still better in the central location, than outside, in view of shorter haulage distances and shortened preproduction development requirements.

APPENDIX F

CAPITAL
COST
DETAILS

14. CAPITAL COSTS

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

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

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

14.1.1 Basis For the Report

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

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

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

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

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

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

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

14.2 Exploration Phase

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

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

10% contingency is added. All the expenditures occur in years -3 to -1. The total is estimated at 24 480 million.

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

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

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

14.3 Production Phase

14.3.1 Shaft

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

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

14.3.2 Mine Development

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

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

14.3.3 Other Areas

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

- Permits

14.4 Replacement Equipment

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

14.5 Summary

The following table comprises a juxtoposition of the two Cases.

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

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

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

Table 14.5-01 Summary of Capital Expenditures

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

Table 14.01
Grizzly Development Drivage
Case A

R = rock

Grizzly Decision ⇓

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

Average Drivage
7m per day

R = Rock

Approx.
2 crews
2 years

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

Basis - no ore hauled up ramp
All contract mining

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

Table 14.02
Construction Capital Requirements
Case B

Decision ⇓ (between years -1 and +1)

Can $000

Year	-3	-2	-1	+1	+2	+3	+4	+5	+6	+7	+8	+9	+10
Production 000		30				300	900	1200	1200	1200	1200	1200	1200
Feasibility Study, etc.			1,800										
Mine Permit		500	500										
Ramp and UG explore	8,000	14,000											
Shaft + orebin, etc.				11,500	12,000								
Fan + Fan drift House				80	260								
Truck Load					500								
Shaft Drill Hole				200									
Storage Shaft Bottom					2,300								
East (p) Return Airshaft, rock removal in mine shaft cost					3,000	3,000							
Power from Grum (installed)				500									
P. Equip. Surface (installed)					400	102							
P. Equip. UG (installed					542	1,000							
Upgrade Haul Rd						1,200							
Backfill Plant (Surf.)						800							
Backfill Lined Holes					400	600							
Sump UG (in Table 14.B6.1)													
Main Pump + Range					300	336							
Ramp & Shaft Sump (210m) 330m. incl. in ramp to SW.						660							
Air Heating System					200	50							
UG Workshop, etc.						400	200						
UG Warehouse						100							
Conveyor, Ramp to UG Bin					200	600							
Sub-Total	8,000	14,500	2,300	12,280	19,902	5,188	200			Total After decision 38,696			
Use 3% for freight/camp (⅓ normal)-			-	368	597	155	6						
TOTAL	8,000	14,500	2,300	12,648	20,499	5,343	206						

Total Till Decision
24,800

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Table 14.03
Grizzly Development Drivage
Case B

R = rock

Grizzly Decision ⇓

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

Capitalised

Average Drivage
5m per day in first 1.5 years

Basis - no ore hauled up ramp
All employees ARMC

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

GRIZZLY PROJECT

UNDERGROUND EXPLORATION PROGRAM
(Parallel Twin Declines, Access Option Number 10)

Proposal from Procon Mining and Tunnelling Ltd., dated October 10, 1996

1. Mob/Demob & Surface Setup		813,546
2 Access Decline	4,254,091	
Conveyor Decline	2,908,974	
Total Twin Declines		7,163,065
3 Lateral Drifting		3,746,656
4.Support Items		5,659,200
Total Estimated Target Price		17,382,467
Fee for Profit		1,129,860
Total Procon		18,512,327
5. Diamond Drilling, 35,000 metres		2,975,000
Total Program		21,486,327

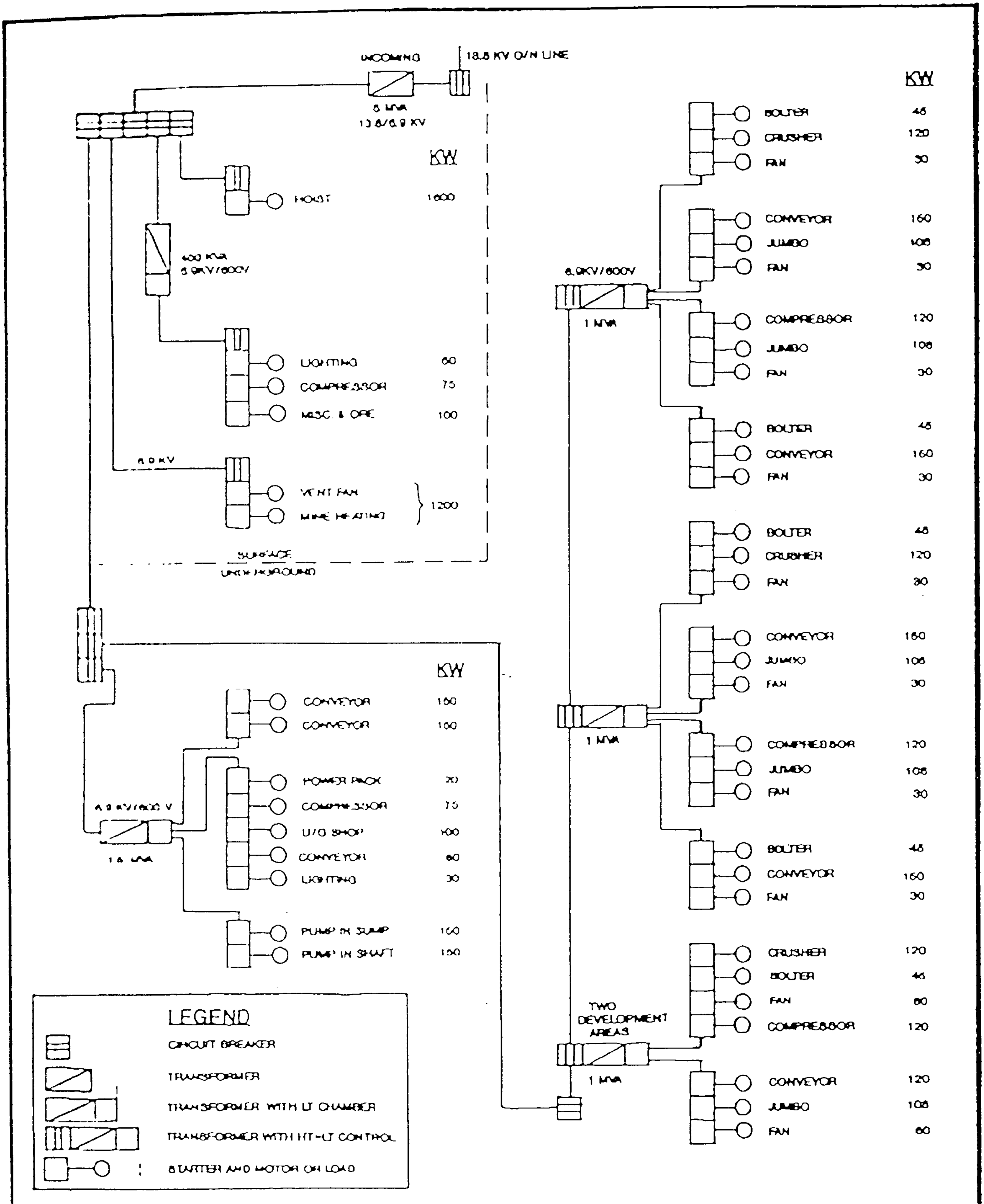
Appendix F

Surface Electrical Costs

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

Underground

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



ANVIL RANGE MINING CORPORATION
GRIZZLY PROJECT
UNDERGROUND MINING STUDY

PROPOSED ELECTRICAL SCEMATIC FOR CASE A MINE PLAN

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

APPENDIX G

ALTERNATE MINING METHODS

12. MINING METHODS

12.1 Introduction

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

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

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

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

12.2 Knowledge Base

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

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

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

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

12.3 Geotechnical Considerations

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

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

Major constraints for a mining method at Grizzly are:

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

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

1. Under 6.5 m:

   Room and pillar mining without backfill in three steps:

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

   Recovery is expected to be 70%.

2 Mining of elongated rooms with backfill:

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

12.4 Stoping Options

12.4.1 Mining Conditions

Mining conditions will vary considerably:

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

12.4.2 Stopes in Ore of Under 6.5 m

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

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

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

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

12.4.3 Stope Performance Under 6.5 Metres Thick

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

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

12.4.4 Stopes in Ore Over 6.5 Metres

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

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

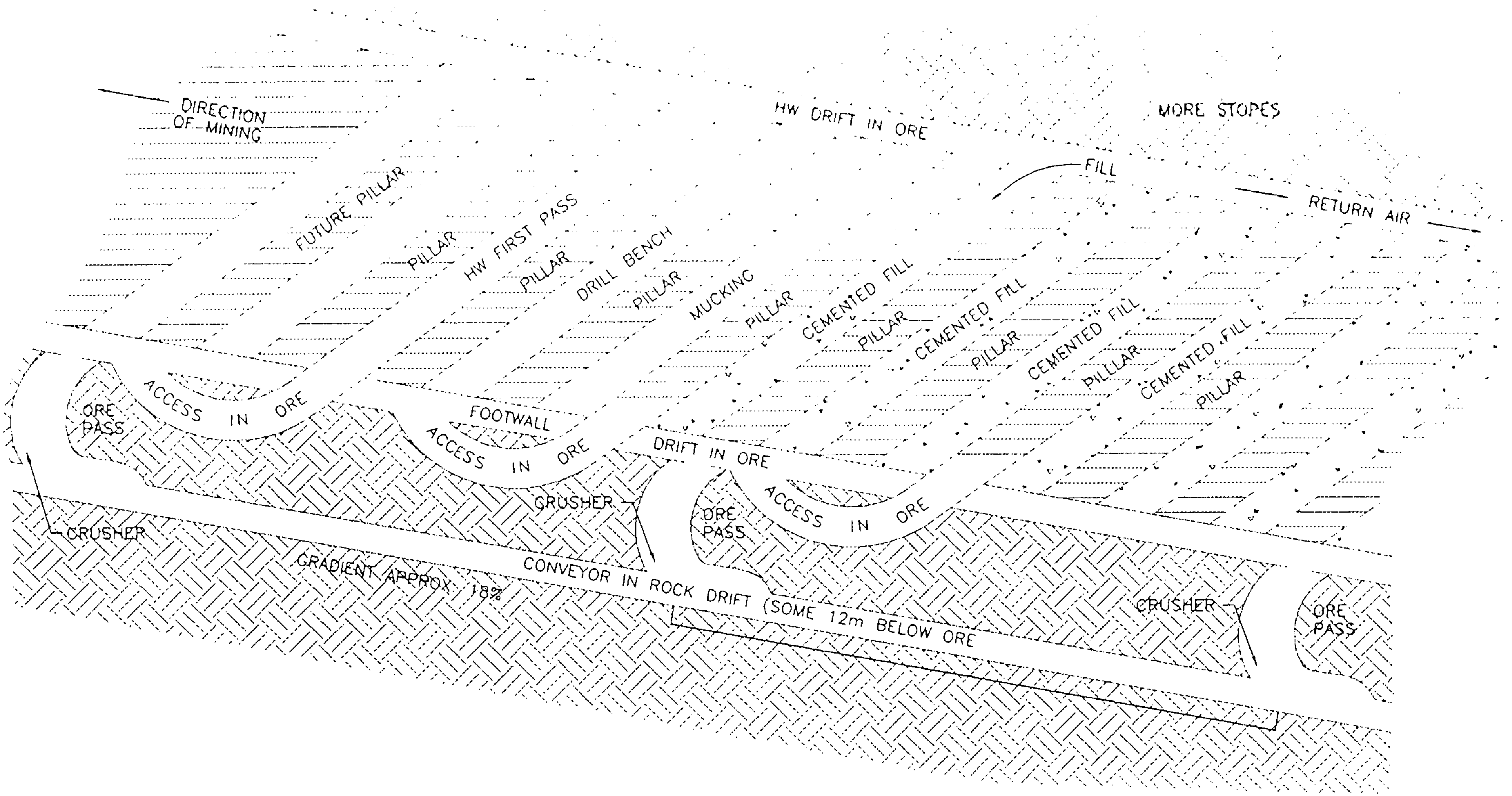
The first operation in the stope is to drive near the hanging wall. Eight foot long rebar bolts and resin are expected to be most suitable for back support. Additional use of straps and/or mesh is anticipated.

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

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

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

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



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GRIZZLY PROJECT
UNDERGROUND EXPLORATION PROGRAM

Fig. 12.4.2 CASE A
Stoping Area In Thick Ore

Anvil Range
MINING CORPORATION

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

12.4.5 Stope Performance Over 6.5 Metres Thick

The three stages of ore extraction are:

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

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

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

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

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

12.5 Backfilling Operations

12.5.1 Introduction

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

12.5.2 General Arrangement

Figure 12.5.01 shows the diagrammatic arrangement of the proposed system.

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

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

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

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

The proportion of different types of material as planned is:

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

A retardent might be considered.

12.5.3 Placement

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

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

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

12.6 Mining Equipment

12.6.1 Introduction

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

12.6.2 Major Items of Equipment

Drills

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

Bench Drill

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

Roof Bolter

A single boom electrohydraulic unit is proposed,

Loader

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

Conveyors are only used in Case A.

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

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

12.6.3 Ancillary Equipment

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

12.7 Ventilation

12.7.1 Aspects of Ventilation for Case A

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

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

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

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

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

Underground, the air splits into two major airways towards the North East. Each drift is planned to be 4 5 metres wide and 5 metres high. The resistance is therefore not high. As the mine expands, the number of splits will increase.

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

12.7.2 Aspects of Ventilation for Case B

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

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

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

12.7.3 Auxiliary Ventilation (Cases A and B)

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

Table 12.A7.1

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

Fresh Air Quantity Requirements

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

Table 12.A7.2

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

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

12.8 Electrical System

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

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

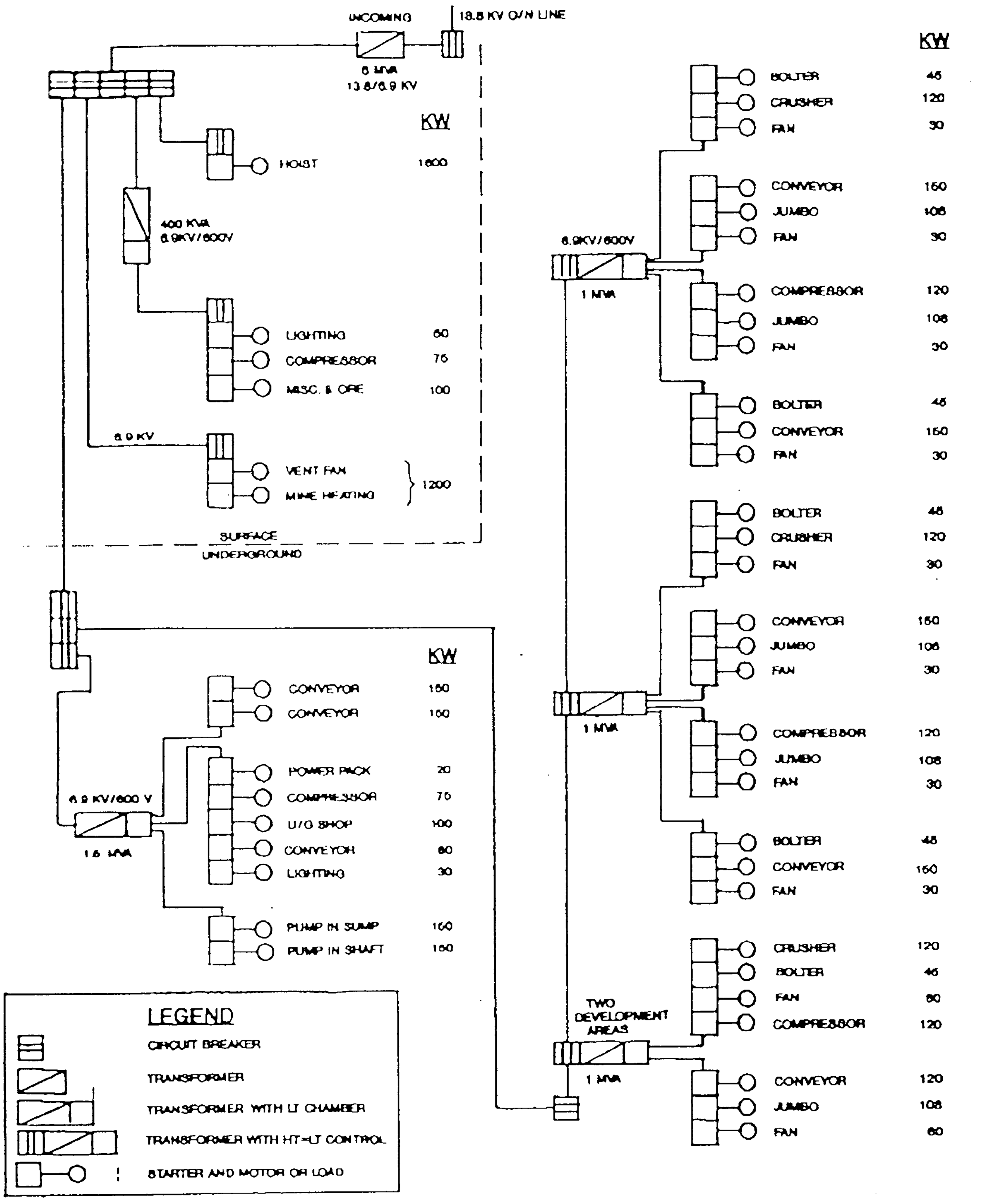
In the mining sections, a single one MVA transformer (6.9 KV to 600 volts) will feed two sections of mining equipment. A second transformer will feed another two sections, and a third similar unit will supply the development drivages.

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

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

There are two differences between Case A and Case B

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



ANVIL RANGE MINING CORPORATION
GRIZZLY PROJECT
UNDERGROUND MINING STUDY

PROPOSED ELECTRICAL SCEMATIC FOR CASE A MINE PLAN

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

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

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

?? too high.

The average works out to 7.4 cents per KWH.

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

12.9 Mine Dewatering

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

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

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

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

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

1.0 Mining Methods

The polygons used to calculate a reserve for the Grizzly deposit can be divided into two groups, as is shown in Tables 1 and 2. Table 1 comprises the thicker portions of the two orebodies, those polygons which are greater than 6 m thick. This accounts for 70% of the reserves. Table 2 comprises the thinner polygons, 6m thick or less, which comprise the remaining 30% of the reserves.

As can be seen on the tables, the polygons demonstrate a strong polarity between the two groups. Excluding one anomaly (block A3), there are no polygons between 6 and 10 m thickness.

The mining method chosen for the thicker polygons is post pillar cut-and-fill (PPCF). Figures 1.1 to 1.5 demonstrate the application of this mining method to a stope which has the average vertical thickness of 15m (13.4m true thickness) and average grade of 27°, as shown on Table 1.

The mining method chosen for the thinner polygons is longitudinal room and pillar (LRP) without backfill replacement of mined out voids. Figures 1.6 to 1.9 demonstrate the application of this mining method to the average vertical thickness of 4m (3.7m true thickness) and average dip of 23°, as shown on Table 2.

1.1 Post Pillar Cut-and-fill (PPCF)

PPCF development would be in the footwall of the orebody. A ramp of 15% grade would loop and spiral as required to provide accesses to the orebody at 15m vertical intervals. A minimum stand-off distance of 10 m from the footwall contact would be maintained.

Each access would set up the mining of three 5m high cuts. Allowing 1m for breaking room, each cut would have a total open height of 6m. This height allows underground trucks to dump loose waste in the stope. The accesses and windows are tentatively sized at 7m width, while the post pillars are sized at 6m by 6m. These dimensions are preliminary only and would be fine-tuned through actual mining. The post pillars would be placed directly atop each other from cut to cut.

The mining of four cuts is demonstrated in Figures 1.1 to 1.4. The mining sequence is demonstrated in Figure 1.5.

A typical mining cut would require two rows of post pillars. The back would be comprised of the ore from the next mining cut. The hangingwall would only be exposed along one edge of each cut. This would allow for a stable back and minimal dilution from the hangingwall. The longitudinal drive closest to the footwall would be used as the primary access for the mining of the panels and windows. It would have a row of post pillars on the hangingwall side and either another row of post pillars or virgin ground on

TABLE 1: POST PILLAR CUT-AND-FILL (PPCF) POLYGONS (> 6m thick)

Block	LENS	Thickness	kTonnes	%Pb	%Zn	Pb + Zn%	DIP
A3	UPPER	-6.8	242	4.8	7.0	11.8	60
A4	UPPER	13.2	1290	6.1	4.9	11.0	24
A4	UPPER	13.2	1290	6.1	4.9	11.0	33
A7	UPPER	15.6	5148	5.2	5.5	10.7	27
A14	LOWER	10.3	621	4.1	8.2	12.3	20
A18	LOWER	15.4	505	5.6	5.0	10.6	26
B3	LOWER	17.8	1204	5.4	9.2	14.6	20
B5	LOWER	16.4	1426	4.3	8.1	12.4	20
B5	LOWER	16.4	1426	4.3	8.1	12.4	33
B7	LOWER	11.6	836	4.2	7.8	12.0	24
Q6	LOWER	16.8	991	3.7	5.2	8.9	31
	AVERAGES	**15.0**	**14979**	**5.0**	**6.4**	**11.4**	**26.9**

TABLE 2: LONGITUDINAL ROOM AND PILLAR (LRP) POLYGONS (<= 6m)

Block	LENS	Thickness	kTonnes	%Pb	%Zn	Pb + Zn%	DIP
A1	UPPER	3.5	614	4.1	7.8	11.9	32
A2	UPPER	3.4	352	5.5	4.6	10.1	28
A5	UPPER	4.3	272	5.4	5.1	10.5	28
A6	UPPER	3.3	366	3.8	6.1	9.9	26
A8	UPPER	5.8	381	8.1	9.0	17.1	19
A9	UPPER	4.1	270	4.2	4.6	8.8	23
B1	UPPER	5.6	354	2.5	6.1	8.6	10
B2	UPPER	3.3	149	3.1	7.0	10.1	17
Q1	UPPER	3.3	356	2.8	6.1	8.9	14
A11	LOWER	3.9	493	3.5	5.5	9.0	19
A13	LOWER	3.4	237	5.2	4.6	9.8	19
A16	LOWER	3.2	214	8.0	5.3	13.3	30
B4	LOWER	4.2	183	6.9	9.9	16.8	22
B4	LOWER	4.2	183	6.9	9.9	16.8	29
B6	LOWER	4.8	292	3.9	6.0	9.9	23
B8	LOWER	3.4	202	4.0	5.1	9.1	35
Q5	LOWER	3.6	940	3.9	6.4	10.3	18
Q7	LOWER	4.9	530	3.4	5.4	8.8	27
	AVERAGES	**4.0**	**6388**	**4.5**	**6.3**	**10.8**	**22.8**

the footwall side. This drift would be driven to the end of the stope, a maximum distance of 300m from the main access point.

As the panels are advanced toward the hangingwall, a wedge condition would be created in the back. The ore would have to be testholed by the jumbo each round to determine the ore wedge thickness. This would be done simply by the colour of the cuttings.

The options for mining this wedge would be to breast down and remove it to the hangingwall or to leave an ore skin. In either case, the panels should be driven from the footwall drift to the hangingwall contact prior to mining the windows.

Each cut moves laterally away from the post pillars of the previous cut as mining proceeds up-dip. For this reason, it may be possible to rob or trim the post pillars which contact the hangingwall on each cut. This would depend on whether or not the practice destabilizes the next cut.

A mining recovery of approximately 80% would be possible with this mining method without pillar robbing. Pillar robbing could increase this figure to as high as 84%.

Dilution would come from three sources:

- internal dilution caused by mining into the footwall (approximately 2% by weight, based on Figures 1.1 to 1.4)
- waste dilution from the hangingwall (approximately 1% by weight, based on 0.5m overbreak), and
- backfill dilution from remucking fill material during floor clean up (approximately 5% by weight based on 0.5m overmucking depth)

for a total dilution factor of 8% by weight, based on specific gravities of 3.92 for the ore, 2.7 for the waste rock, and 2.0 for the backfill.

Leaving an ore skin of 1m width along the back is a practice which has reportedly worked with success at the Faro underground operation. In PPCF, the hangingwall contact is only partially exposed at the end of each panel. The ore skin would not be required for stability, as the footwall drive would be stable, having a massive sulphide back, and would provide access for the entire stope length. Such an oreskin would account for only 4% of the ore on the cut and it may be found that leaving it stabilizes the panels such that a higher recovery of the windows and pillars is possible, justifying the practice.

1.2 Longitudinal Room and Pillar (LRP)

An LRP stope would be comprised of a series of flat lying LRP drifts intersecting a main access ramp at -15% grade. Both the drifts and the ramp would be driven in ore, contact to contact, and have shanty backs to conform to the ore/waste contact.

For the average polygon dip of 23 degrees, a 15% grade is found at 22 degrees from strike. Thus, the intersection of the LRP drifts and the access ramp would be very shallow. Figure 4 shows the detail of this intersection in plan and on section. The up-dip LRP drive would be accessed exactly on its ore waste contact. The down-dip LRP drift would require approximately 20 m to flatten out from its downward orientation then climb back up to the ore/waste contact. The ore left in the back of the down-dip LRP drift would help to stabilize the intersection.

Mining would proceed along the LRP drifts to the end of the stope. Each would then be slashed in the up-dip direction by jumbo or solo drilling leaving permanent rib pillar running the length of the stope. Upon the completion of this slashing, the pillars at each intersection would be recovered in a retreating fashion. The ore left in the back of the down-dip LRP drift at the main ramp intersection would also be breasted at this time.

LRP stopes would not be backfilled.

Ore recovery is estimated to be as high as 73 %, based on LRP drifts of 3.5 m width spaced every 11.5 m, with permanent rib pillar of 3m width. These dimensions are preliminary only, and would be fine-tuned through actual mining.

LRP stopes would receive dilution from two sources:

- waste dilution from hangingwall overbreak (approximately 12 % by weight, based on 0.5m in the bolted backs, 1.0m in the unbolted slash), and
- internal dilution caused by mining into the footwall (approximately 2% by weight, based on Figures 1.6 to 1.9)

for a total estimated dilution factor of 14%, based on specific gravities of 3.92 for the ore and 2.7 for the waste rock.

The practice of leaving an ore skin along the hangingwall is not practical in the LRP mining method, as the orebodies are too thin. Recovery of the orebody would decrease to approximately 54%. Dilution from the back would be eliminated, but the LRP drifts would be forced into the footwall to maintain a mining height of 3.5m for the mechanized equipment. Although the total dilution is decreased, the lower recovery causes an *increase* in the dilution factor to 20%.

1.3 Alternative Mining Methods

1.3.1 Concrete Pillar Mining (CPM)

Steffen, Robertson, and Kirsten (SRK) recommend concrete pillar mining (CPM) in their August 6, 1992 report. The ore is replaced by high strength pillars which support the back as mining progresses, yielding extraction ratios as high as 90 -100%. A reinforceable sulphide skin is left against the back to reduce dilution from the ore/waste contact.

Quoting the report "Tight placement of high quality fill is a must for concrete pillars to function properly." Both of these requirements would be difficult and costly using a crushed rock fill. This mining method is not considered to be practical for this reason.

1.3.2 Sublevel Stoping

Sublevel stoping could be accomplished using a footwall and hangingwall drift arrangement as shown in Figure 1.10. The hangingwall drift is driven along the ore/waste contact. It provides access for cablebolting, production drilling, blasting, and becomes the dump point for filling. The footwall drift is used for production drilling, blasting, and mucking. To achieve a high recovery of the orebody, this drift is placed partly in the footwall waste.

The stope shown on Figure 1.10 assumes that no cement is used in the fill material. Ore skins of 2m width have been left to keep the loose fill material from flowing into the adjacent stope's muck pile. The stope as shown would achieve 80% recovery. Adding cement to make the fill self-standing could increase the recovery to almost 100 %. In this case, primary and secondary stopes would be employed to reduce cement requirements.

The method is hampered by excessive development. To achieve the numerous of mining elevations would require either a cost-prohibitive amount of development outside of the orebody or else the sterilizing of significant reserves by developing in ore. Further, the fill cycle would require that the longitudinal dimension of the stope be quite small (nominally 20 to 30 m) to reduce the difficulties of fill placement, which would have to be done remotely. This would necessitate that the numerous mining elevations be accessed every stoping block, and that additional pillars be left between stoping blocks if cemented tailings are not used. In the LRP and PPCF methods, the longitudinal stope length is determined only by practical mucking limits (nominally 300 m maximum).

The development could be lessened somewhat by compromising the placement of the drifts. The hangingwall drift of one stope could be driven on the same elevation as the previous stope, placing it approximately 4m below the hangingwall contact. Similarly, the footwall mucking drift could be placed at approximately the same elevation of the previous mucking drift, which would place it entirely in waste beneath the orebody. The result of these practices would be much higher dilution, external dilution from the unbolted back in the former case and internal dilution from the footwall in the latter.

1.3.3 Room and Pillar Mining

Room and pillar mining could be employed in the thicker portions of the orebody without the need for backfill. This would be realized at the expense of poor orebody recovery.

The uppermost cut of the stope would be mined first and carefully bolted and screened. Vertical or horizontal benching would then be used to mine down to the footwall, bolting and screening the walls with each cut. The rooms would be as large as possible, as determined by the maximum allowable span of the hangingwall. The pillars would also be large to maintain a reasonable width-to-height ratio.

SRK estimated that the recoveries would not exceed 50% of the mineable reserve using this method.

2.0 Backfill

At mining rate of 4300 t/d (1.5 MT/a), 1097 m3 of void would be generated each day. Only PPCF stopes would require fill, LRP would not. Thus, 70% of the mined out voids would require backfill or 770 m3/day. At an s.g. of 2.0 for fill material, 1536 t/d of fill would be required.

PPCF does not require a high strength fill material. As the walls would not be exposed, the fill does not have to be self-supporting, and the bulk of the fill can be uncemented. The only requirement from the backfill is that a flat and reasonably hard mucking floor be made for the next cut. Ideally, it should be a slurry or paste which is pumped and piped into the poursite with cement added.

For any fill option chosen, development waste would be used to augment the fill system. Assuming 20,000 m is the total of all waste development for the life of the mine, 756 KTonnes of waste would be generated over the life of the mine. This accounts for 12% of the fill which would be required over the life of the mine. Thus, an external source of fill is required to meet 88% of the fill requirements of the mine. Assuming that one round of development waste is generated per day and used as fill material, the requirement for fill to be brought into the mine is estimated to be 1400 t/d. The fill system would be designed to accommodate a fill rate 2000 t/d, with a utilization of 70% to account for disruptions due to equipment and stope availability.

Three possible sources of fill material exist: tailings from the mill, crushed rock from surface (quarried), or a fine grain waste rock and sand mixture from surface.

2.1 Mill Tailings

The mill is 20 km from the minesite. Transportation of tailings would be accomplished either as a slurry in a pipeline or by filtering the tails and transporting the cake by truck. Either option would be quite expensive from a capital cost perspective. The latter would also have high operating costs.

Once brought to the minesite, the fill would be mixed with cement in a surface batching plant and sent by borehole underground as either a paste or hydraulic fill.

2.2 Crushed Rock Fill

Crushed rock would be acceptable as a fill material and presumably could be quarried near the minesite. It is employed in several mines, usually in bulk mining applications, however.

Placement would be expensive, as the PPCF stopes would have several corners for the trucks to maneuver around to properly place the fill. Further, a bulldozer or scooptram with a pusher plate would have to laboriously pack the fill material close to the back. Unlike a bulk mining application, this would have to be done for every 5m cut with high operating costs.

The final result would be a somewhat irregular floor for the next cut, which would cause ore loss and backfill dilution on mucking the next cut. A smooth floor would only be possible by packing the fill material to the back during filling, but this would eliminate the undercut and force drifting rather than breasting on the next cut.

2.2 Surface Waste Rock and Sand

If a fine alluvial sand and gravel mixture could be located near the minesite, it would make a suitable fill material. The mixture would be dug and hauled to a separation plant located on surface near the portal. The material would be screened in stages and separated into coarse and fine products.

The coarse fraction would either be rejected and stockpiled on surface or sent into the mine on the conveyor and dumped into a 1000 tonne capacity underground bin for use as fill material. All development waste stockpiled near the portal could also be returned underground for use as fill. A waste pass would also be driven from the bin to establish a dump point for LHD and truck dumping of development waste. The bin would have a chute at the bottom such that trucks may load and haul the material to the stopes for use as fill. Cement would not be added.

The fine fraction would be fed into a small surface backfill plant housed in the same building as the separation plant. The material would be slurried, cement would be added, and then the mixture would be pumped through a 6" diameter pipeline located in the access or conveyor drift. At the end of the access ramp a second pump may be required to deliver the slurry to the stopes on the northern up-dip extent of the orebody.

The characteristics of the fine fraction should make it a good hydraulic fill material. Being coarser than a mill grind, the dewatering characteristics should be excellent, allowing a short decant time between the pour and the start of the next mining cut. As the fines would not be a sulphide material, it should be less corrosive than the mill tails. As a high strength material will not be required, the mixture could have a low pulp density, which would reduce the friction and pumping requirements.

A minimum of 20% of the backfill (the top 1m of a 5m cut) would be the fine hydraulically placed material from the surface backfill plant.. The remaining 80% of the pour could be the coarse material from the separation plant, development waste, or hydraulic fines from the backfill plant. It would be uncemented regardless of which material is placed. The operating costs associated with the hydraulically placed fine fraction will be less than the coarse fraction because underground truck haulage is not required. For this reason, the excavated material from surface should be as fine as possible, increasing the fine fraction as much as possible.

An alternate location for the separation and backfill plant would be directly above the orebody. This would make hydraulic placement via borehole possible. The coarse fraction could be delivered to the waste bin via a 4ft diameter borehole. This larger borehole would be difficult and costly to place and, in fact, a long alimak raise to surface may be more practical. Further, there will be no possibility of maintaining this waste pass once in place. If spalling of the walls blocks or a hang-up ensues, it will have to be replaced.

The best location for the plant structure will depend upon the distance from the fill material source, the surface topography, and the ability to access and service an alternate location. As road access and electrical power is provided to the portal, this location is assumed.

Although a paste backfill may be possible with the finer fraction, it would not be practical for this application. The usual advantages of a paste backfill system are lost, and other complications arise for the following reasons:

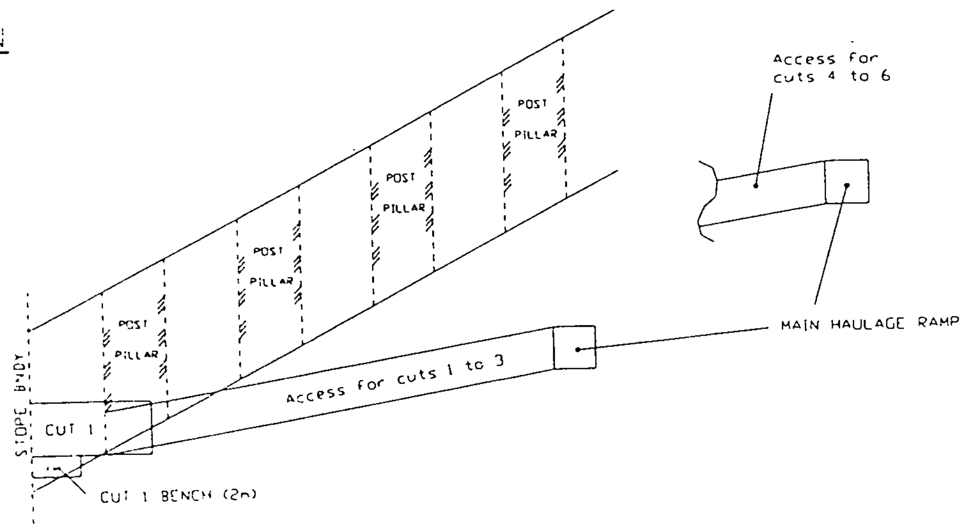
1. A paste fill generally provides more strength for less cement. In this application, high strength is not required. Several of the hydraulic pours will not require cement. Paste backfill always requires a minimum amount of cement to fully hydrate the water, so the cement usage may actually be higher with paste backfill than with a cemented hydraulic backfill.

2. A paste system virtually eliminates decant water, but this advantage is minimal given that the bulk of each pour would be the coarser fraction or development waste, both placed dry.

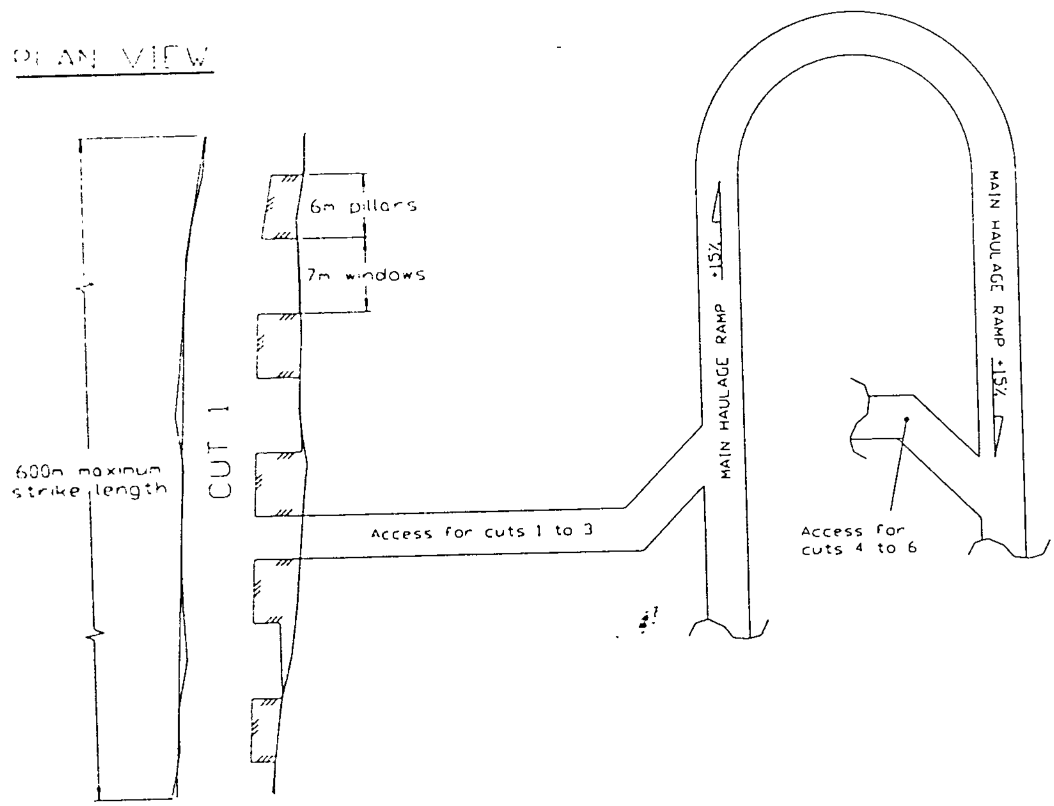
3. A paste backfill system would require additional capital expenditure, particularly for the positive displacement pumps, which would be fairly large to accommodate the horizontal distances required. Because of the higher viscosity, the pipeline will run at a higher pressure with a paste and be constructed of heavier, more expensive pipe.

4. The lack of homogeneity in the surface material's grainsize and colloidal properties would make it impossible to manufacture a consistent paste. It's strength, rheology, and cement requirements will vary, as will its ability to remain a paste.

SECTION



PLAN VIEW



Areas

Ore = 659 m2
Mining = 547 m2
Recovery = 83%

Fig 1.1

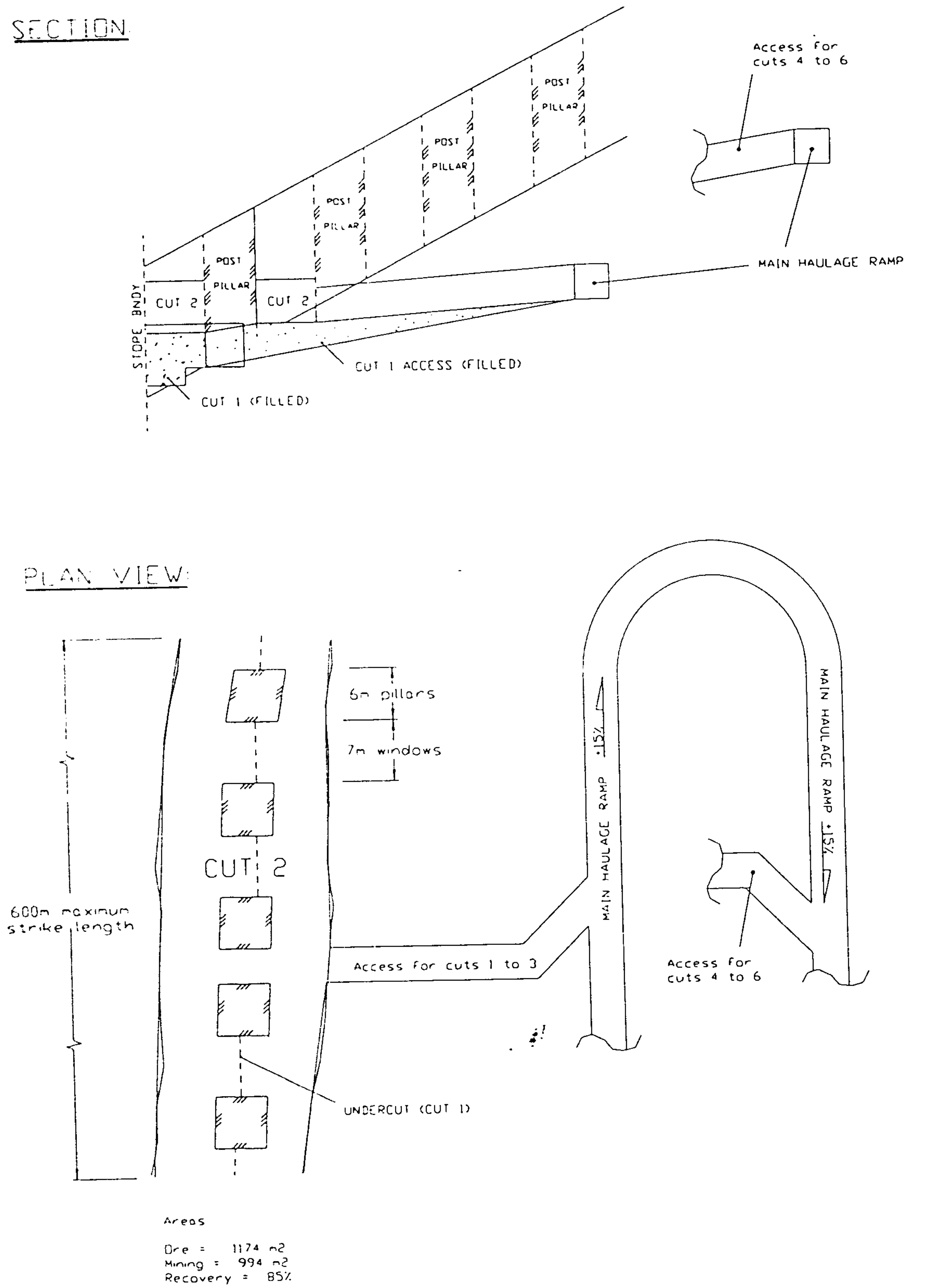
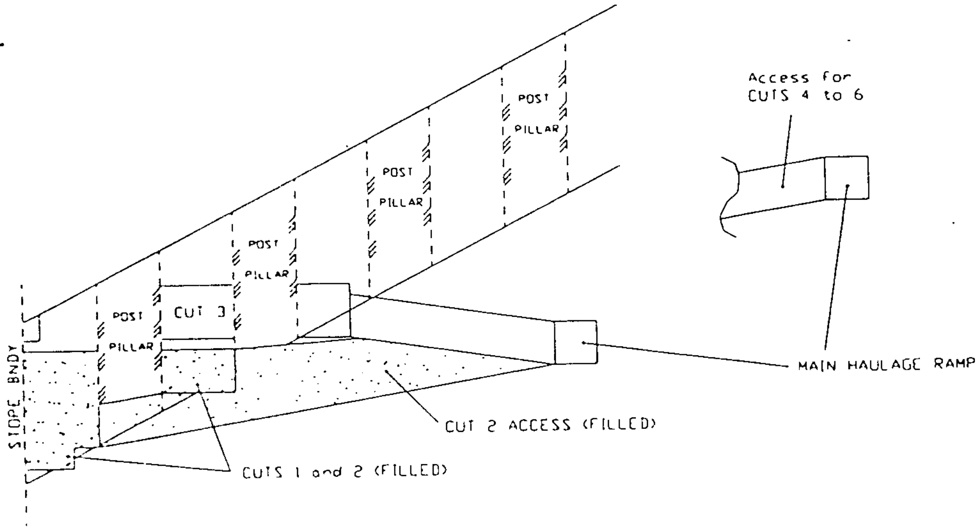
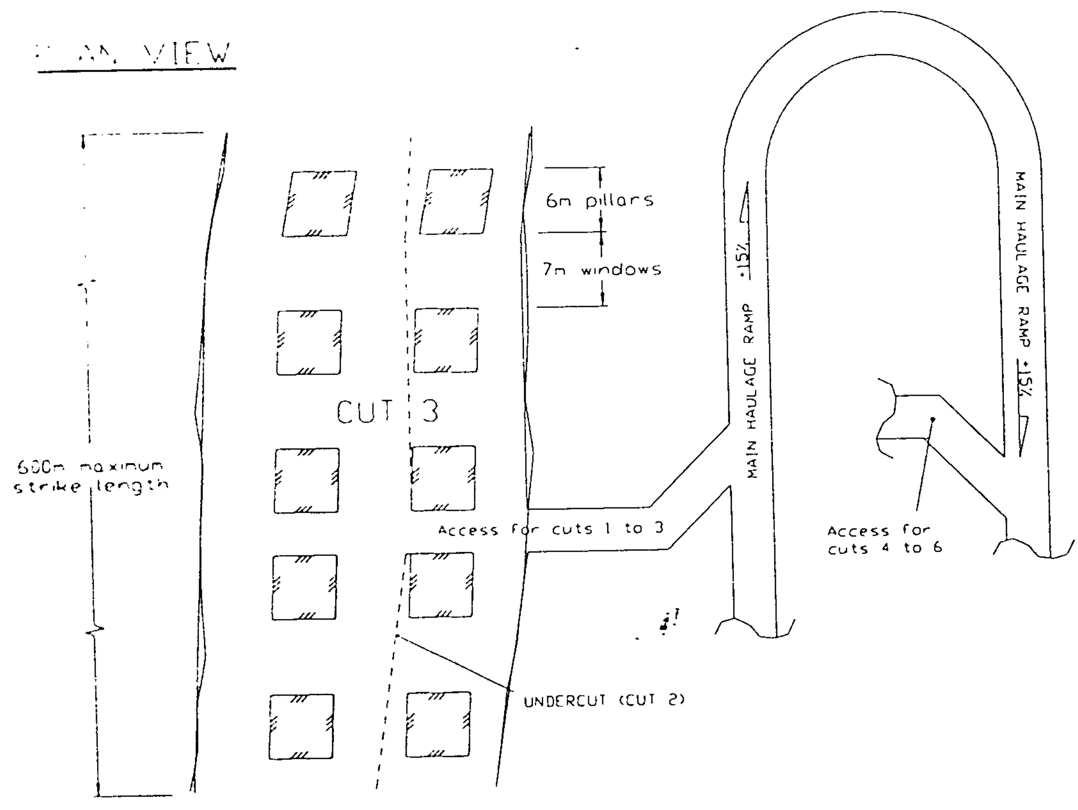

SECTION
POST PILLAR
POST PILLAR
POST PILLAR
POST PILLAR
POST PILLAR
Access for cuts 4 to 6
STOPE BNDY
CUT 2
CUT 2
MAIN HAULAGE RAMP
CUT 1 ACCESS (FILLED)
CUT 1 (FILLED)
PLAN VIEW
6m pillars
7m windows
CUT 2
600m maximum strike length
MAIN HAULAGE RAMP +15%
MAIN HAULAGE RAMP +15%
Access for cuts 1 to 3
Access for cuts 4 to 6
UNDERCUT (CUT 1)
Areas
Ore = 1174 m2
Mining = 994 m2
Recovery = 85%

Fig 1.2

SECTION



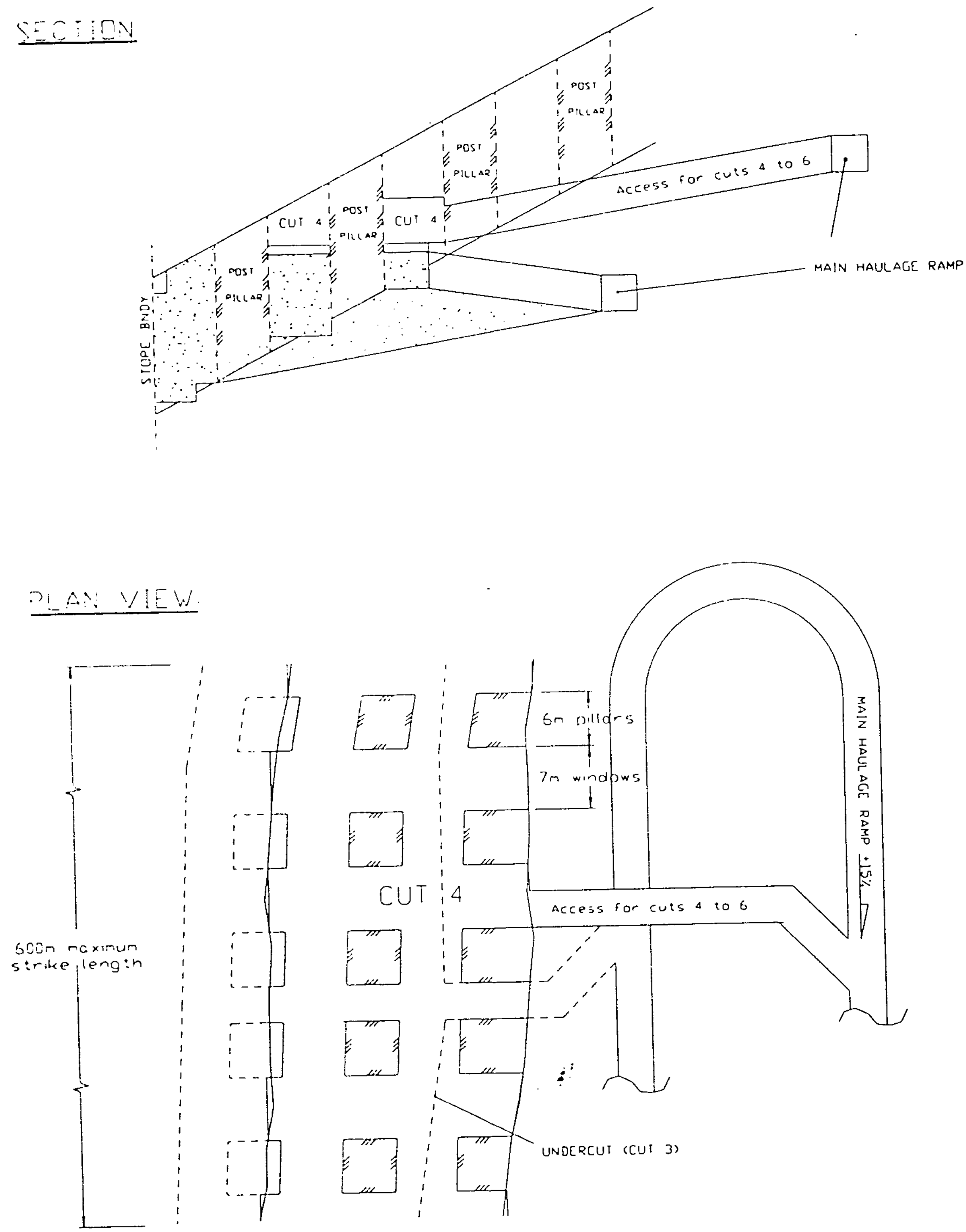
PLAN VIEW



Areas

Ore = 1865 m2
Mining = 1411 m2
Recovery = 76%

Fig 1.3



Areas

Ore = 1760 m2
Mining = 1297 m2
Recovery = 74%

Fig 1.4

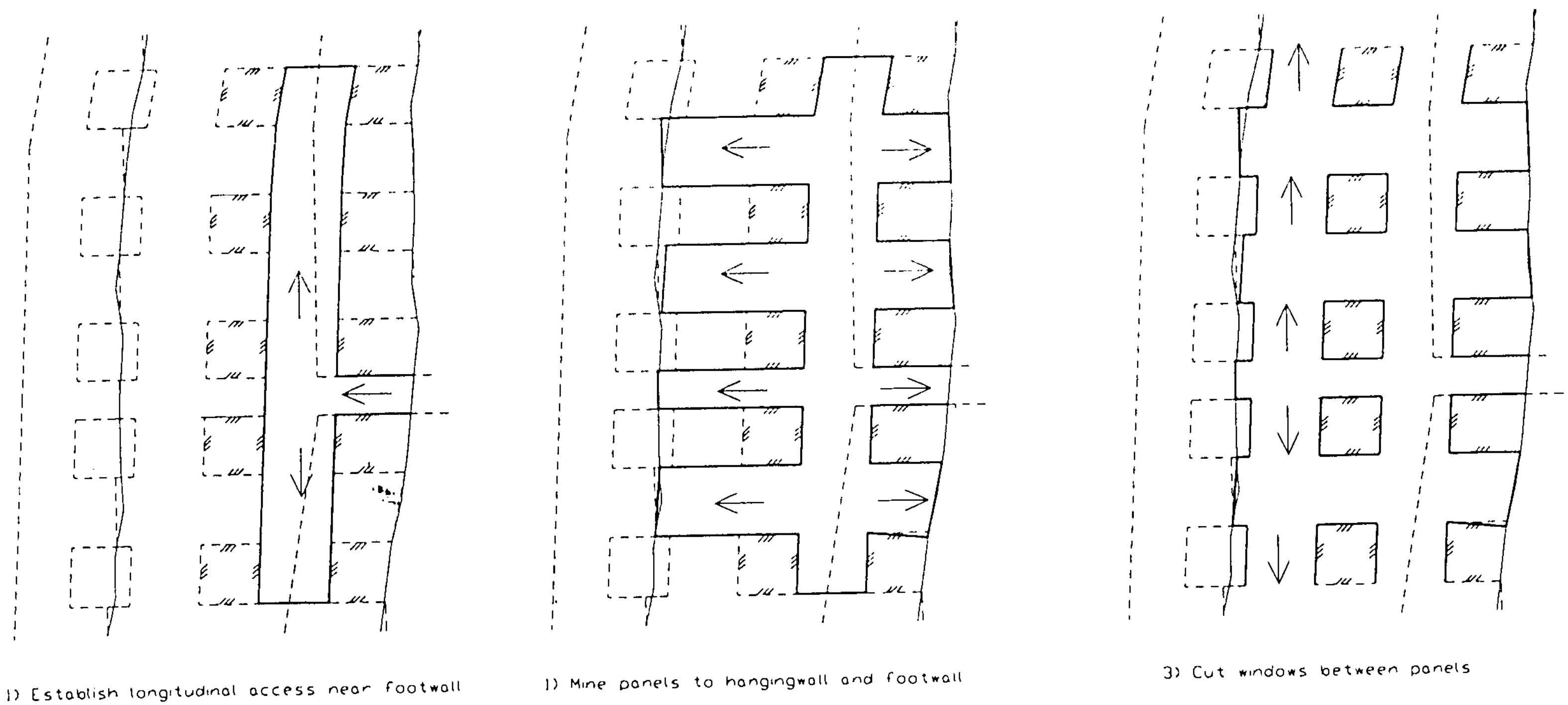


Fig 1.5 MINING SEQUENCE for POST PILLAR CUT-AND-FILL

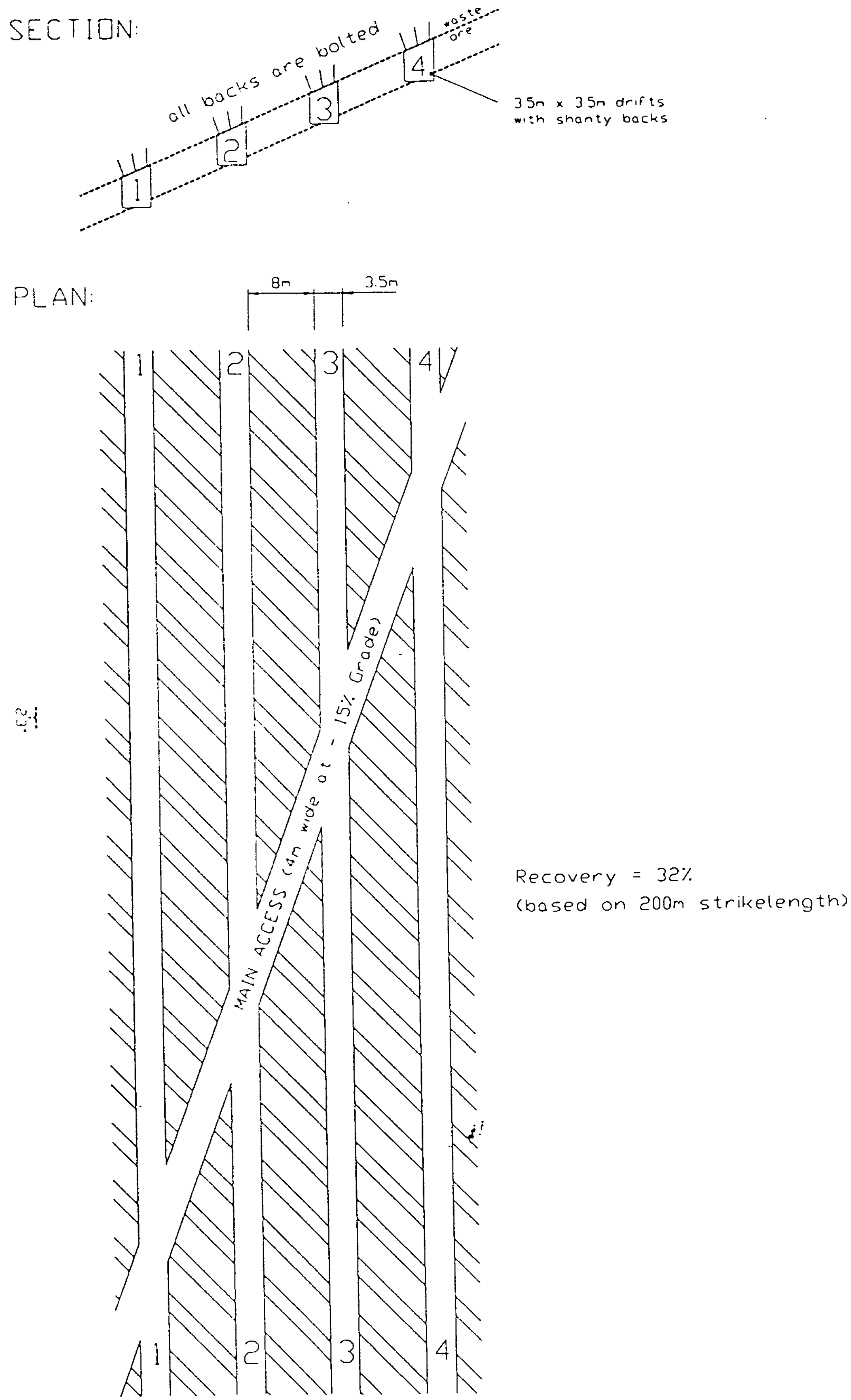


Figure 1: Plan and Section of Longitudinal Room and Pillar Stope showing primary mining only

Fig 1.6

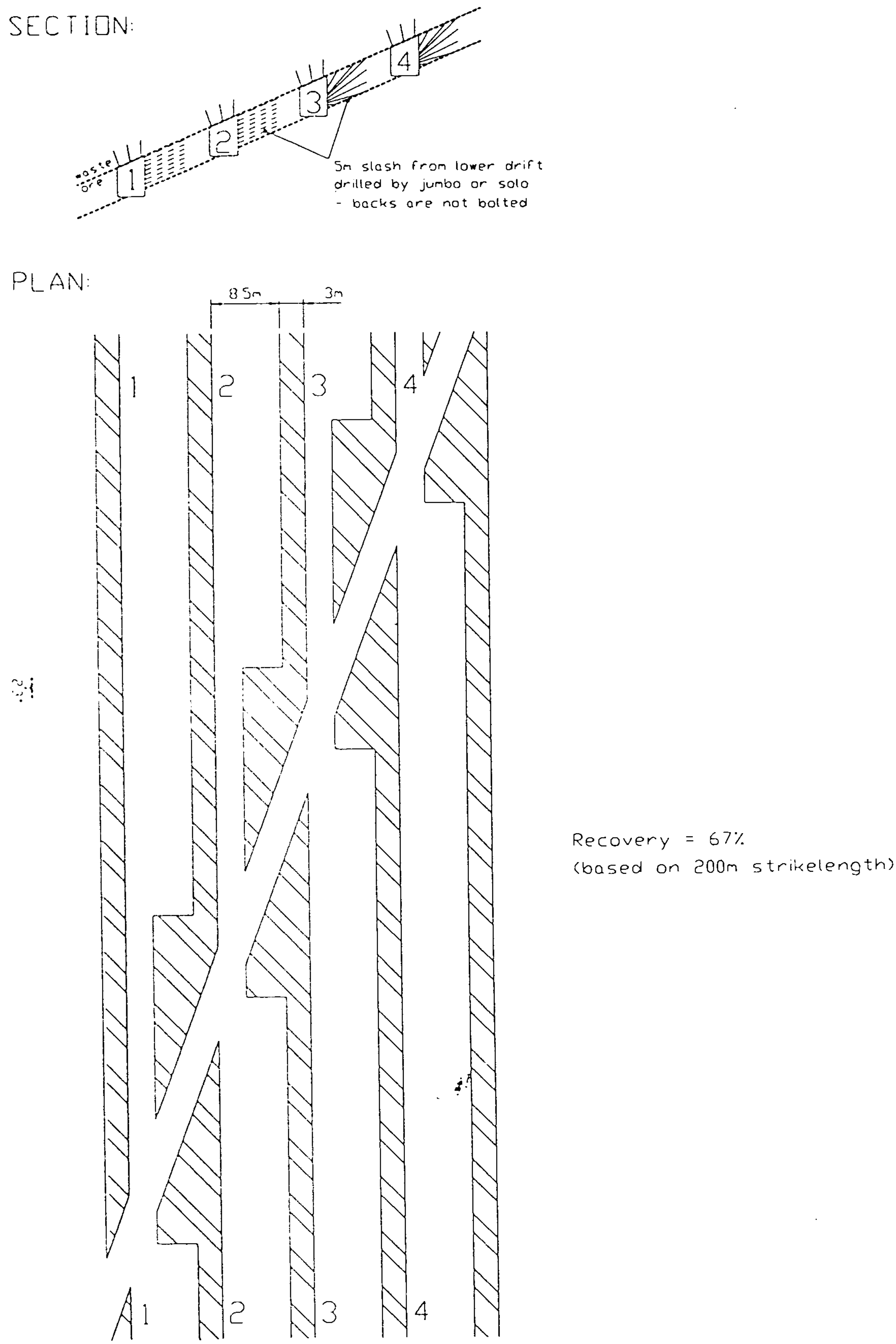


Figure 2: Plan and Section of Longitudinal Room and Pillar Stope after slashing primary drift walls

Fig 1.7

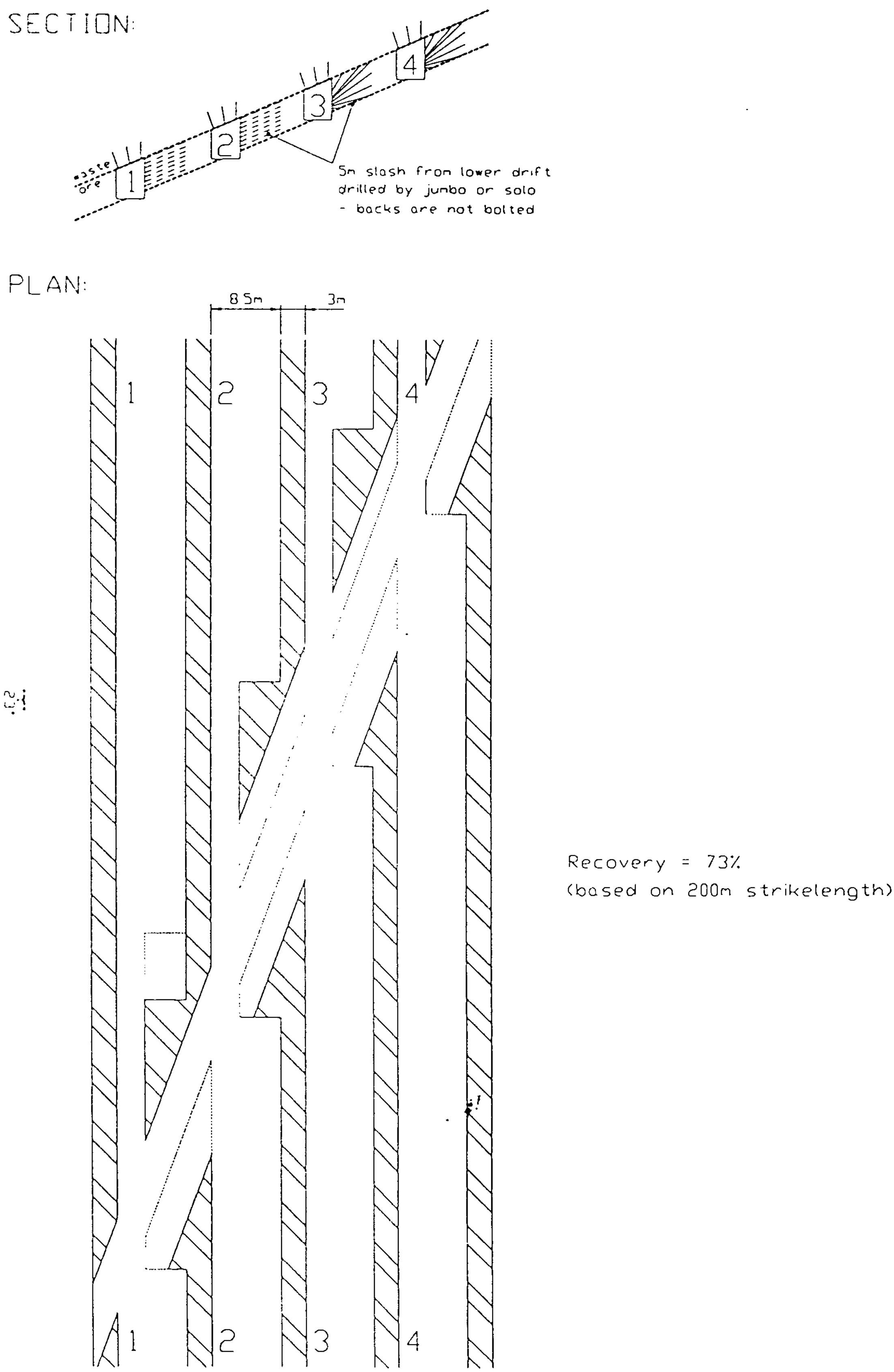
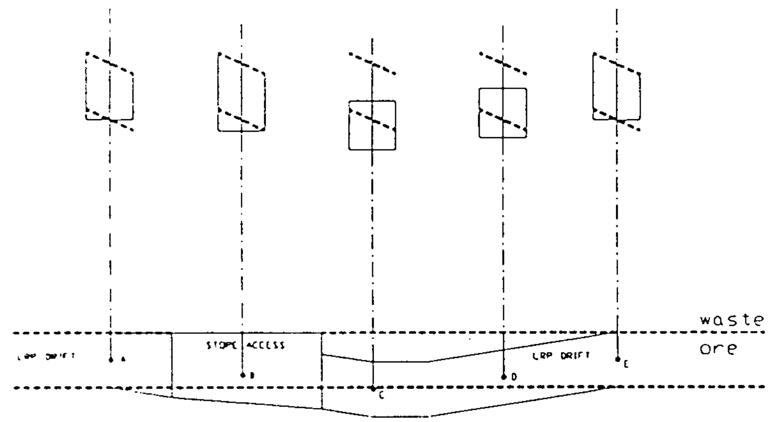


Figure 3: Plan and Section of Longitudinal Room and Pillar Stope after pillar robbing

Fig 1.8

CENTERLINE PROFILE and SECTIONS through LRP DRIFT



PLAN VIEW of INTERSECTION of STOPE ACCESS and LRP DRIFT:

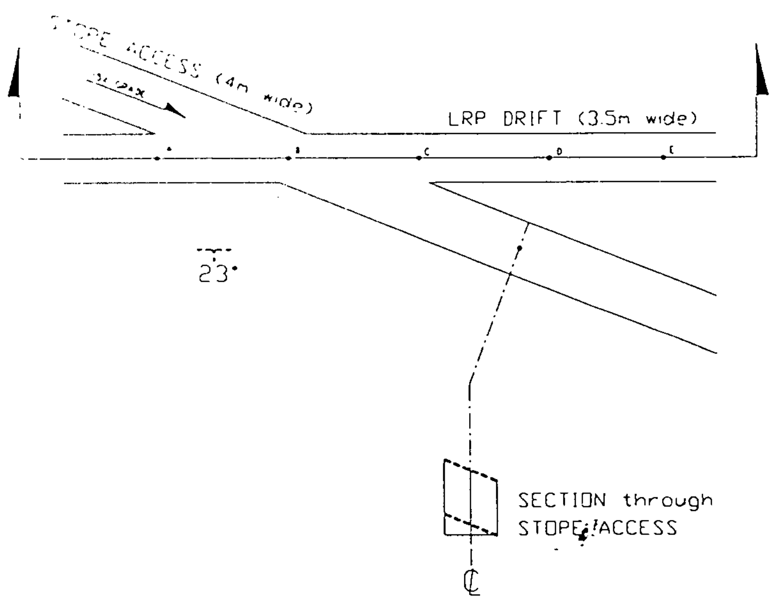


Figure 4: Plan and Section of Longitudinal Room and Pillar Stope showing detail of typical drift intersection

Fig 1.9

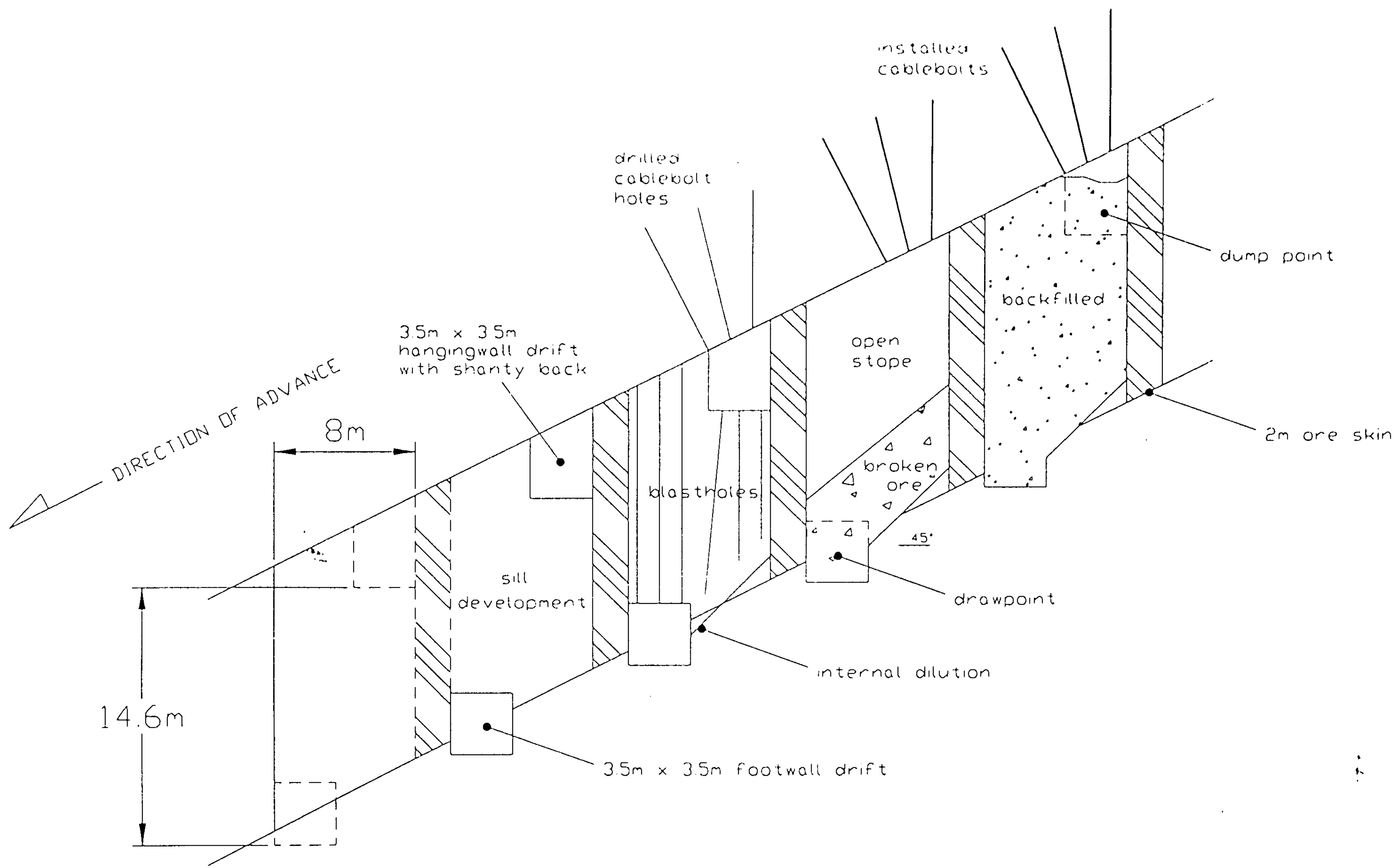

installed cablebolts
drilled cablebolt holes
3.5m x 3.5m hangingwall drift with shanty back
DIRECTION OF ADVANCE
8m
open stope
backfilled
dump point
2m ore skin
broken ore
45°
blastholes
sill development
drawpoint
internal dilution
14.6m
3.5m x 3.5m footwall drift

Fig 1.10

FILE NOTE

August 28, 1996

SUBJECT: DYE/GRIZZLY

I spent a day with Fritz Prugger (yesterday and today) going over current thinking. Highlights:

1. A double entry, small decline system IF it does not cost too much, is an excellent idea (my opinion).

2. A hanging wall drift 80 metres above and a footwall drift 60 metres below the ore provides optimal drill stations...but it does not show you what the ore and hanging wall looks like. We debated pro's and con's of putting one of these drifts in ore or locating them closer to the ore to reduce drilling.

3. They have outlined a 13,000 metre, $700,000 - $1,000,000 drilling program. This will penetrate ore on a 50^M x 50^M pattern over about 20 - 25% of orebody plan. It is located in the heart of the good grade, good thickness ore (will cover ±50% of the "known" metal I suspect).

   We debated spending less money here and cover more of the total area. I think their idea to raise confidence level of ore reserves is good. As planned, it should give a "probable" ore classification to the area involved.

   This leads, of course, to the questions: How do you raise confidence for the other ore inventory? And when?

4. They are thinking/looking at lowering the cutoff grade currently at 9% combined to 6%, 7% and 8%. It doubles the tonnes and increases the metal substantially as well. Looking at the logs, particularly in the current "non ore" areas, it makes good sense to look at this but be careful how far you push incremental costing.

   The prefeasibility study that is underway will allow a good analysis of this.

5. We debated what harm a drift in ore will cause future mining: I do not think it is a serious problem.

   If you know enough about where to drive a hanging wall drift (in the ore), it could be useful in future mining - but I suspect, you will have an uneven hanging wall caused by faulting and folding (S_1 - S_2)

6. They have an idea to drive the decline from the hanging wall drift to the footwall drift eastward through a major fault and then westward back through the same fault. Purpose: To establish a drill station east of the fault to test the faulted upward extension theory.

   The theory is good, but I think I would avoid putting a decline through it twice. Consider swinging the decline and a few degrees to the east to get the bottom closer to the fault then simply drift east to establish the drill station. If it gets too tough, you simply give up. You then spiral down to the footwall or ore horizon west of the fault.

7. Their plan to mine a large tonnage of ore for a good pilot test, but from a single area, should be thought through. You are not getting a "representative sample".

8. We debated - swing the decline a few degrees to the west to get it closer to the "centre" of the ore zones and thus allowing more broad coverage to explore the "entire" area. They had already examined this...it is properly abandoned.

9. We debated various mining systems. Fritz is thinking room and pillar in the 3^M thick ore and a longitudinal panel/fill system using an overcut and an undercut, long hole benching and remote mucking for the +20 metre thick areas. He is not sure where one changes over to the other?

10. We examined the cores +20 holes. I was pleasantly surprised at how competent the hanging wall appears. Cores are +15 years old and the phyllite has NOT turned to mud. It looks quite good.

    One hole intersected a fault at the hanging wall and it looked terrible; one or two others had some weathered phyllite in the hanging wall, but it was certainly manageable. Friz has also been down the Grum decline...and the ground appears pretty good.

11. I am a room and pillar fan! I suggested that Fritz visit two companies:

    (i) **Boliden** - who were technical managers at the Black Angel Mine in Greenland. Examine the "contour" room and pillar thinking and planning in areas dipping 20 - 25 degrees. Fritz has visited Greenland and knows how to make contact to see if maps and planners are still around.

    (ii) **Doe Run, Missouri** - are using a panel pillar and fill system in good grade areas which allows ±95% ore recovery...plus they room and pillar mine ±100 feet high (in very good ground).

    If Grizzly ground proves to be better than we think, you could mine it very successfully!

12. We visited the portal site. It looks good...BUT it is very close to Blind Creek, a sacred salmon spawning stream. Greg Jilson questioned whether we can get it permitted.

    Greg and Fritz seemed to get along very well! I suspect Fritz was not fully aware of how well informed and valuable Greg can be?

Conclusion

I liked what I saw and heard. I hope I contributed a little.

I think Grizzly is shaping up to be a tough decision:

(i) The decline/drifting/drilling as planned will be expensive - something less than $20.0 mm I expect.

(ii) By the time you explore the other areas of the known Grizzly foot print - you will spend $X million more.

(iii) Then you do the metallurgical testing, definitive mine planning and final feasibility details, permitting, etc. and you spend another $Y million.

(iv) When it comes time to sink the shaft (small tonnage, say 1.0 mm/year) or the conveyor declines (larger tonnage, say 2.0 mm/year) you have likely spent +50% of the required capital, i.e. you should have an easier time to justify the (remaining) project...SUNK COSTS ARE IGNORED.

(v) The current prefeasibility, therefore, will be a very important first step. The board must have a good sneak preview of the entire project **before** launching the Phase I initial work.

Selecting appropriate metal prices for this prefeasibility study will not be easy. We are going through a zinc drought (low prices and high inventories). It will not be easy to forecast a 55c or 57c long term price today, but I suspect we will need that to justify Grizzly! I think I could accept prices like this.

MNA:bk

cc: Kurt Forgaard
Fritz Prugger

ANDERSON
& ASSOCIATES

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October 19, 1996

Mr. Fritz Prugger
Anvil Range Mining Corporation
117 Industrial Road
Whitehorse, Yukon
Y1A 2T8

Dear Fritz:

Subject: Grizzly Meeting Today
Attendees Fritz, Gregg, Nick and I

Highlights:

1 We discussed the concept of increasing decline dip from 15% to 20% and possibly swinging or bending it to intersect more in the middle of the "footprint". All sounded good IF the geometry works out.

2. We reviewed the current "exploration plan". Again, it looks good. It covers about 66% - 75% of the footprint. It covers the A and B best zones reasonably well and it penetrates the ore sufficiently, I think, to extract a representative metallurgical sample and to test mining conditions.

3. We heard the latest numbers from contractors:

- Decline and development - $19.0 million plus contingency and drilling say $21.0 million.
- Contract mining (including amortization) $55 - $60/tonne (not definitive).

4. Fritz mentioned a possible shaft location on north end which has some attractive features (geometry). Need to confirm ground conditions, but NO decision necessary yet.

5. Panel pillar mining is planned. I suggested we show an area or two, where bulk mining and the system, would (likely) be appropriate.

6. My No. 1 objective at this early date would be to lock in on tonnes and grade for the prefeasibility:

(i) The simple way to look at it would be a 9% cut-off and the tonnes and grade straight off the polygons.

(ii) You cannot mine to a 9.0% cut-off...you will have low grade in the +9% polygons and you will have high grade in the -9% polygons. Therefore, a more practical answer is required. Nick and Gregg will check out "something" akin to my October 7 memo.

(iii) Should we test sensitivity (best case scenario) to being able to mine some heretofore unidentified, high grade pods X tonne of Y% grade...as has been and is being done at Faro, Vanguard and Grum. I think the answer was yes.

Gregg and Nick will spend some time on this to identify a chart that may look like this:

Case	Mineable Tons Undiluted (Tonnes)	Grade % (Pb + Zn)	Cut-off (% Pb + Zn)	Mining Cost ($/Tonne)	Annual Production Tonne
Nick's Reserves	14.0 M	12.13	9.0%	$30	1.0 M
Norm's ** Noodling	21.0 M	10.22	Mixed 9 and 6	$28	1.5 M
Add Hi Grade Pods*	X M	Y%			

* Add this to both Nick's and Norm's cases.

** Nick and Gregg will work out a base case based on "something like this".

REVIEW OF PREVIOUS NOTES

The following question needs addressing for the prefeasibility:

1. How much has been spent on Dye including by others to date?

2. Need contractor's contingency for water, bad ground, etc. ($2 million or so, I suspect).

3. What metallurgy will you use in study? Where did it come from?

4. Here is an example of the kind of pressure you will likely get from the Board re 55¢ zinc.

The Globe and Mail, Monday, October 21, 19..

Inmet Mining

Inmet Mining Corp. says 48-per-cent-owned Société Minière de Bougrine SA has indefinitely suspended mining and milling operations at its zinc/lead mine in Tunisia.

Toronto-based Inmet said an evaluation of the capital required to lower operating costs has been completed, and "the investors in Bougrine have determined that the required investment is not justifiable in light of current zinc prices."

Inmet said it made a partial write-down of its investment in Bougrine June 30. It said it expects to make payments on about $33-million of loans made by two international development banks to Bougrine. The loans have been guaranteed by Inmet.

Norm

MNA:bk

To: Fritz Prugger

From: Norm Anderson

Date: October 7, 1996

Subject: Grizzly

FAX: 926-3446

You and I reviewed your latest thinking and planning based on 1:2500 plans by NDR dated 9/06/96 and 10/06/96. Observations:

1. Grizzly 1 - Upper B Zone 9% Cut-off

Three blocks B1, B2 and Q1 (Upper B) do not make mining sense. They are three discrete pods, totalling 861,000 tonnes at 8.95% combined and separated from each other by 200 - 500 meters of low grade. Difficult to mine!

When you look at Grizzly 1, upper zone - 6% cut-off, you have a more mineable package: B1, B2, Q1 plus Q2, Q3 and Q4.

We then separated the tonnes and grade from your mining inventory chart at 10% dilution as follows:

Upper B Zone at 9% Cut-off

Block	Av. Thick (M)	Tonnes	Comb. Grade (%)
B1	5.6	355,000	8.5
B2	3.3	150,000	10.04
Q1	3.3	356,000	8.95
Av.	4.07	861,000	8.95%

Upper B Zone at 6% Cut-off

Block	Av. Thick (M)	Tonnes	Comb. Grade (%)
B1	5.6	355,000	8.5
B2	3.3	150,000	10.04
Q1	16.12	2,655,000	6.44
Q2	3.75	161,000	5.94
Q3	6.98	631,000	7.26
Q4	4.65	841,000	6.15
Av.	6.74	4,793,000	6.75%
DIFFERENCE	2.67	3,932,000	2.20

If one were to mine the Upper B zone to a sensible, geological cut-off (whatever it may be), I would guess you have to mine at least 2.5 mm tonnes at about 7.5 - 8.0% combined IF dilution is 10%.

2. We then repeated my "funny logic" on the Lower B zone as follows:

Lower B Zone at 9% Cut-off

Block	Av. Thick (M)	Tonnes	Comb. Grade (%)
B3	17.83	1,204,000	14.53
B4	4.2	366,000	16.83
B5	16.4	2,852,000	12.38
B6	4.8	292,000	9.85
B7	11.6	836,000	12.03
B8	3.4	202,000	9.08
Q5	3.9	940,000	10.31
Q6	16.8	991,000	8.84
Q7	4.9	530,000	8.78
	Av. 9.31	8,213,000	11.79

Lower B Zone at 6% Cut-off

Block	Av. Thick (M)	Tonnes	Comb. Grade (%)
B3	17.83	1,204,000	14.53
B4	4.2	365,000	16.82
B5	17.9	3,114,000	11.84
* B6	4.6	553,000	8.71
B7	15.0	1,089,000	10.61
B8	5.1	300,000	8.00
Q5	4.8	1,252,000	9.39
Q6	29.5	1,739,000	7.00
Q7	11.6	1,244,000	7.00
** Q8	3.3	354,000	5.60
** Q9	9.7	2,112,000	5.73
Q10	3.6	943,000	7.59
	Av. 10.59	14,269,000	9.34

* This does not look right compared to B6 at 9% cut-off?

**Omit Q8 and Q9	Av. 11.4	11,803,000	10.10
**DIFFERENCE	2.09	3,590,000	1.69

**Q8 and Q9 may prove uneconomic and can sensibly be left out of the mining plan.

This indicates a better correlation in grade and tonnes between 6% and 9% cut-offs than with the Upper B zone (item 1 above).

It also clearly shows better tonnage and better grade. I would guess you might mine 10.0 mm tonnes at 11.0% combined in the lower B zone if you can maintain a 10% dilution.

3. We then looked at the Upper A zone:

Upper A Zone at 9% Cut-off

	Block	Av. Thick (M)	Tonnes	Comb. Grade (%)
	A1	3.5	614,000	11.93
	A2	3.4	352,000	10.06
	A3	6.75	242,000	11.75
	A4	13.2	2,580,000	10.97
	A5	4.25	272,000	10.61
	A6	3.3	366,000	9.81
	A7	15.56	5,148,000	10.72
•	A8	5.8	381,000	17.06
	A9	4.0	270,000	8.80
		Av. 6.64	10,225,000	11.01

• A8 at 17.06 is pretty high grade! May need cutting.

Upper A Zone at 6% Cut-off

	Block	Av. Thick (M)	Tonnes	Comb. Grade (%)
	A1	4.6	816,000	10.4
	A2	7.6	782,000	7.57
	A3	6.75	244,000	11.75
	A4	18.52	3,601,000	9.68
	A5	7.71	495,000	8.55
	A6	4.8	530,000	8.88
	A7	16.9	5,594,000	10.37
••	A8	21.5	1,396,000	8.62
	A9	4.1	272,000	8.80
	A10	10.2	746,000	6.45
		Av. 10.27	14,476,000	9.55
	DIFFERENCE	3.63	4,251,000	1.46

•• and • - A simple "cut" would be to substitute A8 at 6% cut-off for A8 at 9% cut-off. This adds 1,015,000 tonnes and brings the 9% cut-off grade from 11.01% to 10.51%.

It shows remarkable good correlation (tonnes, grade and thickness) with the Lower B zone above!

I would guess we might mine 11 to 13 million tonnes of 10% to 11.0% grade...depending on the actual grade of A8 (and dilution, of course).

4. Finally we looked at the Lower A zone:

Lower A Zone at 9% Cut-off

Block	Av. Thick (M)	Tonnes	Comb. Grade (%)
A11	3.9	493,000	8.97
A13	3.4	237,000	9.82
A14	10.3	621,000	12.35
A16	3.2	214,000	13.32
A18	15.4	505,000	10.57
	Av. 7.24	2,070,000	10.92

Lower A Zone at 6% Cut-off

A11	10.9	569,000	7.49
A13	9.5	1,312,000	7.48
A14	15.2	916,000	10.22
A16	3.2	214,000	13.32
A18	15.4	505,000	10.57
A12	4.0	303,000	8.98
A15	5.6	631,000	7.29
A17	5.2	281,000	7.39
A19	3.6	263,000	6.05
A20	4.8	739,000	6.89
	Av. 7.74	5,733,000	8.32
DIFFERENCE	0.50	3,663,000	2.60

- This does not look right - 6% cut-off versus 9% cut-off thickness and tonnes.

Here again, the tonnes and grade of the Lower A from 9% to 6% cut-off is very similar to Upper B zone. Why? (Although it appears to be better grade than Upper B.)

I would guess you may mine 3.5 - 4.0 mm tonnes of 9% combined.

CONCLUSIONS

1. Upper A zone, I guess, will mine out at 11 - 13 million tonnes of 10 - 11% combined.

   Lower B zone, I guess, will mine out at 10 million tonnes of 11% combined.

   These appear to be the two best zones and need to be in the underground exploration and confirmed as No. 1 priority.

2. The Upper B zone, which I guess mines out 2.5 millions at 7.75% combined - appears to be the least likely to succeed = lowest priority.

3. The Lower A zone is in the middle, it appears at 3.5 - 4.0 million tonnes of say 9% (my guess) and at the right metal prices, it should make it.

4. Compiling these:

Zone	Tonnes (mm)	Grade		Cumulative Tonnes (mm)	Grade
Upper A	12.0	10.50%	(Priority 1)	12.0	10.50
Lower B	10.0	11.00%	(Priority 1)	22.0	10.73
Lower A	4.0	9.00%	(Priority 2)	26.0	10.46
Upper B	2.5	7.75%	(Priority 3)	28.5	10.22
		Mine Recovery 75%		21.4	

5. I have NOT examined the source data for these tonnes and grades. This should be done before picking a prefeasibility grade...preferably by a good ore reserve geologist (Gregg Jilson?).

6. Is the Upper A zone and the Lower B zone the same zone? In the two sections we saw today, they are stratigraphically the same. We should check some additional sections. This may indicate that Upper A or Lower B may be overturned.

MNA:bk

To: Fritz Prugger

From Norm Anderson

Date. October 8, 1996

Subject: Design and Operating Parameters
Grizzly Report

As discussed, I have read your six page itemized synopsis. It is a good, comprehensive list of items that will require attention. My comments (reactions) are as follows:

I. PAGE 1

1. In view of our doodling yesterday on tonnage and grade, I suspect you might push for a base case of 1.5 million tonnes per year MAXIMUM. Remembering how difficult it was to sustain production over 1.0 mm TPA at Rubiales Mine, I am cautious to go higher because of the limited infrastructure and the potentially difficult ground.

   Grade estimate for a base case will be an interesting exercise...need more discussion and expert help. What grade of dilution was included in your 10% dilution addition?

2. Mining Contractor - In the prefeasibility, I am happy to go along with the mining contractors for the first two years of operation...BUT...I am wary of bad results from contractors and would always want the option to boot them out.

   I am not sure, depending on the availability of cash, whether we should not own the underground equipment. If the contractor owns it, we will be paying amortization, the equipment may not be "up to snuff" and it makes it tougher for us to boot them out.

3. We should lay out a mine plan for feasibility purposes and not worry too much about it. After we have seen the ore, determined rock competence, chocked water flows, etc. we should then optimize the mine plan, shaft location, sump details, vertical distance from haulage to ore, etc.

4. Computerized mine planning - I know you are keen about 3D modelling, etc., but be cautious, please.

II. PAGE 2

5. High level of automation - provided the cost is justified!

6. A pump station in the low point of the mine means the pump station gets flooded first in a power outage. We are going to need to think this through careful, if we have a wet mine. There are ways to mitigate.

7. Hoisting - As discussed yesterday, we have to consider an emergency escape via the shaft because access to the twin decline may be blocked.

III. PAGE 3

8. **Auto Hoisting** is a good concept for our prefeasibility. We may, however, buy a used, unautomated hoist/headframe and would want to revisit cost of automating.

   Pumping Controls are inexpensive and necessary. Various other controls require justification at the point of final approval to purchase.

9. "Cage over Skip Combination" - I would prefer to say "skip cage combination". A cage over skip will require an extra high headframe (unless we have a below surface receiving bin.) A cage under the skip, of course, requires a deeper shaft = trade offs.

10. Conveyor haulage of waste up the decline may or may not be best way. We should likely have the option to be able to hoist waste too...depending on hoisting capacity.

IV. PAGE 4

11. Whether or not we develop mining levels with conveyors is a decision I think we will make after assessing how well we have done on the twin decline. If we go with Redpath's single, large decline, we will have another problem in deciding on conveyor.

    I acknowledge you are vastly more familiar than I with conveyors.

12. Underground Infrastructure:

    (i) We may use electric hydraulic jumbos which places more emphasis on power distribution and less on compressed air lines.

    (ii) Underground cement storage - If we use a lot of cement and can get bulk deliveries, we might want a surface storage plant and mixer and pipe slurry down the shaft.

    (iii) Do you want to mention a backfill pit on surface plus screening plant, etc.?

13. The term "drop raises" for backfill is interesting. As discussed yesterday, we might drill an 8 inch hole for backfill. A "drop raise" I thought was a short (50 feet) raise drilled from an upper level to a lower level with an air track.

V. PAGE 5

14. Under ore body characteristics, I have underlined everything to do with rock quality. It will have an enormous effect on mine planning. I would add "faulting" to this list.

15. Your current plan to mine panels is the right way to start...subject to revisions as we find out more about rock quality, securing the back, height to be mined, undulations/faulting of the ore, water, etc. Size will be determined later.

VI PAGE 6

16. You might want to consider a bonus system if we do our own mining.

Larry

MNA bk

The #3 gold-bearing zone, which is mined underground, is located in felsic tuffs dipping steeply to the South (85°). The tuffs exhibit a high degree of schistosity with sericite concentrated along the schist planes.

Considering the ore zone width, which is 13 to 28 meters, a transverse mining method has been selected. The "TRANSVERSE OPEN STOPING" permits a hundred percent (100%) ore recovery at a minus five percent (5%) dilution.

In this mining method, the ore zone is divided in transverse primary (6m) and secondary (10m) stopes with sublevels located at 20 meter vertical intervals. The primary stopes will be backfilled with cemented rockfill and the secondary with rockfill only. Rockfill is provided from mine development and the open pit waste rock, which is stockpiled near the fill raise on surface.

The cemented rockfill (5% to 7% CEMENT) is a minimum of rockfill with cement in pulp (60% solid). The slurry is prepared at the surface plant which consists of 2 silos (220 mt) containing dry Portland Cement, a mixing tank ($17 m^3$), a water line and a retardant tank (4,000 ℓ).

The dry Portland Cement (14.4 mt) is fed into the mixing tank by screw conveyor (36 mt/h, 8 BHP) and is then mixed with water ($9.6\ m^3$). Mixing is done with a LIGHTNIN agitator (25 HP) equipped with an impeller of one meter diameter. Retardant (DARATARD #17) is also added at a ratio of 185 ml/100 Kg. This admixture retards the setting time for 3 hours after the initial mix.

The time required to prepare the slurry ($14.5\ m^3$) is one half hour. Pulp is 60% solid (1.69 pulp density).

The cement consumption is controlled weekly. Both silos are equipped with a BINDICATOR YO-YO, which consists of a 2 vessel mounted sensor unit and a remote digital readout.

The slurry is piped by gravity into a backfill line, (430 m) drilled in the rock, to the underground reservoir (17 m^3) located at the backfill station. All drilled holes are cased with a metal tubing (10 cm ⌀) which prevent the hole from collapsing. A spare line is also operational.

The underground backfill station consists of a reservoir (17 m^3), a slurry pump SRL - 3X3 (5 HP) and a measuring tank in which the slurry level is automatically controlled by a floating switch. Pulp is kept in suspension by 2 LIGHTNIN agitators (3 hp). A pneumatic BUTTERFLY valve, fixed at the end of the measuring tank, controls the slurry output.

Rockfill is loaded into the TELEDUMP trucks (2 DUX TD-20), which are designed to dump in 3.2 meters high back and then, the slurry is added to the right proportion (5% to 7% cement). All switches and valve controls are located next to the operator's side wall.

Loading and dumping time are respectively 30 and 60 seconds. Hauling time varies from 4 to 6 minutes, depending on the travelling distance to the stopes. Productivity is 400 metric tons per 8 hour shift.

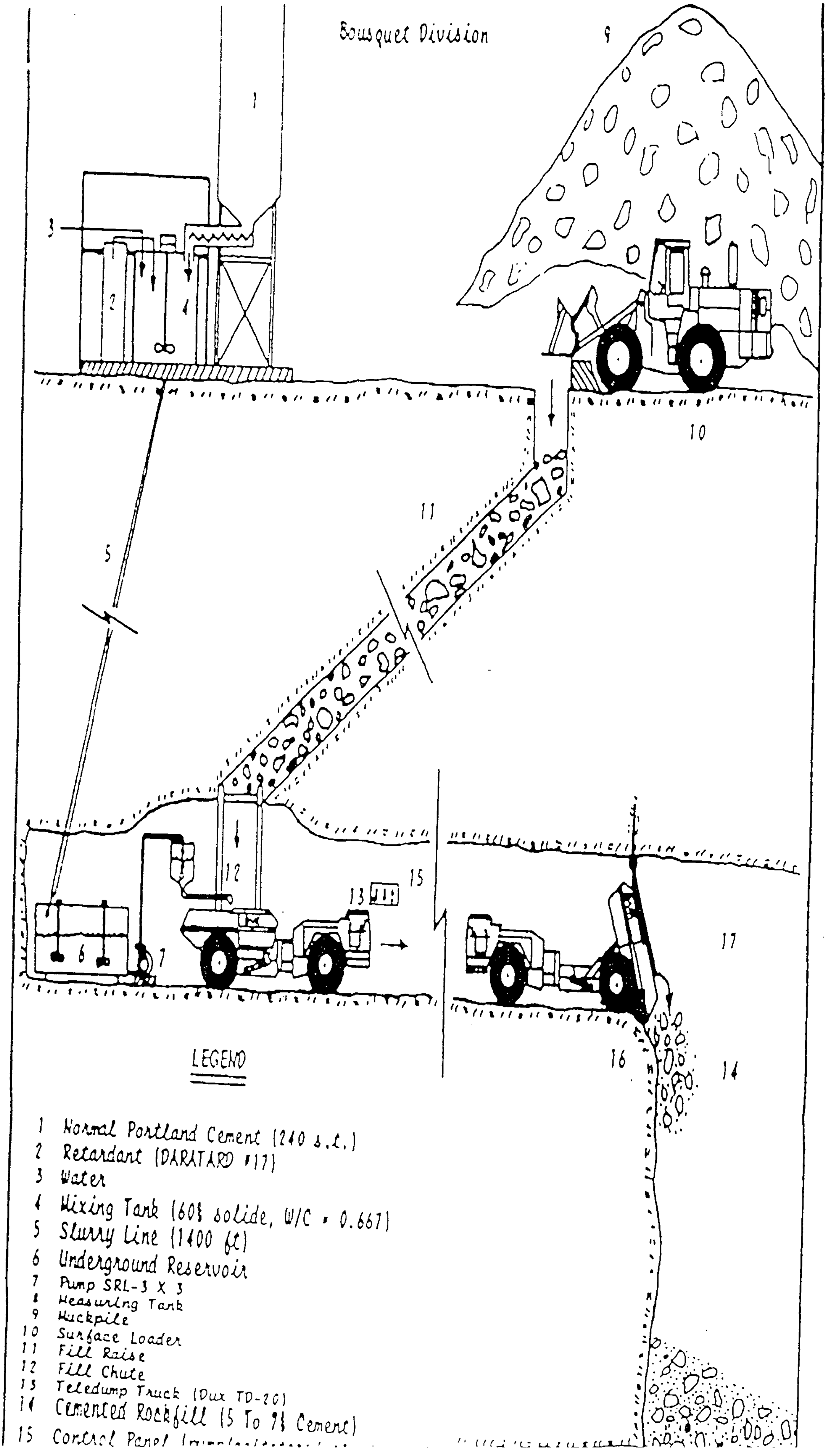
Surface (1) and underground (1) operators may communicate during the shift with the mine phone system. This is particularly important for the washing procedure.

The backfill station and the stopes are ventilated with electrical fans (60 cm ∅, 40 hp) which permit a secondary ventilation (9.5 m^3/s). Concrete bumper and dust safety control curtain are used at the dumping area. Special sumps were installed to eliminate the cement from infiltrating the mine drainage circuit.

The Bousquet backfilling system is actually in its starting phase. Both surface and underground operators are gaining confidence in the system and with time, the forecasted production rate will be obtained.

Claude Bouchard
Mine Engineer

CB/dt


Bousquet Division
1
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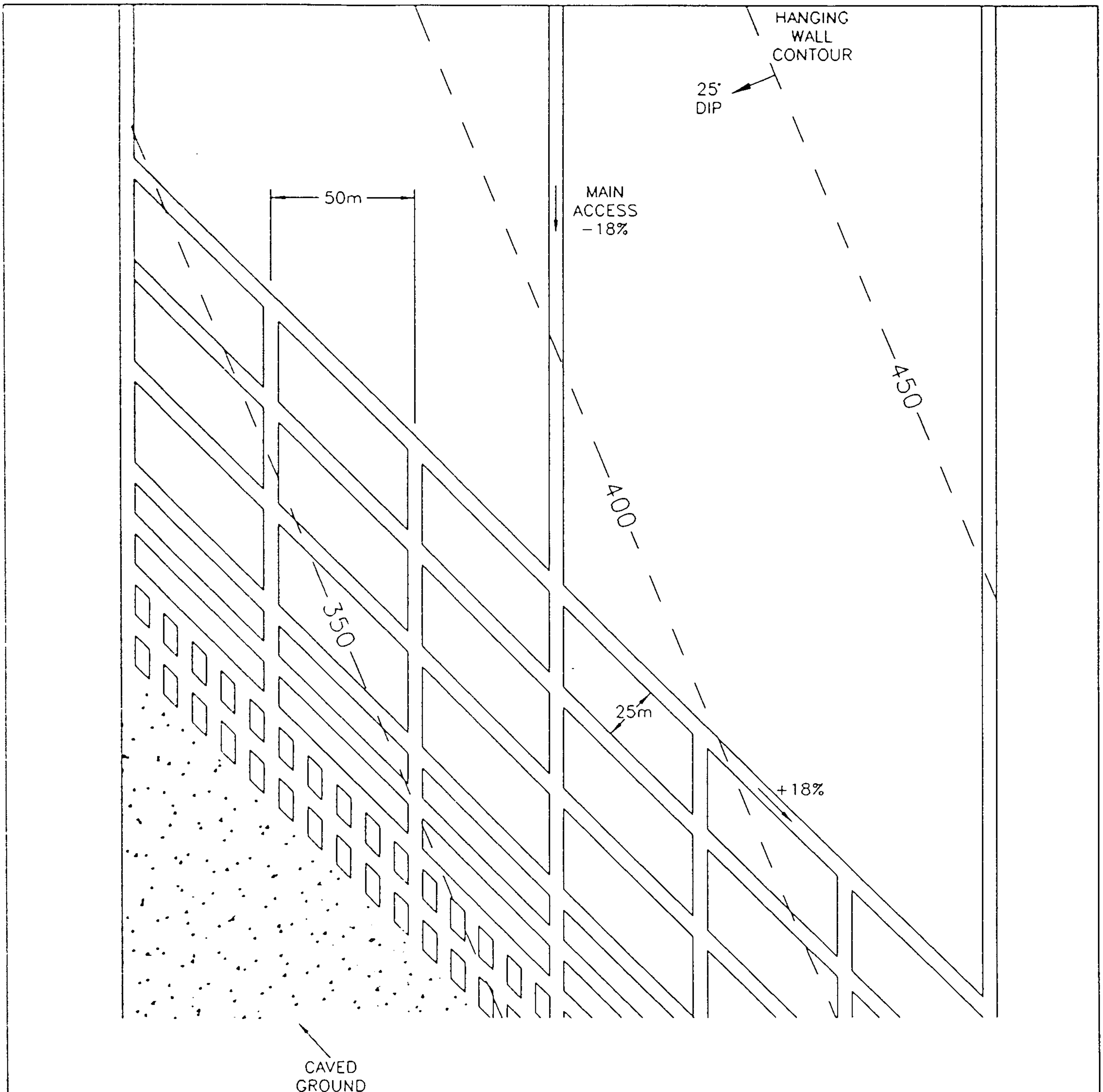
17
LEGEND
1 Normal Portland Cement (240 s.t.)
2 Retardant (DARATARD #17)
3 Water
4 Mixing Tank (60% solide, W/C = 0.667)
5 Slurry Line (1400 ft)
6 Underground Reservoir
7 Pump SRL-3 X 3
8 Measuring Tank
9 Muckpile
10 Surface Loader
11 Fill Raise
12 Fill Chute
13 Teledump Truck (Dux TD-20)
14 Cemented Rockfill (5 To 9% Cement)
15 Control Panel



GRIZZLY PROJECT

PRE-FEASIBILITY STUDY

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

Anvil Range
MINING CORPORATION

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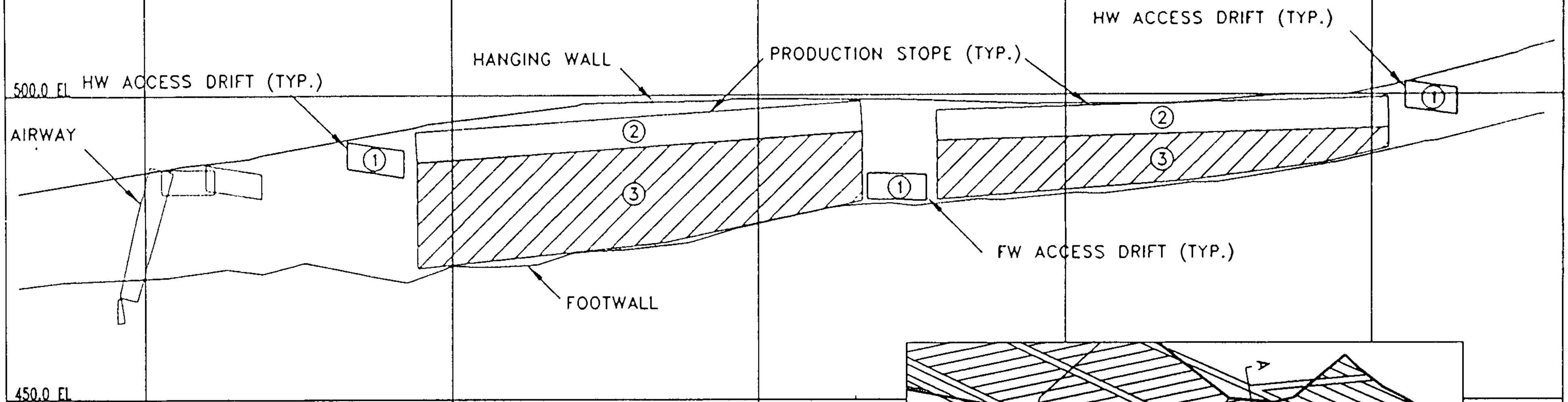
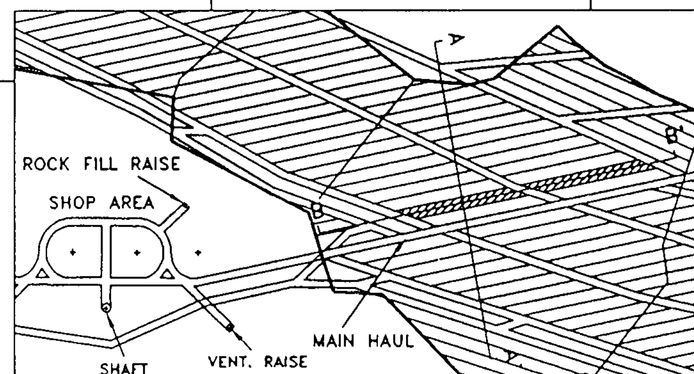
DRAWING. 7301MOD.DWG	CHECKED	DATE 96/11



LEGEND

1. DRIFT DEVELOPMENT
2. STOPE DEVELOPMENT (DRIFT & SLASH)
3. STOPE PRODUCTION (BENCHING)
4. CEMENTED ROCKFILL OF AREAS 2 & 3



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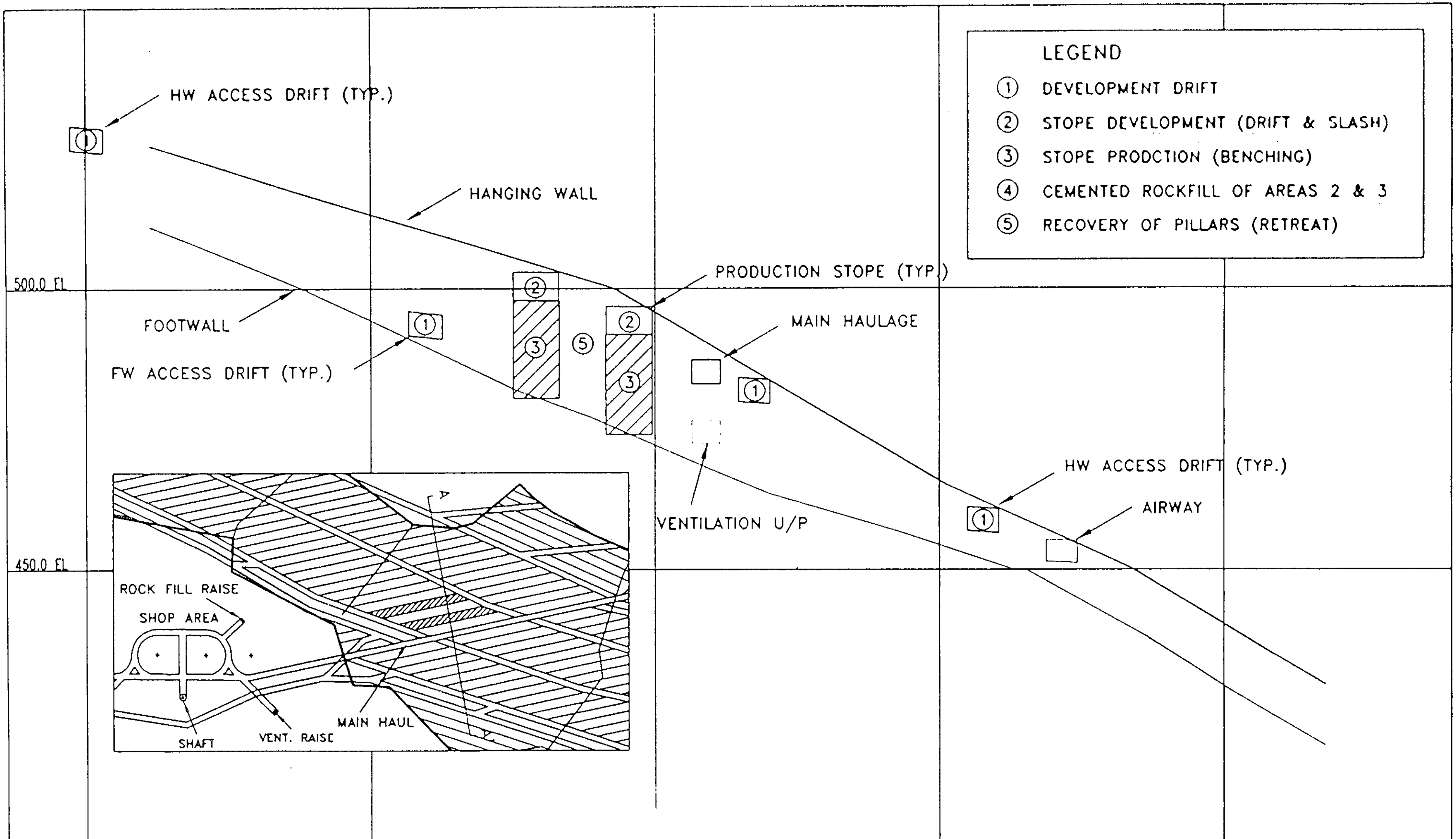
DATE 96/11

GRIZZLY PROJECT

PRE-FEASIBILITY STUDY

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

Anvil Range MINING CORPORATION



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APPROVED

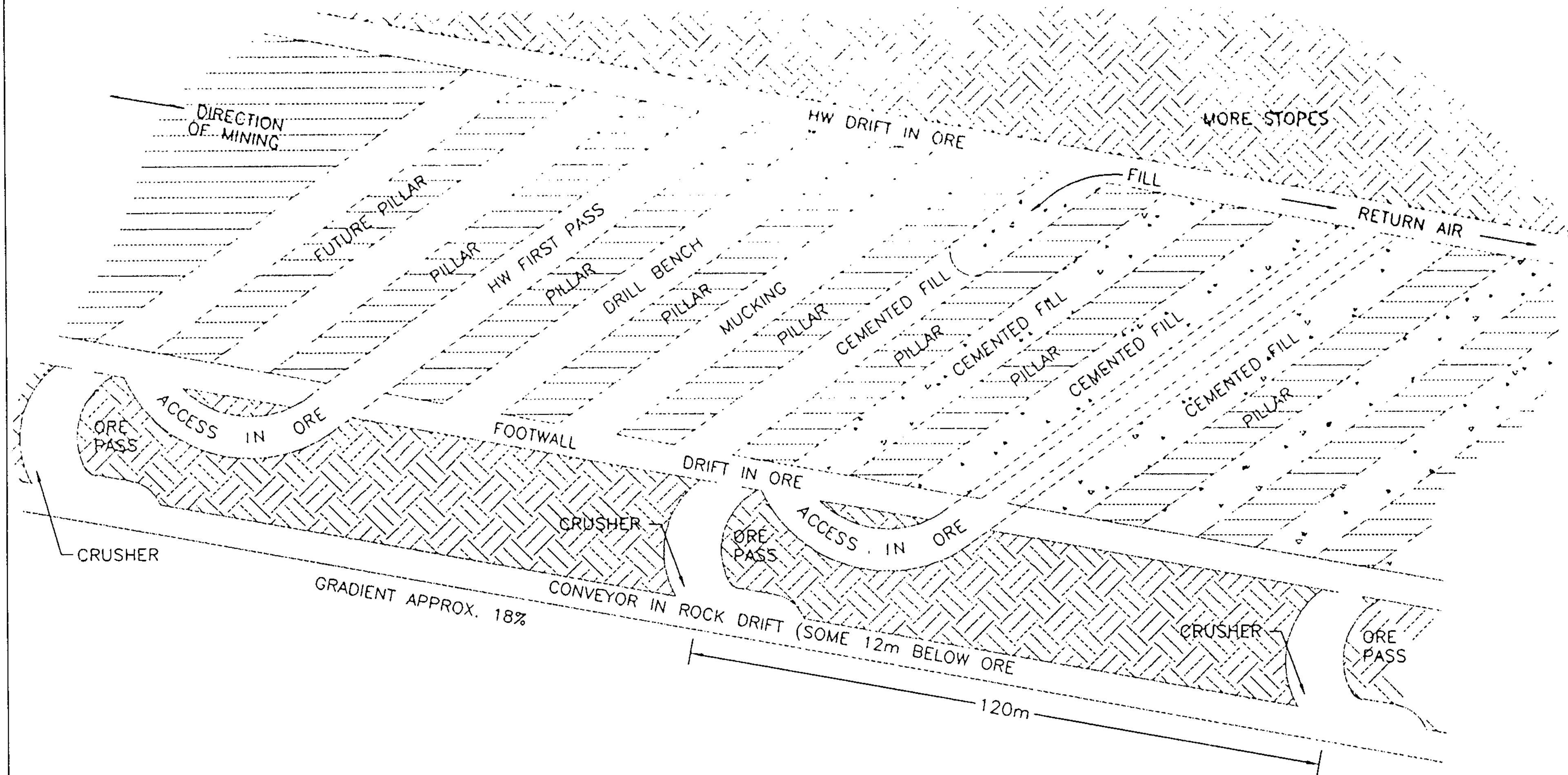
DATE 96/11

GRIZZLY PROJECT

PRE-FEASIBILITY STUDY

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

Anvil Range MINING CORPORATION



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DATE	96/11

GRIZZLY PROJECT

PRE-FEASIBILITY STUDY

Fig. 12–02 CASE A
Stoping In Thick Ore

Anvil Range MINING CORPORATION

APPENDIX H

PITEAU & ASSOCIATES: OREBODY - GEOTECHNICAL CONSIDERATIONS

7. OREBODY - GEOTECHNICAL CONSIDERATIONS

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

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

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

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

7.1 PROPOSED DECLINE FOR EXPLORATION AND UNDERGROUND ACCESS

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

7.1.1 Geomechanical Core Logging Data

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

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

Table 7.1
Summary of Weighted Average Geomechanical
Core Logging Data From Drillholes Along Decline Azimuth

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

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

7.1.2 Estimation of Support Requirements for Proposed Decline

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

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

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

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

7.1.3 Assessment of Kinematically Possible Wedges Based on Surface Mapping

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

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

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

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

7.2 GEOTECHNICAL CONSIDERATIONS FOR UNDERGROUND MINING

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

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

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

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

7.2.1 In-situ Stresses

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

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

7.2.2 Rock Competency and Rock Mass Strength

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

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

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

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

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

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

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

7.3 SELECTION OF A MINING METHOD

The depth and intermediate dip of the Grizzly deposit ultimately pose the greatest challenge in choosing a suitable mining method. Conventional room and pillar at Grizzly is possible, but extraction will be limited due to its depth. Table 7.4 illustrates the maximum theoretical recoveries for varying safety factors in different portions of the orebody (see SRK report, Appendix 7).

Table 7.4
Maximum Theoretical Recoveries
Standard Room and Pillar

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

As indicated from Table 7.4, theoretical recoveries (based on tributary area theory) indicate that only approximately half of the ore is recoverable using standard room and pillar mining, as was conducted at the Faro underground.

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

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

where:

K = DRMS (MPa)

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

H = pillar height

Pillar stress is calculated by:

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

where:

σ_v = vertical stress field

σ_h = horizontal stress field

α = dip of ore (degrees)

e = extraction ratio

The Factor of Safety equals:

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

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

7.3.1 Proposed Mining of Thin Ore

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

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

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

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

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

Secondary extraction on retreat is to consist of mining 5m wide breakthroughs on 10m centrelines between the 7.5m spaced drifts, leaving 7.5m long by 5m wide pillars, thus arriving at 70% extraction. These pillars would act as post pillars, with safety factors of 0.4 to 0.7, and would be designed to fail or crush as the mining front retreats. Secondary extraction areas would require mucking by remote controlled machines.

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

7.3.2 Proposed Mining of Thick Ore

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

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

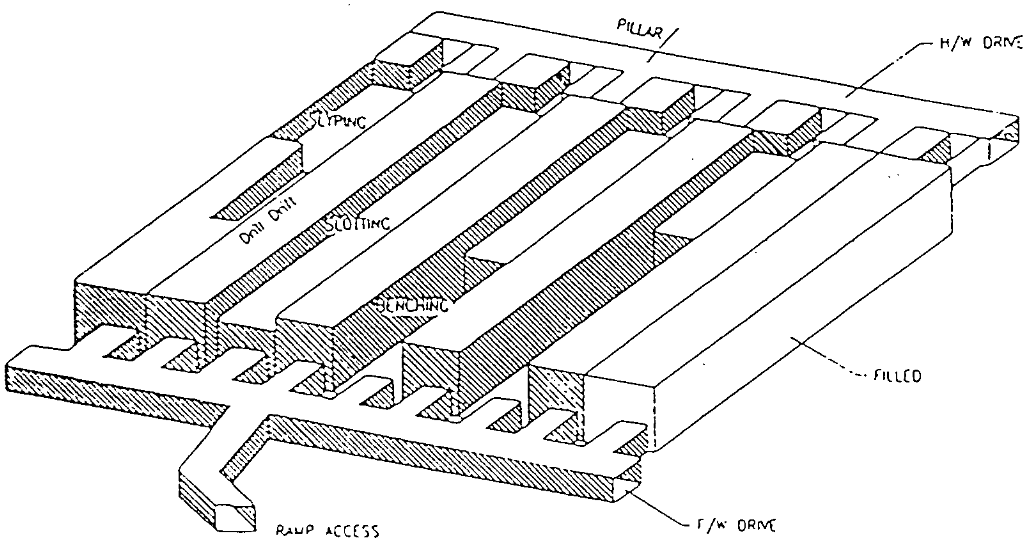


Figure 7.3-02 - Concrete Pillar Mining

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

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

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

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

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

7.4 SHAFT ACCESS AND SHAFT PILLAR DESIGN

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

7.4.1 Shaft Location and Pillar - Case B Mine Plan

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

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

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

7.4.2 Shaft Location and Pillar - Case A Mine Plan

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

Consideration was also given to the fact that the majority of ore blocks in the vicinity of the shaft pillar would be in thick ore and that backfilling of these areas would occur.

APPENDIX I

DETAILS OF ECONOMIC ANALYSIS

Summary Of Economics

23-Jan-97

Case : Grizzly Development – 1,500,000 Tonnes Annually

	---Before Tax---		---After Tax---	
	Cashflow	NPV@10%	Cashflow	NPV@10%
1. Base Case	$230,984	$44,616	$172,643	$27,988
2 Sensitivity Analysis				
Zinc +$.01	$16,619	$5,636	$12,126	$4,208
Lead +$.01	$15,655	$5,076	$11,424	$3,760
Silver +$.50	$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 +$1	($2,032)	($689)	($1,475)	($516)
Lead TC +$10	($1,424)	($462)	($1,034)	($343)

Grizzly Mining Plan

Cashflows After Tax - US$0.55 /lb Zinc & $ 0.32 /lb Lead
23-Jan-97 **Case : Grizzly Development - 1,500,000 Tonnes Annually**
Cdn Dollar

			1997	1998	1999	2000	2001	2002	2003	2004	2005	2006	2007	2012	2017	All Years
A. Prices (US)																
	Zn	/lb	$0.55	$0.55	$0.55	$0.55	$0.55	$0.55	$0.55	$0.55	$0.55	$0.55	$0.55	$0.55	$0.55	$0.55
	Pb	/lb	$0.32	$0.32	$0.32	$0.32	$0.32	$0.32	$0.32	$0.32	$0.32	$0.32	$0.32	$0.32	$0.32	$0.32
	Cu	/lb	$1.15	$1.15	$1.15	$1.15	$1.15	$1.15	$1.15	$1.15	$1.15	$1.15	$1.15	$1.15	$1.15	$1.15
	Ag	/oz	$5.50	$5.50	$5.50	$5.50	$5.50	$5.50	$5.50	$5.50	$5.50	$5.50	$5.50	$5.50	$5.50	$5.50
	Au	/oz	$400.00	$400.00	$400.00	$400.00	$400.00	$400.00	$400.00	$400.00	$400.00	$400.00	$400.00	$400.00	$400.00	$400.00
Exchange Rate		US/Cdn	0.740	0.740	0.740	0.740	0.740	0.740	0.740	0.740	0.740	0.740	0.740	0.740	0.740	0.74
B. Smelter Revenue		(Cdn 000)						$77,982	$179,125	$228,002	$203,815	$200,504	$200,570	$173,646		$2,269,658
Less Treatment Charges								($27,920)	($62,808)	($80,452)	($70,156)	($69,197)	($69,364)	($57,941)		($781,719)
Less Transportation								($12,408)	($26,726)	($31,634)	($28,866)	($28,819)	($29,012)	($25,182)		($329,991)
Less Operating								($28,022)	($59,537)	($68,206)	($68,206)	($68,206)	($68,206)	($68,206)		($799,027)
C. Operating Cash Flow								$9,632	$30,053	$47,709	$36,587	$34,282	$33,987	$22,317		$358,921
Less Capital			($8,800)	($15,400)	($2,200)	($17,178)	($30,302)	($21,219)						($2,000)		($106,099)
Less Working Capital								($7,950)	($9,076)	($5,242)	$2,724	$512	$46	($288)		
Less Acquisition Cost																
Less Interest																
Less Financing																
Less Income Taxes												($9,299)	($9,478)	($6,028)		($64,018)
Less Yukon Royaltieses									($1,238)	($3,032)	($1,829)	($1,611)	($1,583)	($721)		($16,160)
D. Net Cashflow After Income Tax			($8,800)	($15,400)	($2,200)	($17,178)	($30,302)	($19,537)	$19,740	$39,435	$37,482	$23,885	$22,971	$13,279		$172,643
E. Cumulative Cashflow			($8,800)	($24,200)	($26,400)	($43,578)	($73,880)	($93,417)	($73,677)	($34,243)	$3,239	$27,124	$50,095	$138,356	$172,643	
Payback	3 years															
Present Values		10.00%	($8,000)	($20,727)	($22,380)	($34,113)	($52,928)	($63,956)	($53,827)	($35,430)	($19,534)	($10,326)	($2,274)	$21,647	$27,988	
		12.00%	($7,857)	($20,134)	($21,700)	($32,617)	($49,811)	($59,709)	($50,780)	($34,853)	($21,336)	($13,646)	($7,043)	$11,681	$16,293	
		15.00%	($7,652)	($19,297)	($20,743)	($30,565)	($45,631)	($54,077)	($46,656)	($33,765)	($23,110)	($17,206)	($12,269)	$817	$3,708	
IRR at	16.18%															

Summary Page - US$0.55 /lb Zinc & $ 0.32 /lb Lead
23-Jan-97 Case : Grizzly Development - 1,500,000 Tonnes Annually
US Dollars

				1997	1998	1999	2000	2001	2002	2003	2004	2005	2006	2007	2012	2017	All Years
A.	Prices																
		Zn	/lb	$0.55	$0.55	$0.55	$0.55	$0.55	$0.55	$0.55	$0.55	$0.55	$0.55	$0.55	$0.55	$0.55	$0.55
		Pb	/lb	$0.32	$0.32	$0.32	$0.32	$0.32	$0.32	$0.32	$0.32	$0.32	$0.32	$0.32	$0.32	$0.32	$0.32
		Cu	/lb	$1.15	$1.15	$1.15	$1.15	$1.15	$1.15	$1.15	$1.15	$1.15	$1.15	$1.15	$1.15	$1.15	$1.15
		Ag	/oz	$5.50	$5.50	$5.50	$5.50	$5.50	$5.50	$5.50	$5.50	$5.50	$5.50	$5.50	$5.50	$5.50	$5.50
		Au	/oz	$400.00	$400.00	$400.00	$400.00	$400.00	$400.00	$400.00	$400.00	$400.00	$400.00	$400.00	$400.00	$400.00	$400.00
	Exchange Rate		US/Cdn	0.740	0.740	0.740	0.740	0.740	0.740	0.740	0.740	0.740	0.740	0.740	0.740	0.740	$0.74
B.	Annual Ore Production (000 t)								500	1,200	1,500	1,500	1,500	1,500	1,500		17,240
	Ore Grades	Zn							8.71%	8.16%	8.41%	7.06%	6.35%	5.97%	5.52%		6.30%
		Pb							4.87%	4.57%	4.62%	4.40%	5.13%	5.66%	4.03%		4.86%
		Cu															
		Ag	g/t						68.3	72.9	69.7	71.3	75.4	75.9	73.3		72.6
		Au	g/t						0.4	0.6	0.6	0.8	0.7	0.8	0.9		0.8
C.	Total Concentrate		(000 t)						101.4	228.1	291.9	255.9	255.3	257.8	212.8		2,876
	Zinc		(000 t)						66.9	150.4	193.7	162.4	146.3	137.5	127.2		1,691
	Lead		(000 t)						34.5	77.7	98.2	93.5	109.0	120.3	85.6		1,185
	Copper		(000 t)														
D.	Contained Metal																
	Zinc		(000 t)						37.5	84.2	108.5	90.9	81.9	77.0	71.2		947
	Lead		(000 t)						20.7	46.6	58.9	56.1	65.4	72.2	51.4		711
	Copper		(000 t)														
	Silver		(000 oz)						976	2,500	2,987	3,056	3,232	3,253	3,142		35,777
	Gold		(oz)						4,056	13297	16,324	22260	21,667	23,448	25228		257,220
E.	Payable Metal																
	Zinc		(000 lbs)						70,184	157,804	203,299	170,423	153,501	144,315	133,437		1,774,382
	Lead		(000 lbs)						43,348	97,627	123,369	117,494	136,988	151,140	107,614		1,489,338
	Copper		(000 lbs)														
	Silver		(000 oz)						823	2,107	2,519	2,576	2,725	2,743	2,649		30,162
	Gold in Base Conc		(oz)						2,288	8,639	10515	14915	14517	15,710	16913		171,742
	In Gold Conc		(oz)														
F.	Smelter Revenue		(000)						$57,707	$132,552	$168,721	$150,823	$148,373	$148,422	$128,498		$1,679,547
	Less Treatment Charges								($20,661)	($46,478)	($59535)	($51915)	($51205)	($51,330)	($42876)		($578,472)
	Less Transportation								($9,182)	($19,777)	($23,410)	($21,361)	($21,326)	($21,469)	($18,635)		($244,193)
	Less Operating								($20,736)	($44,058)	($50,472)	($50,472)	($50,472)	($50,472)	($50,472)		($591,280)
	Less Capital			($6,512)	($11,396)	($1,628)	($12,712)	($22,424)	($15,702)						($1,480)		($78,514)
	Less Yukon Royalties									($1,238)	($3,032)	($1,829)	($1,611)	($1,583)	($721)		($16,160)
	Less Working Capital Change								($5,883)	($6,716)	($3,879)	$2,016	$379	$34	($213)		
G.	Cashflow before Income Tax		(000)	($6,512)	($11,396)	($1,628)	($12,712)	($22,424)	($14,457)	$14,286	$28,393	$27,261	$24,137	$23,601	$14,100		$170,928
H.	Cumulative			($6,512)	($17,908)	($19,536)	($32,248)	($54,672)	($69,129)	($54,843)	($26,450)	$811	$24,948	$48,549	$140,564	$170,928	$170,928
	Payback	3 years															
	NPV @		10.00%	($5,920)	($15,338)	($16,561)	($25,244)	($39,167)	($47,328)	($39,997)	($26,751)	($15,190)	($5,884)	$2,388	$27,340	$33,016	$33,016
			12.00%	($5,814)	($14,899)	($16,058)	($24,136)	($36,860)	($44,185)	($37,723)	($26,255)	($16,424)	($8,653)	($1,868)	$17,665	$21,800	$21,800
			15.00%	($5,663)	($14,280)	($15,350)	($22,618)	($33,767)	($40,017)	($34,646)	($25,365)	($17,615)	($11,649)	($6,576)	$7,077	$9,677	$9,677
	IRR bt		18.71%														
	Site value	/t ore							$55.73	$55.25	$57.18	$51.70	$50.56	$50.42	$44.66		$49.70
	Op Cost	/t ore							($41.47)	($36.71)	($33.65)	($33.65)	($33.65)	($33.65)	($33.65)		($34.30)
	Return	/t ore							$14.26	$18.53	$23.54	$18.06	$16.91	$16.77	$11.01		$15.41

Financing Plan BT - US$0.55 /lb Zinc & $ 0.32 /lb Lead

23-Jan-97 Case : Grizzly Development - 1,500,000 Tonnes Annually

Cdn Dollar

				1997	1998	1999	2000	2001	2002	2003	2004	2005	2006	2007	2012	2017	All Years
A.	Prices (US)																
		Zn	/lb	$0.55	$0.55	$0.55	$0.55	$0.55	$0.55	$0.55	$0.55	$0.55	$0.55	$0.55	$0.55	$0.55	$0.55
		Pb	/lb	$0.32	$0.32	$0.32	$0.32	$0.32	$0.32	$0.32	$0.32	$0.32	$0.32	$0.32	$0.32	$0.32	$0.32
		Cu	/lb	$1.15	$1.15	$1.15	$1.15	$1.15	$1.15	$1.15	$1.15	$1.15	$1.15	$1.15	$1.15	$1.15	$1.15
	Exchange Rate		US/Cdn	0.74	0.74	0.74	0.74	0.74	0.74	0.74	0.74	0.74	0.74	0.74	0.74	0.74	0.74
B.	Cashflow		(Cdn 000)						$9,632	$28,381	$43,611	$34,115	$32,106	$31,848	$21,342		$337,083
	Net Smelter Revenue																
	Base Metals								$70,631	$158,792	$203,598	$176,603	$172,406	$171,693	$144,817		$1,952,646
	Precious Metals								$7,351	$20,333	$24,404	$27,211	$28,098	$28,877	$28,829		$317,012
	Less Treatment Charges								($27,920)	($62,808)	($80,452)	($70,156)	($69,197)	($69,364)	($57,941)		($781,719)
	Less Transportation								($12,408)	($26,726)	($31,634)	($28,866)	($28,819)	($29,012)	($25,182)		($329,901)
	Less Operating								($28,022)	($59,537)	($68,206)	($68,206)	($68,206)	($68,206)	($68,206)		($799,027)
	Less Yukon Royalties									($1,672)	($4,098)	($2,472)	($2,177)	($2,140)	($975)		($21,838)
C.	Capital Requirement		(Cdn 000)	$8,800	$15,400	$2,200	$17,178	$30,302	$29,169	$9,076	$5,242	($2,724)	($512)	($46)	$2,288		$106,099
	Working Capital								$7,950	$9,076	$5,242	($2,724)	($512)	($46)	$288		
	Capital			$8,800	$15,400	$2,200	$17,178	$30,302	$21,219						$2,000		$106,099
		CCA					$3,265	$5,527	$8,308						$2,000		$28,100
		CDE		$8,800	$15,400	$2,200	$13,913	$24,776	$12,911								$78,000
		CEE															
D.	Cash Required		(Cdn 000)	($8,800)	($15,400)	($2,200)	($17,178)	($30,302)	($19,537)	$19,305	$38,369	$36,839	$32,617	$31,893	$19,054		$230,984
	Op. Contribution								$9,632	$28,381	$43,611	$34,115	$32,106	$31,848	$21,342		$337,083
	Capital			($8,800)	($15,400)	($2,200)	($17,178)	($30,302)	($29,169)	($9,076)	($5,242)	$2,724	$512	$46	($2,288)		($106,099)
	Acquisition																
E.	**Base Metal Operations**		(Cdn 000)														
	Concentrate Loan @																
	Corporate Loan																
	Development Loan																
	Mine Equipment Loan																
	Transport Equipment																
	Interest																
	Interest Breakdown																
	Concentrate Int @																
	Corporate Int @																
	Development Int @																
	Mine Equip Int @																
	Transport Equip Int @																
F.	**Net Cashflow**		(Cdn 000)	($8,800)	($15,400)	($2,200)	($17,178)	($30,302)	($19,537)	$19,305	$38,369	$36,839	$32,617	$31,893	$19,054		$230,984
	Cumulative			($8,800)	($24,200)	($26,400)	($43,578)	($73,880)	($93,417)	($74,112)	($35,743)	$1,096	$33,714	$65,607	$189,951	$230,984	
	Payback	3 years															
	NPV @		10.00%	($8,000)	($20,727)	($22,380)	($34,113)	($52,928)	($63,956)	($54,050)	($36,150)	($20,527)	($7,952)	$3,227	$36,946	$44,616	
			12.00%	($7,857)	($20,134)	($21,700)	($32,617)	($49,811)	($59,709)	($50,976)	($35,480)	($22,195)	($11,693)	($2,525)	$23,872	$29,460	
			15.00%	($7,652)	($19,297)	($20,743)	($30,565)	($45,631)	($54,077)	($46,819)	($34,276)	($23,804)	($15,742)	($8,887)	$9,564	$13,077	
	IRR bt	18.71%															

Statistics - US$0.56 /lb Zinc & $ 0.32 /lb Lead

23-Jan-97 Case : Grizzly Development - 1,500,000 Tonnes Annually

US Dollar

			1997	1998	1999	2000	2001	2002	2003	2004	2005	2006	2007	2012	2017	All Years
1. Financial																
Revenue		(Cdn 000)						$37,664	$89,591	$115,915	$104,793	$102,488	$102,193	$90,523		$1,157,948
Cashflow before Tax		(Cdn 000)	($8,800)	($15,400)	($2,200)	($17,178)	($30,302)	($19,537)	$19,305	$38,369	$36,839	$32,617	$31,893	$19,064		$230,984
Cumulative cashflow bt		(Cdn 000)	($8,800)	($24,200)	($26,400)	($43,578)	($73,880)	($93,417)	($74,112)	($35,743)	$1,096	$33,714	$65,607	$169,647	$198,962	
PVbt		10.00%	($8,000)	($20,727)	($22,380)	($34,113)	($52,928)	($63,956)	($54,050)	($36,150)	($20,527)	($7,952)	$3,227	$36,946	$44,616	
		12.00%	($7,857)	($20,134)	($21,700)	($32,617)	($49,811)	($59,709)	($50,976)	($35,480)	($22,195)	($11,693)	($2,525)	$23,872	$29,460	
		15.00%	($7,652)	($19,297)	($20,743)	($30,565)	($45,631)	($54,077)	($46,819)	($34,276)	($23,804)	($15,742)	($8,887)	$9,564	$13,077	
2. Values																
Site Net Back		US/t						$55.73	$55.25	$57.18	$51.70	$50.56	$50.42	$44.66		$49.70
Operating Costs		US/t						($41.47)	($36.71)	($33.65)	($33.65)	($33.65)	($33.65)	($33.65)		($34.30)
Ore Value		US/t						$14.26	$18.53	$23.54	$18.05	$16.91	$16.77	$11.01		$15.41
Concentrate Value		US/t						$274.86	$290.69	$293.85	$303.03	$297.08	$293.34	$314.80		$297.94
Contained Metal in Ore		(000 t)						500	1,200	1,500	1,500	1,500	1,500	1,500		17,240
Ore Grade	- Zn	(000 t)						44	98	126	106	95	90	83		1,101
	- Pb	(000 t)						24	55	69	66	77	85	60		837
	- Cu	(000 t)														
	- Ag	(000 oz)						1,098	2,813	3,361	3,439	3,636	3,660	3,535		40,256
	- Au	(000 oz)						7	22	27	36	35	38	41		418
Paid	- Zn							73.10%	73.10%	73.10%	73.10%	73.10%	73.10%	73.10%		73.10%
	- Pb							80.75%	80.75%	80.75%	80.75%	80.75%	80.75%	80.75%		80.75%
	- Cu															
	- Ag							74.93%	74.93%	74.93%	74.93%	74.93%	74.93%	74.93%		74.93%
	- Au							34.72%	39.98%	39.64%	41.24%	41.24%	41.24%	41.26%		41.09%
C. Simple Cost Comparison																
1. Byproducts																
Revenue		(US 000)						$19,106	$45,760	$56,907	$57,090	$63,947	$69,048	$55,108		$703,637
Treatment Charges		(US 000)						($6,184)	($13,927)	($17,599)	($16,761)	($19,542)	($21,561)	($15,352)		($212,461)
Transportation		(US 000)						($3,124)	($6,737)	($7,873)	($7,805)	($9,107)	($10,016)	($7,499)		($100,649)
Net Revenue		(US 000)						$9,798	$25,096	$31,435	$32,525	$35,299	$37,471	$32,257		$390,526
2. Costs to Zinc																
Treatment Charges		(US 000)						$14,477	$32,551	$41,935	$35,154	$31,664	$29,769	$27,525		$366,011
Transportation		(US 000)						$6,058	$13,040	$15,536	$13,556	$12,220	$11,453	$11,135		$143,544
Operating Costs		(US 000)						$20,736	$44,058	$50,472	$50,472	$50,472	$50,472	$50,472		$591,280
Royalties		(US 000)							$1,238	$3,032	$1,829	$1,611	$1,583	$721		$16,160
Capital		(US 000)	$6,512	$11,396	$1,628	$12,712	$22,424	$15,702						$1,480		$78,514
3. Payable Zinc		(000 lbs)						70,184	157,804	203,299	170,423	153,501	144,315	133,437		1,774,382
4. Payable Zinc Cost		US/lb zn						$0.448	$0.417	$0.391	$0.402	$0.395	$0.387	$0.432		$0.409
Treatment Charges		US/lb zn						$0.206	$0.206	$0.206	$0.206	$0.206	$0.206	$0.206		$0.206
Transportation		US/lb zn						$0.086	$0.083	$0.076	$0.080	$0.080	$0.079	$0.083		$0.081
Operating Costs		US/lb zn						$0.295	$0.279	$0.248	$0.296	$0.329	$0.350	$0.378		$0.333
Royalties		US/lb zn							$0.008	$0.015	$0.011	$0.010	$0.011	$0.005		$0.009
By Product Credit		US/lb zn						($0.140)	($0.159)	($0.155)	($0.191)	($0.230)	($0.260)	($0.242)		($0.220)
Capital		US/lb zn						$0.224						$0.011		$0.044
5. Total Cost		US/lb zn						$0.672	$0.417	$0.391	$0.402	$0.395	$0.387	$0.443		$0.454

Yukon Royalties - US$0.55 /lb Zinc & $ 0.32 /lb Lead

23-Jan-97 Case : Grizzly Development - 1,500,000 Tonnes Annually

Cdn Dollars

	1997	1998	1999	2000	2001	2002	2003	2004	2005	2006	2007	2012	2017	All Years
A. Yukon Royalties Payable	No	No	No	No	No	Yes	Yes	Yes	Yes	Yes	Yes	Yes	No	
B. Yukon Royalties														
Minesite Revenue						$37,654	$89,591	$115,915	$104,793	$102,488	$102,193	$90,523		$1,157,948
Less Operating Costs						($28,022)	($59,537)	($68,206)	($68,206)	($68,206)	($68,206)	($68,206)		($799,027)
Operating Profit						$9,632	$30,063	$47,709	$36,587	$34,282	$33,987	$22,317		$358,921
Taxable Operating Profit						$9,632	$30,063	$47,709	$36,587	$34,282	$33,987	$22,317		$358,921
Less Depreciation						(2,552)	(2,552)	(2,552)	(2,552)	(2,552)	(2,552)	(1,493)		($24,622)
Preproduction Allowance	($8,800)	($15,400)	($2,200)	($13,913)	($24,776)	($12,911)								($78,000)
Exploration														
Net Profit before PA	($8,800)	($15,400)	($2,200)	($13,913)	($24,776)	($5,831)	$27,501	$45,157	$34,035	$31,730	$31,435	$20,824		$256,299
Processing Allowance							(8,000)	(8,000)	(8,000)	(8,000)	(8,000)	(8,000)		($88,712)
Taxable Profit	(8,800)	(15,400)	(2,200)	(13,913)	(24,776)	(5,831)	19,501	37,157	26,035	23,730	23,435	12,824		$167,587
Rate of Taxation							6.35%	8.16%	7.03%	6.79%	6.76%	5.62%		9.64%
Total Tax Payable							$1,238	$3,032	$1,829	$1,611	$1,583	$721		$16,160
C. Predevelopment	8,800	15,400	2,200	13,913	24,776	12,911								$78,000
D. Exploration and Development														
E. Processing Allowance														
Initial Mill Processing Base	$100,000	$100,000	$100,000	$100,000	$100,000	$100,000	$100,000	$100,000	$100,000	$100,000	$100,000	$100,000		$1,900,000
Additions														
Final Processing Base	$100,000	$100,000	$100,000	$100,000	$100,000	$100,000	$100,000	$100,000	$100,000	$100,000	$100,000	$100,000	$108,000	$2,116,000
Processing Rate	8.00%	8.00%	8.00%	8.00%	8.00%	8.00%	8.00%	8.00%	8.00%	8.00%	8.00%	8.00%	$14,925	$29,852
Allowance	$8,000	$8,000	$8,000	$8,000	$8,000	$8,000	$8,000	$8,000	$8,000	$8,000	$8,000	$8,000		$152,000

Yukon Royalty Deductions – US$0.55 /lb Zinc & $ 0.32 /lb Lead
23-Jan-97 Case : Grizzly Development – 1,500,000 Tonnes Annually
Cdn Dollars

	1997	1998	1999	2000	2001	2002	2003	2004	2005	2006	2007	2012	2017	All Years
A. Depreciation						($2,562)	($2,562)	($2,562)	($2,562)	($2,562)	($2,562)	($1,493)		($24,822)
• 1997 Additions														
Allowable														
Claimed														
• 1998 Additions														
Allowable														
Claimed														
• 1999 Additions														
Allowable														
Claimed														
• 2000 Additions				$3,265	$3,265	$3,265	$2,778	$2,290	$1,803	$1,316	$828			
Allowable				$487	$487	$487	$487	$487	$487	$487	$487			
Claimed						($487)	($487)	($487)	($487)	($487)	($487)			($3,265)
• 2001 Additions					$5,527	$5,527	$4,702	$3,877	$3,052	$2,227	$1,402			
Allowable					$825	$825	$825	$825	$825	$825	$825			
Claimed						($825)	($825)	($825)	($825)	($825)	($825)			($5,527)
• 2002 Additions						$8,308	$7,068	$5,828	$4,588	$3,348	$2,108			
Allowable						$1,240	$1,240	$1,240	$1,240	$1,240	$1,240			
Claimed						($1,240)	($1,240)	($1,240)	($1,240)	($1,240)	($1,240)			($8,308)
• 2003 Additions														
Allowable														
Claimed														
• 2004 Additions														
Allowable														
Claimed														
• 2005 Additions														
Allowable														
Claimed														
• 2006 Additions														
Allowable														
Claimed														
• 2007 Additions														
Allowable														
Claimed														
• 2008 Additions												$806		
Allowable												$299		
Claimed												($299)		($2,000)
• 2009 Additions												$1,104	$209	
Allowable												$299	$209	
Claimed												($299)		($1,791)
• 2010 Additions												$1,403	$507	
Allowable												$299	$299	
Claimed												($299)		($1,493)
• 2011 Additions												$1,701	$806	
Allowable												$299	$299	
Claimed												($299)		($1,194)
• 2012 Additions												$2,000	$1,104	
Allowable												$299	$299	
Claimed												($299)		($896)
• 2013 Additions														
Allowable														
Claimed														($149)
• 2017 Additions														
Allowable														
Claimed														

Income Tax Calculations – US$0.55 /lb Zinc & $ 0.32 /lb Lead

23-Jan-97 Case : Grizzly Development – 1,500,000 Tonnes Annually

	Cdn Dollars	1997	1998	1999	2000	2001	2002	2003	2004	2005	2006	2007	2012	2017	All Years
A. Operating Profit before Tax							$9,632	$28,381	$43,611	$34,115	$32,106	$31,848	$21,342		$337,083
CCA 41 (b)							($2,420)	($2,854)	($2,140)	($1,605)	($1,204)	($903)	($1,661)		($20,814)
CCA Other															
CCA 41 (a)							($3,265)								($3,265)
Resource Profits							$3,947	$25,527	$41,471	$32,510	$30,902	$30,945	$19,681		$313,004
Resource Allowance							($987)	($6,382)	($10,368)	($8,128)	($7,725)	($7,736)	($4,920)		($78,251)
Cdn Exploration Expense							($2,960)	($19,145)	($31,103)	($24,383)	($408)				($78,000)
Cdn Development Expense															
Interest Expense															
Net Income											$22,768	$23,208	$14,761		$156,753
Prior Year Losses															
Taxable Income											$22,768	$23,208	$14,761		$156,753
Federal Tax before ITC	28.84%										$6,566	$6,693	$4,257		$45,208
ITC															
Net Federal Income Tax											$6,566	$6,693	$4,257		$45,208
Provincial Income Tax	12.00%										$2,732	$2,785	$1,771		$18,810
Total Income Tax											$9,299	$9,478	$6,028		$64,018
B. Breakdown of Capital Expenditures															
1. Income Tax Categories		$8,800	$15,400	$2,200	$17,178	$30,302	$21,219						$2,000		$106,099
CCA41a					$3,265										$3,265
CCA41b						$5,527	$8,308						$2,000		$24,835
CCA-Other															
CEE		$8,800	$15,400	$2,200	$13,913	$24,776	$12,911								$78,000
CDE															

CCA, CCE, CDE, and Loss Carryforwards - US$0.55 /lb Zinc & $ 0.32 /lb Lead

23-Jan-97 Case Grizzly Development - 1,500,000 Tonnes Annually
Cdn Dollars

		1997	1998	1999	2000	2001	2002	2003	2004	2005	2006	2007	2012	2017	All Years
1. Opening UCC41b							2,763	7,261	8,561	6,421	4,816	3,612	4,642	4,021	
Additions						5,527	8,308						2,000		$24 835
Usable CCA Pool						2,763	9,681	11,415	8,561	6,421	4,816	3,612	6,642	4,021	
Available CCA	25.00%					691	2,420	2,854	2,140	1,605	1,204	903	1,661	1,005	
CCA claimed							(2,420)	(2,854)	(2,140)	(1,605)	(1,204)	(903)	(1,661)		($20,814)
Closing balance						2,763	7,261	8,561	6,421	4,816	3,612	2,709	4,982	4,021	
Accumulated CCA							(2,420)	(5,274)	(7,414)	(9,019)	(10,223)	(11,126)	(17,853)	(20,814)	
2. Opening balance CCA-other															
Additions															
Usable CCA Pool															
Available CCA															
CCA claimed															
Closing balance															
Accumulated CCA															
3. Opening balance CCA41a						1,632	3,265								
Additions					3,265										$3,265
Usable CCA Pool					1,632	3,265	3,265								
Available CCA	100.00%				1,632	3,265	3,265								
CCA claimed							(3,265)								($3,265)
Closing balance					1,632	3,265									
Accumulated CCA							(3,265)	(3,265)	(3,265)	(3,265)	(3,265)	(3,265)	(3,265)	(3,265)	
4. Opening balance CDE															
Additions to pool															
Usable CDE Pool															
Available CDE	30.00%														
CDE claimed															
Closing balance															
Accumulated CDE															
5. Opening Balance CEE			8,800	24,200	26,400	40,313	65,089	75,039	55,894	24,790	408				
Additions to Pool		8,800	15,400	2,200	13,913	24,776	12,911								$78,000
Usable CEE Pool		8,800	24,200	26,400	40,313	65,089	78,000	75,039	55,894	24,790	408				
Available CEE	100.00%	8,800	24,200	26,400	40,313	65,089	78,000	75,039	55,894	24,790	408				
CEE claimed							(2,960)	(19,145)	(31,103)	(24,383)	(408)				($78,000)
Closing balance		8,800	24,200	26,400	40,313	65,089	75,039	55,894	24,790	408					
Accumulated CEE							(2,960)	(22,106)	(53,209)	(77,592)	(78,000)	(78,000)	(78,000)	(78,000)	
6. Opening Balance Loss Carried Forward															
Additions to Pool															
Available LCF															
LCF claimed															
Expired Non Capital Losses															
Closing balance															
Accumulated LCF															

Note
Initial tax pools will have to be determined.

Production Page – US$0.55 /lb Zinc & $ 0.32 /lb Lead
23-Jan-97 Case : Grizzly Development – 1,500,000 Tonnes Annually

				1997	1998	1999	2000	2001	2002	2003	2004	2005	2006	2007	2012	2017	All Years
A.	Mining Rate (tpd)								1,370	3,288	4,110	4,110	4,110	4,110	4,110		3,633
	Working Days								365	365	365	365	365	365	365		4,745
	Ore Production (000)								500	1,200	1,500	1,500	1,500	1,500	1,500		17,240
	Ore Grade	–Zn							8.7%	8.2%	8.4%	7.1%	6.4%	6.0%	5.5%		6.4%
		–Pb							4.9%	4.6%	4.6%	4.4%	5.1%	5.7%	4.0%		4.9%
		–Cu															
		–Ag	g						68.3	72.9	69.7	71.3	75.4	75.9	73.3		72.6
		–Au	g						0.4	0.6	0.6	0.8	0.7	0.8	0.9		0.8
B.	Contained Metal																
		–Zn	(000 t)						44	98	126	106	95	90	83		1,101
		–Pb	(000 t)						24	55	69	66	77	85	60		837
		–Cu	(000 t)														
		–Ag	(000 oz)						1,098	2,813	3,361	3,439	3,636	3,660	3,535		40,256
		–Au	oz						6,591	21,605	26,524	36,170	35,205	38,099	40,992		417,940
C.	Recovery																
	Zn	Zn Cons							86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%		86.0%
		Pb Cons							4.0%	4.0%	4.0%	4.0%	4.0%	4.0%	4.0%		4.0%
		Cu Cons															
		Overall							90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%		90.0%
	Pb	Zn Cons							7.0%	7.0%	7.0%	7.0%	7.0%	7.0%	7.0%		7.0%
		Pb Cons							85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%		85.0%
		Cu Cons															
		Overall							92.0%	92.0%	92.0%	92.0%	92.0%	92.0%	92.0%		92.0%
	Cu	Zn Cons															
		Pb Cons															
		Cu Cons															
		Overall															
	Ag	Zn Cons							10.0%	10.0%	10.0%	10.0%	10.0%	10.0%	10.0%		10.0%
		Pb Cons							78.9%	78.9%	78.9%	78.9%	78.9%	78.9%	78.9%		78.9%
		Cu Cons															
		Overall							88.9%	88.9%	88.9%	88.9%	88.9%	88.9%	88.9%		88.9%
	Au	Zn Cons							10.0%	10.0%	10.0%	10.0%	10.0%	10.0%	10.0%		10.0%
		Pb Cons							51.5%	51.5%	51.5%	51.5%	51.5%	51.5%	51.5%		51.5%
		Cu Cons															
		Overall							61.5%	61.5%	61.5%	61.5%	61.5%	61.5%	61.5%		61.5%
D.	Tailings																
	Metal in tails	–Zn		100.0%	100.0%	100.0%	100.0%	100.0%	10.0%	10.0%	10.0%	10.0%	10.0%	10.0%	10.0%	100.0%	10.0%
		–Pb		100.0%	100.0%	100.0%	100.0%	100.0%	8.0%	8.0%	8.0%	8.0%	8.0%	8.0%	8.0%	100.0%	8.0%
		–Cu		100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%
		–Ag		100.0%	100.0%	100.0%	100.0%	100.0%	11.1%	11.1%	11.1%	11.1%	11.1%	11.1%	11.1%	100.0%	11.1%
		–Au		100.0%	100.0%	100.0%	100.0%	100.0%	38.5%	38.5%	38.5%	38.5%	38.5%	38.5%	38.5%	100.0%	38.5%

Concentrates - US$0.55 /lb Zinc & $ 0.32 /lb Lead
23-Jan-97 Case . Grizzly Development - 1,500,000 Tonnes Annually

			1997	1998	1999	2000	2001	2002	2003	2004	2005	2006	2007	2012	2017	All Years
1 Zn Concentrates		(000 t)						67	150	194	162	146	138	127		1,691
	Zn							56.0%	56.0%	56.0%	56.0%	56.0%	56.0%	56.0%		56.00%
	Pb							2.5%	2.6%	2.5%	2.8%	3.7%	4.3%	3.3%		3.40%
	Cu															
	Ag	g						51.06	58.17	53.97	65.86	77.32	82.79	86.47		74.05
	Au	g						0.31	0.45	0.43	0.69	0.75	0.86	1.00		0.77
Metal	Zn	(000 t)						37	84	108	91	82	77	71		947
	Pb	(000 t)						2	4	5	5	5	6	4		59
	Cu	(000 t)														
	Ag	(000 oz)						110	281	336	344	364	366	353		4,026
	Au	(000 oz)						0.7	2.2	2.7	3.6	3.5	3.8	4.1		42
2 Pb Concentrate		(000 t)						34	78	98	94	109	120	86		1,185
	Pb							60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%		60.00%
	Zn							5.0%	5.0%	5.1%	4.5%	3.5%	3.0%	3.9%		3.72%
	Cu															
	Ag	g						780.82	888.12	839.95	902.19	818.30	746.60	1012.65		833
	Au	g						3.06	4.46	4.33	6.20	5.18	5.08	7.67		5.65
Metal	Pb	(000 t)						21	47	59	56	65	72	51		711
	Zn	(000 t)						2	4	5	4	4	4	3		44
	Cu	(000 t)														
	Ag	(000 oz)						866	2,218	2,651	2,712	2,868	2,887	2,788		31,751
	Au	(000 oz)						3.4	11.1	13.7	18.6	18.1	19.6	21.1		215
3 Cu Concentrate		(000 t)														
	Cu															
	Zn															
	Pb															
	Ag	g														
	Au	g														
Metal	Cu	(000 t)														
	Zn	(000 t)														
	Pb	(000 t)														
	Ag	(000 oz)														
	Au	(000 oz)														
4 Tailings																
Metal in tails	-Zn	(000 t)						4	10	13	11	10	9	8		110
	-Pb	(000 t)						2	4	6	5	6	7	5		67
	-Cu	(000 t)														
	-Ag	(000 oz)						122	313	374	383	405	407	393		4,479
	-Au	oz						2,535	8,308	10,200	13,909	13,538	14,651	15,764		160,720
Value in tails	Total							$8,341	$20,013	$25,344	$24,215	$23,533	$23,750	$21,920		$269,644
	-Zn	(US 000)						$5,281	$11,873	$15,296	$12,823	$11,549	$10,858	$10,040		$133,503
	-Pb	(US 000)						$1,374	$3,095	$3,911	$3,725	$4,343	$4,792	$3,412		$47,216
	-Cu	(US 000)														
	-Ag	(US 000)						$672	$1,721	$2,057	$2,104	$2,225	$2,240	$2,163		$24,636
	-Au	(US 000)						$1,014	$3,323	$4,080	$5,564	$5,415	$5,860	$6,305		$64,288

Transportation - US$0.55 /lb Zinc & $ 0.32 /lb Lead

23-Jan-97 Case : Grizzly Development - 1,500,000 Tonnes Annually

US Dollars

			1997	1998	1999	2000	2001	2002	2003	2004	2005	2006	2007	2012	2017	All Years
A. Transportation																
Volume		(000 dmt)						101	228	292	256	255	258	213		2 876
Transport Cost		$/dmt						$90 57	$86 72	$80 20	$83.47	$83.54	$83 28	$87 57		$84 91
Total Cost		(000)						$9,182	$19,777	$23,410	$21,361	$21,326	$21,469	$18,635		$244,193
Concentrates	- Zn	(000)						$6,058	$13,040	$15,536	$13,556	$12,220	$11,453	$11,135		$143,544
	- Pb	(000)						$3,124	$6,737	$7,873	$7,805	$9,107	$10,016	$7,499		$100,649
	-Cu	(000)														
Trans Shipping		(000)														
B. Transportation Costs																
Loading Costs		$/wmt						$1 48	$1 37	$1 27	$1 32	$1 32	$1 32	$1 40		
Cost to port		$/wmt						$33 30	$30 17	$28.87	$29.53	$29.54	$29.49	$30 60		
Port Costs		$/wmt						$16 00	$15.70	$13.03	$14.37	$14 40	$14.29	$16.00		
Ocean Freight		$/wmt						$30 00	$29 97	$28.01	$29.00	$29 02	$28 94	$30 00		
IAS		$/wmt						$3 00	$3 00	$3 00	$3 00	$3 00	$3 00	$3 00		
2 Zinc Concentrates		(000 dmt)						67	150	194	162	146	138	127		1,691
Land Handling		/wmt						$1.48	$1 37	$1 27	$1 32	$1 32	$1 32	$1 40		$1 35
Land Transportation		/wmt						$33 30	$30.17	$28.87	$29.53	$29 54	$29 49	$30.60		$30 00
Port		/wmt						$16.00	$15.70	$13.03	$14.37	$14.40	$14.29	$16 00		$14 86
Ocean Freight		/wmt						$30.00	$29.97	$28.01	$29.00	$29.02	$28.94	$30.00		$29 32
IAA		/wmt						$3.00	$3.00	$3 00	$3 00	$3 00	$3.00	$3.00		$3 00
Transport Cost		/wmt						$83.78	$80.21	$74.18	$77.21	$77.27	$77.03	$81.00		$78.53
Moisture								7.50%	7.50%	7.50%	7.50%	7.50%	7.50%	7.50%		7.50%
Transport Cost		/dmt						$90.57	$86.72	$80.20	$83.47	$83.54	$83.28	$87.57		$84.89
3 Lead Concentrates		(000 dmt)						34	78	98	94	109	120	86		1,185
Land Handling		/wmt						$1.48	$1 37	$1 27	$1 32	$1 32	$1 32	$1.40		$1 35
Land Transportation		/wmt						$33.30	$30.17	$28.87	$29.53	$29.54	$29.49	$30.60		$29.98
Port		/wmt						$16.00	$15.70	$13.03	$14.37	$14.40	$14.29	$16.00		$14.88
Ocean Freight		/wmt						$30.00	$29.97	$28.01	$29.00	$29.02	$28.94	$30.00		$29.34
IAA		/wmt						$3.00	$3.00	$3.00	$3.00	$3.00	$3.00	$3.00		$3.00
Transport Cost		/wmt						$83.78	$80.21	$74.18	$77.21	$77.27	$77.03	$81.00		$78.56
Moisture								7.50%	7.50%	7.50%	7.50%	7.50%	7.50%	7.50%		7.50%
Transport Cost		/dmt						$90.57	$86.72	$80.20	$83.47	$83.54	$83.28	$87.57		$84.92
Copper Concentrates		(000 dmt)														
Land Handling		/wmt						$1.48	$1 37	$1 27	$1 32	$1 32	$1 32	$1.40		
Land Transportation		/wmt						$33.30	$30.17	$28.87	$29.53	$29.54	$29.49	$30.60		
Port		/wmt						$16.00	$15.70	$13.03	$14.37	$14.40	$14.29	$16.00		
Ocean Freight		/wmt						$30.00	$29.97	$28.01	$29.00	$29.02	$28.94	$30.00		
IAA		/wmt						$3.00	$3.00	$3.00	$3.00	$3.00	$3.00	$3.00		
Transport Cost		/wmt						$83.78	$80.21	$74.18	$77.21	$77.27	$77.03	$81.00		
Moisture																
Transport Cost		/dmt						$83.78	$80.21	$74.18	$77.21	$77.27	$77.03	$81.00		

Cash Costs - US$0.55 /lb Zinc & $ 0.32 /lb Lead

23-Jan-97 **Case : Grizzly Development - 1,500,000 Tonnes Annually**

Cdn Dollars

			1997	1998	1999	2000	2001	2002	2003	2004	2005	2006	2007	2012	2017	All Years
A. Production	Ore	(000 t)						500	1,200	1,500	1,500	1,500	1,500	1,500		17,240
	Tailing	(000 t)														
	Waste	(000 t)														
B. Site Costs		Cdn/t						$53 46	$47 56	$42 89	$42 89	$42 89	$42 89	$42 89		$43 72
Mining		Cdn/t						$28 95	$24 37	$21.64	$21 64	$21 64	$21 64	$21 64		$22 27
Milling		Cdn/t						$15 00	$14 17	$13 33	$13.33	$13 33	$13 33	$13 33		$13 49
Surface Transportation		Cdn/t						$3 50	$3 50	$3 50	$3 50	$3 50	$3 50	$3 50		$3 50
Administration		Cdn/t						$6 00	$5 52	$4 42	$4.42	$4 42	$4 42	$4 42		$4 46
C. Annual Charges																
Mining		(000 Cdn)						$14,480	$29,238	$32,457	$32,457	$32,457	$32,457	$32,457		383,926
Milling		(000 Cdn)						$7,500	$17,000	$20,000	$20,000	$20,000	$20,000	$20,000		232,600
Surface Transportation		(000 Cdn)						$1,750	$4,200	$5,250	$5,250	$5,250	$5,250	$5,250		60,340
Administration		(000 Cdn)						$3,000	$5,299	$6,624	$6,624	$6,624	$6,624	$6,624		76,924
Site Costs		(000 Cdn)						$26,730	$55,737	$64,331	$64,331	$64,331	$64,331	$64,331		$753,790
Environmental		(000 Cdn)						$125	$300	$375	$375	$375	$375	$375		$4,310
Head Office & Marketing		(000 Cdn)						$1,167	$3,500	$3,500	$3,500	$3,500	$3,500	$3,500		$40,927
Capital		(000 Cdn)	$8,800	$15,400	$2,200	$17,178	$30,302	$21,219						$2,000		$106,099
Other		(000 Cdn)														
Total Costs		(000 Cdn)	$8,800	$15,400	$2,200	$17,178	$30,302	$49,241	$59,537	$68,206	$68,206	$68,206	$68,206	$70,206		$905,126
Exchange Rate		US/Cdn	0.74	0.74	0.74	0.74	0.74	0.74	0.74	0.74	0.74	0.74	0.74	0.74	0.74	0.74
Total Cost		(000 $)	6,512	11,396	1,628	12,712	22,424	36,438	44,058	50,472	50,472	50,472	50,472	51,952		669,794
D. Total Cost		US$/t						$72.88	$36.71	$33.65	$33.65	$33.65	$33.65	$34.63		$38.85
E. Capital Costs Breakdown		(000 Cdn)	$8,800	$15,400	$2,200	$17,178	$30,302	$21,219						$2,000		$106,099
1. Existing Operation			$8,800	$15,400	$2,200	$17,178	$30,302	$21,219						$2,000		$106,099
Mine Equipment	CCA41					$3,265	$5,527	$8,308						$2,000		$28,100
Bond	CCA41															
Pipeline	CCA41															
Mill Equipment	CCA41															
Development	CEE		$8,800	$15,400	$2,200	$13,913	$24,776	$12,911								$78,000
Drivage	CEE															
2. Reserve Extended		(000 Cdn)														
Mine Equipment	CCA41															
Bond	CCA41															
Pipeline	CCA41															
Mill Equipment	CCA41															
Development	CEE															
Drivage	CEE															

Smelter Page - US$0.55 /lb Zinc & $ 0.32 /lb Lead

23-Jan-97 Case : Grizzly Development - 1,500,000 Tonnes Annually

US Dollars

				1997	1998	1999	2000	2001	2002	2003	2004	2005	2006	2007	2012	2017	All Years
A.	Zinc Concentrates		(000 t)						67	150	194	162	146	138	127		1691
	Base TC		/t	$190.00	$83.76	$83.76	$83.76	$83.76	$83.76	$83.76	$83.76	$83.76	$83.76	$83.76	$83.76	$83.76	84
	Price Level		/t Zn	$1,000.00													
	Up Scale		/$	$0.13	$0.10	$0.10	$0.10	$0.10	$0.10	$0.10	$0.10	$0.10	$0.10	$0.10	$0.10	$0.10	$0.10
	Down Scale		/$	$0.080													
	Penalty		/t						$6.600	$6.600	$6.600	$6.600	$6.600	$6.600	$6.600		$6.60
	NSR Zinc Concentrate																
	Paid	Zn	lb/t						1,049	1,049	1,049	1,049	1,049	1,049	1,049		1,049
		Ag	g/t														
		Au	g/t												0		0
	Revenue	Zn	/t						$577.16	$577.16	$577.16	$577.16	$577.16	$577.16	$577.16		577.2
		By-prod	/t												$0.03		$0.12
	Less TC		/t						$216.46	$216.46	$216.46	$216.46	$216.46	$216.46	$216.46		$216.46
	Zn NSR								$360.70	$360.70	$360.70	$360.70	$360.70	$360.70	$360.73		$360.83
	Per T pay zn								$454.75	$454.75	$454.75	$454.75	$454.75	$454.75	$454.75		$454.75
	TC/lb zn								$0.21	$0.21	$0.21	$0.21	$0.21	$0.21	$0.21		$0.21
	Zn		K Lbs						70,184	157,804	203,299	170,423	153,501	144,315	133,437		************
B.	Lead Concentrates		(000 t)						34	78	98	94	109	120	86		1,185
	Base TC		/t	$170.00	$129.88	$129.88	$129.88	$129.88	$129.88	$129.88	$129.88	$129.88	$129.88	$129.88	$129.88	$129.88	130
	Price Level		/t Pb	$617.29													
	Up Scale		/$	$0.13	$0.07	$0.07	$0.07	$0.07	$0.07	$0.07	$0.07	$0.07	$0.07	$0.07	$0.07	$0.07	$0.07
	Penalty		/t														
	NSR Lead Concentrate																
	Paid	Pb	lb/t						1,257	1,257	1,257	1,257	1,257	1,257	1,257		1,257
		Ag	g/t						742	844	798	857	777	709	962		792
		Au	g/t						2	3	3	5	4	4	6		4
	Revenue	Pb	/t						$402.12	$402.12	$402.12	$402.12	$402.12	$402.12	$402.12		402.1
		By-prod	/t						$151.74	$186.89	$177.53	$208.47	$184.48	$171.97	$241.33		$191.39
	Less TC		/t						$179.26	$179.26	$179.26	$179.26	$179.26	$179.26	$179.26		$179.26
	Pb NSR		/t						$374.60	$409.74	$400.39	$431.33	$407.34	$394.82	$464.19		$414.25
	Pb TC		/t						$314.50	$314.50	$314.50	$314.50	$314.50	$314.50	$314.50		$314.50
	Pb TC		/lb						$0.14	$0.14	$0.14	$0.14	$0.14	$0.14	$0.14		$0.14
	Pb		K Lbs						$43,348.22	$97,626.96	$123,368.86	************	$136,987.51	************	************		************

Smelter Revenue – US$0.55 /lb Zinc & $ 0.32 /lb Lead

23-Jan-97 Case : Grizzly Development – 1,500,000 Tonnes Annually

US Dollars

		1997	1998	1999	2000	2001	2002	2003	2004	2005	2006	2007	2012	2017	All Years
A. Net Smelter Return															
Total Revenue							$57,707	$132,552	$168,721	$150,823	$148,373	$148,422	$128,498		$1,679,547
Zinc							$38,601	$86,792	$111,814	$93,733	$84,426	$79,373	$73,301		$975,910
Lead							$13,871	$31,241	$39,478	$37,598	$43,836	$48,365	$34,436		$476,588
Copper															
Precious Metals							$5,234	$14,519	$17,429	$19,492	$20,111	$20,683	$20,671		$227,048
Smelter Costs							$20,661	$46,478	$59,535	$51,915	$51,205	$51,330	$42,876		$578,472
Zinc							$14,477	$32,561	$41,935	$35,154	$31,664	$29,769	$27,525		$366,011
Lead							$6,184	$13,927	$17,599	$16,761	$19,542	$21,561	$15,362		$212,461
Copper															
Net Smelter Revenue							$37,046	$86,074	$109,187	$98,908	$97,167	$97,092	$85,622		$1,101,074
B. Penalty Calculations															
1. Zinc Concentrate		$6.60	$6.60	$6.60	$6.60	$6.60	$6.60	$6.60	$6.60	$6.60	$6.60	$6.60	$6.60	$6.60	
Mercury	Actual	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	
	Allowable	0.005%	0.005%	0.005%	0.005%	0.005%	0.005%	0.005%	0.005%	0.005%	0.005%	0.005%	0.005%	0.005%	
	$2.00/100 ppm	660.000%	660.000%	660.000%	660.000%	660.000%	660.000%	660.000%	660.000%	660.000%	660.000%	660.000%	660.000%	660.000%	
Iron	Actual	$0.06	$0.06	$0.06	$0.06	$0.06	$0.06	$0.06	$0.06	$0.06	$0.06	$0.06	$0.06	$0.06	
	Allowable	8.00%	8.00%	8.00%	8.00%	8.00%	8.00%	8.00%	8.00%	8.00%	8.00%	8.00%	8.00%	8.00%	
	$1.5/1%														
2. Lead Concentrate															
Mercury	Actual														
	Allowable	0.005%	0.005%	0.005%	0.005%	0.005%	0.005%	0.005%	0.005%	0.005%	0.005%	0.005%	0.005%	0.005%	
	$2.00/100 ppm														
Arsenic	Actual	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	
	Allowable	0.30%	0.30%	0.30%	0.30%	0.30%	0.30%	0.30%	0.30%	0.30%	0.30%	0.30%	0.30%	0.30%	
	$20/1%														
Antimony	Actual														
	Allowable	0.30%	0.30%	0.30%	0.30%	0.30%	0.30%	0.30%	0.30%	0.30%	0.30%	0.30%	0.30%	0.30%	
	$30/1%														
Bismuth	Actual														
	Allowable	0.30%	0.30%	0.30%	0.30%	0.30%	0.30%	0.30%	0.30%	0.30%	0.30%	0.30%	0.30%	0.30%	
	$20/1%														

Current Reserves
23-Jan-97 Case : Grizzly Development - 1,500,000 Tonnes Annually

			1997	1998	1999	2000	2001	2002	2003	2004	2005	2006	2007	2012	2017	All Years
1. Production																
Ore Production		(000 T)						500	1200	1500	1500	1500	1500	1500		17240
Waste Production		(000 T)														
Ore Grade	-Zn							8.71%	8.16%	8.41%	7.05%	6.35%	5.97%	5.52%		0
	-Pb							4.87%	4.57%	4.62%	4.40%	5.13%	5.66%	4.03%		4.9%
	-Cu															
	-Ag	g/t						6830.00%	7290.00%	6970.00%	7130.00%	7540.00%	7590.00%	7330.00%		7262.8%
	-Au	g/t						0.4	0.6	0.6	0.8	0.7	0.8	0.9		0.8
Recovery																
Zn			86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%		86.00%
	Pb Cons		4.0%	4.0%	4.0%	4.0%	4.0%	4.0%	4.0%	4.0%	4.0%	4.0%	4.0%	4.0%		4.00%
	Cu Cons															
	Overall		90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%		90.00%
Pb	Zn Cons		7.0%	7.0%	7.0%	7.0%	7.0%	7.0%	7.0%	7.0%	7.0%	7.0%	7.0%	7.0%		7.00%
	Pb Cons		85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%		85.00%
	Cu Cons															
	Overall		92.0%	92.0%	92.0%	92.0%	92.0%	92.0%	92.0%	92.0%	92.0%	92.0%	92.0%	92.0%		92.00%
Cu	Zn Cons															
	Pb Cons															
	Cu Cons															
	Overall															
Ag	Zn Cons		10.0%	10.0%	10.0%	10.0%	10.0%	10.0%	10.0%	10.0%	10.0%	10.0%	10.0%	10.0%		10.00%
	Pb Cons		78.9%	78.9%	78.9%	78.9%	78.9%	78.9%	78.9%	78.9%	78.9%	78.9%	78.9%	78.9%		78.87%
	Cu Cons															
	Overall		88.9%	88.9%	88.9%	88.9%	88.9%	88.9%	88.9%	88.9%	88.9%	88.9%	88.9%	88.9%		88.87%
Au	Zn Cons		10.0%	10.0%	10.0%	10.0%	10.0%	10.0%	10.0%	10.0%	10.0%	10.0%	10.0%	10.0%		10.00%
	Pb Cons		51.5%	51.5%	51.5%	51.5%	51.5%	51.5%	51.5%	51.5%	51.5%	51.5%	51.5%	51.5%		51.54%
	Cu Cons															
	Overall		61.5%	61.5%	61.5%	61.5%	61.5%	61.5%	61.5%	61.5%	61.5%	61.5%	61.5%	61.5%		61.54%
Zn Concentrates		(000 t)						67	150	194	162	146	138	127		1,691
			56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%		1
	Pb							2.55%	2.56%	2.50%	2.84%	3.68%	4.32%	3.33%		3.46%
	Cu															
	Ag	g						5106.1%	5817.4%	5396.7%	6585.5%	7731.9%	8278.6%	8646.8%		7405.09%
	Au	g						0.3	0.4	0.4	0.7	0.7	0.9	1.0		1
Pb Concentrate		(000 t)						34	78	98	94	109	120	86		1,185
			1	1	1	1	1	1	1	1	1	1	1	1		1
	Zn							5.05%	5.04%	5.14%	4.52%	3.50%	2.98%	3.87%		3.72%
	Cu															
	Ag	g						78082.3%	88812.1%	83994.6%	90218.9%	81830.4%	74659.7%	101265.1%		83325.98%
	Au	g						3.1	4.5	4.3	6.2	5.2	5.1	7.7		6

Capital Investment Plan

23-Jan-97 Case : Grizzly Development – 1,500,000 Tonnes Annually

(000)	-3 1997	-2 1,998	-1 1,999	1 2,000	2 2,001	3 2,002	4 2,003	5 2,004	6 2,005	7 2,006	8 2,007	13 2,012	18 2,017	All Years
A. Production						500	1200	1500	1500	1500	1500	1500		17240
B. Total Capital Cost	$8,000	$14,500	$2,300	$17,178	$36,552	$20,748	$3,097	$891	$891	$3,036	$2,430	$891		$125,287
Development	$8,000	$14,000	$1,800	$12,648	$20,705	$9,113	$206							$66,473
Development Drivage					$5,000									$5,000
Equipment				$2,968	$5,024	$9,190	$2,470	$636	$636	$2,586	$2,035	$636		$38,679
Recruitment					$500	$500								$1,000
Licensing & Water Bond		$500	$500		$2,000	$58	$139	$174	$174	$174	$174	$174		$5,000
Contingency @ 10.00%				$1,562	$3,323	$1,886	$282	$81	$81	$276	$221	$81		$9,135
Cumulative Capital Costs	$8,000	$22,500	$24,800	$41,978	$78,530	$99,278	$102,375	$103,266	$104,157	$107,193	$109,623	$124,327	$125,287	
C. Development	$8,000	$14,000	$1,800	$12,648	$20,705	$9,113	$206							$66,473
1. Feasibility Study Etc			$1,800											1,800
2 Drilling														
Ramp and UG Explore . Separate Drilling	$8,000	$14,000												22,000
Shaft & Orebody				$11,500	$12,000									23,500
Fan & orebin				$80	$260									340
Truck Load					500									500
Shaft DH				$200										200
Storage Shaft & TM					$2,300									2,300
East Return Airshaft, rock removed in mine shaft					$3,000	3,000								6,000
3 Power from Grum				$500										500
P. Equipment					$400	$102								502
P. Equipment UG					$542	$1,000								1,542
4 Upgrade haul Rd.						$1,200								1,200
P. Equipment Installed						$800								800
P. Equipment UG					$400	$600								1,000
5 Sump UG														
Main Pump & Range					$300	$336								636
Ramp & Shaft Sump						$660								660
6. Air Heating					$200	$50								250
UG Workshops, etc						$400	$200							600
UG Warehouse						$100								100
Conveyor, Ramp to UG bin					$200	$600								800
Use 3% Freight				$368	$603	$265	$6							1,243
D. Development Drivage					$5,000									5,000
Ramp Conv (airway)					$5,000									5,000
Intake B2														
Raise														
B3-1														
B3_2														
B3-3														
Ramps to HW														
Main Mine Conveyor														
Main Access														
NW4 - 1 Conv														
NW4 - 2 FW														
NW4 - 3 HW														
A4 raise Airway														
Raise Airway														
Access Raising														
Access Raising														
Backfill Drift														

Capital Investment Plan (Continued)

23-Jan-97 Cdn Dollars

			-3 1997	-2 1998	-1 1999	1 2000	2 2001	3 2002	2003	2004	2005	2006	2007	2012	2017	All Years
	Production							500	1,200	1,500	1,500	1,500	1,500	1,500		17240
E.	Equipment					$2,968	$5,024	$9,190	$2,470	$636	$636	$2,586	$2,035	$636		$38,679
	Twin drills Jumbo	6				$600	$600	$600	$600			$600	$600			$5,400
	Prod. Loader	6				$630	$630	$1,260	$630			$630	$630			$6,300
	Bolter Electric	6				$520	$520	$1,040				$520	$520			$4,160
	Bench Drill	3				$430	$430	$1,040								$2,940
	26 t Truck for fill	2				$320										$1,180
	Small Scoop	3						$320	$320							$1,600
	Ore Trucks 40 t	4					$640	$1,260				$640				$3,180
	An Loader	2						$360	$360							$1,120
	Man Carrier	2						$200								$400
	Sissors Lift	2						$380								$380
	Mech late truck	1						$230								$460
	Supervisor Vechicles	6						$150	$150							$600
	Crusher/Rock breaker	5					$1,500									$2,000
	Road Grader	1						$350								$550
	UG Core Drill	2					$120	$120					$120			$480
	Stopers, jack etc	1						$250								$250
	Compressor, portable	2						$120								$220
	Mechanics Truck	2						$300								$360
	FF pipe-valve							$250								$250
	Cable replacement								$120	$50	$50	$50	$50			$520
	Fans & vent tube	10						$60	$70							$130
	Face Pumps & pipe	4						$50	$80							$130
	Explosive, Mag	1						$30								$30
	Backfill UG Silo							$300								$400
	Misc.					$300	$300			$550	$550			$600		$3,450
	Freight @	6.00%				$168	$284	$520	$140	$36	$36	$146	$115	$36		$2,189

- Value of spares is included in the invetory estimate of working capital.

T

CALCULATION OF NET SMELTER RETURNS

The calculations of Net Smelter Return values are based on geological data provided by the "polygon system", i.e. the thickness of ore intersection projected to the vertical, and the respective ore grades, are projected half way to the nearest surrounding drillhole locations, thus forming a polygon around each ore intersection.

"In situ" tonnage is calculated by:

tonnes "in situ" = vertical height x polygon area x specific gravity

where tonnes are metric tons, height is in meters, area is in square meters, and the specific gravity is considered constant, having a value of 3.92, regardless of metal contents.

Mine recovery depends on thickness: if it is less than 6.5m, the stopes in the polygon will not be backfilled, and the mine recovery will be 70%; if it is larger than 6.5m, the stope will be backfilled, and the mine recovery will be 85%. "Tonnes milled", therefore, is calculated by using the respective recovery rate.

Next, the grades are multiplied with the "tonnes milled" value to obtain the weight of the metal contents of the ore. The tonnes and the grams are then converted into pounds and ounces, and multiplied with the respective metal prices.

The following prices are applied:

Lead:	$US	0.30/lb
Zinc:	$US	0.55/lb
Silver:	$US	5.50/oz
Gold:	$US	400.00/oz

The sum of the metal values provide the "Gross Revenue" (sometimes called "Payable Metal Value") of the respective polygon. The Gross Revenue divided by the Tonnes Milled value gives the dollars per ton value of the polygon.

The pounds of lead, and the ounces of silver and gold, are converted into tonnes. The total is divided by 0.64, as the above metals make up 64% of the concentrate to be sent to the lead smelter. Similarly, the pounds of zinc are converted into tonnes and divided by 0.53 to get the zinc concentrate tonnage to be sent to the zinc smelter.

The treatment charge of the lead smelter is $US 160.17/tonne of concentrate,
the treatment charge of the zinc smelter is $US 220.54/tonne of concentrate,
and the freight to both smelters is $US 23.73/tonne of concentrate.

Multiplying the above costs with the respective tonnages provides the "Smelting Costs". The sum of these costs deducted from the Gross Revenue gives the "Net Smelter Return" of the polygon in question.

The Net Smelter Return value, divided by the Tonnes Milled, results in the N.S.R. per tonne value, in US dollars, of the polygon.

Using an exchange value of $US 0.73/$Cdn, the N.S.R./tonne is converted into Canadian dollars.

(I)

274/6

Yukon Energy Corporation
Invoice

Sold to:
Anvil Range Mining Corporation
Via Telecopier 403-668-6518

cc Oliver O'Rourke - YEC
Mary Bompas - APL
Wendy Sumanik - YECL

GST # 12300 7494 RT0001
Account # 05-9004600-1-01
Date 28-Oct-96
Terms Net 21 days
Due: 18-Nov-96

		Feeder 1	Feeder 2	Total
Meter reading	Current	1,921.9	1,854.9	
	Previous	1,303.3	1,239.9	
Read date 25-Oct-96	Change	618.6	615.0	
	Multiplier	12,000	12,000	
	Consumption	7,423,200	7,380,000	14,803,200

Energy Charge

Total energy		14,803,200	
Threshold energy - 90% of forecast energy of	15,313,267	13,781,940	
Excess energy (Total energy less threshold energy 0 if negative)		1,021,260	
Base load energy		13,781,940	

	Consumption	Rate	Charge	
Base load energy charge	13,781,940	0.04453	613,709.80	
Excess energy charge	1,021,260	0.10240	104,576.99	
Total energy charge	14,803,200			718,286.79
Fixed Charge				21,448.00

Demand Charge		Peak	Rate	Charge	
a) Metered peak demand current month (KVA)		23,201			
b) Peak demand in last twelve months		23,400			
Peak month was:	26-Feb-96				
Billing demand (maximum of a) or b))		23,400	18.60	435,240.00	435,240.00
Sub-total					1,174,974.79

Riders

	Base	Rate	Charge	
Rider G - 1996 revenue shortfall rider	1,174,974.79	4.16%	48,878.95	48,878.95
Sub-total				1,223,853.74
GST				85,669.76
Current balance due				1,309,523.50

Payment on this account should be made to

The Yukon Electrical Company Limited
P.O. Box 4190
Whitehorse, Yukon
Y1A 3T4
Attention: Brian Milne

APPENDIX J

ACTION PLAN
FOR
UNDERGROUND EXPLORATION
AND
FULL FEASIBILITY STUDY

5. ACTION PLAN

5.1 Options to be Considered for the Final Feasibility Study

5.1.1 Production Access

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

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

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

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

5.1.2 Hoisting Arrangement

- Double drum hoist

- Koepe hoist ground mounted (earth quakes)
- Koepe hoist tower mounted

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

5.1.3 Headframe

- Concrete or steel

5.1.4 Mining Methods

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

5.1.5 Main Underground Haulage

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

5.1.6 Ventilation Raises

5.1.7 Mine Dewatering

5.2 Project Schedule

See Figure 5.2-01.

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

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

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

5.3 Action Plan

5.3.1 Preparation Work

- Re-logging of the existing core
- Creation of three dimensional computer model (Gemcom GS-32)
- Review experience and knowledge from Faro open pit and underground:
  - Geology
  - Mining
- Utilise experience and knowledge from Grum:
  - Review all existing cross sections
  - Review all existing longitudinal sections
  - Develop three dimensional computer model (Gemcom GS-32)
  - Design surface exploration drilling program for Grum underground
  - Schedule Grum underground for production
  - Design Grum underground
- Review all information on Grizzly.
- Visit mines with features similar to Grizzly

5.3.2 Access for Underground Exploration

- Obtain data on:
  - geology
  - geological structures
  - hydrology
  - geotechnical conditions

5.3.3 Underground Exploration

- Obtain information of the orebody concerning:

- Thickness
- Grade
- Continuity
- Dip
- Tectonic
- Geotechnical data of:
  - Ore horizon
  - Hanging wall
  - Footwall
- Hydrology
- Metallurgy

5.3.4 Mine visits to build up expertise

5.3.5 Gathering Information on Mining Equipment and Mining Technology

5.3.6 Evaluation of Data

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

5.4 Bankable Feasibility Study

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

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

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

APPENDIX K

PITEAU & ASSOCIATES: GEOLOGY, ORE RESERVE DETAILS

6. GEOLOGY AND ORE RESERVES

6.1 REGIONAL GEOLOGY

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

6.2 DEPOSIT GEOLOGY

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

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

'he B Zone is generally characterized by relatively consistent high grade, pyritic and pyrrhotitic ıassive sulphide ore with quartz forming the main gangue mineral. The A Zone consists of ıick intervals of pyritic massive sulphides, generally of lower grade and greater variation in lead nd zinc content. Gangue mineralogy in the A Zone is dominated by barite.

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

6.2.1 Structural Geology

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

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

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

6.2.2 Faults

Limited information with respect to high angle faults at Grizzly is evident due to bias from near vertical and widely spaced surface drillholes. Drillholes that have encountered steeply dipping faults indicate that faults with significant displacement faults do occur (e.g. a large scale fault encountered in drillhole 90DY04 on the north side of the deposit). This type of faulting is consistent with near vertical displacement faulting encountered at other deposits within the district. Larger displacement structures of approximately 10 to 20m, or greater, tend to be characterized by thick clay filled gouge and breccia zones which can often transmit water.

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

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

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

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

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

6.2.3 Geology Sections and Structure Contours

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

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

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

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

CRI, and have subsequently been called the UPPER-G and LOWER-G horizons, respectively.

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

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

6.3 GEOLOGICAL RESERVE ESTIMATE AND MINING INVENTORY

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

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

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

6.3.1 Calculation Method

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

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

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

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

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

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

6.3.2 Results

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

Table 6.7
Grizzly Mining Inventory
10% Dilution

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

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

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

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

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

Table 6.12
Grizzly Mining Inventory with Recoveries
10% Dilution

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

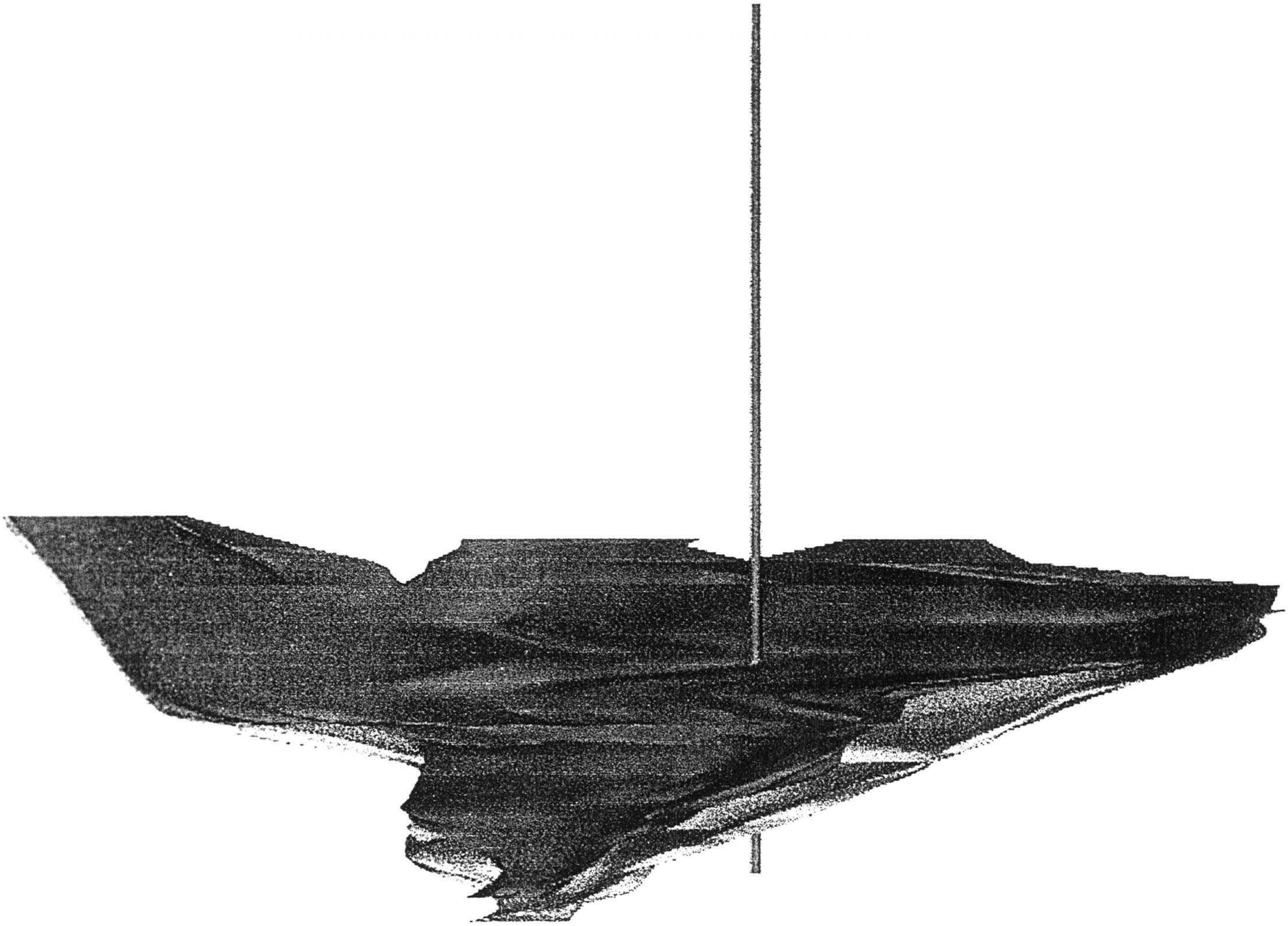
6 3.3 Discussion of Results

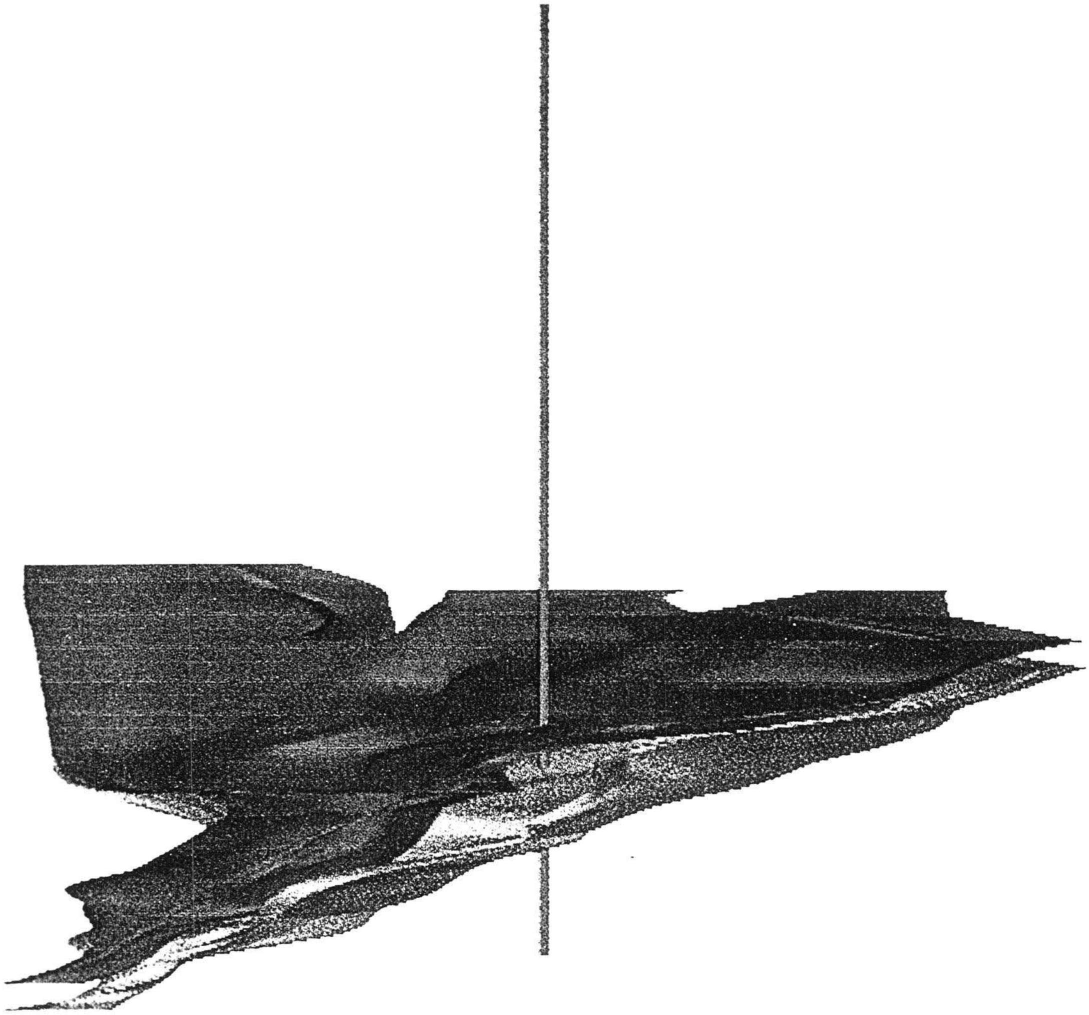
In consideration of mining at Grizzly, the above compositing criteria is suggested to represent a selective mining situation which would be based strictly on an economic cutoff grade with no defined geologic controls, and may require great detail in drillhole definition.

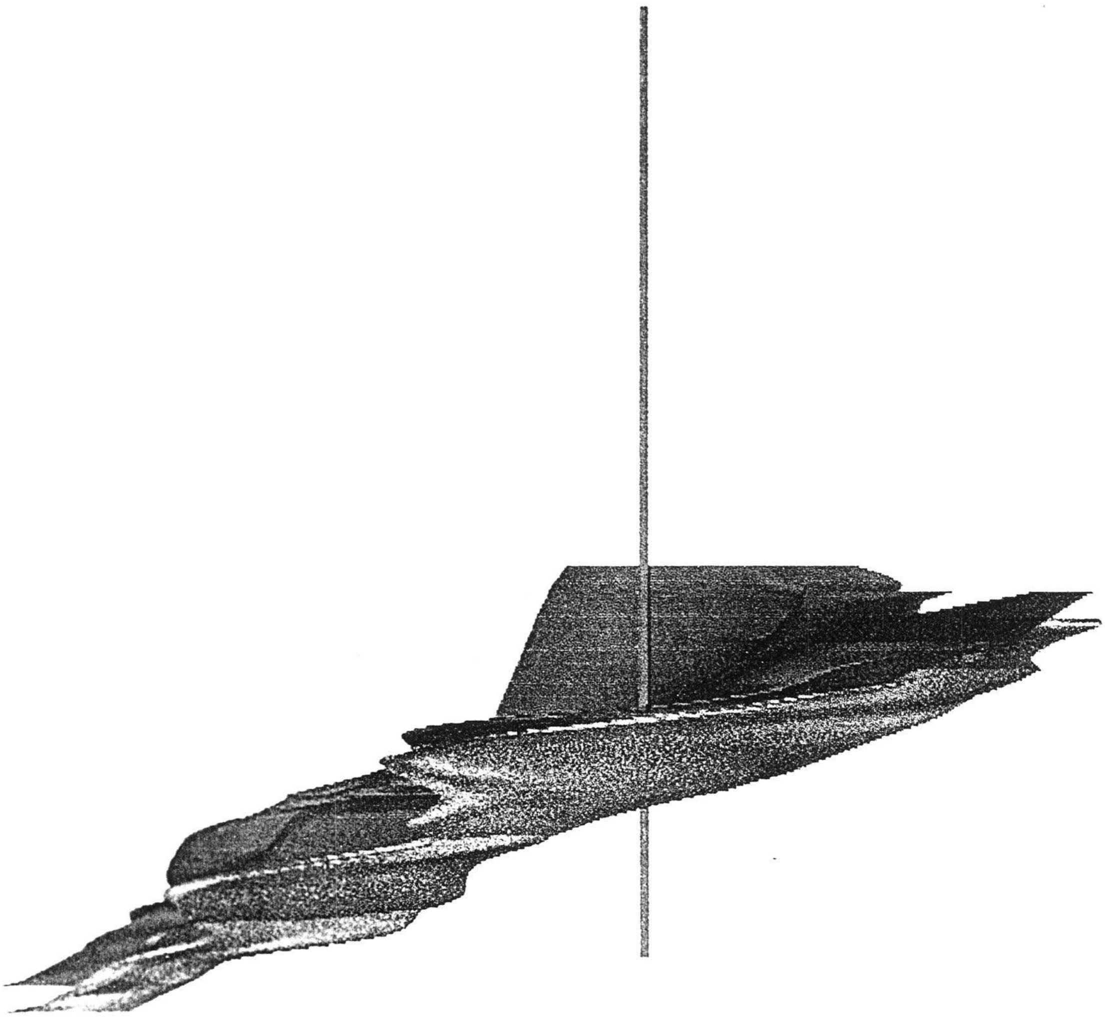
It should be noted that no lithologic constraints were placed on composite intervals, thus all composites were generated on a grade basis only. Therefore, no differentiation was made between quartzite mineralization and massive sulphide ore, and no restriction was placed on separation of massive sulphide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

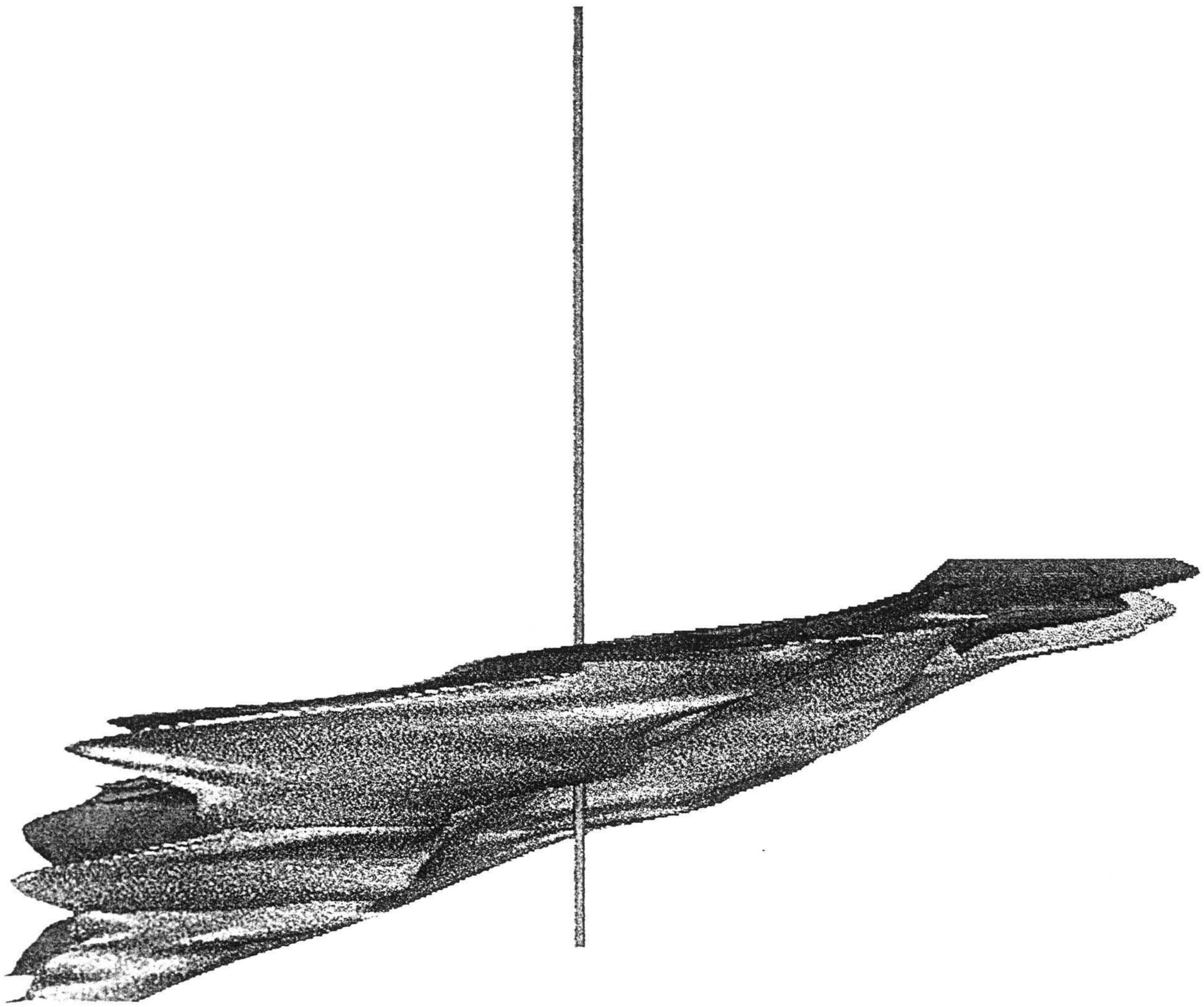
The pre-feasibility study by N.D. Rose of FGC in 1992 involved rigid geologic parameters in defining a mineable criteria based on experience in underground mining at the Faro Underground Mine. The reader is referred to that study as a comparison with the present results in consideration of mining of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

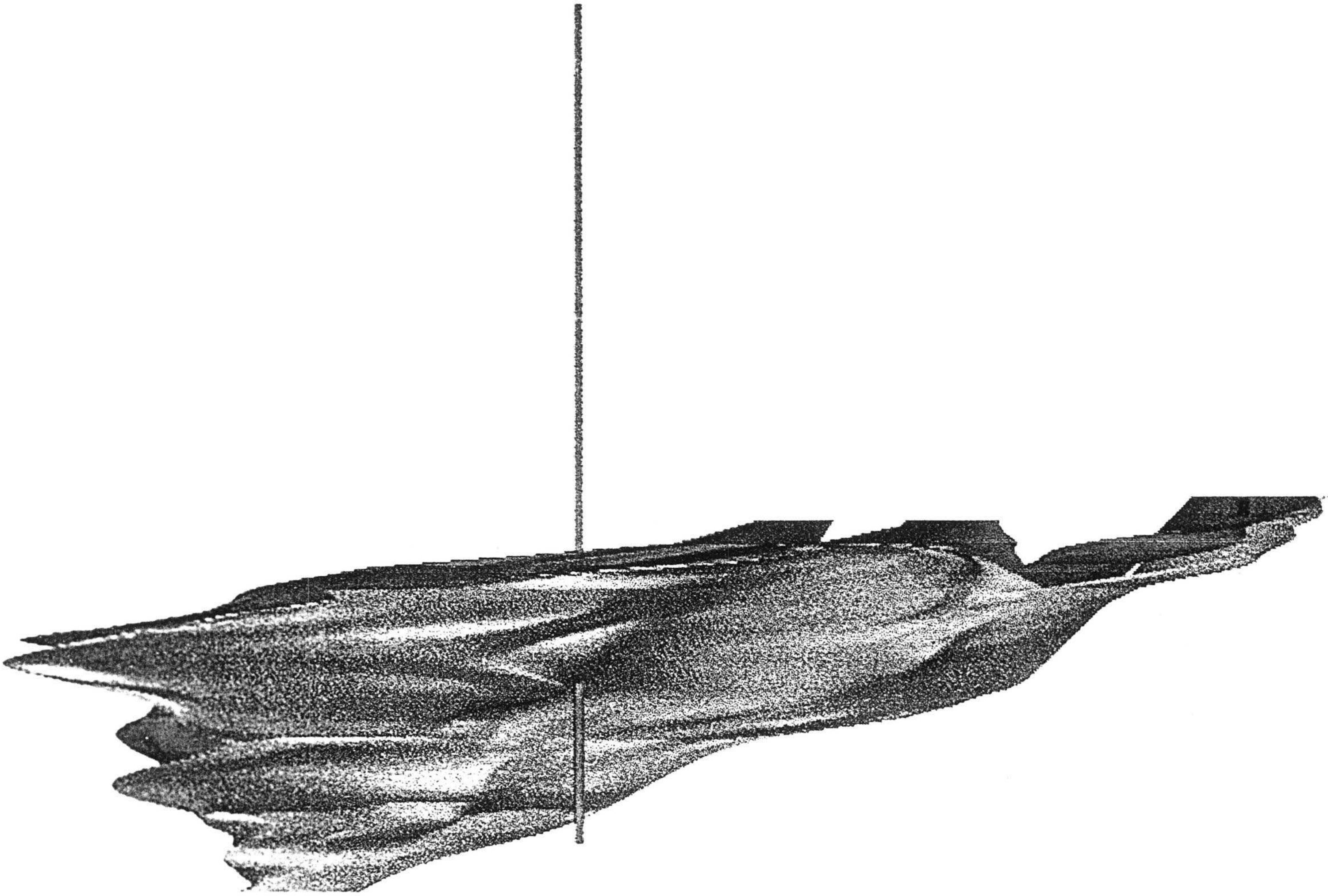
When considering mining methods, the 3.5m or greater waste exclusions should likely be incorporated as dilution or accounted for in mining and haulage of waste. Also, the recognition of possible mixing or blending of lower grade materials should possibly be addressed in the overall mining inventory and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

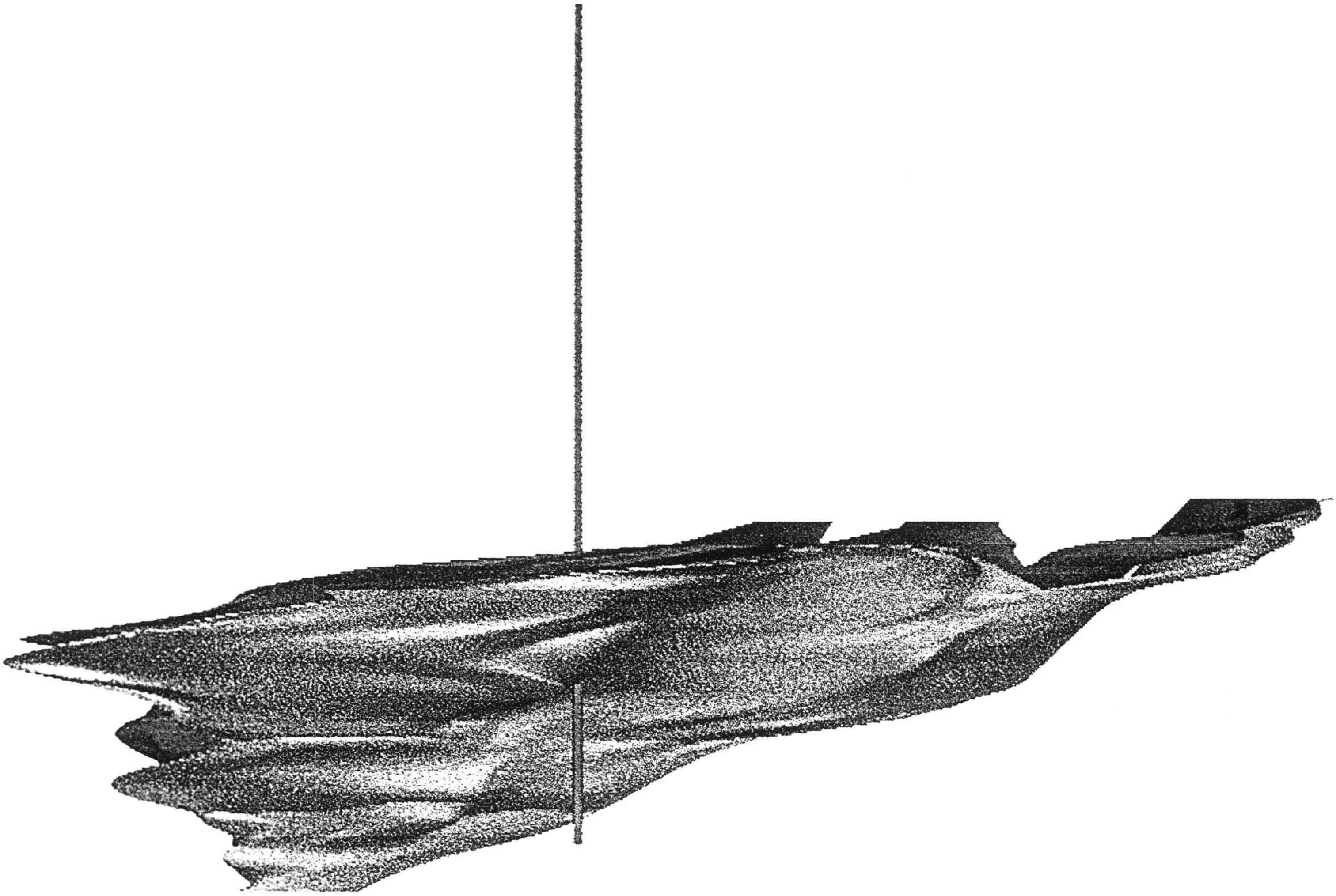


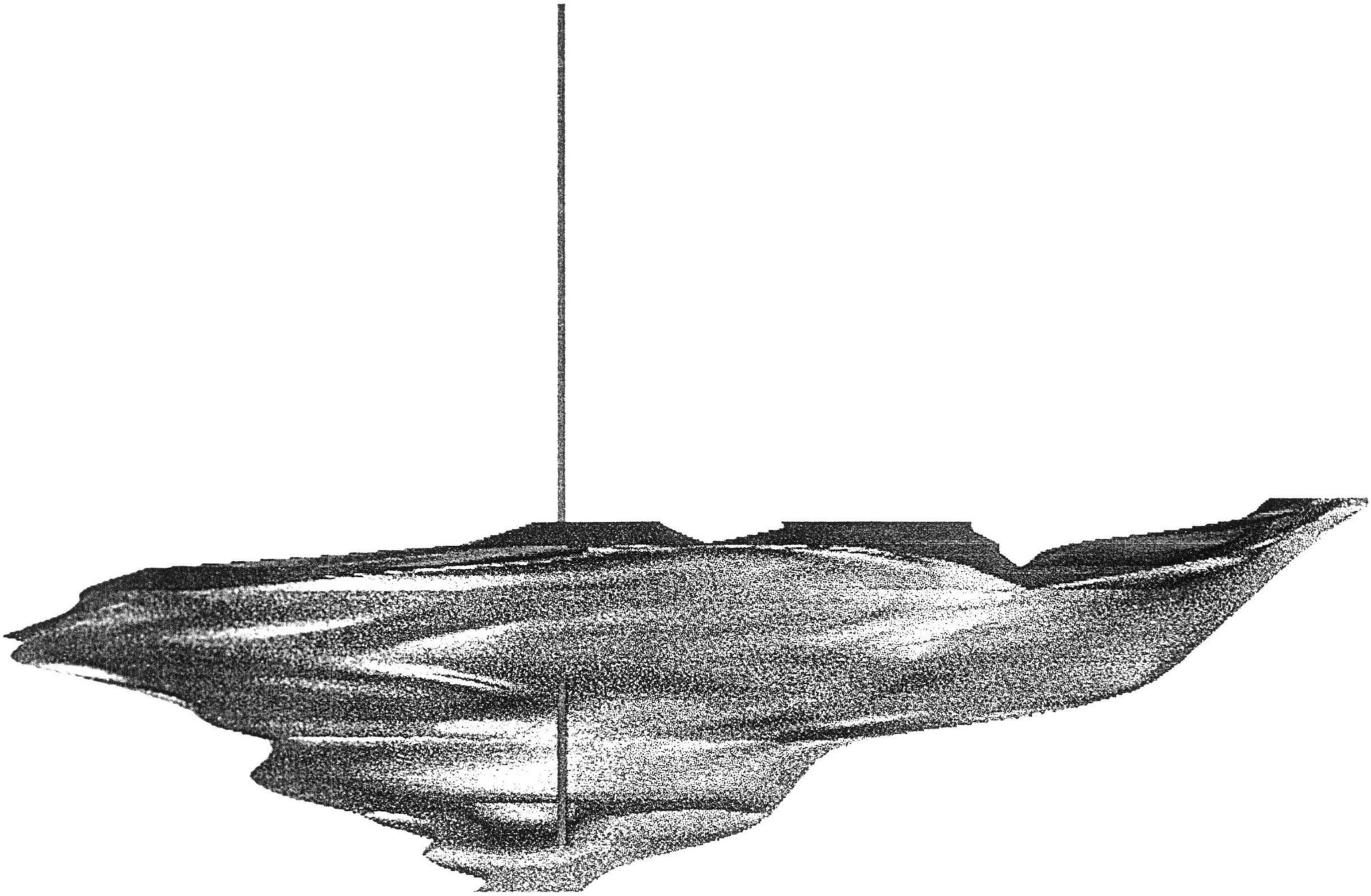


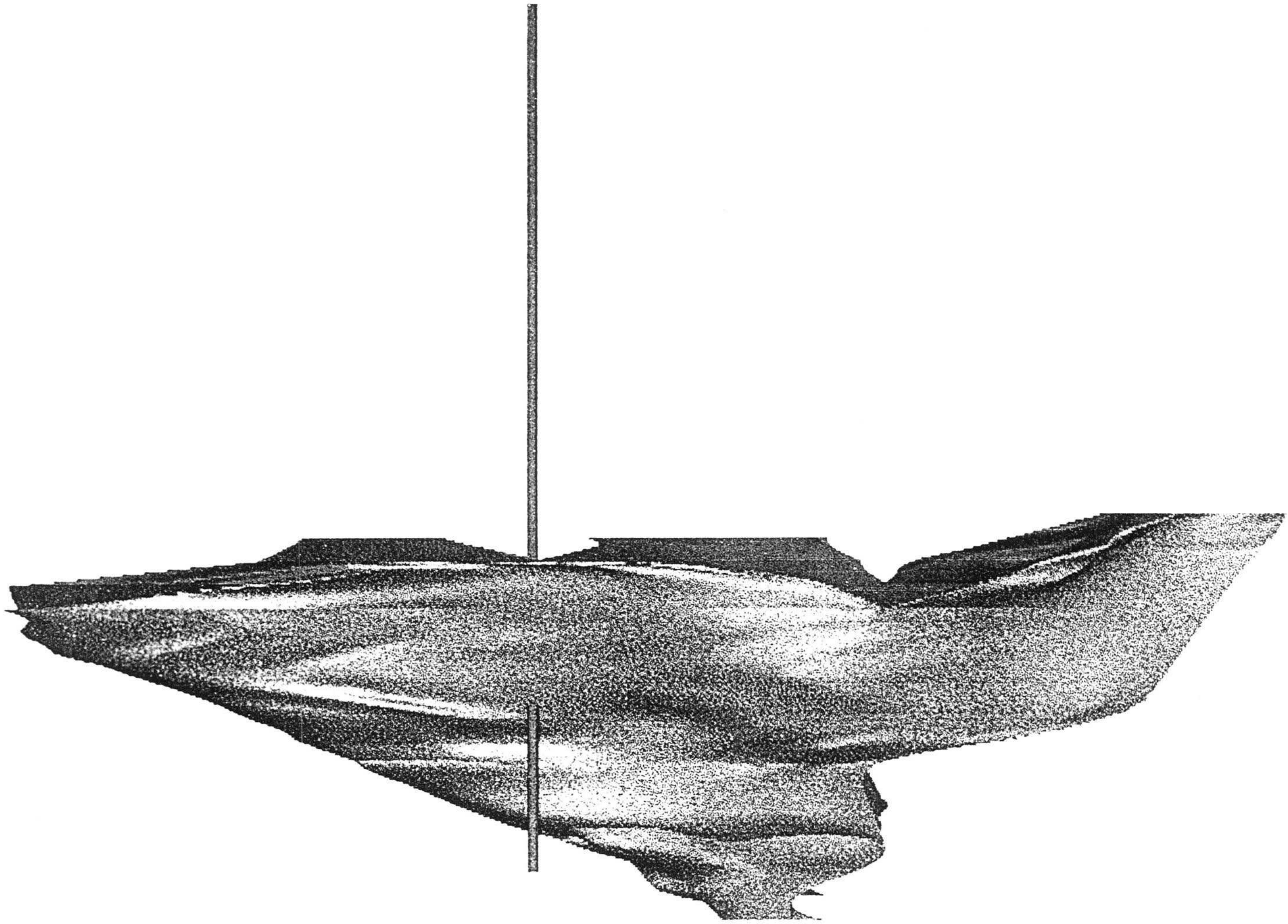


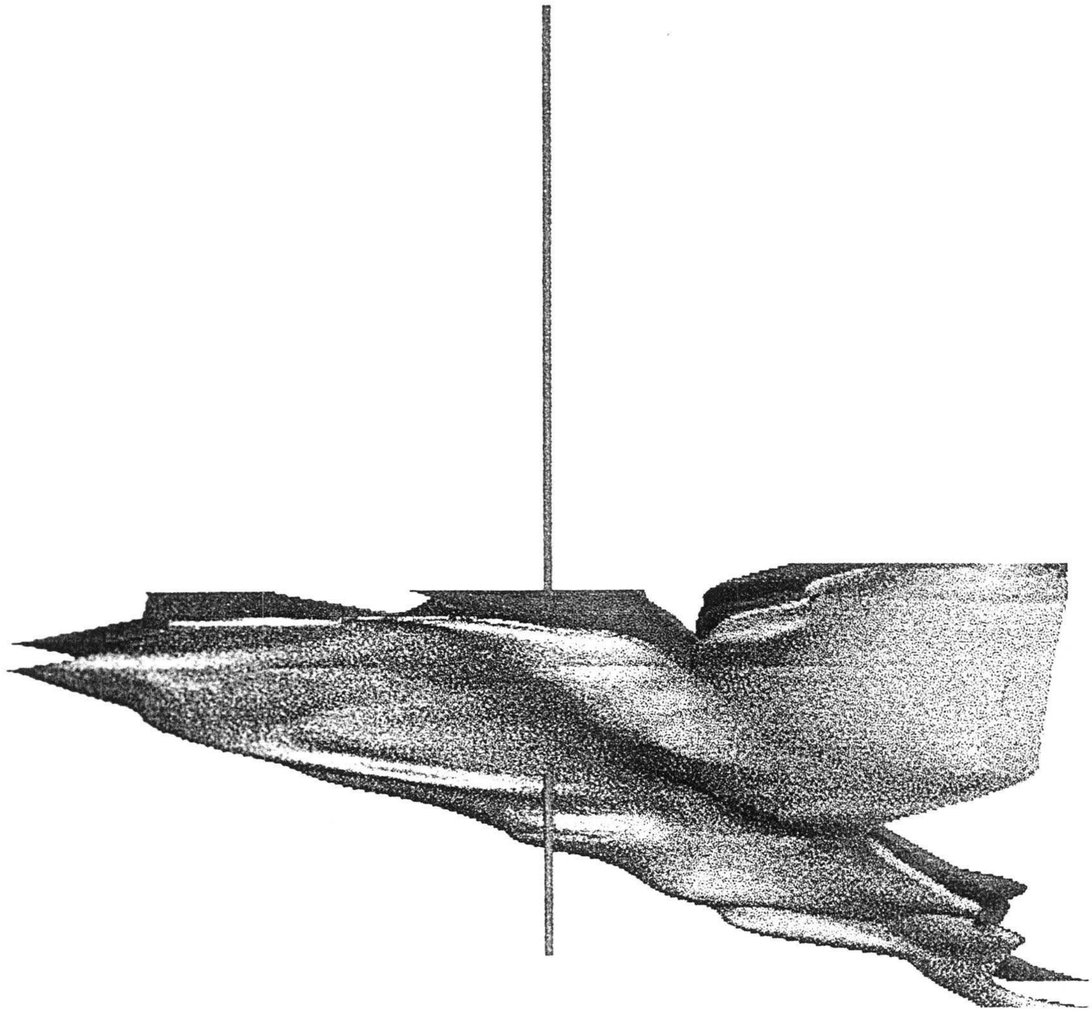


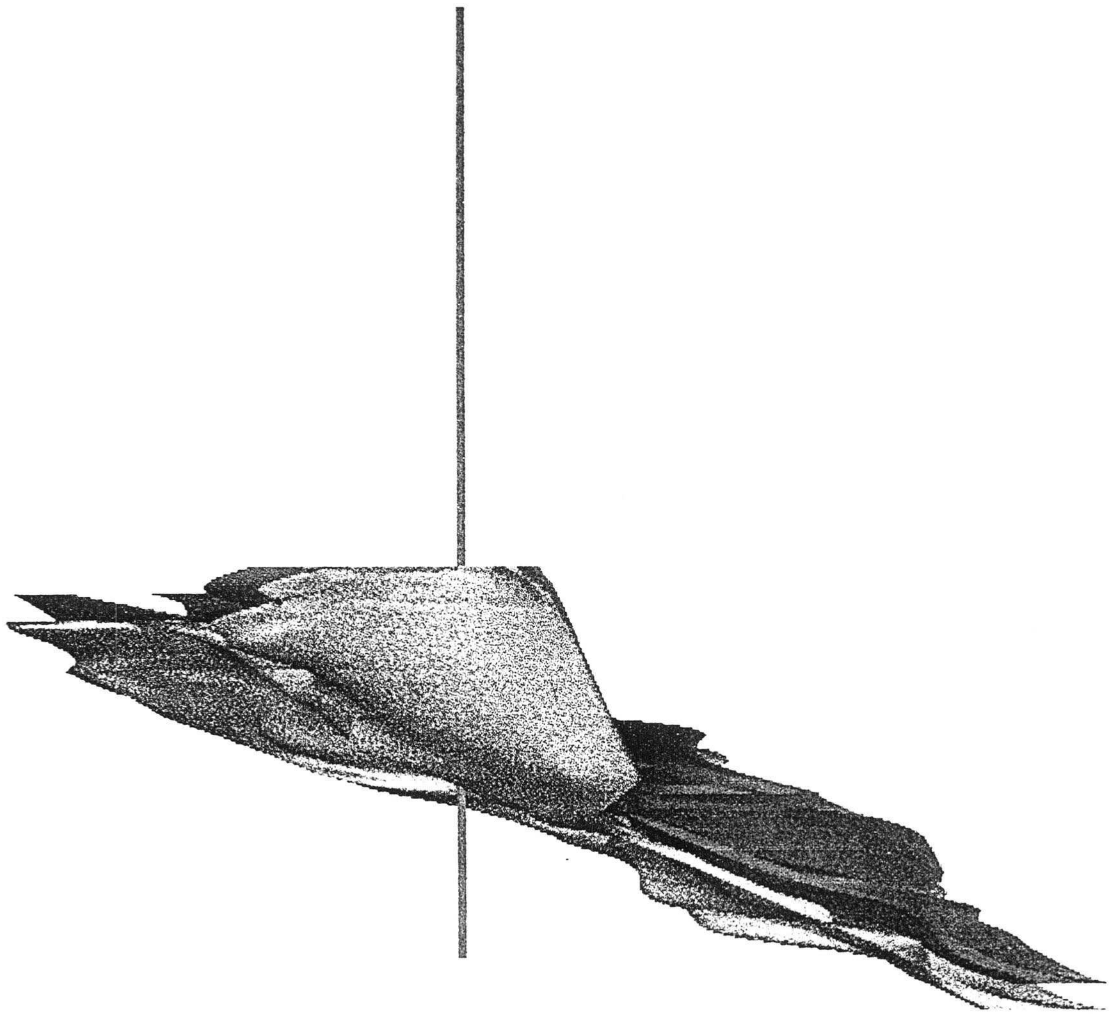


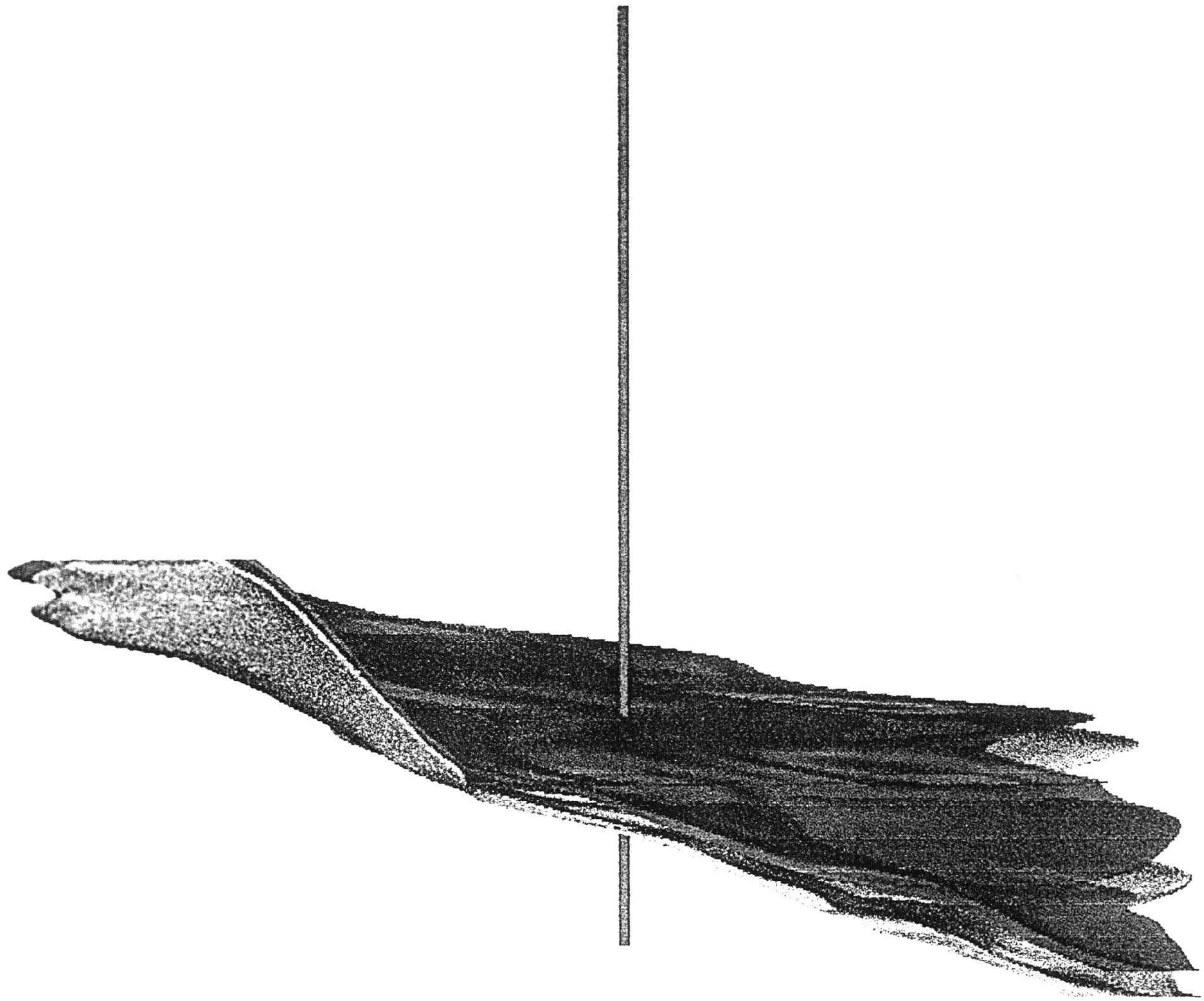


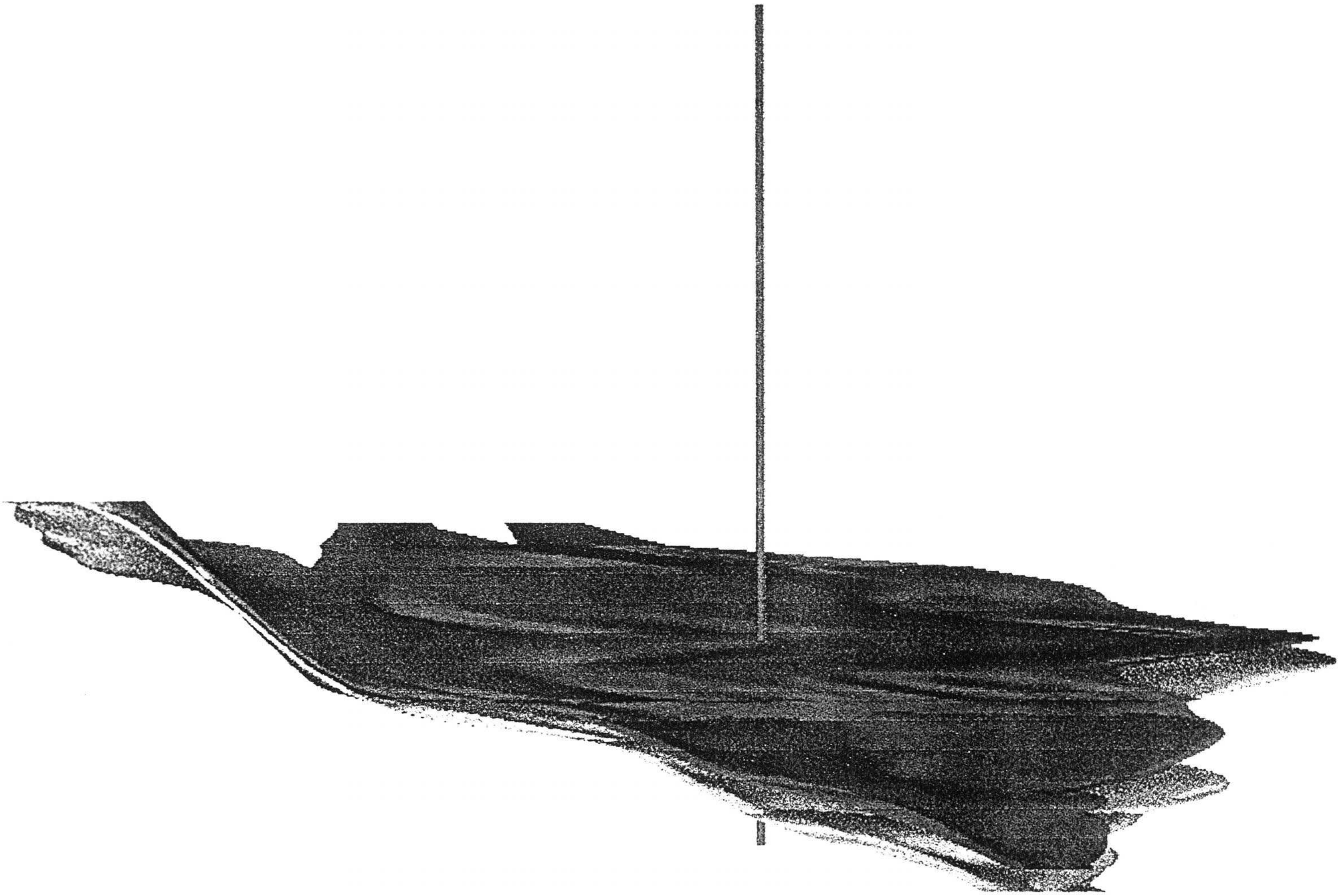


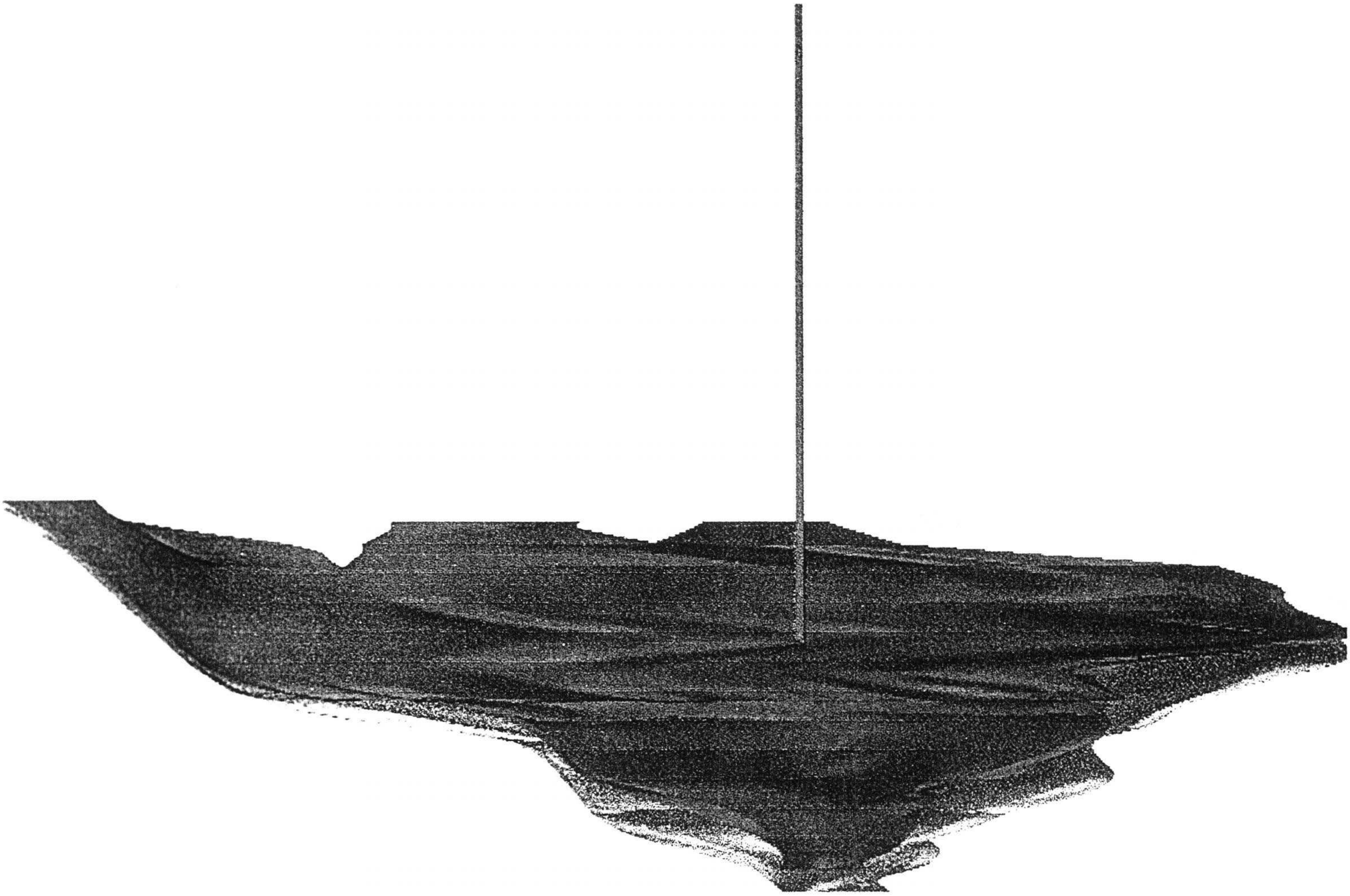


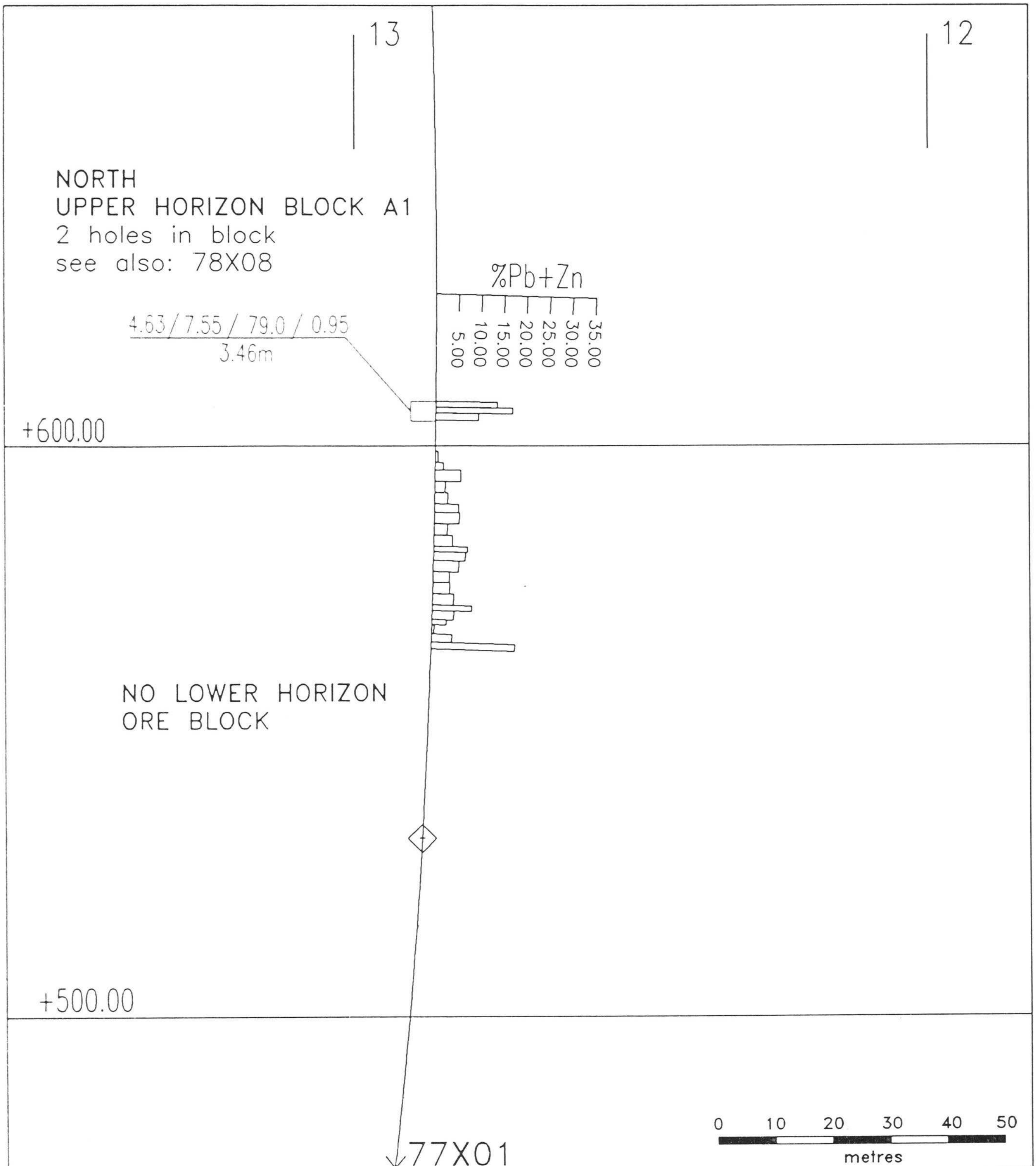




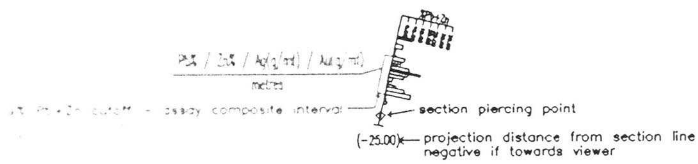








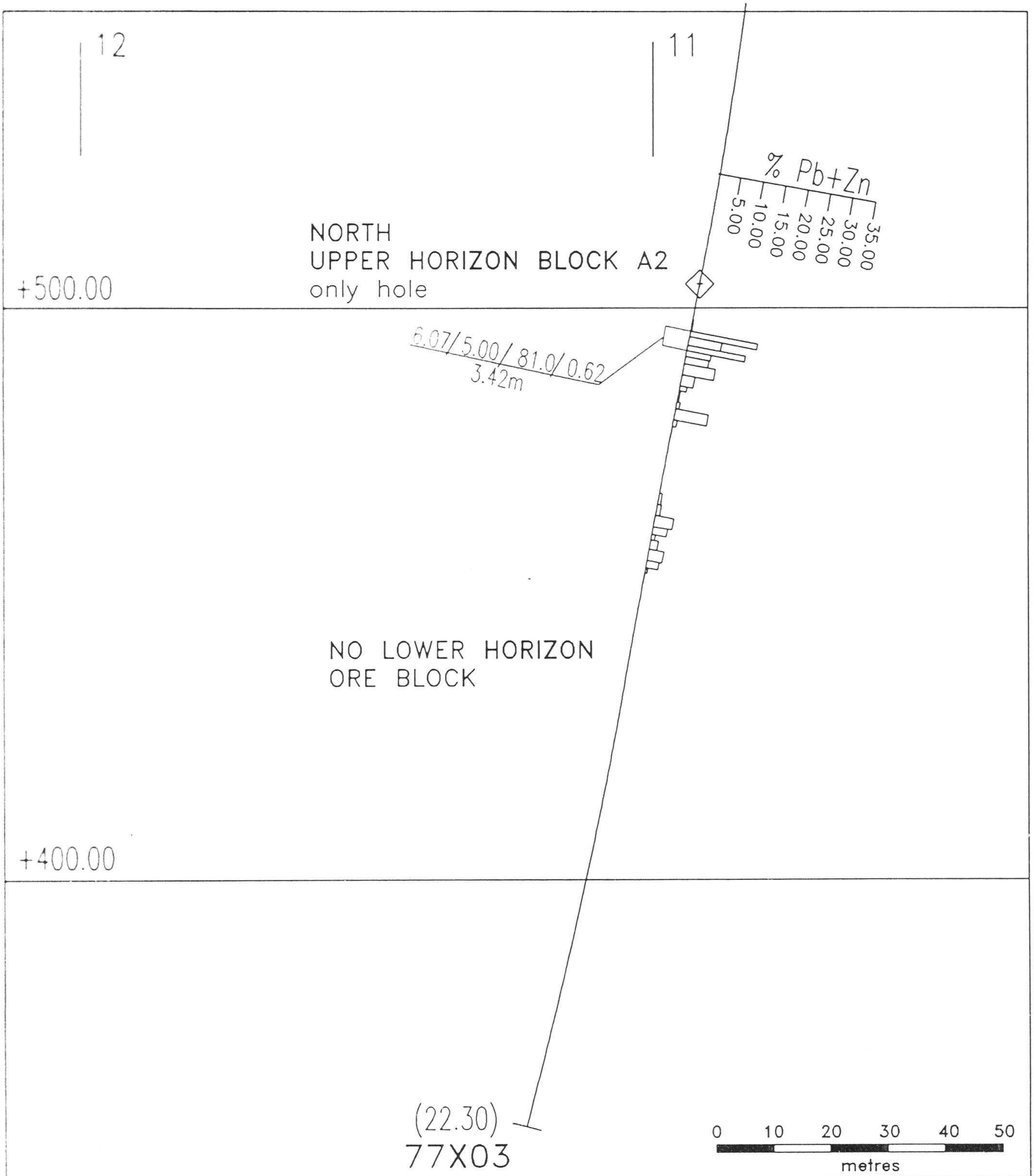
LEGEND



ANVIL RANGE MINING CORPORATION

GRIZZLY
HOLE# 77X01

SCALE:1:1,000	FILE: 76X21.DWG	DATE: 96.10.25
DRAWN: H.D.S.	DWG:	FIGURE



LEGEND

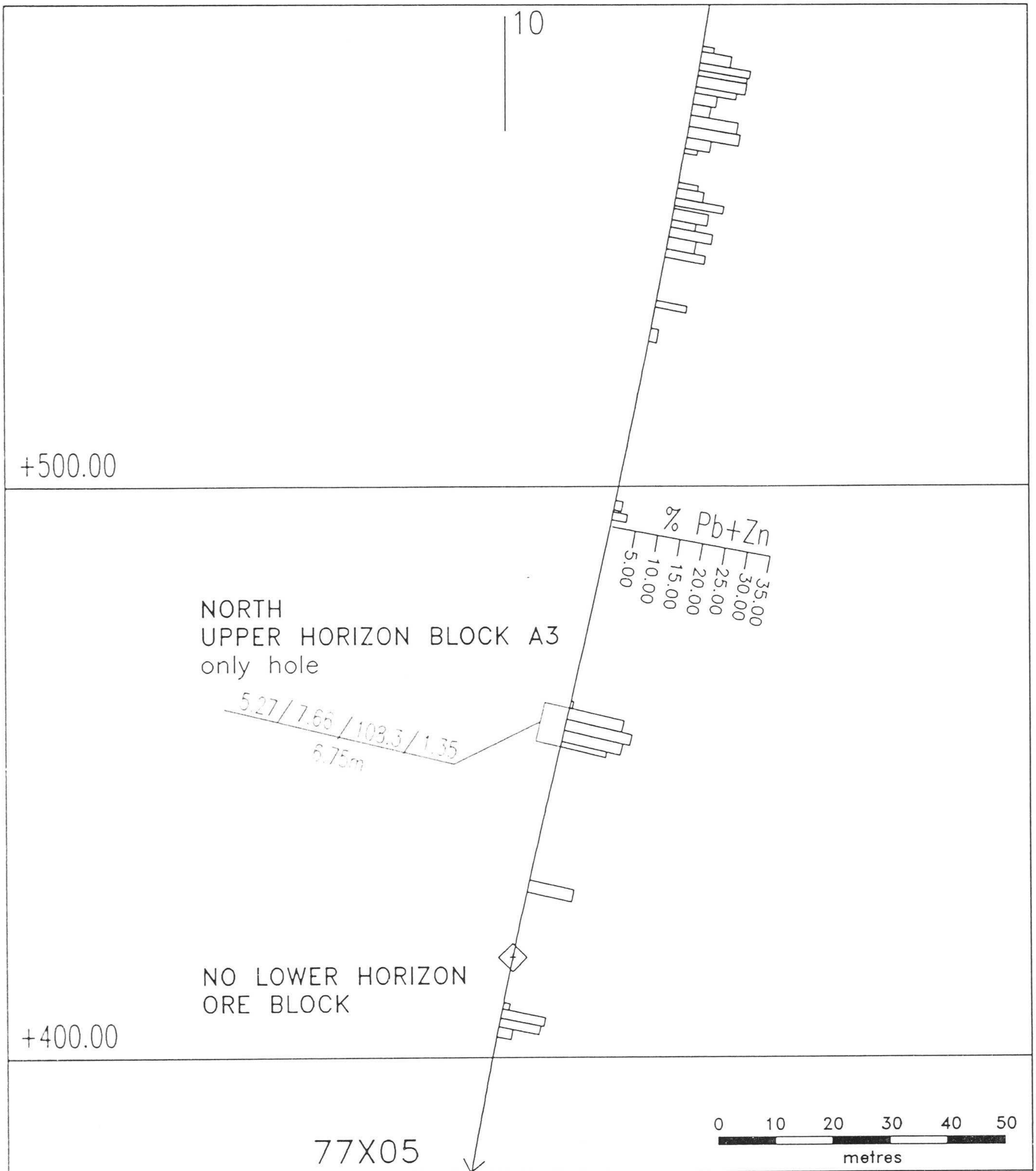
section piercing point

(-25.00) projection distance from section line negative if towards viewer

ANVIL RANGE MINING CORPORATION

GRIZZLY
HOLE# 77X03

SCALE:1:1,000	FILE: 77X03.DWG	DATE: 96.10.25
DRAWN: H.D.S.	DWG:	FIGURE



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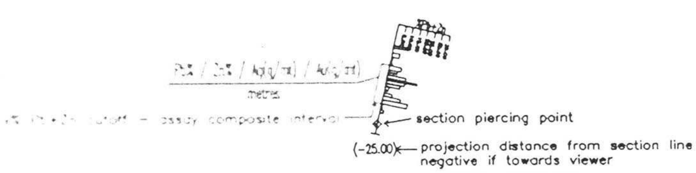

ANVIL RANGE MINING CORPORATION

GRIZZLY HOLE# 77X05

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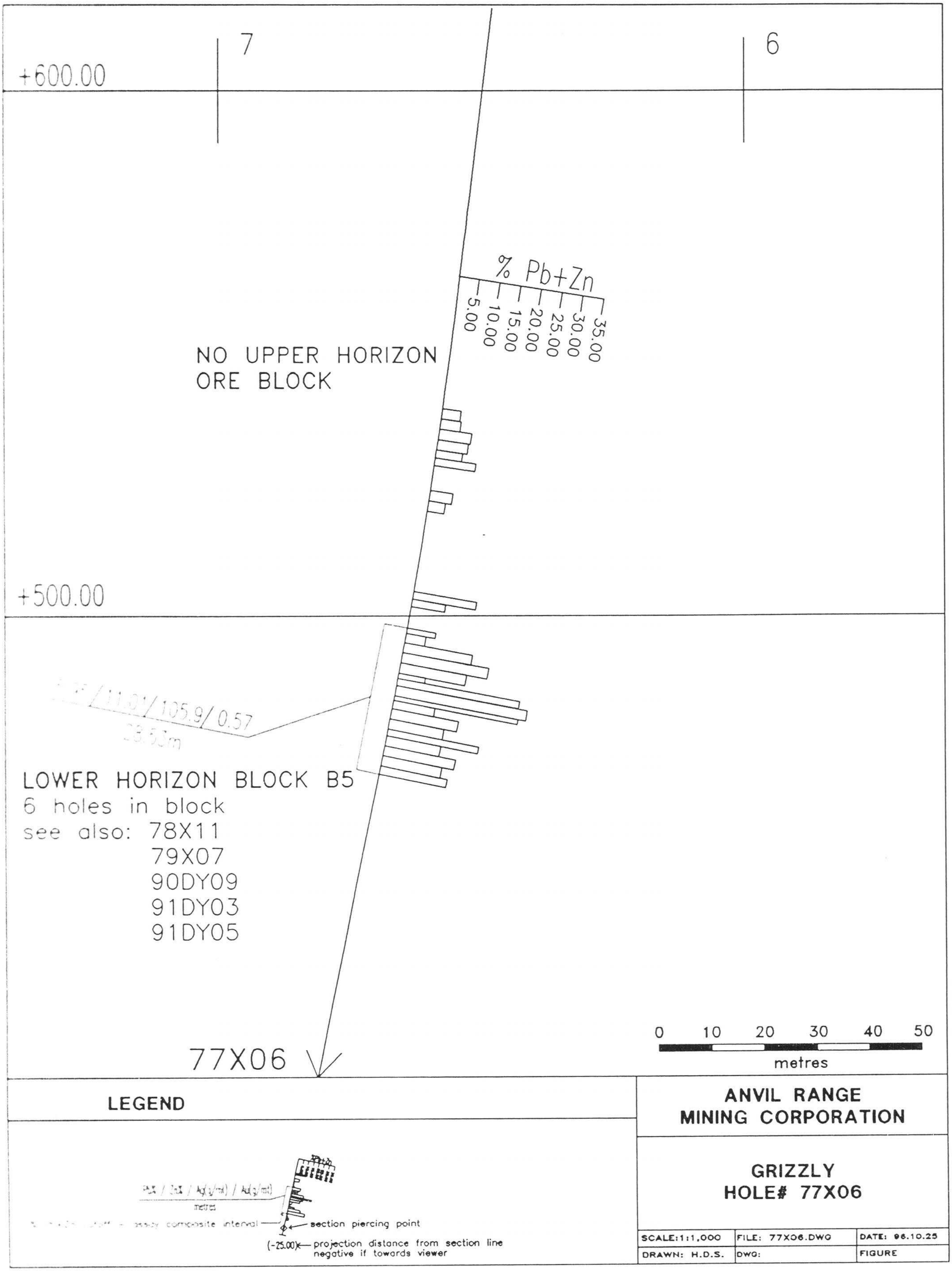
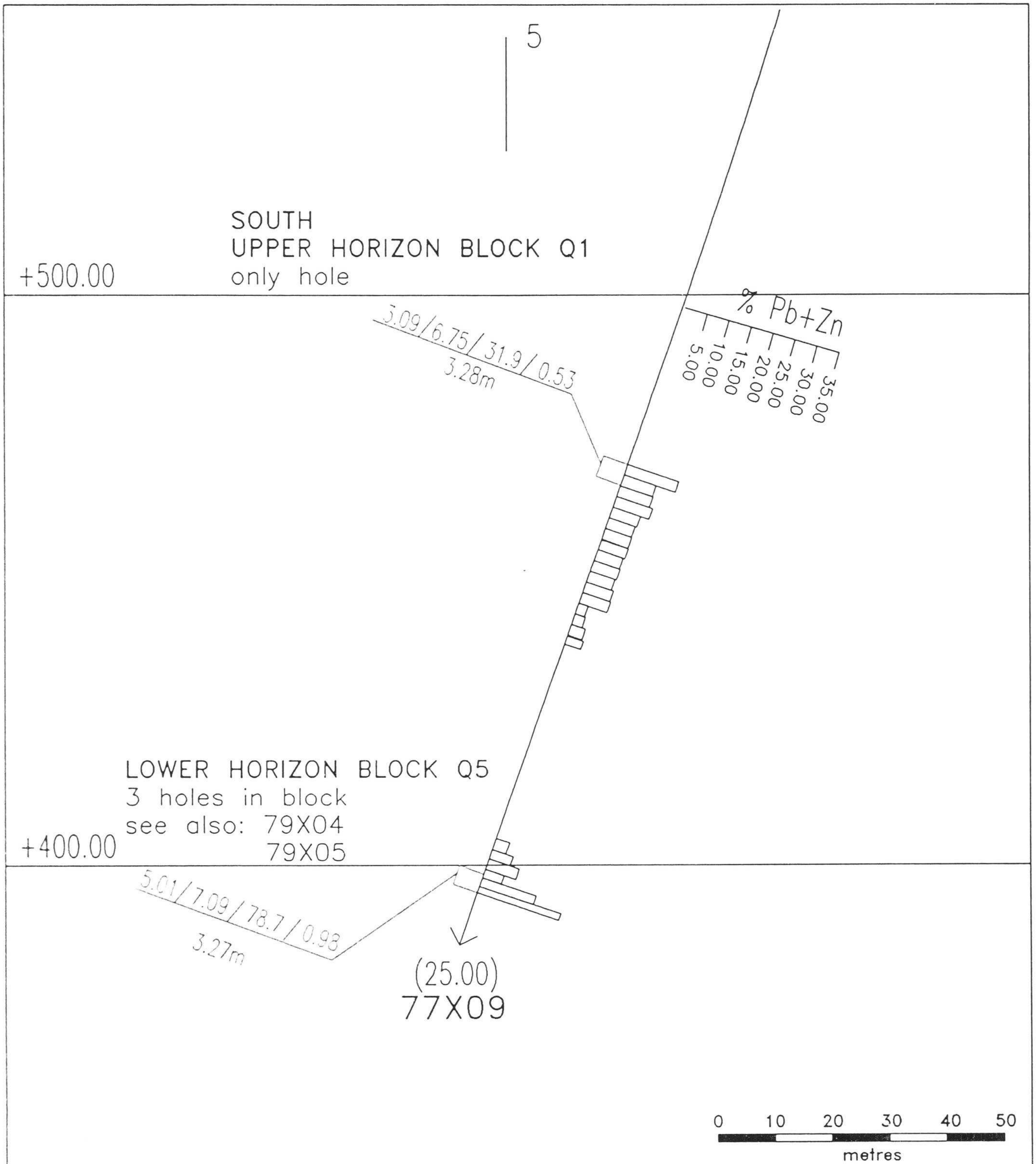
DRAWN: H.D.S.	DWG:	FIGURE



7
6
+600.00
% Pb+Zn
5.00
10.00
15.00
20.00
25.00
30.00
35.00
NO UPPER HORIZON
ORE BLOCK
+500.00
/105.9/ 0.57
LOWER HORIZON BLOCK B5
6 holes in block
see also: 78X11
79X07
90DY09
91DY03
91DY05
0 10 20 30 40 50
metres
77X06
LEGEND
metres
composite interval
section piercing point
(-25.00) projection distance from section line
negative if towards viewer
ANVIL RANGE
MINING CORPORATION
GRIZZLY
HOLE# 77X06
SCALE:1:1,000
FILE: 77X06.DWG
DATE: 96.10.25
DRAWN: H.D.S.
DWG:
FIGURE



LEGEND

section piercing point

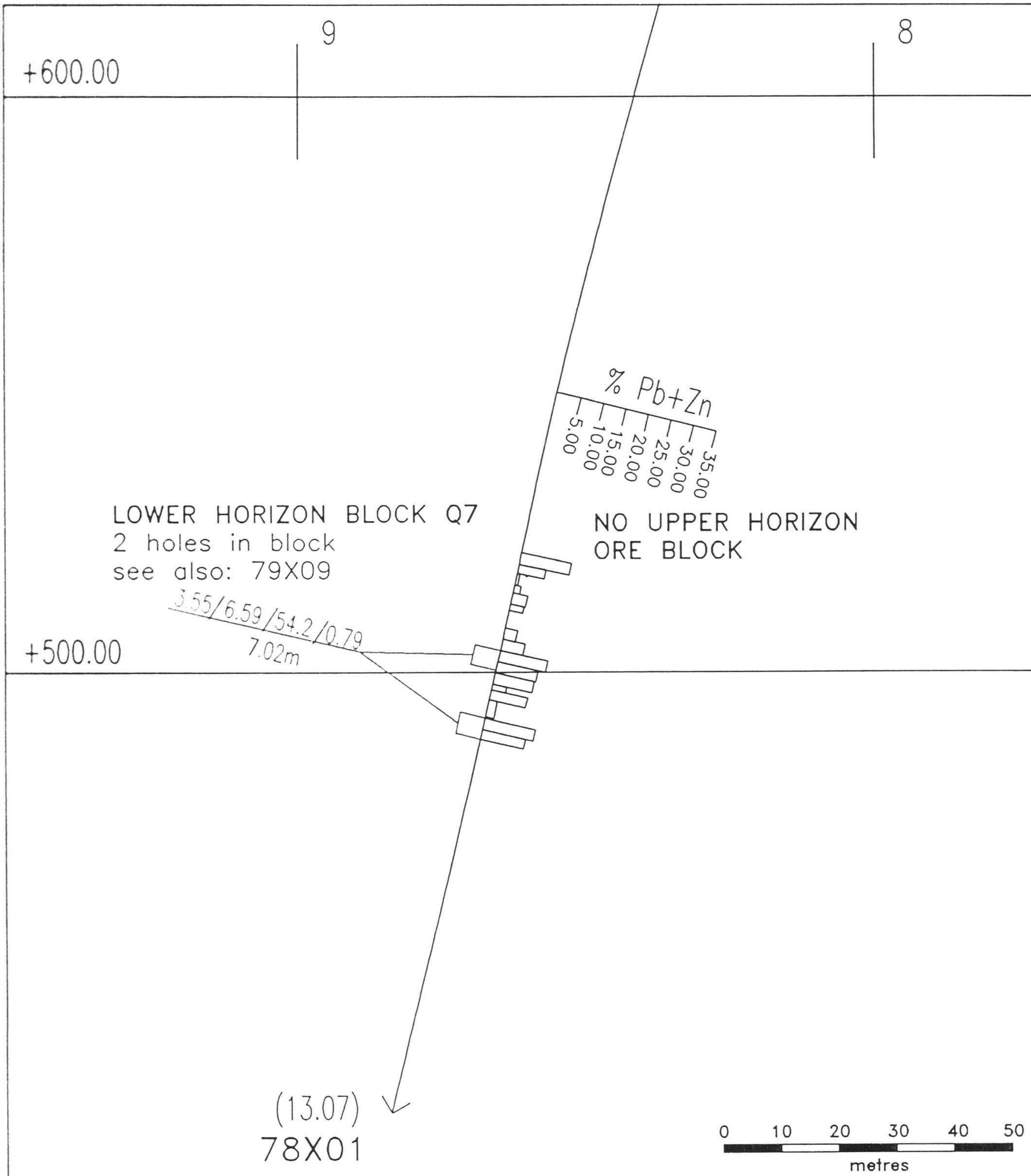
assay composite interval

(-25.00) projection distance from section line negative if towards viewer

ANVIL RANGE MINING CORPORATION

GRIZZLY
HOLE# 77X09

SCALE:1:1,000	FILE: 77X09.DWG	DATE: 96.10.25
DRAWN: H.D.S.	DWG:	FIGURE



LEGEND

Pb% / Zn% / Ag(g/mt) / Au(g/mt)
metres
9% Pb+Zn cutoff – assay composite interval
section piercing point
(-25.00) projection distance from section line negative if towards viewer

ANVIL RANGE MINING CORPORATION

GRIZZLY HOLE# 78X01

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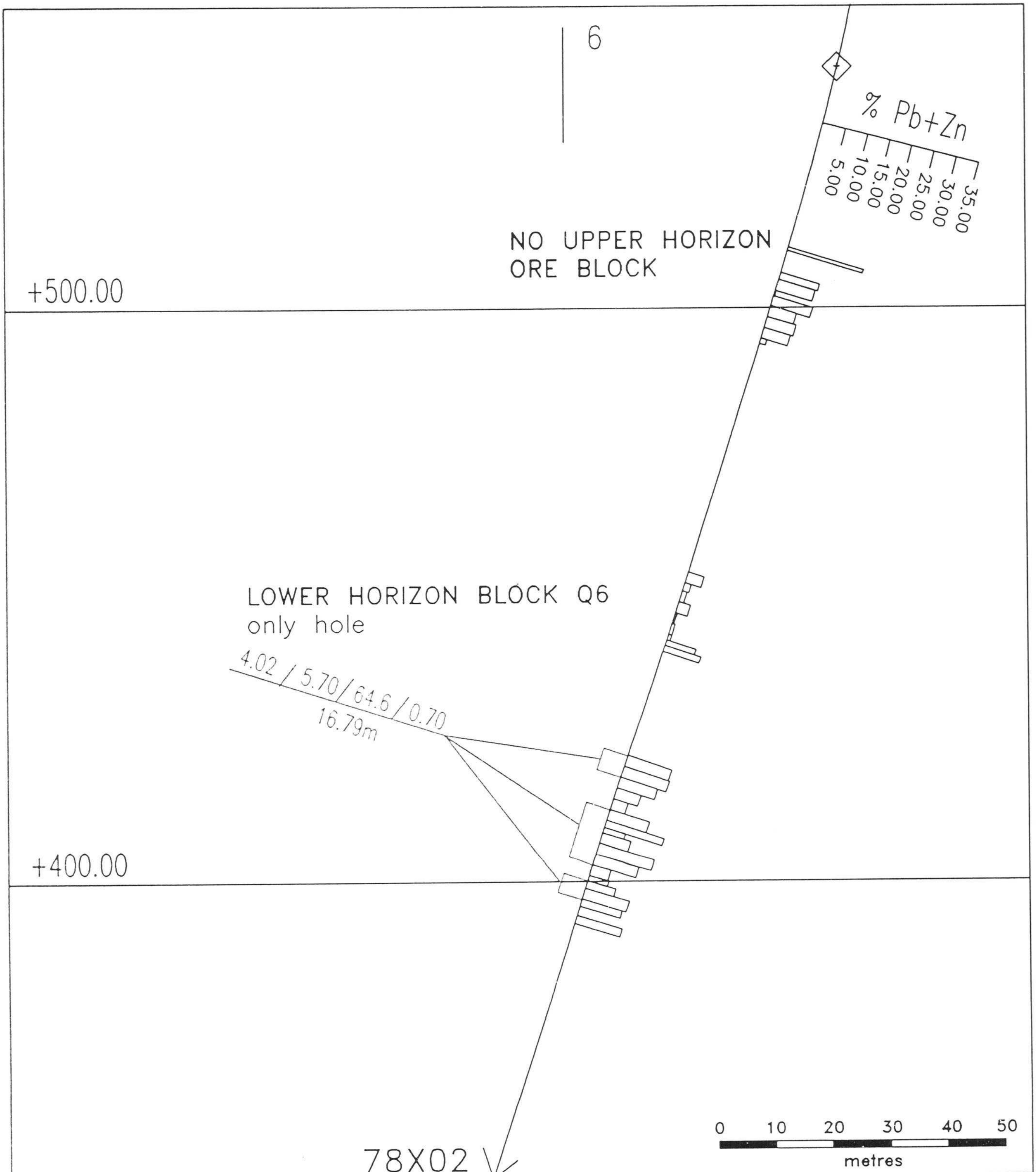
DRAWN: H.D.S.	DWG:	FIGURE



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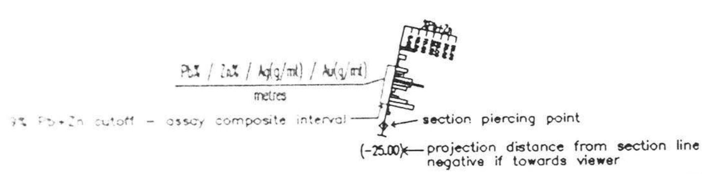

ANVIL RANGE MINING CORPORATION		
GRIZZLY HOLE# 78X02		
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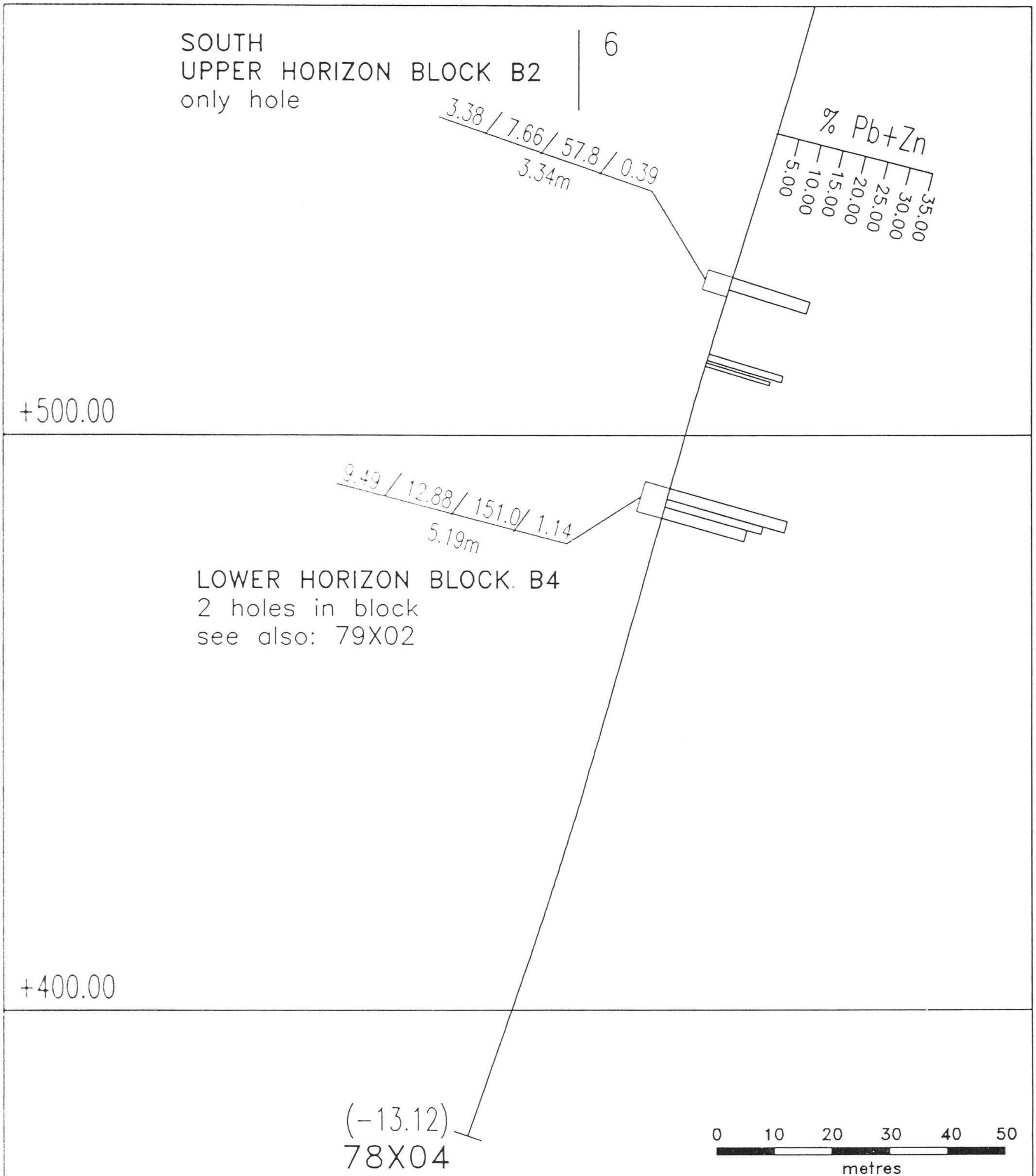
DRAWN: H.D.S.	DWG:	FIGURE



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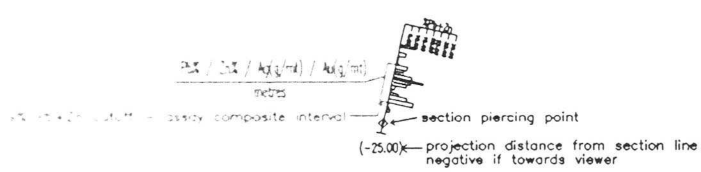

ANVIL RANGE MINING CORPORATION

GRIZZLY HOLE# 78X04

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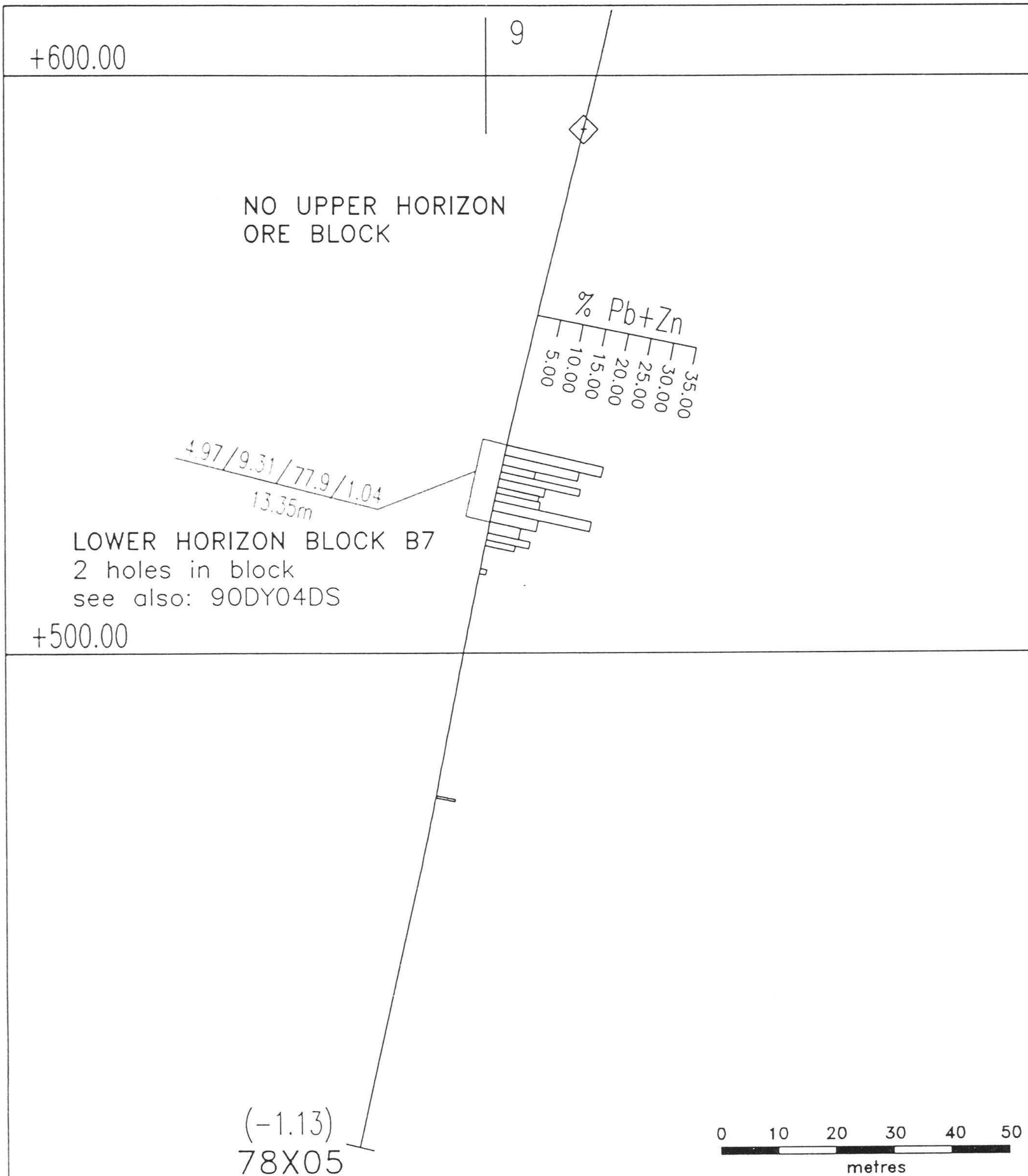
DRAWN: H.D.S.	DWG:	FIGURE



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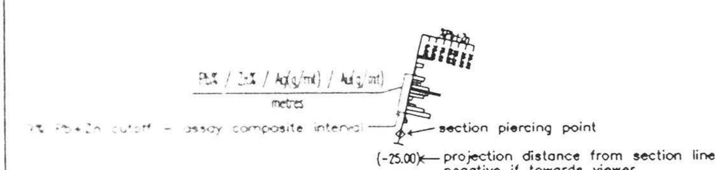

ANVIL RANGE MINING CORPORATION

GRIZZLY
HOLE# 78X05

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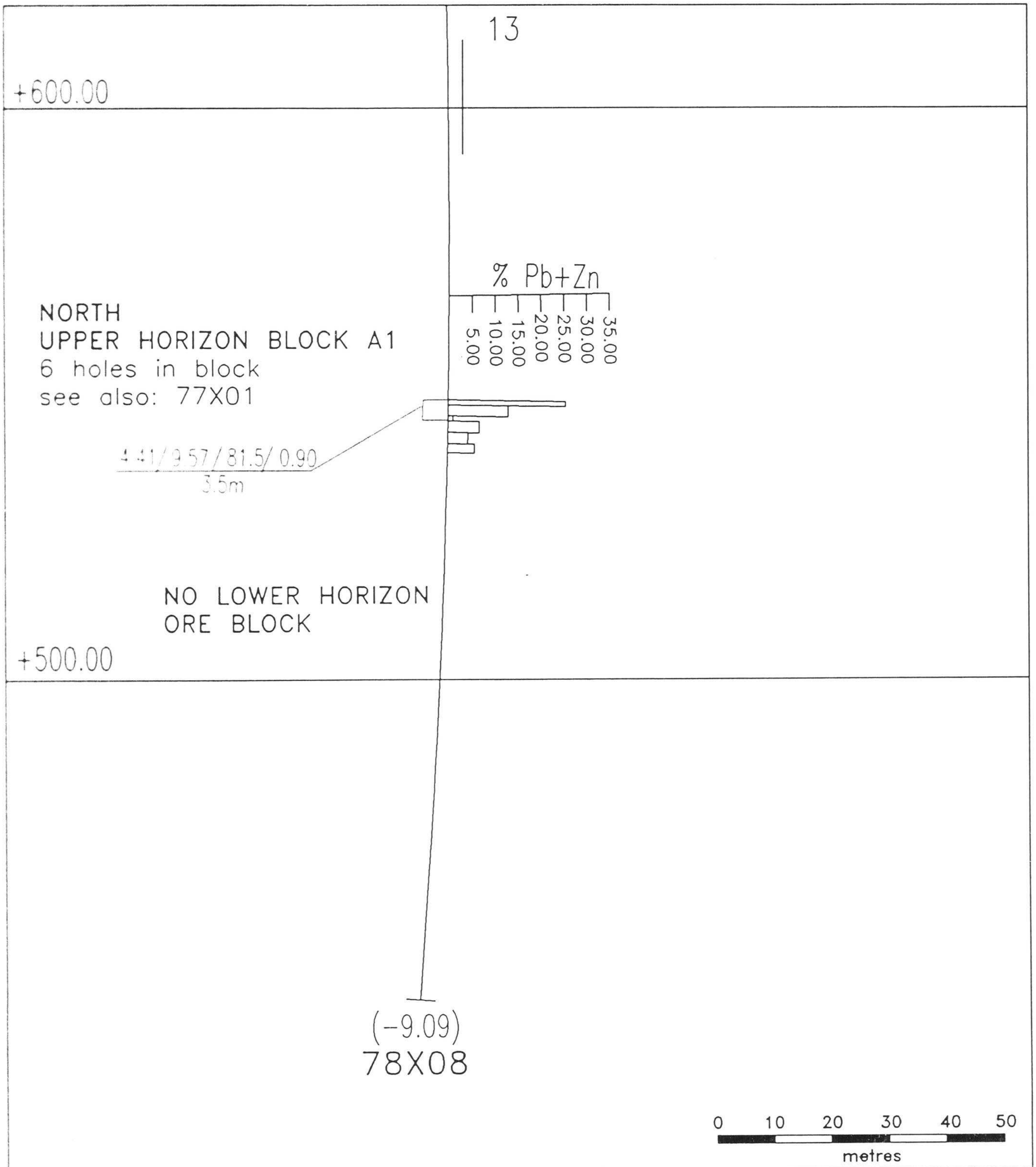
DRAWN: H.D.S.	DWG:	FIGURE



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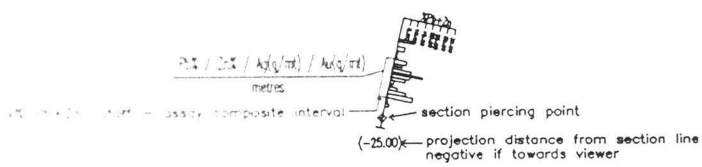

ANVIL RANGE MINING CORPORATION

GRIZZLY
HOLE# 78X08

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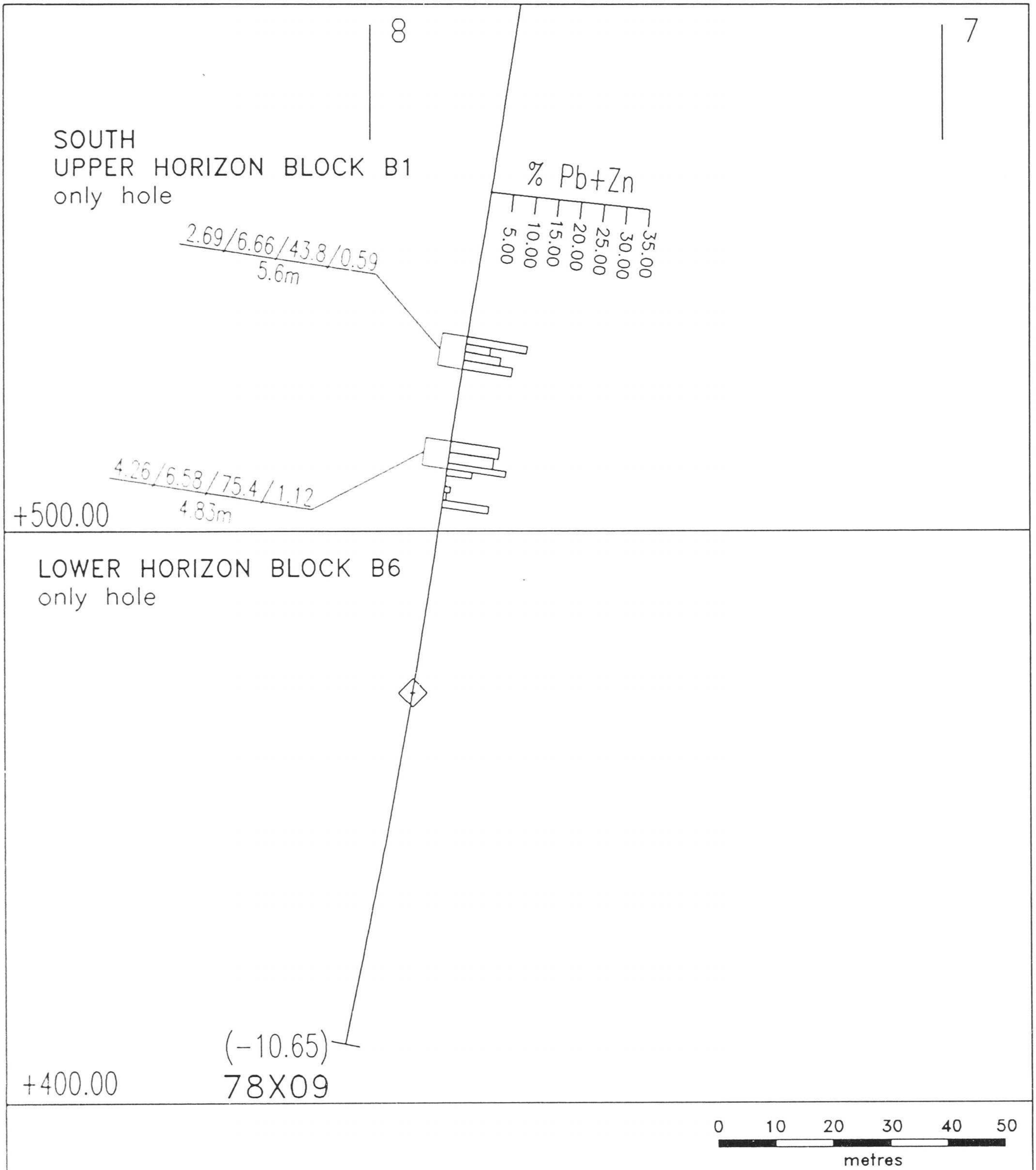
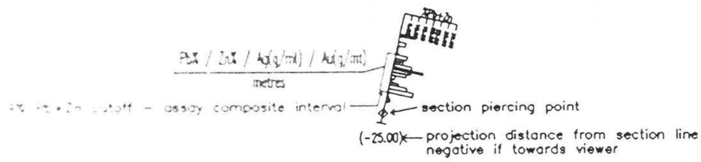
DRAWN: H.D.S.	DWG:	FIGURE



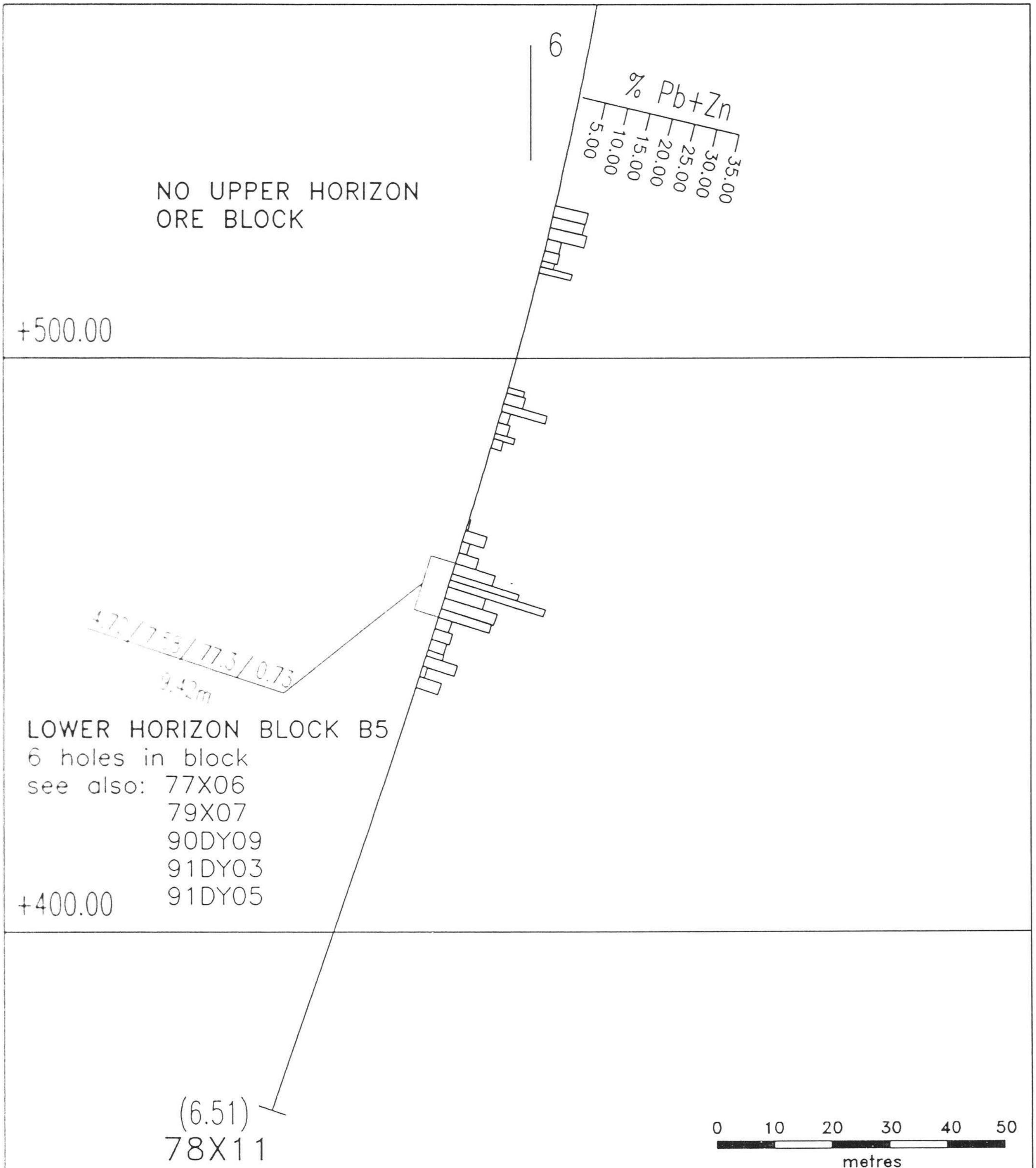
LEGEND



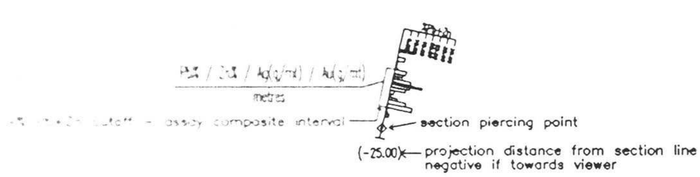
ANVIL RANGE MINING CORPORATION

GRIZZLY
HOLE# 78X09

SCALE:1:1,000	FILE: 78X09.DWG	DATE: 96.10.25
DRAWN: H.D.S.	DWG:	FIGURE



LEGEND



ANVIL RANGE MINING CORPORATION

GRIZZLY
HOLE# 78X11

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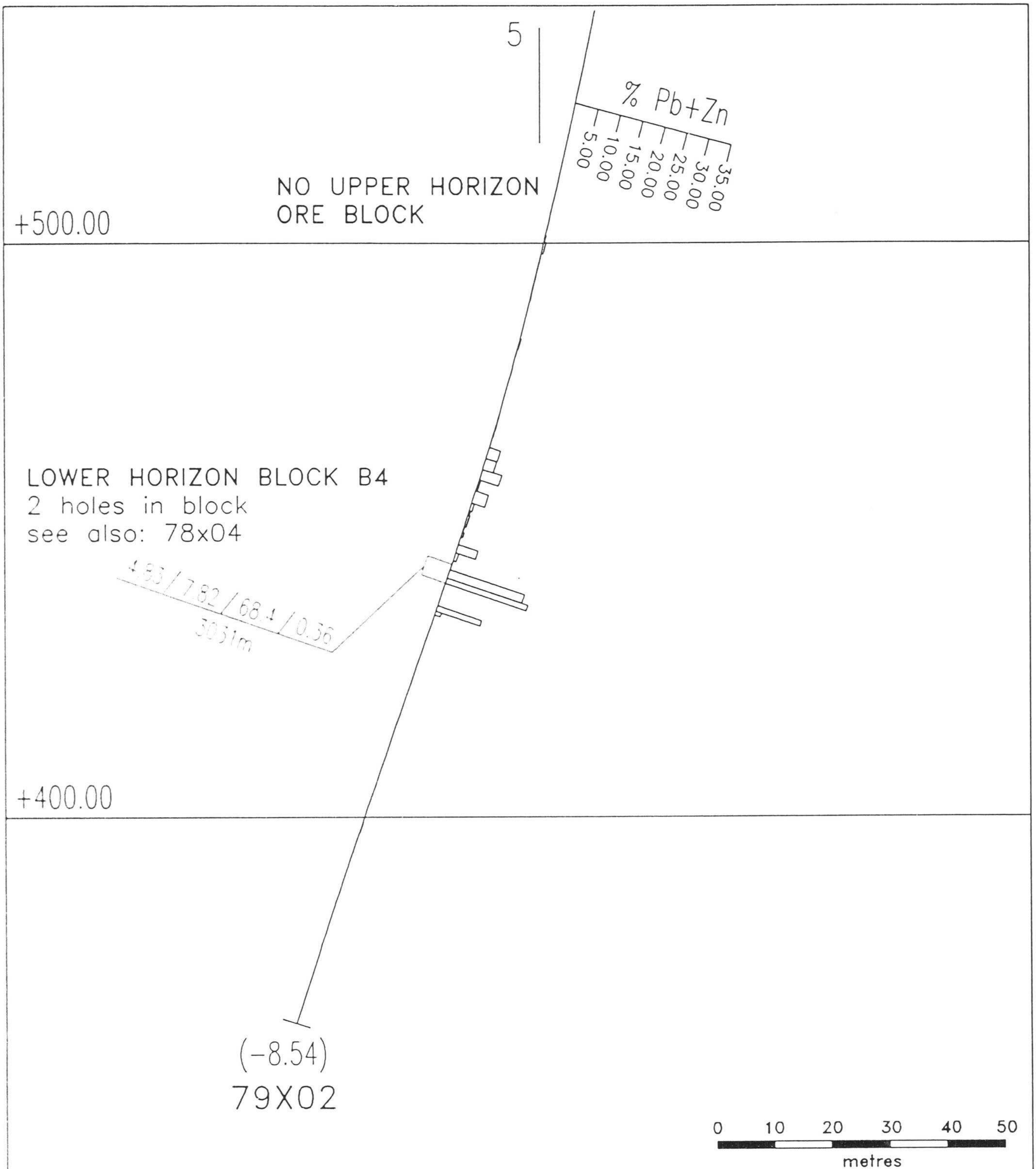
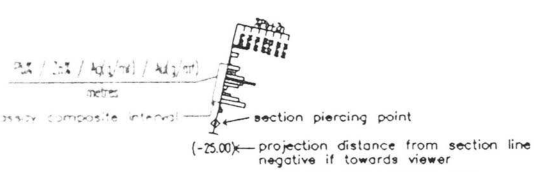
DRAWN: H.D.S.	DWG:	FIGURE



LEGEND



ANVIL RANGE MINING CORPORATION

GRIZZLY
HOLE# 79X02

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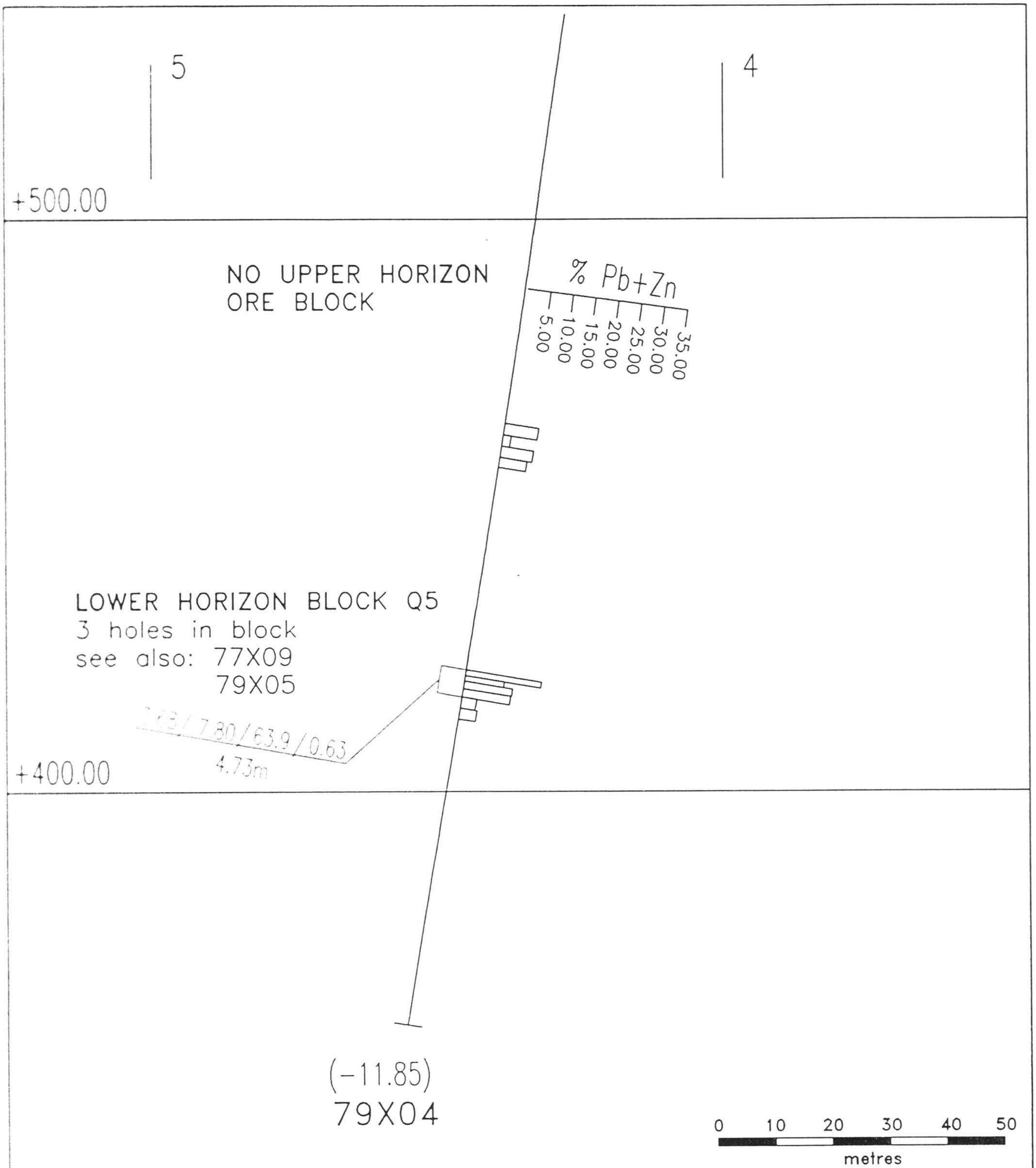
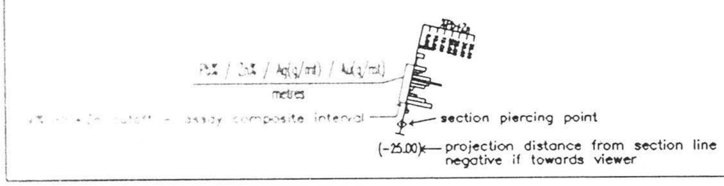
DRAWN: H.D.S.	DWG:	FIGURE



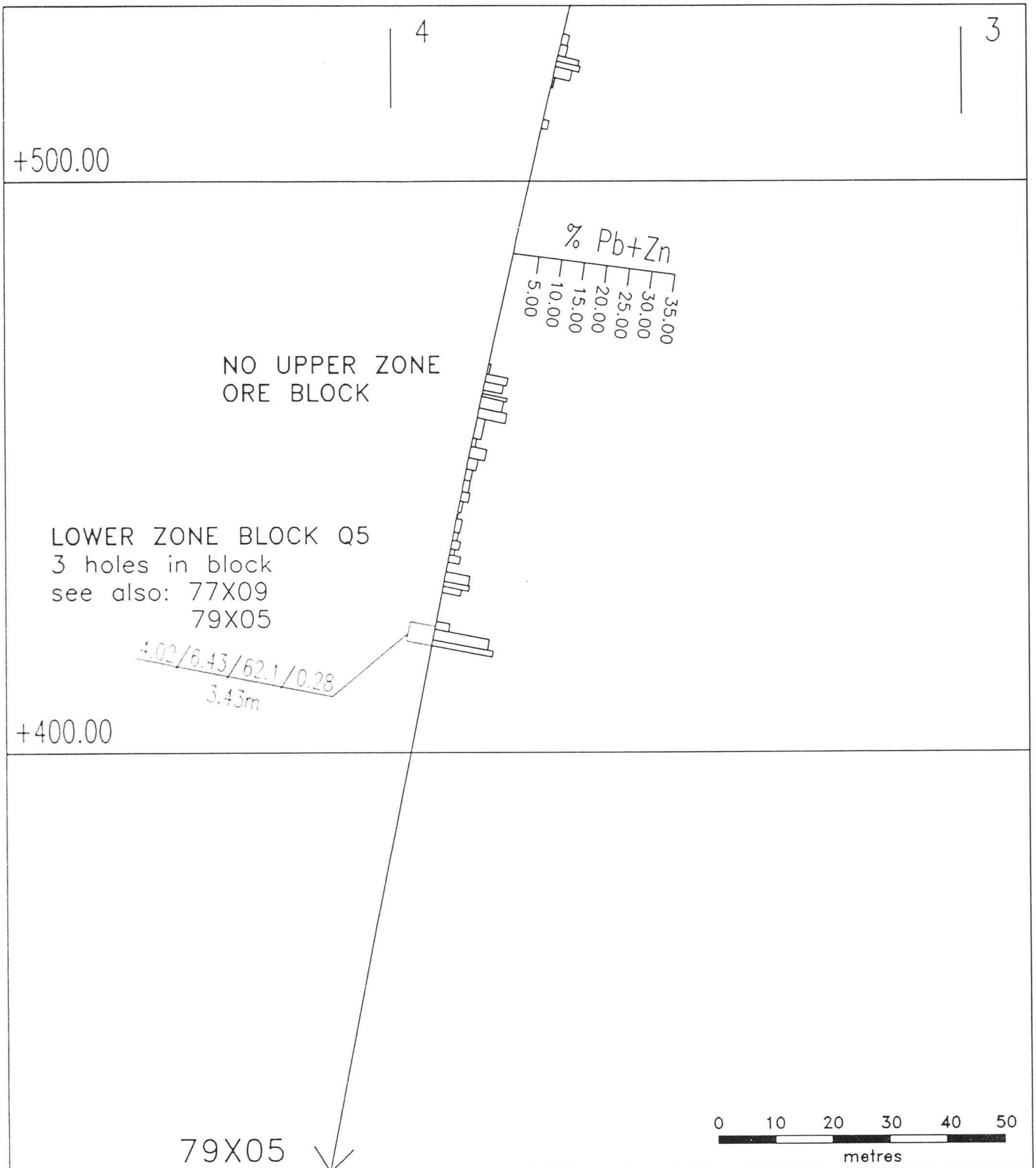
LEGEND



ANVIL RANGE MINING CORPORATION

GRIZZLY
HOLE# 79X04

SCALE:1:1,000	FILE: 79X04.DWG	DATE: 96.10.25
DRAWN: H.D.S.	DWG:	FIGURE



0 10 20 30 40 50
metres

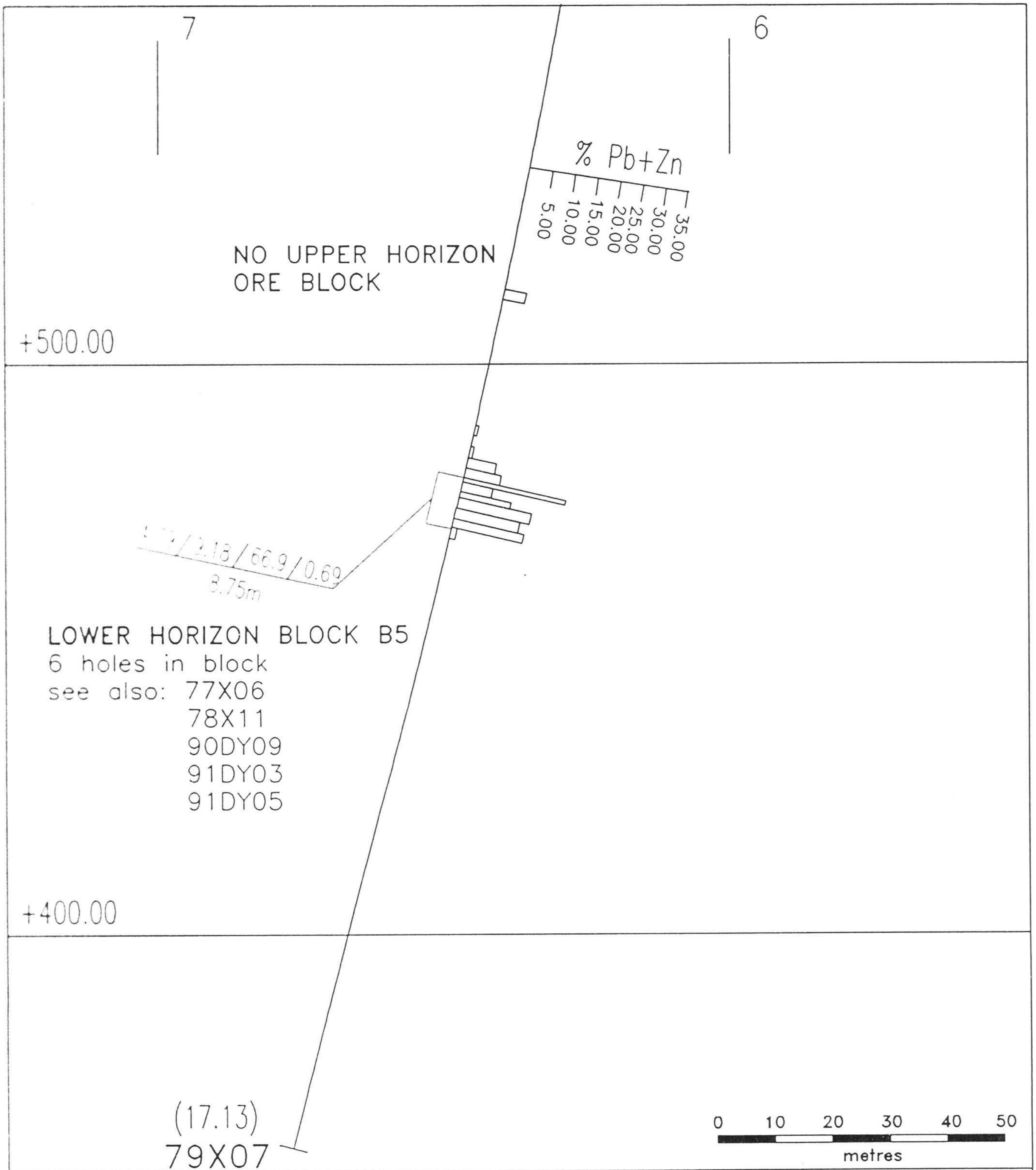
LEGEND

Pb% / Zn% / Ag(g/mt) / Au(g/mt)
metres
4% Pb+Zn cutoff - assay composite interval
section piercing point
(-25.00) projection distance from section line negative if towards viewer

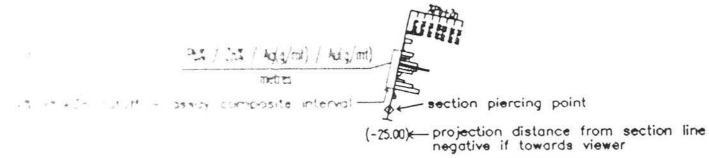
ANVIL RANGE MINING CORPORATION

GRIZZLY HOLE# 79X05

SCALE:1:1,000	FILE: 79X05.DWG	DATE: 96.10.25
DRAWN: H.D.S.	DWG:	FIGURE



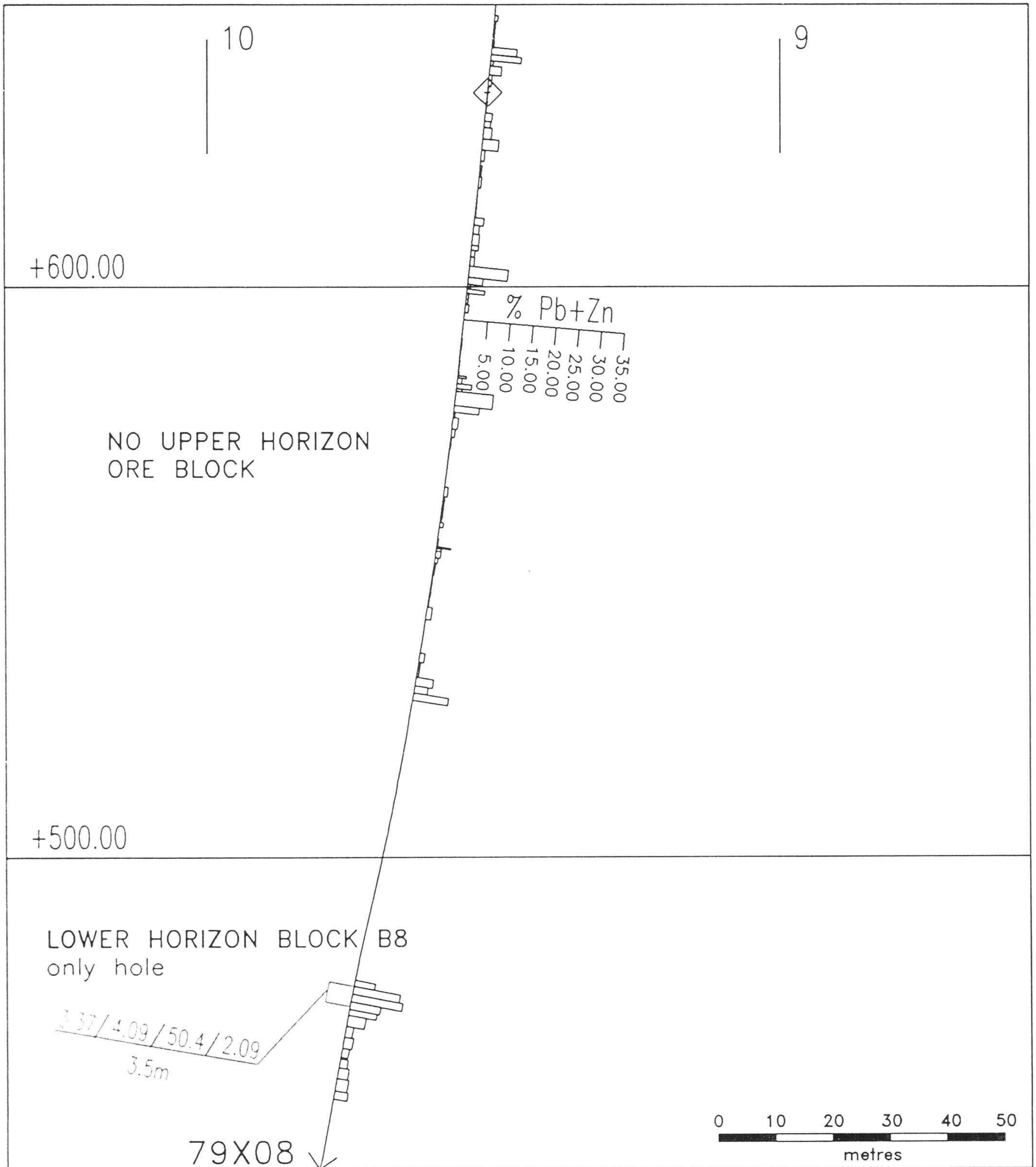
LEGEND



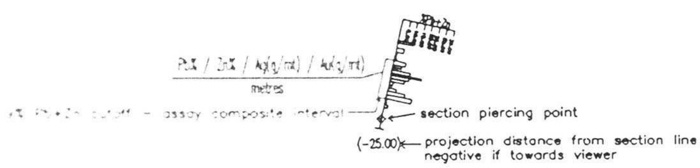
ANVIL RANGE MINING CORPORATION

GRIZZLY
HOLE# 79X07

SCALE:1:1,000	FILE: 79X07.DWG	DATE: 96.10.25
DRAWN: H.D.S.	DWG:	FIGURE



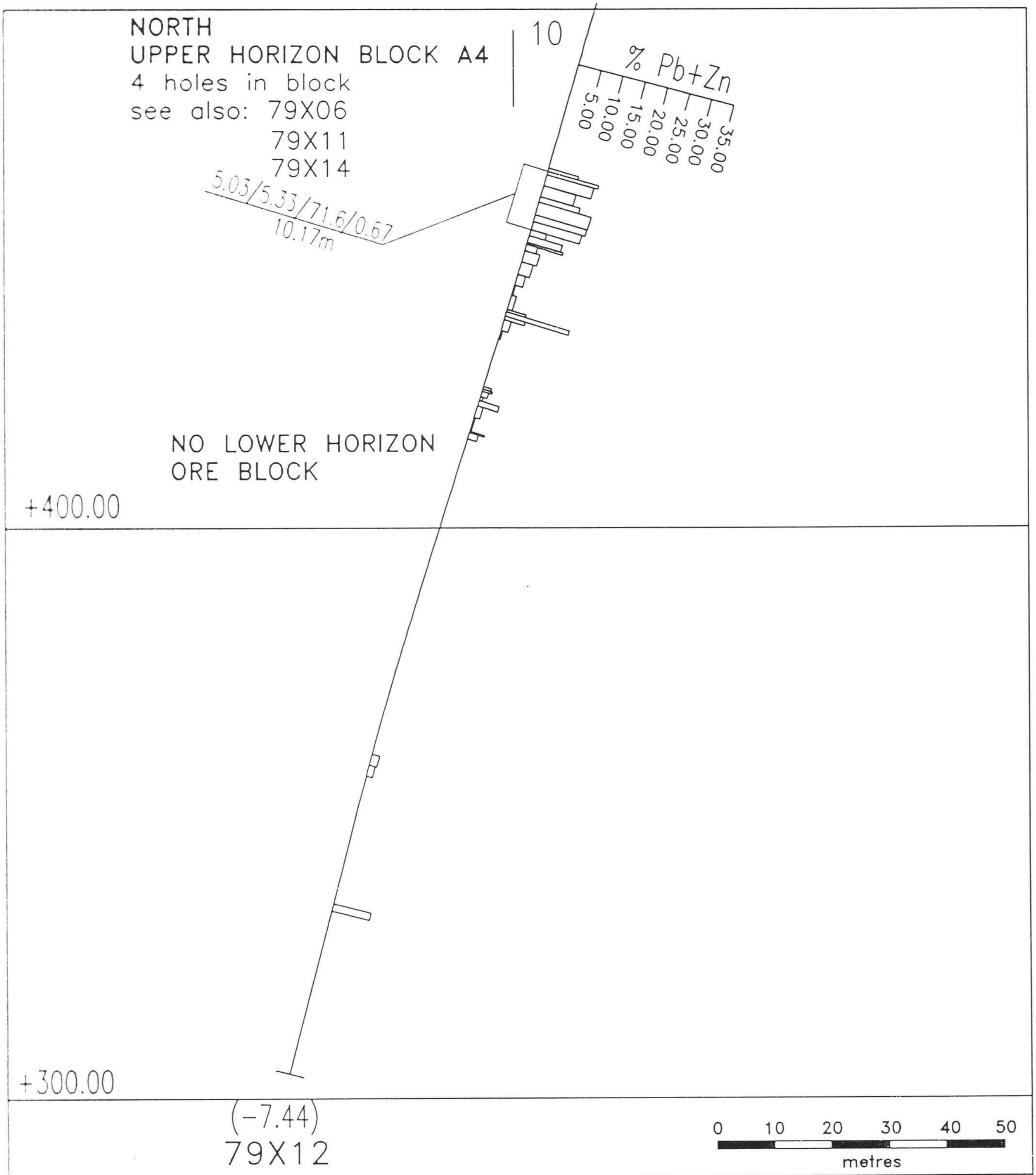
LEGEND



ANVIL RANGE MINING CORPORATION

GRIZZLY
HOLE# 79X08

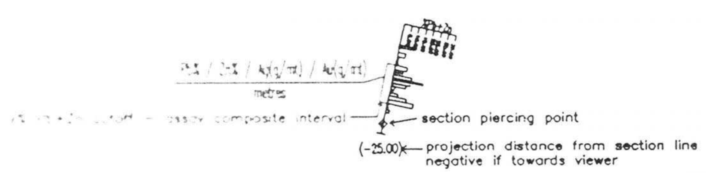
SCALE:1:1,000	FILE: 79X08.DWG	DATE: 96.10.25
DRAWN: H.D.S.	DWG:	FIGURE



0 10 20 30 40 50

metres

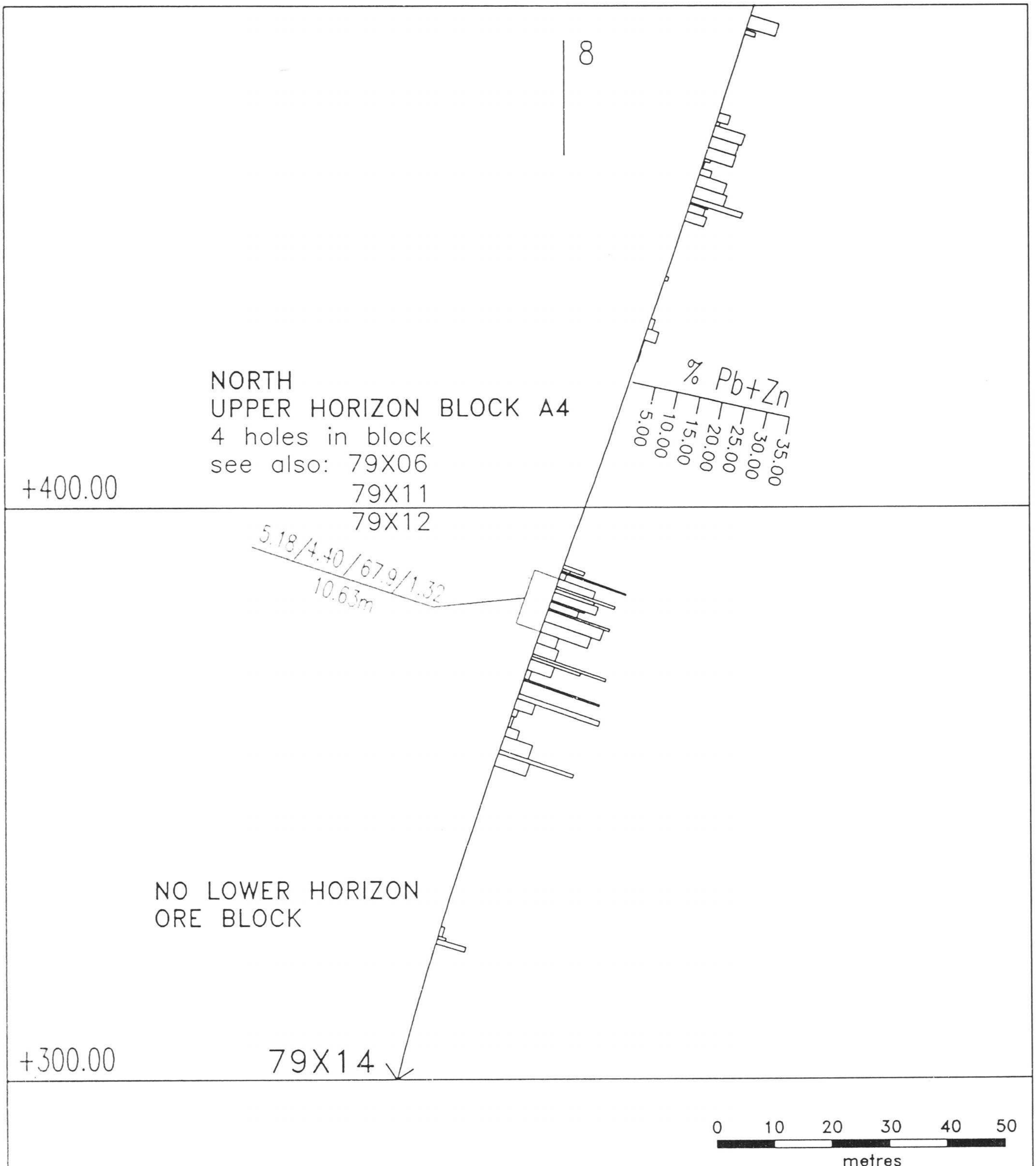
LEGEND



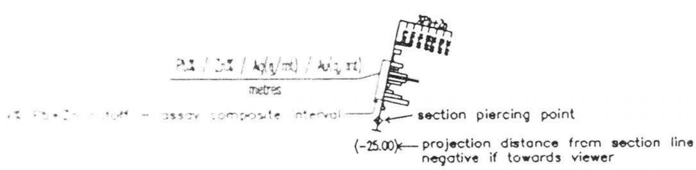
ANVIL RANGE MINING CORPORATION

GRIZZLY HOLE# 79X12

SCALE:1:1,000	FILE: 79X12.DWG	DATE: 96.10.25
DRAWN: H.D.S.	DWG:	FIGURE



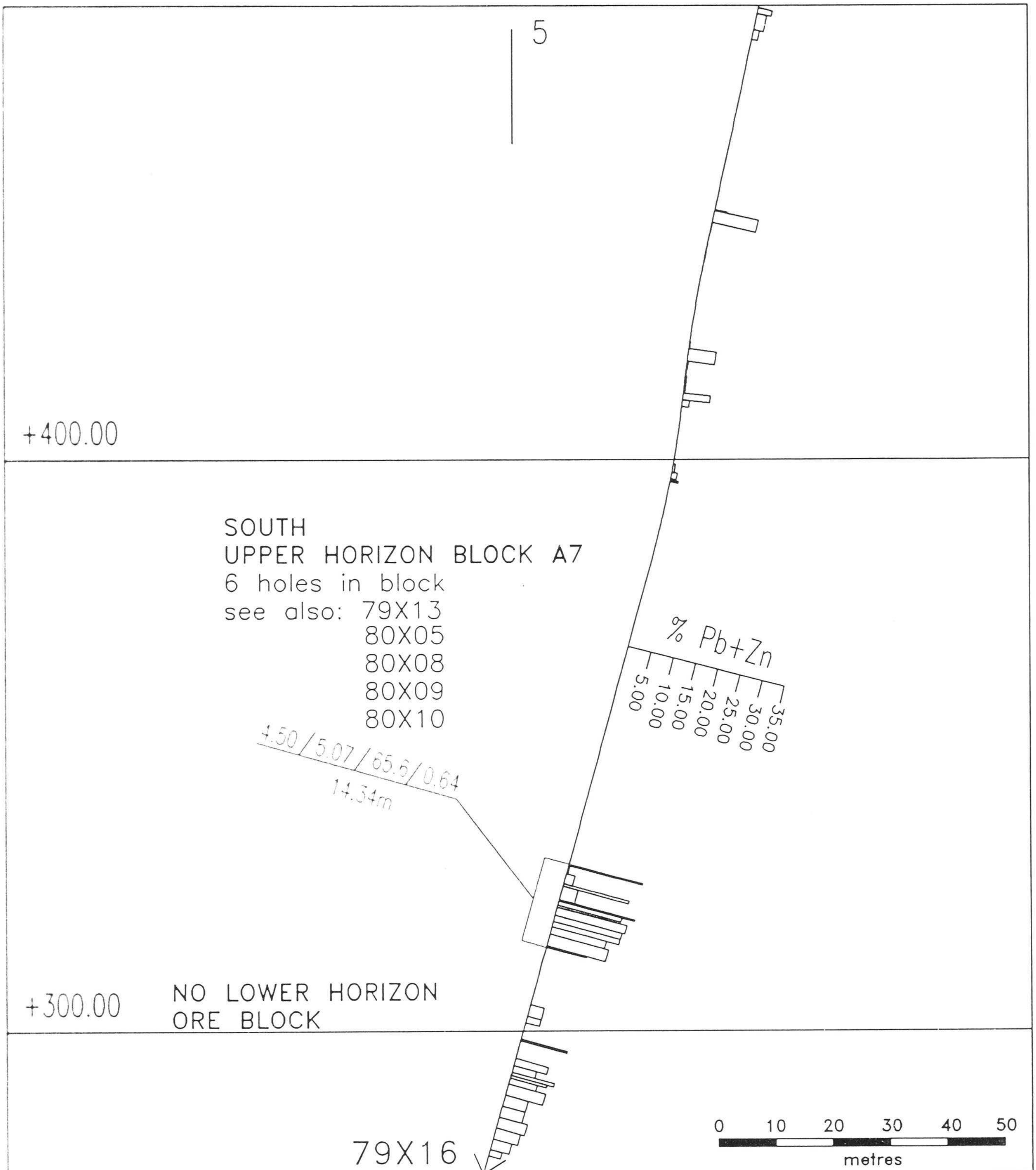
LEGEND



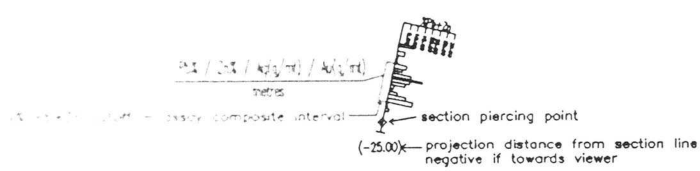
ANVIL RANGE MINING CORPORATION

GRIZZLY
HOLE# 79X14

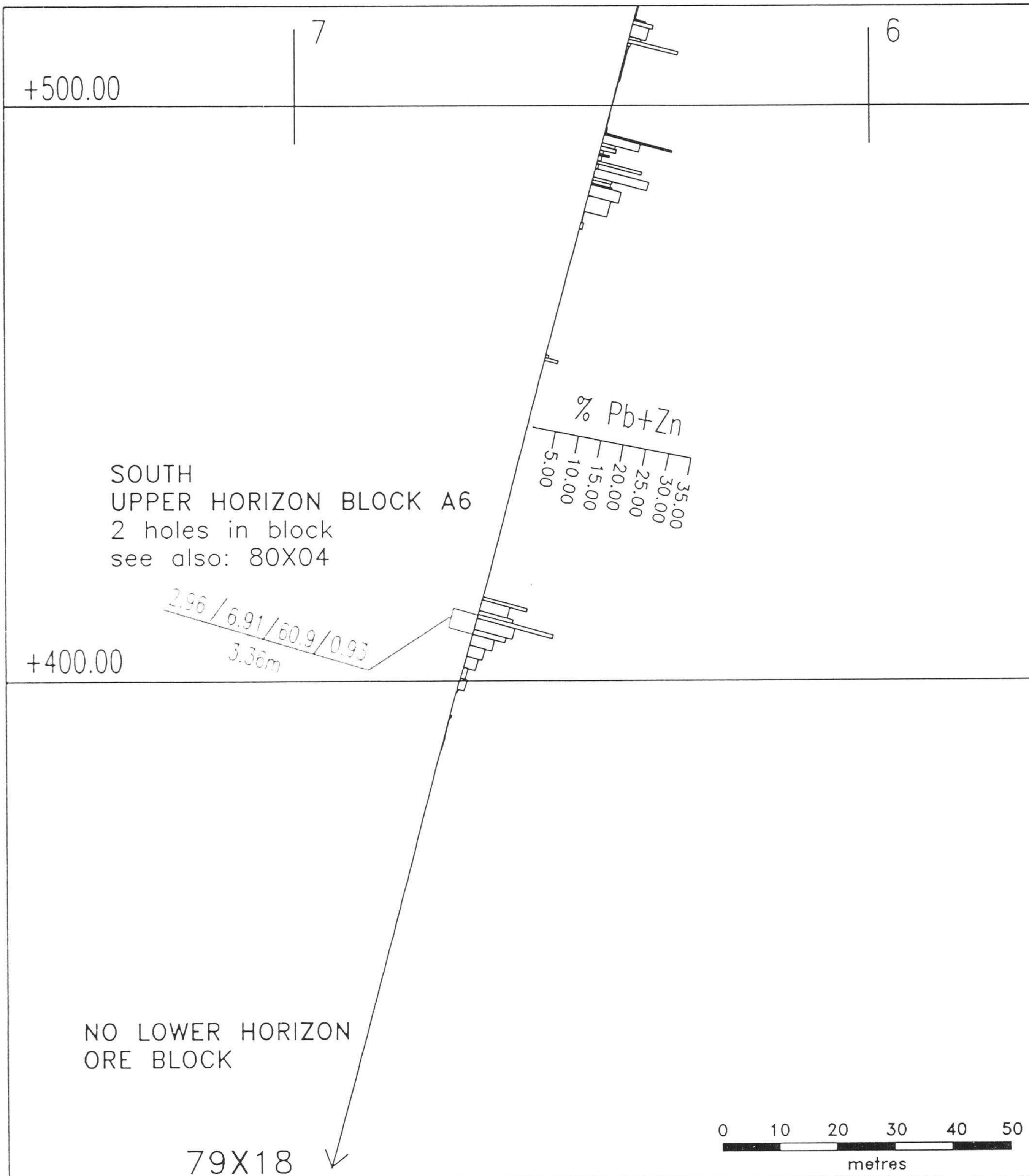
SCALE:1:1,000	FILE: 79X14.DWG	DATE: 96.10.25
DRAWN: H.D.S.	DWG:	FIGURE



LEGEND



ANVIL RANGE MINING CORPORATION		
GRIZZLY HOLE# 79X16		
SCALE: 1:1,000	FILE: 79X16.DWG	DATE: 96.10.25
DRAWN: H.D.S.	DWG:	FIGURE



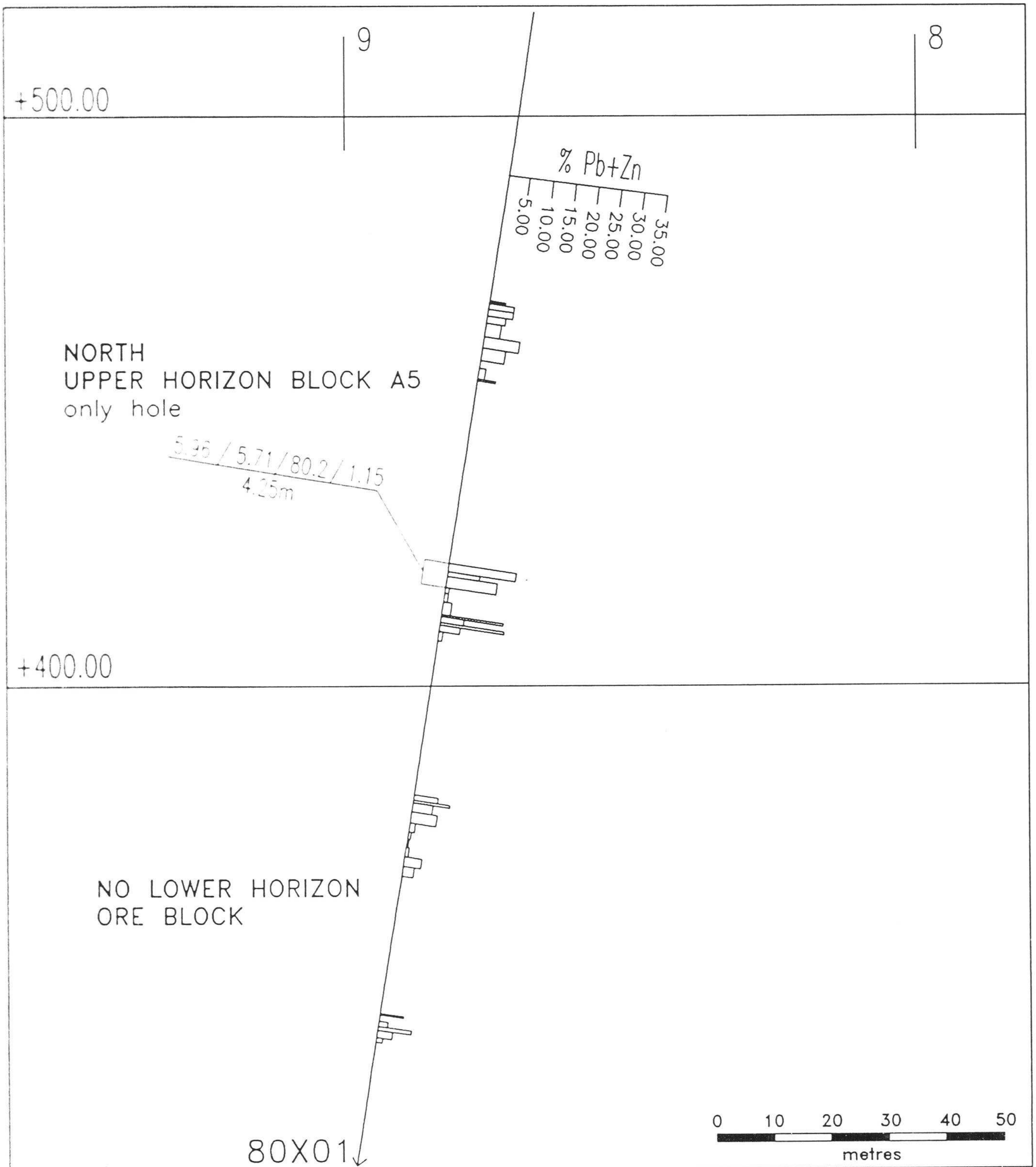
LEGEND

Pb% / Zn% / Ag(g/mt) / Au(g/mt)
metres
% Pb+Zn cutoff – assay composite interval
section piercing point
(-25.00) projection distance from section line negative if towards viewer

ANVIL RANGE MINING CORPORATION

GRIZZLY
HOLE# 79X18

SCALE: 1:1,000	FILE: 79X18.DWG	DATE: 96.10.25
DRAWN: H.D.S.	DWG:	FIGURE



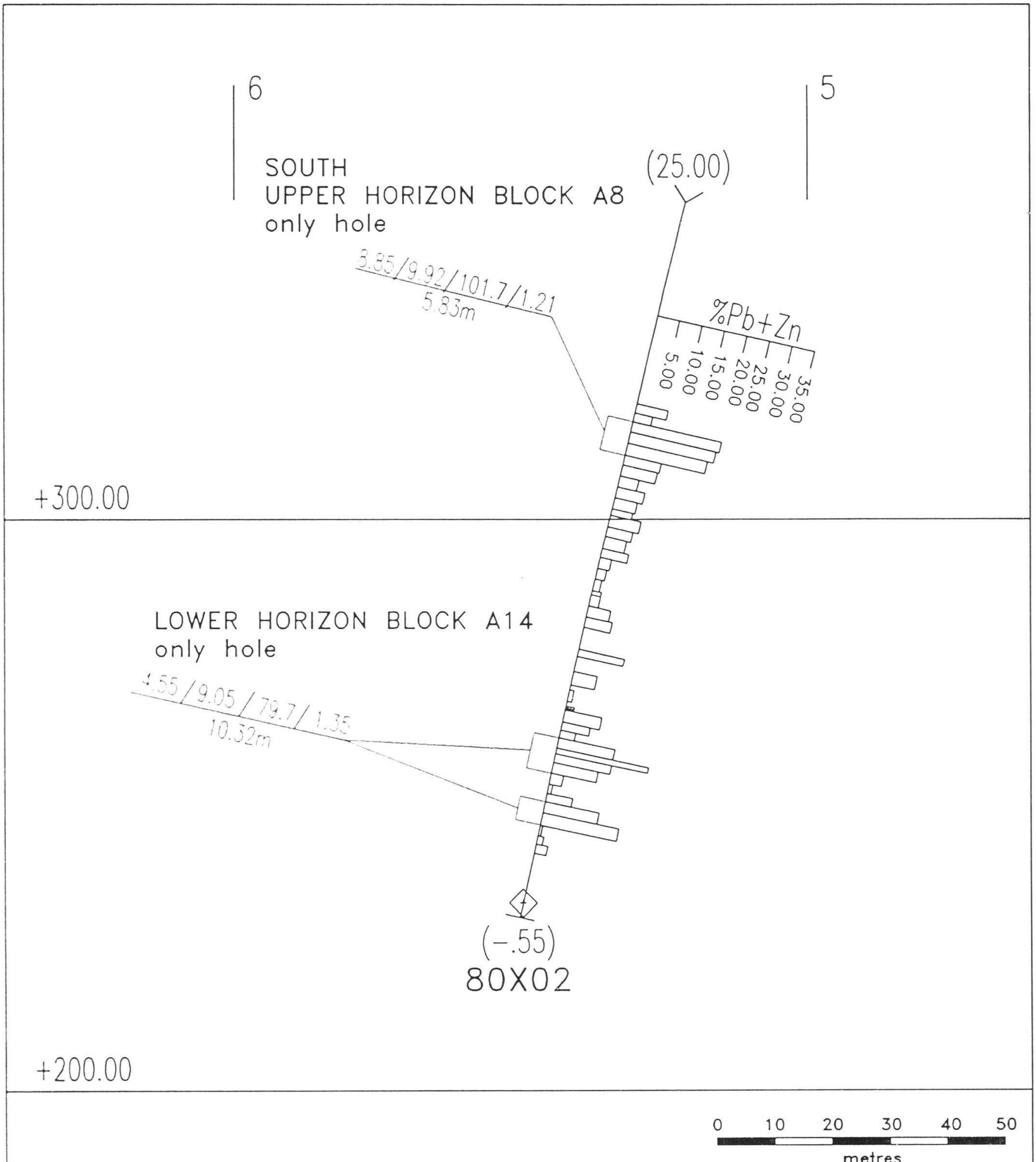
LEGEND

Pb% / Zn% / Ag(g/mt) / Au(g/mt)
metres
composite interval
section piercing point
(-25.00) projection distance from section line
negative if towards viewer

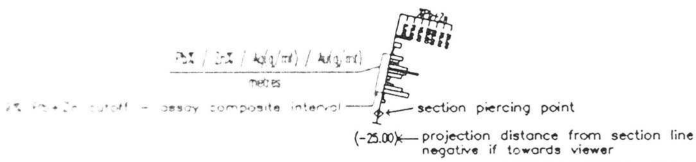
ANVIL RANGE MINING CORPORATION

GRIZZLY
HOLE# 80X01

SCALE: 1:1,000	FILE: 80X01.DWG	DATE: 96.10.25
DRAWN: H.D.S.	DWG:	FIGURE



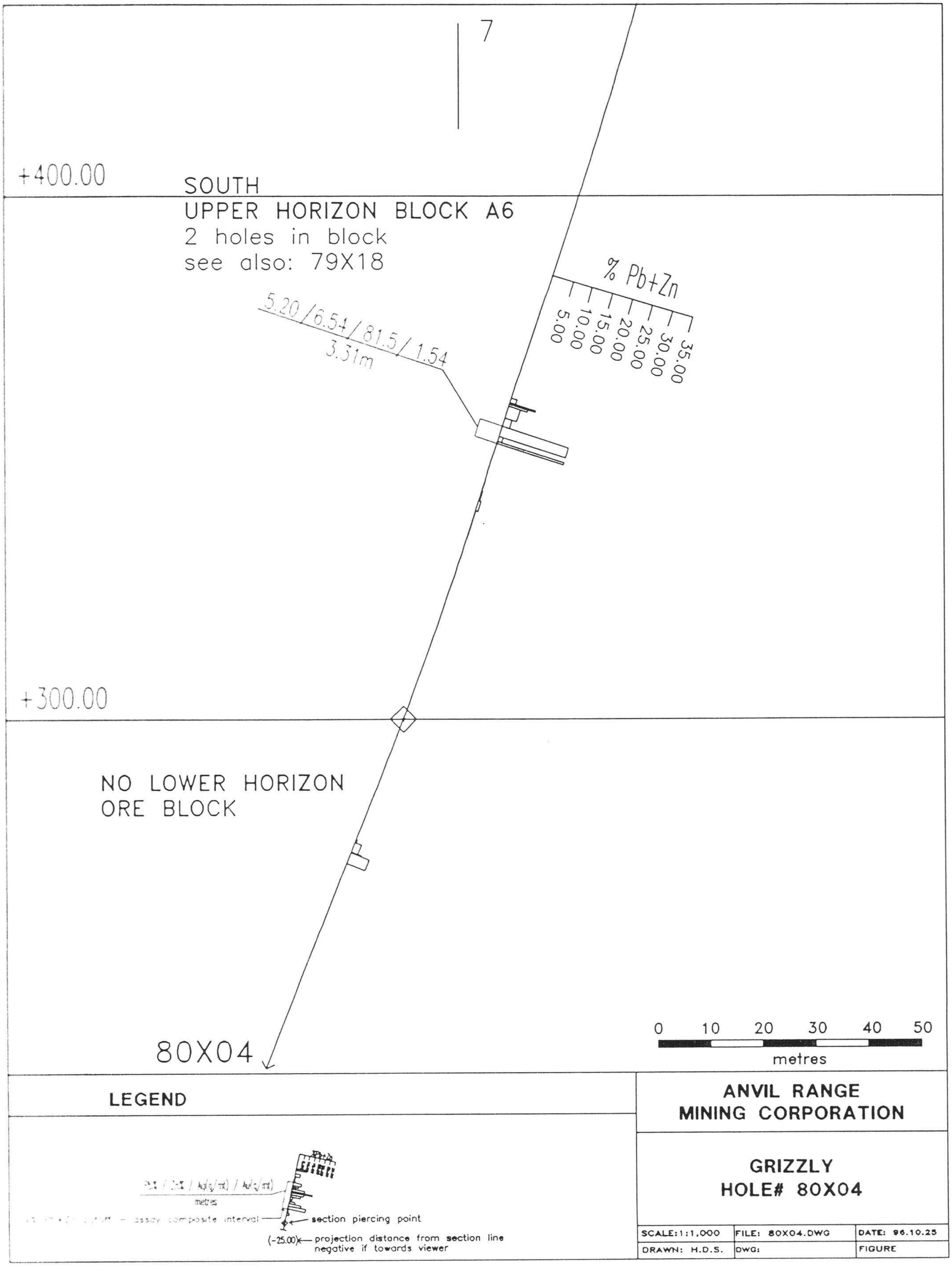
LEGEND

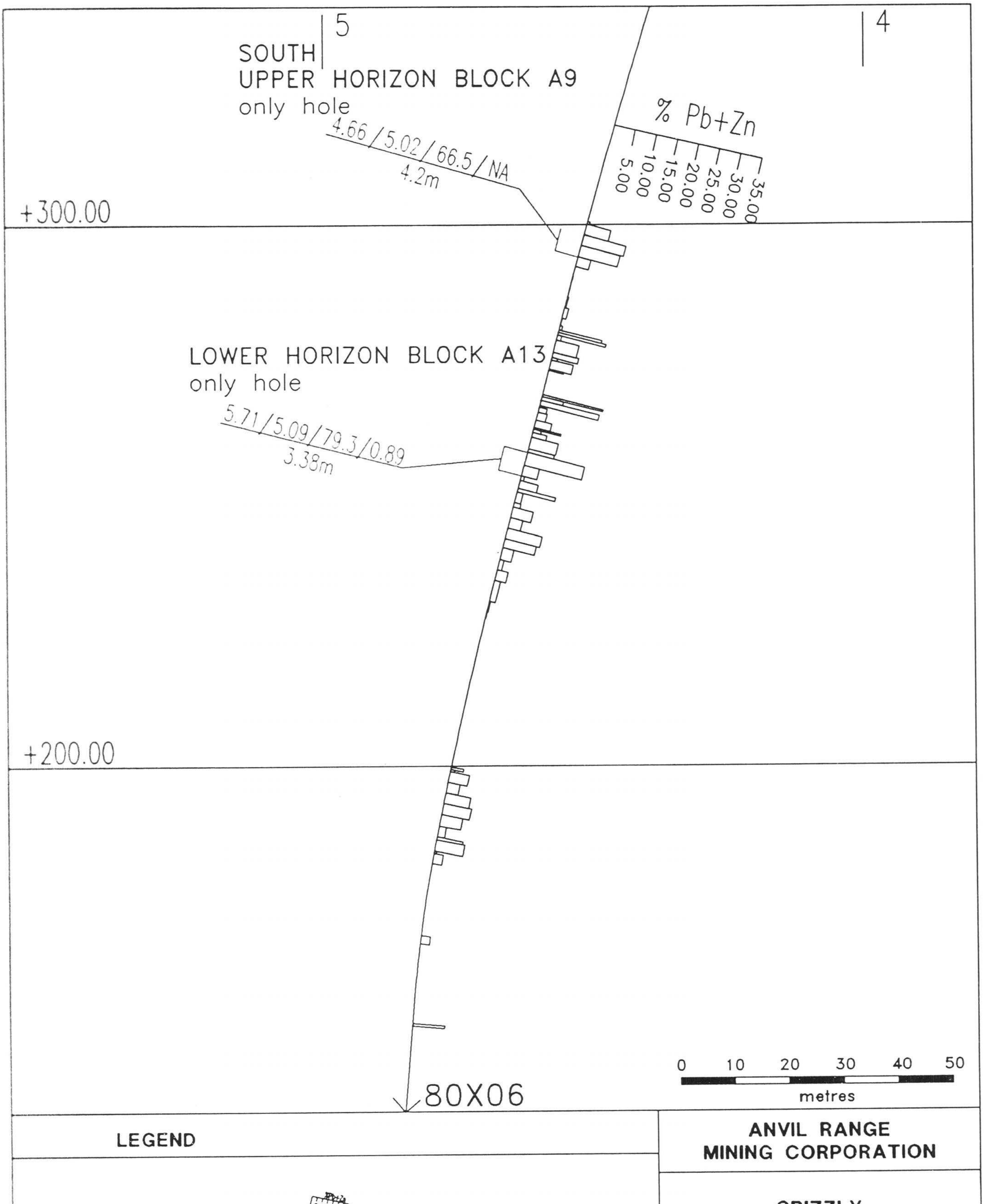


ANVIL RANGE MINING CORPORATION

GRIZZLY
HOLE# 80X02

SCALE: 1:1,000	FILE: 80X02.DWG	DATE: 96.10.25
DRAWN: H.D.S.	DWG:	FIGURE

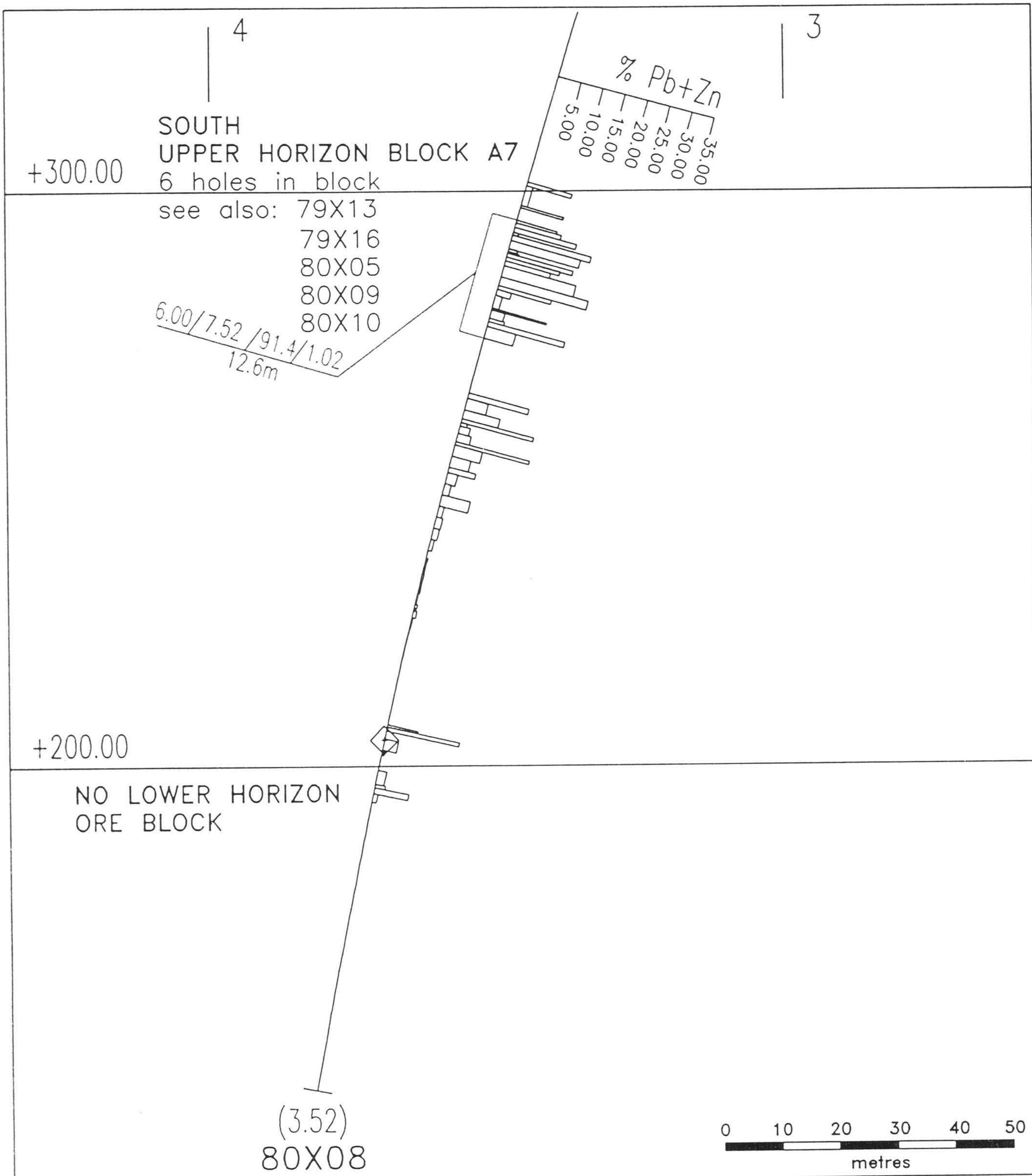

7
+400.00
SOUTH
UPPER HORIZON BLOCK A6
2 holes in block
see also: 79X18
% Pb+Zn
5.00
10.00
15.00
20.00
25.00
30.00
35.00
5.20/6.54/81.5/1.54
3.31m
+300.00
NO LOWER HORIZON
ORE BLOCK
80X04
0 10 20 30 40 50
metres
LEGEND
metres
assay composite interval
section piercing point
(-25.00)
projection distance from section line
negative if towards viewer
ANVIL RANGE
MINING CORPORATION
GRIZZLY
HOLE# 80X04
SCALE:1:1,000
FILE: 80X04.DWG
DATE: 96.10.25
DRAWN: H.D.S.
DWG:
FIGURE



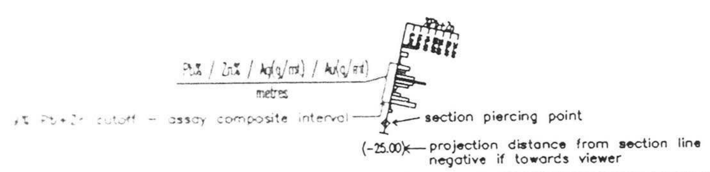
ANVIL RANGE MINING CORPORATION

GRIZZLY
HOLE# 80X06

SCALE: 1:1,000	FILE: 80X06.DWG	DATE: 96.10.25
DRAWN: H.D.S.	DWG:	FIGURE



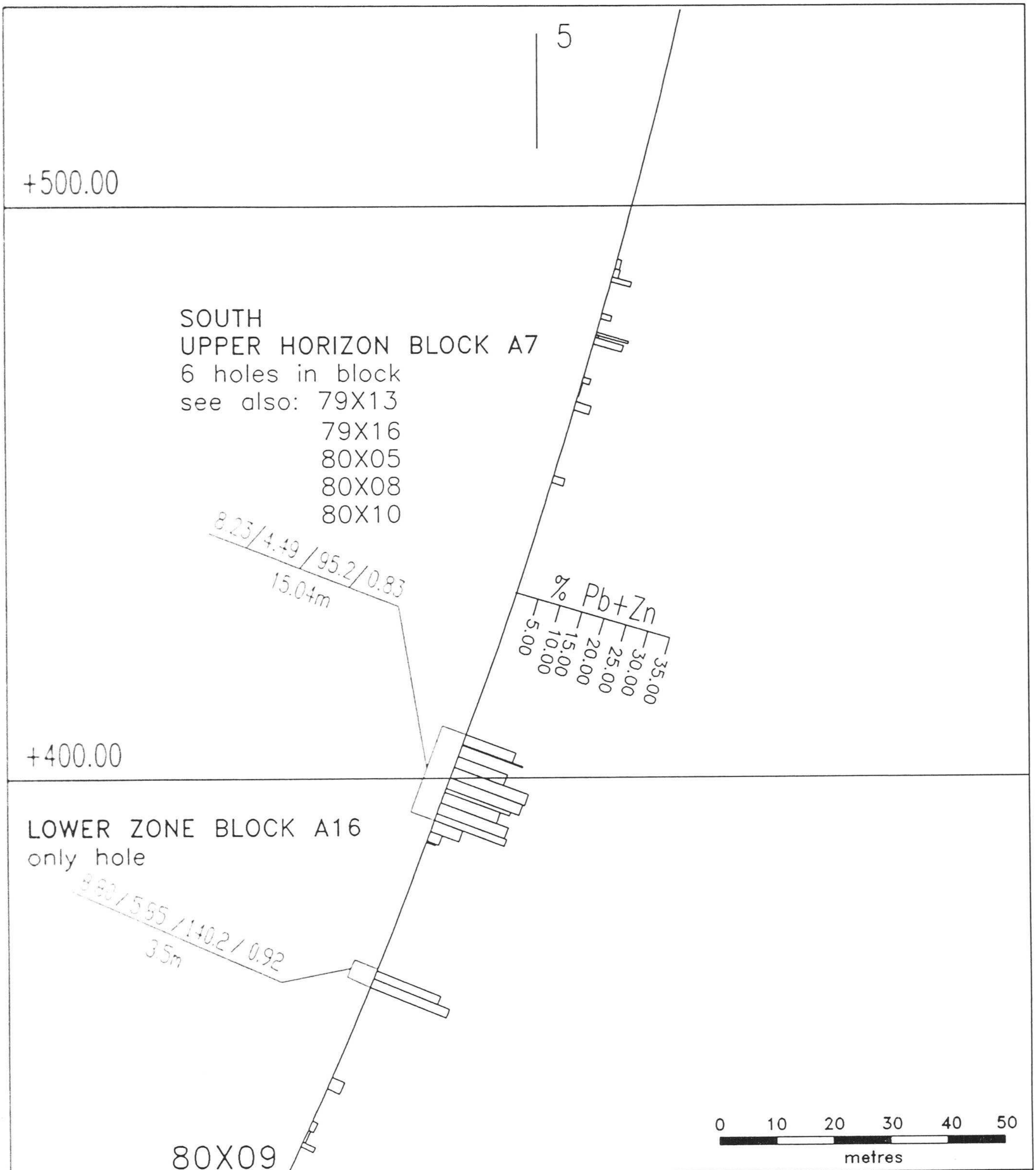
LEGEND



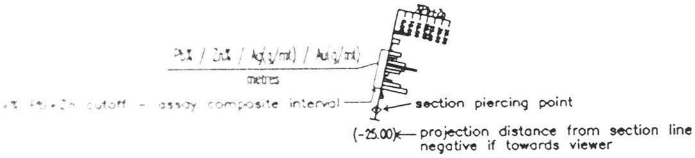
ANVIL RANGE MINING CORPORATION

GRIZZLY
HOLE# 80X08

SCALE:1:1,000	FILE: 80X08.DWG	DATE: 96.10.25
DRAWN: H.D.S.	DWG:	FIGURE



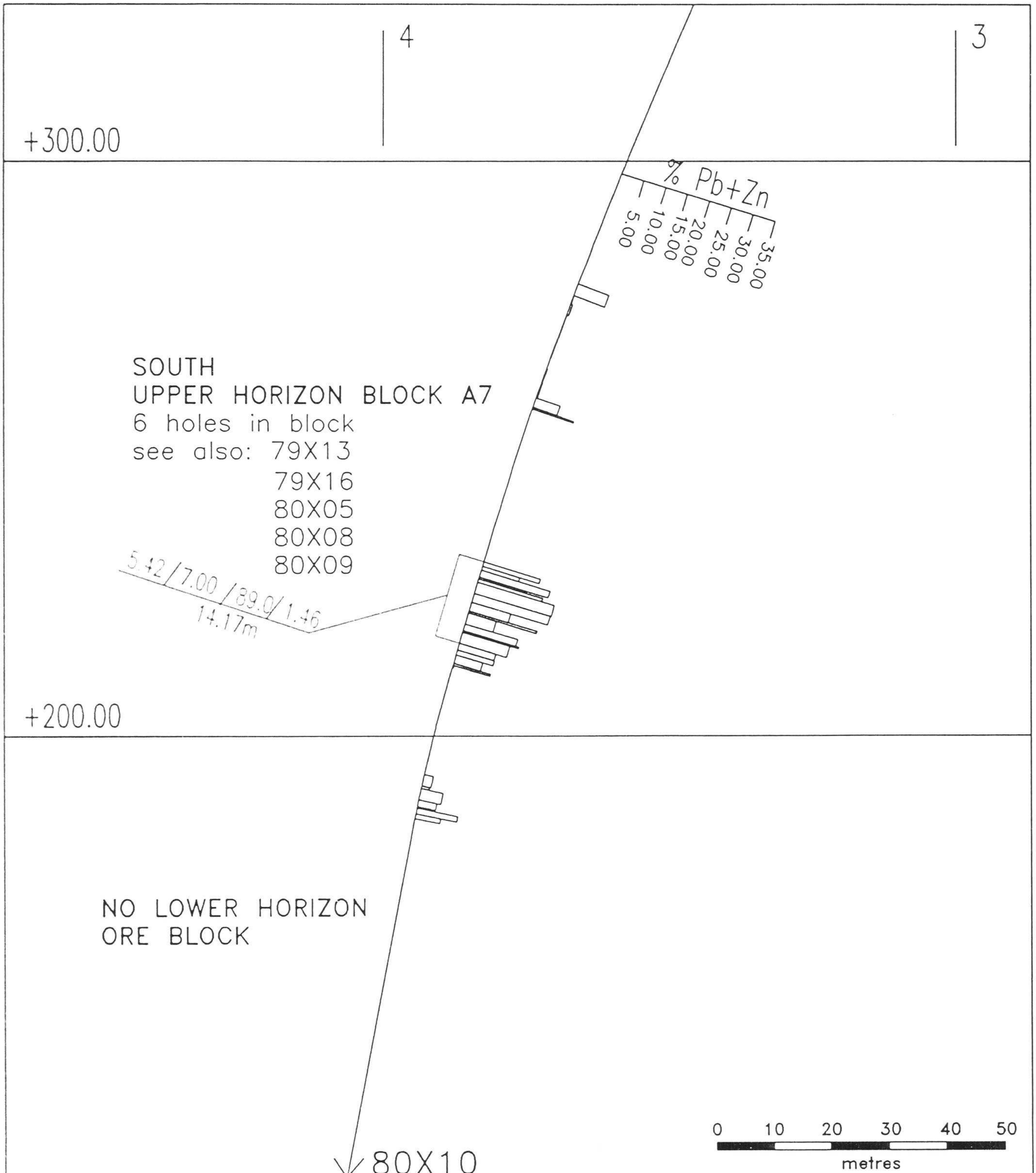
LEGEND



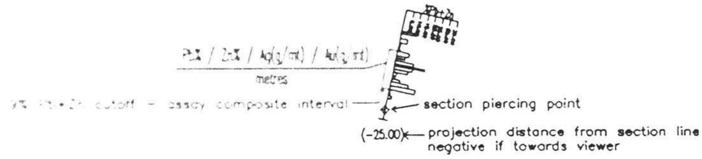
ANVIL RANGE MINING CORPORATION

GRIZZLY
HOLE# 80X09

SCALE:1:1,000	FILE: 80X09.DWG	DATE: 96.10.25
DRAWN: H.D.S.	DWG:	FIGURE



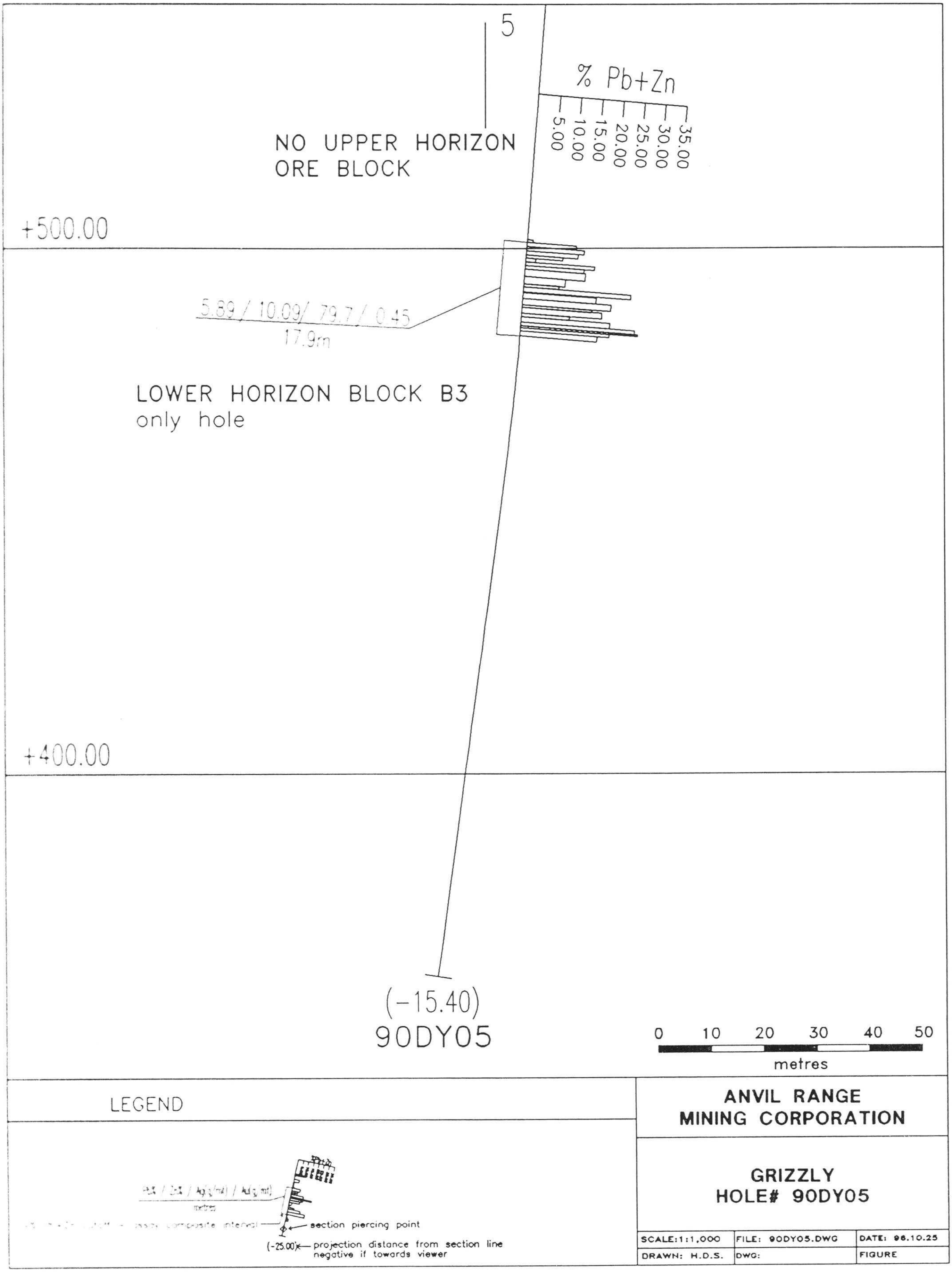
LEGEND

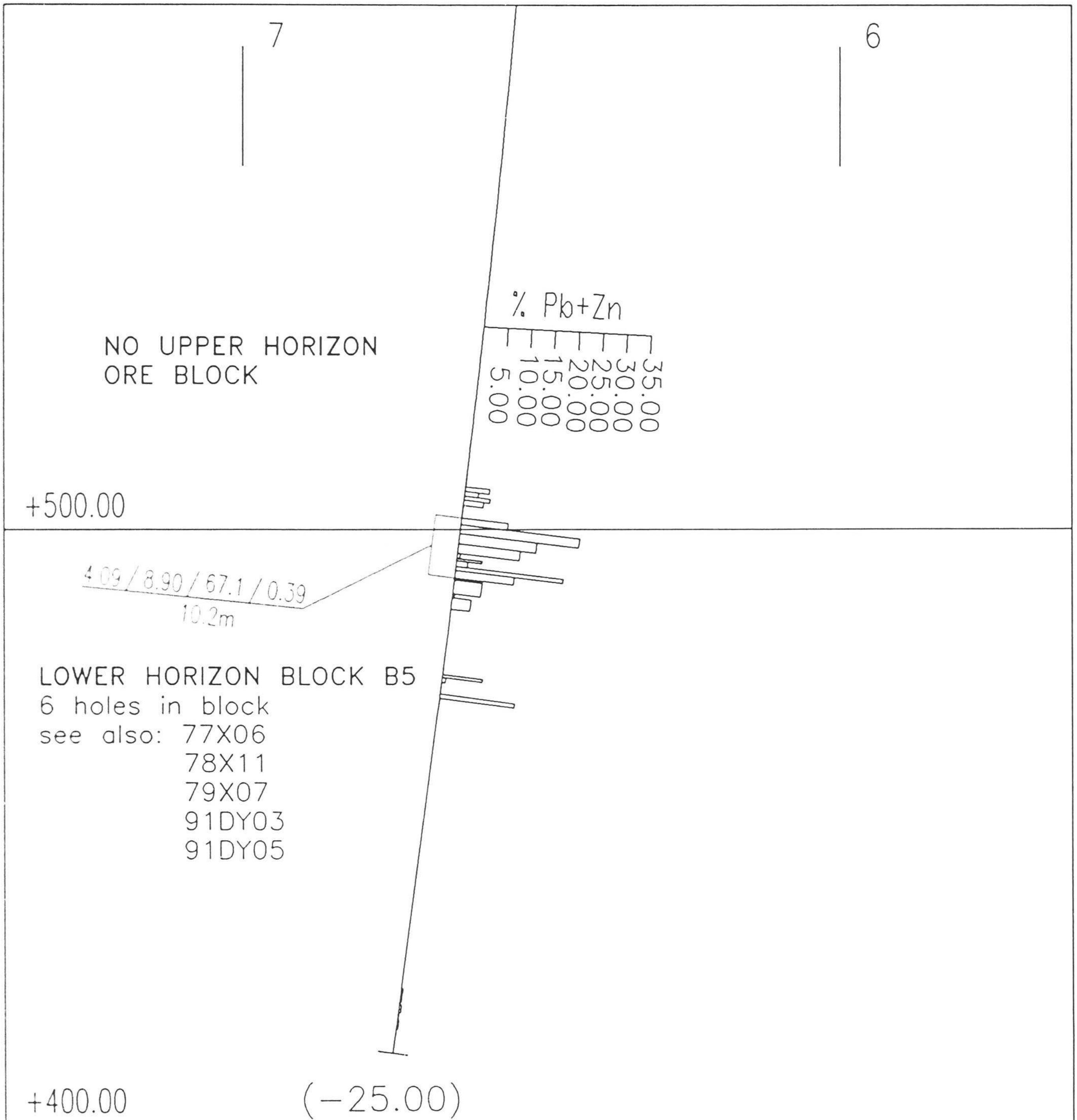


ANVIL RANGE MINING CORPORATION

GRIZZLY
HOLE# 80X10

SCALE:1:1,000	FILE: 80X10.DWG	DATE: 96.10.25
DRAWN: H.D.S.	DWG:	FIGURE


5
% Pb+Zn
5.00
10.00
15.00
20.00
25.00
30.00
35.00
NO UPPER HORIZON ORE BLOCK
+500.00
5.89 / 10.09/ 79.7 / 0.45
17.9m
LOWER HORIZON BLOCK B3
only hole
+400.00
(-15.40)
90DY05
0 10 20 30 40 50
metres
LEGEND
section piercing point
(-25.00) projection distance from section line negative if towards viewer
ANVIL RANGE MINING CORPORATION
GRIZZLY
HOLE# 90DY05
SCALE:1:1,000
FILE: 90DY05.DWG
DATE: 96.10.25
DRAWN: H.D.S.
DWG:
FIGURE

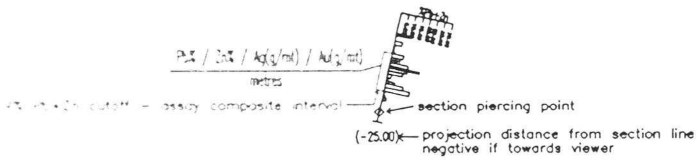


90DY09

0 10 20 30 40 50

metres

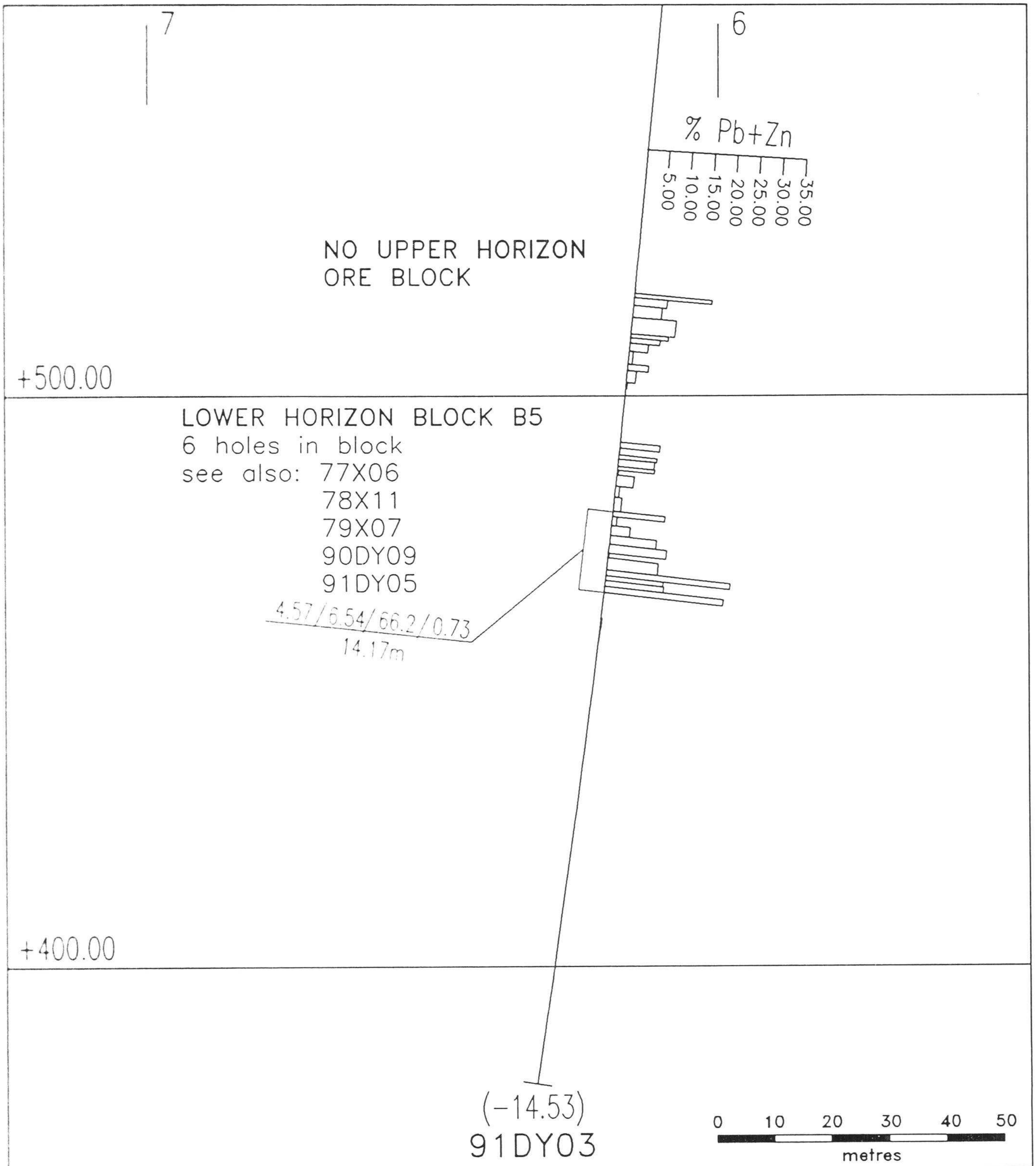
LEGEND



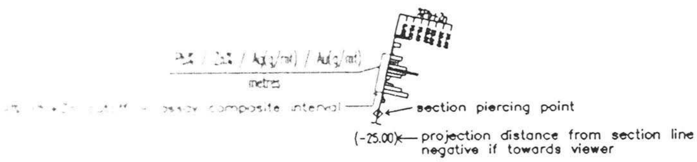
ANVIL RANGE MINING CORPORATION

GRIZZLY
HOLE# 90DY09

SCALE:1:1,000	FILE: 90DY09.DWG	DATE: 96.10.25
DRAWN: H.D.S.	DWG:	FIGURE



LEGEND



ANVIL RANGE MINING CORPORATION

GRIZZLY
HOLE# 91DY03

SCALE:1:1,000	FILE: 91DY03.DWG	DATE: 96.10.25
DRAWN: H.D.S.	DWG:	FIGURE

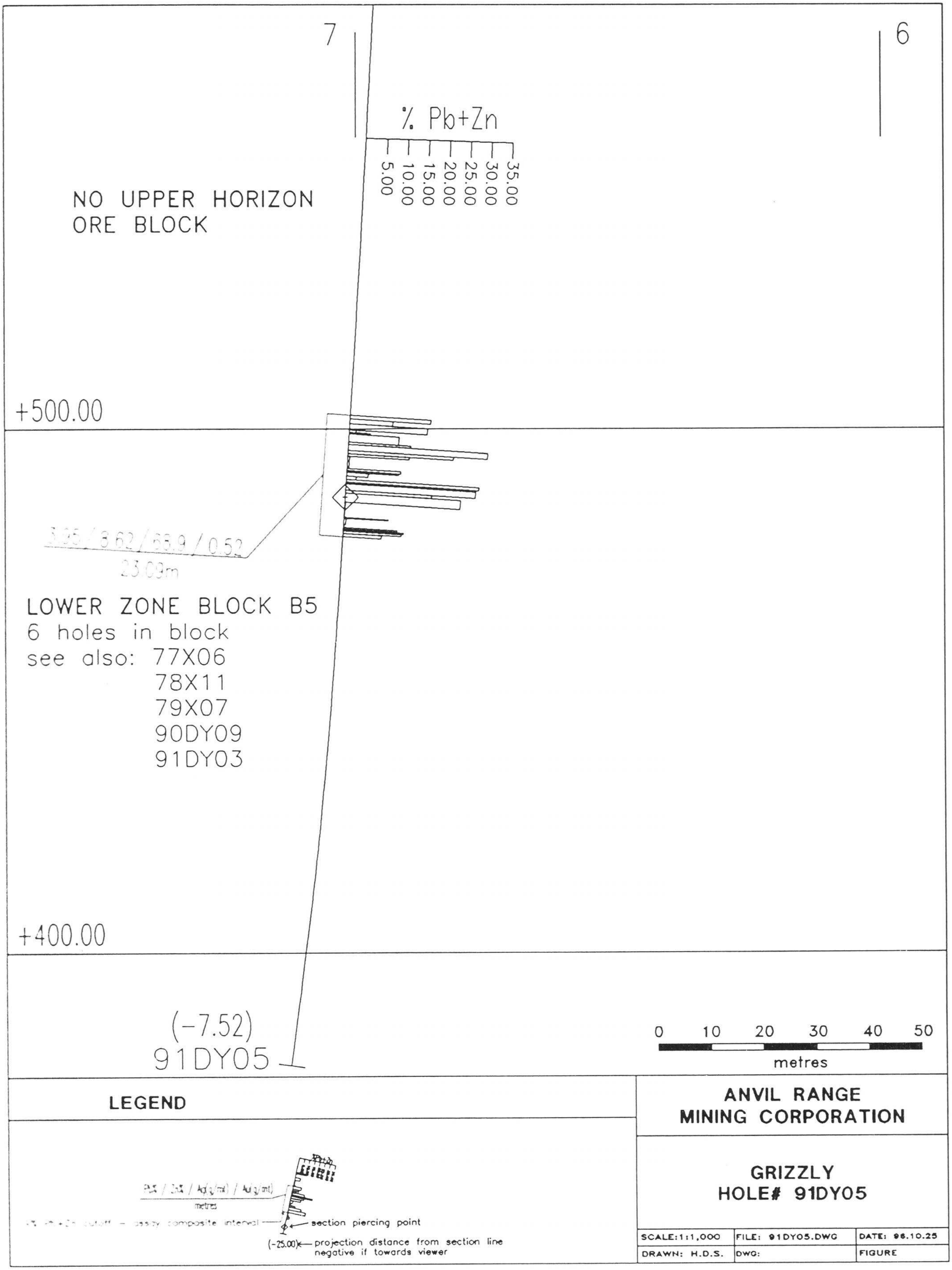

7
6
% Pb+Zn
5.00
10.00
15.00
20.00
25.00
30.00
35.00
NO UPPER HORIZON ORE BLOCK
+500.00
3.95 / 3.62 / 63.9 / 0.52
23.09m
LOWER ZONE BLOCK B5
6 holes in block
see also: 77X06
78X11
79X07
90DY09
91DY03
+400.00
(-7.52)
91DY05
0 10 20 30 40 50
metres
LEGEND
Pb% / Zn% / Ag(g/mt) / Au(g/mt)
metres
section piercing point
(-25.00)
projection distance from section line negative if towards viewer
ANVIL RANGE MINING CORPORATION
GRIZZLY
HOLE# 91DY05
SCALE:1:1,000
FILE: 91DY05.DWG
DATE: 96.10.25
DRAWN: H.D.S.
DWG:
FIGURE

TABLE 6.1
GRIZZLY PROJECT 6% PB+ZN CUTOFF GRADE COMPOSITES

DRILL HOLE	FROM (m)	TO (m)	COMP-ID	COMPLEN (m)	VERT TH. (m)	Pb+Zn (%)	Pb (%)	Zn (%)	Ag (g/t)	Au (g/t)	COMMENTS
76X21	563.0	571.4	UPPER-G	8.4	8.38	0.36	0.18	0.18	4.0	0.20	
76X21	581.7	588.0	LOWER-G	6.3	6.28	8.10	2.93	5.17	55.4	0.66	
77X01	581.4	584.9	UPPER-G	3.5	3.46	12.18	4.63	7.55	79.0	0.95	
77X01	607.0	610.5	LOWER-G	3.5	3.45	6.75	2.62	4.13	49.5	0.20	
77X03	700.1	707.9	UPPER-G	7.8	7.61	8.33	4.62	3.72	60.8	0.39	
77X03	731.2	736.6	LOWER-G	5.4	5.24	2.66	1.50	1.15	23.4	0.20	
77X05	709.0	716.0	UPPER-G	7.0	6.75	12.93	5.27	7.66	108.3	1.35	
77X05	740.2	743.7	LOWER-G	3.5	3.39	6.29	2.84	3.44	53.4	0.72	
77X06	545.3	552.3	UPPER-G	7.0	6.91	7.78	2.78	5.00	53.9	0.20	
77X06	576.6	580.1	LOWER-G	3.5	3.44	9.73	2.91	6.82	54.3	0.22	
77X06	583.7	612.1	LOWER-G	28.4	27.84	16.73	5.94	10.79	105.4	0.58	
	SUBTOTAL		LOWER-G	31.9	31.28	15.96	5.61	10.35	99.8	0.54	(3.6m exclusion)
77X09	625.5	641.5	UPPER-G	16.0	14.99	7.42	2.47	4.95	36.9	0.33	
77X09	647.5	651.5	UPPER-G	4.0	3.74	6.32	2.23	4.09	36.5	0.20	
	SUBTOTAL		UPPER-G	20.0	18.73	7.20	2.42	4.78	36.8	0.30	(6.0m exclusion)
77X09	699.6	706.0	LOWER-G	6.4	5.97	9.89	4.03	5.86	67.4	0.63	
77X11	665.9	679.6	UPPER-G	13.7	13.43	3.21	1.91	1.30	28.5	0.20	
77X11	769.6	779.6	LOWER-G	10.0	9.68	6.30	2.27	4.02	34.3	0.20	
77X04	790.3	793.8	UPPER-G	3.5	3.26	2.84	1.22	1.63	13.2	0.21	
77X04	798.9	802.4	LOWER-G	3.5	3.27	1.89	0.54	1.35	9.8	0.25	
78X01	616.4	619.9	UPPER-G	3.5	3.42	8.82	3.02	5.80	59.7	0.56	
78X01	633.7	649.5	LOWER-G	15.8	15.40	7.44	2.62	4.83	41.3	0.55	
78X02	585.6	597.6	UPPER-G	12.0	11.41	6.76	2.22	4.54	29.1	0.20	
78X02	674.3	705.4	LOWER-G	31.1	29.50	7.81	3.28	4.52	53.5	0.65	
78X04	518.4	521.9	UPPER-G	3.5	3.34	11.04	3.38	7.66	57.8	0.46	
78X04	556.6	562.0	LOWER-G	5.4	5.19	22.36	9.49	12.88	151.0	1.14	
78X05	577.5	581.0	UPPER-G	3.5	3.40	0.00	0.00	0.00	0.0	0.00	
78X05	586.3	604.2	LOWER-G	17.9	17.46	12.70	4.43	8.27	69.6	0.87	
78X06	495.4	498.9	UPPER-G	3.5	3.40	3.82	1.34	2.48	25.5	0.20	
78X06	520.0	523.5	LOWER-G	3.5	3.39	0.00	0.00	0.00	0.0	0.00	
78X07	559.3	571.4	UPPER-G	12.1	11.99	2.05	1.36	0.69	25.8	0.23	
78X07	629.1	632.6	LOWER-G	3.5	3.47	3.67	2.38	1.29	49.2	0.58	
78X08	633.2	638.9	UPPER-G	5.7	5.69	11.03	3.70	7.33	68.9	0.71	
78X08	664.0	667.5	LOWER-G	3.5	3.49	0.00	0.00	0.00	0.0	0.00	
78X09	556.3	562.1	UPPER-G	5.8	5.60	9.35	2.69	6.66	43.8	0.59	
78X09	575.2	580.2	LOWER-G	5.0	4.83	10.83	4.26	6.58	75.4	1.12	
78X10	547.1	550.6	UPPER-G	3.5	3.36	0.00	0.00	0.00	0.0	0.00	
78X10	550.8	559.6	LOWER-G	8.8	8.46	0.00	0.00	0.00	0.0	0.00	
78X11	550.2	556.2	UPPER-G	6.0	5.81	7.58	2.62	4.96	42.3	0.47	
78X11	615.3	625.2	LOWER-G	9.9	9.42	12.30	4.72	7.58	77.3	0.73	
79X01	650.9	654.9	UPPER-G	4.0	3.81	6.89	2.19	4.70	39.0	0.67	
79X01	681.3	684.8	LOWER-G	3.5	3.31	3.58	1.27	2.31	20.3	0.20	
79X02	539.4	544.8	UPPER-G	5.4	5.25	0.23	0.07	0.16	1.9	0.20	
79X02	600.0	603.5	LOWER-G	3.5	3.31	12.65	4.83	7.82	68.4	0.36	
79X03	700.8	706.4	UPPER-G	5.6	5.41	4.51	1.60	2.91	30.4	0.20	
79X03	733.4	741.7	LOWER-G	8.3	8.01	0.71	0.32	0.39	17.7	0.20	
79X04	586.2	590.0	UPPER-G	3.8	3.75	6.53	3.16	3.38	43.2	0.20	
79X04	625.8	630.6	LOWER-G	4.8	4.73	11.49	3.68	7.80	63.9	0.63	
79X05	591.7	596.1	UPPER-G	4.4	4.29	5.55	2.80	2.75	45.7	0.20	
79X05	634.0	637.5	LOWER-G	3.5	3.43	10.44	4.02	6.43	62.1	0.28	
79X06	706.6	741.8	UPPER-G	35.2	33.42	12.24	7.09	5.15	93.0	1.09	
79X06	772.1	776.1	LOWER-G	4.0	3.72	6.91	2.44	4.47	40.0	0.20	
79X07	544.0	547.5	UPPER-G	3.5	3.42	2.46	0.78	1.69	14.4	0.20	
79X07	574.2	586.8	LOWER-G	12.6	12.25	12.03	3.99	8.04	57.2	0.61	
79X08	620.5	624.6	UPPER-G	4.1	4.01	4.83	2.84	1.99	39.0	0.20	
79X08	676.3	681.5	LOWER-G	5.2	5.07	8.80	3.72	5.08	75.0	1.02	
79X09	580.2	586.2	UPPER-G	6.0	5.87	6.17	1.78	4.39	31.7	0.20	
79X09	636.8	646.9	LOWER-G	10.1	9.80	7.88	3.22	4.66	44.7	1.17	
79X11	747.7	755.4	UPPER-G	7.7	7.24	10.79	6.57	4.22	72.9	1.36	
79X11	761.1	770.1	UPPER-G	9.0	8.46	11.03	5.18	5.85	68.8	0.53	
	SUBTOTAL		UPPER-G	16.7	15.70	10.92	5.82	5.10	70.7	0.91	(5.7m exclusion)
79X11	779.8	796.2	LOWER-G	16.4	15.41	11.63	6.10	5.53	90.2	1.01	
79X12	723.8	737.7	UPPER-G	13.9	13.33	9.36	4.55	4.81	65.1	0.58	
79X12	763.6	767.2	LOWER-G	3.6	3.43	2.28	1.28	0.99	23.5	0.20	
79X13	772.5	781.3	UPPER-G	8.8	8.42	14.09	6.73	7.36	90.8	1.26	
79X13	786.0	791.6	UPPER-G	5.6	5.33	12.57	6.32	6.25	81.6	0.45	
	SUBTOTAL		UPPER-G	14.4	13.75	13.50	6.57	6.93	87.2	0.95	(4.7m exclusion)
79X13	803.4	806.9	LOWER-G	3.5	3.30	4.02	1.75	2.27	46.5	0.20	
79X14	788.8	800.1	UPPER-G	11.3	10.63	9.58	5.18	4.40	67.9	1.32	
79X14	808.7	812.4	UPPER-G	3.7	3.49	6.56	3.01	3.54	35.0	0.20	
	SUBTOTAL		UPPER-G	15.0	14.12	8.83	4.64	4.19	59.8	1.04	(8.6m exclusion)
79X14	819.3	824.8	LOWER-G	5.5	5.20	8.13	3.25	4.88	50.9	0.85	

TABLE 6.1 continued
GRIZZLY PROJECT 6% PB+ZN CUTOFF GRADE COMPOSITES

DRILL HOLE	FROM (m)	TO (m)	COMP-ID	COMPLEN (m)	VERT TH (m)	Pb+Zn (%)	Pb (%)	Zn (%)	Ag (g/t)	Au (g/t)	COMMENTS
79X16	805.0	820.2	UPPER-G	15.2	14.63	9.56	4.49	5.07	65.5	0.64	
79X16	840.4	847.1	LOWER-G	6.7	6.45	7.35	2.66	4.70	47.0	0.72	
79X17	499.3	502.8	UPPER-G	3.5	3.39	0.00	0.00	0.00	0.0	0.00	
79X17	526.8	531.3	LOWER-G	4.5	4.34	8.19	2.35	5.84	37.8	0.25	
79X18	738.0	744.8	UPPER-G	6.8	6.53	8.59	2.87	5.72	72.1	1.14	
79X18	759.5	763.0	LOWER-G	3.5	3.36	0.20	0.13	0.07	0.1	0.20	
80X01	757.3	761.6	UPPER-G	4.3	4.25	11.67	5.96	5.71	80.2	1.15	
80X01	766.4	769.9	UPPER-G	3.5	3.46	6.60	2.97	3.63	47.3	0.61	
	SUBTOTAL		UPPER-G	7.8	7.71	9.39	4.62	4.78	65.4	0.91	(4.8m exclusion)
80X01	799.3	812.9	LOWER-G	13.6	13.42	2.80	1.59	1.20	29.8	0.20	
80X02	827.8	850.3	UPPER-G	22.5	21.49	9.48	3.97	5.51	51.2	0.49	
80X02	883.8	895.4	LOWER-G	11.6	11.09	9.98	3.53	6.44	61.2	0.89	
80X02	900.6	904.9	LOWER-G	4.3	4.11	14.65	4.81	9.84	85.1	1.23	
	SUBTOTAL		LOWER-G	15.9	15.20	11.24	3.88	7.36	67.7	0.98	(4.3m exclusion)
80X03	543.0	546.5	UPPER-G	3.5	3.47	1.33	0.54	0.79	10.3	0.20	
80X03	566.5	570.0	LOWER-G	3.5	3.46	0.00	0.00	0.00	0.0	0.00	
80X04	808.4	811.9	UPPER-G	3.5	3.31	11.74	5.20	6.54	81.5	1.54	
80X04	819.4	831.2	LOWER-G	11.8	11.19	0.19	0.11	0.08	1.0	0.20	
80X05	846.5	861.2	UPPER-G	14.7	13.81	14.08	6.15	7.93	91.1	0.91	
80X05	896.1	901.1	LOWER-G	5.0	4.82	8.87	3.78	5.09	59.9	1.63	
80X06	844.5	848.7	UPPER-G	4.2	4.05	9.68	4.66	5.02	66.5	0.20	
80X06	875.0	878.5	LOWER-G	3.5	3.38	7.38	3.31	4.07	49.2	0.82	
80X06	883.2	888.6	LOWER-G	5.4	5.22	9.73	4.27	5.46	99.6	1.44	
	SUBTOTAL		LOWER-G	8.9	8.60	8.81	3.89	4.91	79.8	1.20	(4.7m exclusion)
80X07	743.0	753.4	UPPER-G	10.4	10.21	7.10	3.22	3.88	46.2	0.80	
80X07	810.9	815.0	LOWER-G	4.1	4.04	9.88	3.74	6.14	48.9	0.79	
80X08	826.5	850.6	UPPER-G	24.1	23.13	9.21	4.12	5.09	64.7	0.78	
80X08	860.5	865.8	LOWER-G	5.3	5.10	8.91	4.20	4.71	60.4	1.30	
80X08	869.3	874.9	LOWER-G	5.6	5.40	6.61	3.12	3.49	50.4	0.55	
	SUBTOTAL		LOWER-G	10.9	10.50	7.73	3.64	4.08	55.3	0.91	(3.5m exclusion)
80X09	725.0	743.1	UPPER-G	18.1	16.91	12.05	7.80	4.25	91.3	0.94	
80X09	769.4	772.9	LOWER-G	3.5	3.23	14.65	8.80	5.85	140.2	0.94	
80X10	909.8	928.6	UPPER-G	18.8	18.00	11.37	4.86	6.52	81.1	1.41	
80X10	950.8	956.2	LOWER-G	5.4	5.25	5.47	1.96	3.51	28.6	0.20	
80X11	695.6	699.1	UPPER-G	3.5	3.29	0.00	0.00	0.00	0.0	0.00	
80X11	735.6	739.1	LOWER-G	3.5	3.25	6.16	2.20	3.95	30.4	0.20	
80X12	848.0	865.2	UPPER-G	17.2	15.52	4.29	1.58	2.71	27.3	0.23	
80X12	876.4	882.5	LOWER-G	6.1	5.51	3.92	2.14	1.78	34.9	0.20	
80X13	733.9	737.4	UPPER-G	3.5	3.26	4.21	1.76	2.44	29.4	0.20	
80X13	782.0	788.7	LOWER-G	6.7	6.43	9.00	3.85	5.15	50.6	0.66	
80X13	794.9	799.5	LOWER-G	4.6	4.44	7.14	2.67	4.47	39.8	0.20	
	SUBTOTAL		LOWER-G	11.3	10.87	8.24	3.37	4.87	46.2	0.47	(6.2m exclusion)
90DY04DS	545.5	549.0	UPPER-G	0.0	3.50	0.00	0.00	0.00	0.0	0.00	
90DY04DS	554.3	565.9	LOWER-G	11.6	11.59	9.55	3.23	6.33	40.9	0.45	
90DY05	495.9	501.5	UPPER-G	0.0	5.58	0.00	0.00	0.00	0.0	0.00	
90DY05	516.7	534.6	LOWER-G	17.9	17.83	15.98	5.89	10.09	79.7	0.45	
90DY07	587.4	592.4	UPPER-G	5.0	4.99	4.60	1.60	3.00	18.2	0.06	
90DY07	596.0	599.8	LOWER-G	3.8	3.79	8.76	3.66	5.10	55.8	0.39	
90DY09	536.3	539.8	UPPER-G	0.0	3.46	0.00	0.00	0.00	0.0	0.00	
90DY09	556.5	566.7	LOWER-G	10.2	10.08	12.99	4.09	8.90	67.1	0.39	
91DY03	545.5	554.3	UPPER-G	8.8	8.73	8.55	2.96	5.59	61.8	0.48	
91DY03	571.8	576.8	LOWER-G	5.0	4.96	6.31	2.22	4.09	29.5	0.31	
91DY03	584.1	598.4	LOWER-G	14.3	14.17	11.11	4.57	6.54	66.2	0.73	
	SUBTOTAL		LOWER-G	19.3	19.13	9.87	3.96	5.90	56.7	0.62	(7.3m exclusion)
91DY04	685.5	693.4	UPPER-G	8.0	7.78	2.12	1.38	0.74	27.2	0.85	
91DY05	544.1	549.2	UPPER-G	5.1	5.08	3.85	1.20	2.64	25.2	0.19	
91DY05	584.9	608.1	LOWER-G	23.2	23.09	12.57	3.95	8.62	68.9	0.52	
EA81X01	827.1	830.6	UPPER-G	3.5	3.29	4.34	1.96	2.38	33.9	0.20	
EA81X01	873.4	882.5	LOWER-G	9.1	8.92	1.76	1.19	0.57	28.4	0.20	
EA81X02	590.4	593.9	UPPER-G	3.5	3.38	4.97	2.46	2.51	29.3	0.20	
EA81X02	604.4	607.9	LOWER-G	3.5	3.38	7.69	3.53	4.16	58.9	0.20	
EA81X03	931.3	934.8	UPPER-G	3.5	3.28	0.81	0.54	0.27	15.8	0.20	
EA81X03	958.2	978.3	LOWER-G	20.1	18.82	2.93	1.60	1.34	33.5	0.20	

file TABLE6-1.XLS

Notes

1. Drillhole composites based on minimum 3.5m core length at a 6% Pb+Zn cutoff grade. Intersections less than 3.5m in length were diluted to a minimum 3.5m core length using footwall material.
2. Intervals of waste of greater than 3.5m were excluded from weight average composites, whereas intervals of waste of greater than 3.5m length were included.
3. A background value (average) of 0.2 g/t Au was assigned as a default in composite calculations to account for assays which did not include Au analyses. This value was derived from the CRI 1991 Mineral Inventory Report.

TABLE 6.2
GRIZZLY PROJECT 9% PB+ZN CUTOFF GRADE COMPOSITES

DRILL HOLE	FROM (m)	TO (m)	COMP-ID	COMPLEN (m)	VERT TH (m)	Pb+Zn (%)	Pb (%)	Zn (%)	Ag (g/t)	Au (g/t)	COMMENTS
76X21	563.0	571.4	UPPER-G	8.40	8.38	0.36	0.18	0.18	4.0	0.20	
76X21	581.7	588.0	LOWER-G	6.30	6.28	8.10	2.93	5.17	55.4	0.66	
77X01	581.4	584.9	UPPER-G	3.50	3.46	12.18	4.63	7.55	79.0	0.95	
77X01	607.0	610.5	LOWER-G	3.50	3.45	6.75	2.62	4.13	49.5	0.20	
77X03	700.1	703.6	UPPER-G	3.50	3.42	11.07	6.07	5.00	81.0	0.62	
77X03	731.2	736.6	LOWER-G	5.40	5.24	2.66	1.50	1.15	23.4	0.20	
77X05	709.0	716.0	UPPER-G	7.00	6.75	12.93	5.27	7.66	108.3	1.35	
77X05	740.2	743.7	LOWER-G	3.50	3.39	6.29	2.84	3.44	53.4	0.72	
77X06	545.3	552.3	UPPER-G	7.00	6.91	7.78	2.78	5.00	53.9	0.20	
77X06	576.6	580.1	LOWER-G	3.50	3.44	9.73	2.91	6.82	54.3	0.22	
77X06	586.5	612.1	LOWER-G	25.60	25.09	17.95	6.37	11.58	112.9	0.62	
	SUBTOTAL		LOWER-G	29.10	28.53	16.96	5.95	11.01	105.8	0.57	(6.4m exclusion)
77X09	625.5	629.0	UPPER-G	3.50	3.28	9.84	3.09	6.75	31.9	0.53	
77X09	703.2	706.7	LOWER-G	3.50	3.27	12.10	5.01	7.09	78.7	0.98	
77X11	665.9	679.6	UPPER-G	13.70	13.43	3.21	1.91	1.30	28.5	0.20	
77X11	769.6	779.6	LOWER-G	10.00	9.68	6.30	2.27	4.02	34.3	0.20	
77X04	790.3	793.8	UPPER-G	3.50	3.26	2.84	1.22	1.63	13.2	0.21	
77X04	798.9	802.4	LOWER-G	3.50	3.27	1.89	0.54	1.35	9.8	0.25	
78X01	616.4	619.9	UPPER-G	3.50	3.42	8.82	3.02	5.80	59.7	0.56	
78X01	633.7	637.2	LOWER-G	3.50	3.42	9.82	3.32	6.50	47.4	0.89	
78X01	645.8	649.5	LOWER-G	3.70	3.60	10.45	3.77	6.68	60.7	0.69	
	SUBTOTAL		LOWER-G	7.20	7.02	10.14	3.55	6.59	54.2	0.79	(8.6m exclusion)
78X02	585.6	597.6	UPPER-G	12.00	11.41	6.76	2.22	4.54	29.1	0.20	
78X02	674.3	678.3	LOWER-G	4.00	3.77	10.29	5.99	4.30	71.5	0.57	
78X02	684.3	694.5	LOWER-G	10.20	9.68	9.56	3.38	6.18	62.8	0.78	
78X02	698.7	702.2	LOWER-G	3.50	3.34	9.53	3.61	5.92	61.9	0.62	
	SUBTOTAL		LOWER-G	17.70	16.79	9.72	4.01	5.71	64.6	0.70	(6m & 4.4m exclusion)
78X04	518.4	521.9	UPPER-G	3.50	3.34	11.04	3.38	7.66	57.8	0.46	
78X04	556.6	562.0	LOWER-G	5.40	5.19	22.36	9.49	12.88	151.0	1.14	
78X05	577.5	581.0	UPPER-G	3.50	3.40	0.00	0.00	0.00	0.0	0.00	
78X05	586.3	600.0	LOWER-G	13.70	13.35	14.28	4.97	9.31	77.9	1.04	
78X06	495.4	498.9	UPPER-G	3.50	3.40	3.82	1.34	2.48	25.5	0.20	
78X06	520.0	523.5	LOWER-G	3.50	3.39	0.00	0.00	0.00	0.0	0.00	
78X07	559.3	571.4	UPPER-G	12.10	11.99	2.05	1.36	0.69	25.8	0.23	
78X07	629.1	632.6	LOWER-G	3.50	3.47	3.67	2.38	1.29	49.2	0.58	
78X08	633.2	636.7	UPPER-G	3.50	3.50	13.98	4.41	9.57	81.5	0.90	
78X08	664.0	667.5	LOWER-G	3.50	3.49	0.00	0.00	0.00	0.0	0.00	
78X09	556.3	562.1	UPPER-G	5.80	5.60	9.35	2.69	6.66	43.8	0.59	
78X09	575.2	580.2	LOWER-G	5.00	4.83	10.83	4.26	6.58	75.4	1.12	
78X10	547.1	550.6	UPPER-G	3.50	3.36	0.00	0.00	0.00	0.0	0.00	
78X10	550.8	559.6	LOWER-G	8.80	8.46	0.00	0.00	0.00	0.0	0.00	
78X11	550.2	556.2	UPPER-G	6.00	5.81	7.58	2.62	4.96	42.3	0.47	
78X11	615.3	625.2	LOWER-G	9.90	9.42	12.30	4.72	7.58	77.3	0.73	
79X01	650.9	654.9	UPPER-G	4.00	3.81	6.89	2.19	4.70	39.0	0.67	
79X01	681.3	684.8	LOWER-G	3.50	3.31	3.58	1.27	2.31	20.3	0.20	
79X02	539.4	544.8	UPPER-G	5.40	5.25	0.23	0.07	0.16	1.9	0.20	
79X02	600.0	603.5	LOWER-G	3.50	3.31	12.65	4.83	7.82	68.4	0.36	
79X03	700.8	706.4	UPPER-G	5.60	5.41	4.51	1.60	2.91	30.4	0.20	
79X03	733.4	741.7	LOWER-G	8.30	8.01	0.71	0.32	0.39	17.7	0.20	
79X04	586.2	590.0	UPPER-G	3.80	3.75	6.53	3.16	3.38	43.2	0.20	
79X04	625.8	630.6	LOWER-G	4.80	4.73	11.49	3.68	7.80	63.9	0.63	
79X05	591.7	596.1	UPPER-G	4.40	4.29	5.55	2.80	2.75	45.7	0.20	
79X05	634.0	637.5	LOWER-G	3.50	3.43	10.44	4.02	6.43	62.1	0.28	
79X06	718.2	739.8	UPPER-G	21.60	20.49	15.19	9.24	5.95	119.7	1.18	
79X06	772.1	776.1	LOWER-G	4.00	3.72	6.91	2.44	4.47	40.0	0.20	
79X07	544.0	547.5	UPPER-G	3.50	3.42	2.46	0.78	1.69	14.4	0.20	
79X07	577.8	586.8	LOWER-G	9.00	8.75	13.97	4.79	9.18	66.9	0.69	
79X08	620.5	624.6	UPPER-G	4.10	4.01	4.83	2.84	1.99	39.0	0.20	
79X08	676.3	679.8	LOWER-G	3.50	3.41	9.99	4.35	5.64	92.6	1.09	
79X09	580.2	586.2	UPPER-G	6.00	5.87	6.17	1.78	4.39	31.7	0.20	
79X09	636.8	640.9	LOWER-G	4.10	3.98	9.28	3.85	5.43	48.8	0.98	
79X11	749.7	755.4	UPPER-G	5.70	5.36	11.98	6.60	5.38	85.8	1.20	
79X11	761.1	770.1	UPPER-G	9.00	8.46	11.03	5.18	5.85	68.8	0.53	
	SUBTOTAL		UPPER-G	14.70	13.82	11.40	5.73	5.67	75.4	0.79	(5.7m exclusion)
79X11	779.8	796.2	LOWER-G	16.40	15.41	11.63	6.10	5.53	90.2	1.01	

TABLE 6.2 continued
GRIZZLY PROJECT 9% PB+ZN CUTOFF GRADE COMPOSITES

DRILL HOLE	FROM (m)	TO (m)	COMP-ID	COMPLEN (m)	VERT TH (m)	Pb+Zn (%)	Pb (%)	Zn (%)	Ag (g/t)	Au (g/t)	COMMENTS
79X12	724.4	735.0	UPPER-G	10.6	10.17	10.36	5.03	5.33	71.6	0.67	
79X12	763.6	767.2	LOWER-G	3.6	3.43	2.28	1.28	0.99	23.5	0.20	
79X13	772.5	781.3	UPPER-G	8.8	8.42	14.09	6.73	7.36	90.8	1.26	
79X13	786.0	791.6	UPPER-G	5.6	5.33	12.57	6.32	6.25	81.6	0.45	
	SUBTOTAL		UPPER-G	14.4	13.75	13.50	6.57	6.93	87.2	0.95	(4.7m exclusion)
79X13	803.4	806.9	LOWER-G	3.5	3.30	4.02	1.75	2.27	46.5	0.20	
79X14	788.8	800.1	UPPER-G	11.3	10.63	9.58	5.18	4.40	67.9	1.32	
79X14	819.3	824.8	LOWER-G	5.5	5.20	8.13	3.25	4.88	50.9	0.85	
79X16	805.0	819.9	UPPER-G	14.9	14.34	9.57	4.50	5.07	65.6	0.64	
79X16	840.4	847.1	LOWER-G	6.7	6.45	7.35	2.66	4.70	47.0	0.72	
79X17	499.3	502.8	UPPER-G	3.5	3.39	0.00	0.00	0.00	0.0	0.00	
79X17	526.8	531.3	LOWER-G	4.5	4.34	8.19	2.35	5.84	37.8	0.25	
79X18	741.2	744.7	UPPER-G	3.5	3.36	9.87	2.96	6.91	60.9	0.93	
79X18	759.5	763.0	LOWER-G	3.5	3.36	0.20	0.13	0.07	0.1	0.20	
80X01	757.3	761.6	UPPER-G	4.3	4.25	11.67	5.96	5.71	80.2	1.15	
80X01	799.3	812.9	LOWER-G	13.6	13.42	2.80	1.59	1.20	29.8	0.20	
80X02	831.1	837.2	UPPER-G	6.1	5.83	18.77	8.85	9.92	101.7	1.21	
80X02	888.9	895.4	LOWER-G	6.5	6.21	12.89	4.38	8.52	76.1	1.43	
80X02	900.6	904.9	LOWER-G	4.3	4.11	14.65	4.81	9.84	85.1	1.23	
	SUBTOTAL		LOWER-G	10.8	10.32	13.59	4.55	9.05	79.7	1.35	(4.3m exclusion)
80X03	543.0	546.5	UPPER-G	3.5	3.47	1.33	0.54	0.79	10.3	0.20	
80X03	566.5	570.0	LOWER-G	3.5	3.46	0.00	0.00	0.00	0.0	0.00	
80X04	808.4	811.9	UPPER-G	3.5	3.31	11.74	5.20	6.54	81.5	1.54	
80X04	819.4	831.2	LOWER-G	11.8	11.19	0.19	0.11	0.08	1.0	0.20	
80X05	846.5	861.2	UPPER-G	14.7	13.81	14.08	6.15	7.93	91.1	0.91	
80X05	896.1	901.1	LOWER-G	5.0	4.82	8.87	3.78	5.09	59.9	1.63	
80X06	844.5	848.7	UPPER-G	4.2	4.05	9.68	4.66	5.02	66.5	0.20	
80X06	886.1	889.6	LOWER-G	3.5	3.38	10.80	5.71	5.09	79.3	0.89	
80X07	743.0	753.4	UPPER-G	10.4	10.21	7.10	3.22	3.88	46.2	0.80	
80X07	810.9	814.4	LOWER-G	3.5	3.45	10.04	3.79	6.25	48.5	0.80	
80X08	826.5	848.5	UPPER-G	22.0	21.11	9.46	4.21	5.25	66.1	0.69	
80X08	865.1	869.9	LOWER-G	4.8	4.62	6.23	2.95	3.28	48.8	0.74	
80X09	725.0	741.1	UPPER-G	16.1	15.04	12.72	8.23	4.49	95.2	0.83	
80X09	769.4	772.9	LOWER-G	3.5	3.23	14.65	8.80	5.85	140.2	0.94	
80X10	909.8	924.6	UPPER-G	14.8	14.17	12.42	5.42	7.00	89.0	1.46	
80X10	950.8	956.2	LOWER-G	5.4	5.25	5.47	1.96	3.51	28.6	0.20	
80X11	695.6	699.1	UPPER-G	3.5	3.29	0.00	0.00	0.00	0.0	0.00	
80X11	735.6	739.1	LOWER-G	3.5	3.25	6.16	2.20	3.95	30.4	0.20	
80X12	848.0	865.2	UPPER-G	17.2	15.52	4.29	1.58	2.71	27.3	0.23	
80X12	876.4	882.5	LOWER-G	6.1	5.51	3.92	2.14	1.78	34.9	0.20	
80X13	733.9	737.4	UPPER-G	3.5	3.26	4.21	1.76	2.44	29.4	0.20	
80X13	782.0	786.7	LOWER-G	4.7	4.51	9.68	3.97	5.71	52.6	0.70	
90DY04DS	545.5	549.0	UPPER-G	0.0	3.50	0.00	0.00	0.00	0.0	0.00	
90DY04DS	556.8	565.9	LOWER-G	9.1	9.09	11.13	3.87	7.26	48.3	0.53	
90DY05	495.9	501.5	UPPER-G	0.0	5.58	0.00	0.00	0.00	0.0	0.00	
90DY05	516.7	534.6	LOWER-G	17.9	17.83	15.98	5.89	10.09	79.7	0.45	
90DY07	587.4	592.4	UPPER-G	5.0	4.99	4.60	1.60	3.00	18.2	0.06	
90DY07	596.0	599.8	LOWER-G	3.8	3.79	8.76	3.66	5.10	55.8	0.39	
90DY09	536.3	539.8	UPPER-G	0.0	3.46	0.00	0.00	0.00	0.0	0.00	
90DY09	556.5	566.7	LOWER-G	10.2	10.08	12.99	4.09	8.90	67.1	0.39	
91DY03	545.5	554.3	UPPER-G	8.8	8.73	8.55	2.96	5.59	61.8	0.48	
91DY03	584.1	598.4	LOWER-G	14.3	14.17	11.11	4.57	6.54	66.2	0.73	
91DY04	685.5	693.4	UPPER-G	8.0	7.78	2.12	1.38	0.74	27.2	0.85	
91DY05	544.1	549.2	UPPER-G	5.1	5.08	3.85	1.20	2.64	25.2	0.19	
91DY05	584.9	608.1	LOWER-G	23.2	23.09	12.57	3.95	8.62	68.9	0.52	
EA81X01	827.1	830.6	UPPER-G	3.5	3.29	4.34	1.96	2.38	33.9	0.20	
EA81X01	873.4	882.5	LOWER-G	9.1	8.92	1.76	1.19	0.57	28.4	0.20	
EA81X02	590.4	593.9	UPPER-G	3.5	3.38	4.97	2.46	2.51	29.3	0.20	
EA81X02	604.4	607.9	LOWER-G	3.5	3.38	7.69	3.53	4.16	58.9	0.20	
EA81X03	931.3	934.8	UPPER-G	3.5	3.28	0.81	0.54	0.27	15.8	0.20	
EA81X03	958.2	978.3	LOWER-G	20.1	18.82	2.93	1.60	1.34	33.5	0.20	

TABLE6-2.XLS

Notes

1. Drillhole composites based on minimum 3.5m core length at a 9% Pb+Zn cutoff grade. Intersections less than 3.5m in length were diluted to a minimum 3.5m core length using footwall material.
2. Intervals of waste of greater than 3.5m were excluded from weight average composites, whereas intervals of waste of greater than 3.5m length were included.
3. A background value (average) of 0.2 g/t Au was assigned as a default in composite calculations to account for assays which did not include Au analyses. This value was derived from the CRI 1991 Mineral Inventory Report.

TABLE 6.3
GRIZZLY DEPOSIT
UPPER-G HORIZON 6% Pb+Zn CUTOFF GRADE POLYGONS

HOLE ID	V THICK (m)	Pb+Zn (%)	Pb (%)	Zn (%)	Ag (g/t)	Au (g/t)	S.G.	AREA (m^2)	VOLUME (m^3)	TONNAGE
77X01	3.46	12.18	4.63	7.55	79.0	0.95	3.92	19516.4	67526.8	264705
77X03	7.61	8.33	4.62	3.72	60.8	0.39	3.92	23842.7	181442.7	711255.5
77X05	6.75	12.93	5.27	7.66	108.3	1.35	3.92	8368.2	56485.5	221423
77X06	6.91	7.78	2.78	5.00	53.9	0.20	3.92	6070.2	41945	164424.3
77X09	18.73	7.20	2.42	4.78	36.8	0.30	3.92	24583.3	460445.8	1804947.5
78X01	3.42	8.82	3.02	5.80	59.7	0.56	3.92	8257.6	28241	110704.8
78X02	11.41	6.76	2.22	4.54	29.1	0.20	3.92	13605.1	155233.9	608517
78X04	3.34	11.04	3.38	7.66	57.8	0.46	3.92	10363.2	34613	135682.9
78X08	5.69	11.03	3.70	7.33	68.9	0.71	3.92	21389.2	121704.3	477081
78X09	5.60	9.35	2.69	6.66	43.8	0.59	3.92	14679.8	82207.1	322251.9
78X11	5.81	7.58	2.62	4.96	42.3	0.47	3.92	8783.6	51032.7	200048.2
79X01	3.81	6.89	2.19	4.70	39.0	0.67	3.92	14870.6	56657	222095.6
79X04	3.75	6.53	3.16	3.38	43.2	0.20	3.92	9948.6	37307.1	146243.8
79X06	33.42	12.24	7.09	5.15	93.0	1.09	3.92	10342.5	345645.5	1354930.4
79X09	5.87	6.17	1.78	4.39	31.7	0.20	3.92	18760.1	110122	431678.2
79X11	15.70	10.92	5.82	5.10	70.7	0.91	3.92	6993.7	109800.9	430419.5
79X12	13.33	9.36	4.55	4.81	65.1	0.58	3.92	15614.7	208144.6	815926.9
79X13	13.75	13.50	6.57	6.93	87.2	0.95	3.92	14374	197642.3	774757.6
79X14	14.12	8.84	4.64	4.19	59.8	1.04	3.92	12149.3	171548.7	672471
79X16	14.63	9.56	4.49	5.07	65.5	0.64	3.92	9248.7	135308.8	530410.6
79X18	6.53	8.59	2.87	5.72	72.1	1.14	3.92	11820.5	77188.2	302577.7
80X01	7.71	9.40	4.62	4.78	65.4	0.91	3.92	14885.6	114768	449890.6
80X02	21.49	9.48	3.97	5.51	51.2	0.49	3.92	15061	323660.8	1268750.2
80X04	3.31	11.74	5.20	6.54	81.5	1.54	3.92	13834.8	45793	179508.7
80X05	13.81	14.08	6.15	7.93	91.1	0.91	3.92	12830.6	177189.9	694584.5
80X06	4.05	9.68	4.66	5.02	66.5	0.20	3.92	15555.2	62998.5	246954
80X07	10.21	7.10	3.22	3.88	46.2	0.80	3.92	16937.5	172932.2	677894.4
80X08	23.13	9.21	4.12	5.09	64.7	0.78	3.92	15083.4	348880	1367609.5
80X09	16.91	12.05	7.80	4.25	91.3	0.94	3.92	12892.4	218009.8	854598.6
80X10	18.00	11.37	4.86	6.52	81.1	1.41	3.92	12241.5	220346.3	863757.4
91DY03	8.73	8.55	2.96	5.59	61.8	0.48	3.92	6122.2	53446.8	209511.3
		9.74	4.43	5.31	64.1	0.73		419,026	4,468,268	17,515,612

file TABLE6-3.XLS

Note Grades are insitu and undiluted.

TABLE 6.4
GRIZZLY DEPOSIT
LOWER-G HORIZON 6%Pb+Zn CUTOFF GRADE POLYGONS

HOLE ID	V. THICK (m)	Pb+Zn (%)	Pb (%)	Zn (%)	Ag (g/t)	Au (g/t)	S.G.	AREA (m^2)	VOLUME (m^3)	TONNAGE
76X21	6.28	8.10	2.93	5.17	55.4	0.66	3.92	16702.5	104891.8	411175.9
77X01	3.45	6.75	2.62	4.13	49.5	0.20	3.92	19269	66478.2	260594.5
77X05	3.39	6.29	2.84	3.44	53.4	0.72	3.92	7263.8	24624.4	96527.7
77X06	31.28	15.96	5.61	10.35	99.8	0.54	3.92	6262.9	195902.8	767939.1
77X09	5.97	9.89	4.03	5.86	67.4	0.63	3.92	26705.9	159434.3	624982.5
77X11	9.68	6.30	2.27	4.02	34.3	0.20	3.92	50597	489779.3	1919935
78X01	15.40	7.44	2.62	4.83	41.3	0.55	3.92	7839	120720.3	473223.6
78X02	29.50	7.81	3.28	4.52	53.5	0.65	3.92	13669.1	403239.3	1580698.1
78X04	5.19	22.36	9.49	12.88	151.0	1.14	3.92	9816	50944.8	199703.7
78X05	17.46	12.70	4.43	8.27	69.6	0.87	3.92	9741.5	170086.9	666740.8
78X09	4.83	10.83	4.26	6.58	75.4	1.12	3.92	14009.5	67666.1	265251.2
78X11	9.42	12.30	4.72	7.58	77.3	0.73	3.92	7163.7	67482.1	264529.7
79X02	3.31	12.65	4.83	7.82	68.4	0.36	3.92	10150.2	33597.3	131701.3
79X04	4.73	11.49	3.68	7.80	63.9	0.63	3.92	10271.1	48582.4	190443.1
79X05	3.43	10.44	4.02	6.43	62.1	0.28	3.92	24003.8	82332.9	322745.1
79X06	3.72	6.91	2.44	4.47	40.0	0.20	3.92	9769.8	36343.8	142467.8
79X07	12.25	12.03	3.99	8.04	57.2	0.61	3.92	2722.1	33345.2	130713.3
79X08	5.07	8.80	3.72	5.08	75.0	1.02	3.92	13744.2	69683.2	273158
79X09	9.80	7.88	3.22	4.66	44.7	1.17	3.92	17120.3	167779.1	657694.2
79X11	15.41	11.63	6.10	5.53	90.2	1.01	3.92	7605.5	117200.8	459427.3
79X14	5.20	8.13	3.25	4.88	50.9	0.85	3.92	12549	65254.7	255798.6
79X16	6.45	7.35	2.66	4.70	47.0	0.72	3.92	12673.7	81745.3	320441.7
79X17	4.34	8.19	2.35	5.84	37.8	0.25	3.92	13946.7	60528.6	237272
80X02	15.20	11.24	3.88	7.36	67.7	0.98	3.92	13977.2	212452.7	832814.7
80X05	4.82	8.87	3.78	5.09	59.9	1.63	3.92	13409.5	64633.6	253363.6
80X06	8.60	8.81	3.89	4.91	79.8	1.20	3.92	16221.5	139504.9	546859.2
80X07	4.04	9.88	3.74	6.14	48.9	0.79	3.92	17407.1	70324.8	275673.2
80X08	10.50	7.73	3.64	4.08	55.3	0.91	3.92	15685.6	164698.8	645619.3
80X09	3.23	14.65	8.80	5.85	140.2	0.94	3.92	15331.8	49521.6	194124.6
80X11	3.25	6.16	2.20	3.95	30.4	0.20	3.92	25285.4	82177.4	322135.4
80X13	10.87	8.24	3.37	4.87	46.2	0.47	3.92	12132.4	131879.6	516968.2
90DY04DS	11.59	9.55	3.23	6.33	40.9	0.45	3.92	7106.2	82361.3	322856.4
90DY05	17.83	15.98	5.89	10.09	79.7	0.45	3.92	15663	279271.1	1094742.6
90DY07	3.79	8.76	3.66	5.10	55.8	0.39	3.92	35250.5	133599.3	523709.4
90DY09	10.08	12.99	4.09	8.90	67.1	0.39	3.92	8296.3	83626.6	327816.1
91DY03	19.13	9.87	3.96	5.91	56.7	0.62	3.92	6558.4	125461.3	491808.2
91DY05	23.09	12.57	3.95	8.62	68.9	0.52	3.92	9369.1	216331.8	848020.8
EA81X02	3.38	7.69	3.53	4.16	58.9	0.20	3.92	25189.6	85141	333752.6
		9.95	3.84	6.11	61.2	0.63		560,480	4,638,629	18,183,429

file TABLE6-4 XLS

Note Grades are insitu and undiluted.

TABLE 6.5
GRIZZLY DEPOSIT
UPPER-G HORIZON 9% Pb+Zn CUTOFF GRADE POLYGONS

HOLE ID	V THICK (m)	Pb+Zn (%)	Pb (%)	Zn (%)	Ag (g/t)	Au (g/t)	S.G.	AREA (m^2)	VOLUME (m^3)	TONNAGE
77X01	3.46	12.18	4.63	7.55	79.0	0.95	3.92	19519	67535.8	264740.5
77X03	3.42	11.07	6.07	5.00	81.0	0.62	3.92	23884.1	81683.5	320199.2
77X05	6.75	12.93	5.27	7.66	108.3	1.35	3.92	8314	56119.2	219987.3
77X09	3.28	9.84	3.09	6.75	31.9	0.53	3.92	25166.2	82545.1	323576.8
78X04	3.34	11.04	3.38	7.66	57.8	0.46	3.92	10363.2	34613	135682.9
78X08	3.50	13.98	4.41	9.57	81.5	0.90	3.92	21412.7	74944.5	293782.4
78X09	5.60	9.35	2.69	6.66	43.8	0.59	3.92	14679.8	82207.1	322251.9
79X06	20.49	15.19	9.24	5.95	119.7	1.18	3.92	10272.1	210474.8	825061.1
79X11	13.82	11.40	5.73	5.67	75.4	0.79	3.92	7196.8	99459.8	389882.3
79X12	10.17	10.36	5.03	5.33	71.6	0.67	3.92	15583.5	158484.2	621258
79X13	13.75	13.50	6.57	6.93	87.2	0.95	3.92	14408.6	198118	776622.6
79X14	10.63	9.58	5.18	4.40	67.9	1.32	3.92	12222.5	129924.9	509305.6
79X16	14.34	9.57	4.50	5.07	65.6	0.64	3.92	9221.6	132238.1	518373.4
79X18	3.36	9.87	2.96	6.91	60.9	0.93	3.92	11773.2	39558	155067.2
80X01	4.25	11.67	5.96	5.71	80.2	1.15	3.92	14874.6	63216.9	247810.3
80X02	5.83	18.77	8.85	9.92	101.7	1.21	3.92	15149.5	88321.5	346220.4
80X04	3.31	11.74	5.20	6.54	81.5	1.54	3.92	13711.9	45386.5	177915
80X05	13.81	14.08	6.15	7.93	91.1	0.91	3.92	12844.8	177387.3	695358.1
80X06	4.05	9.68	4.66	5.02	66.5	0.20	3.92	15461.6	62619.6	245468.9
80X08	21.11	9.46	4.21	5.25	66.1	0.69	3.92	15089	318527.9	1248629.5
80X09	15.04	12.72	8.23	4.49	95.2	0.83	3.92	12916.5	194264.1	761515.1
80X10	14.17	12.42	5.42	7.00	89.0	1.46	3.92	12238.7	173422.3	679815.4
		11.93	5.71	6.22	80.4	0.91		316,304	2,571,052	10,078,524

TABLE6-5.XLS

Note Grades are insitu and undiluted.

TABLE 6.6
GRIZZLY DEPOSIT
LOWER-G HORIZON 9% Pb+Zn CUTOFF GRADE POLYGONS

HOLE ID	V. THICK	Pb+Zn (%)	PB% (%)	ZN% (%)	Ag (g/t)	Au (g/t)	DENSITY	AREA (m^2)	VOLUME (m^3)	TONNAGE
77X06	28.53	16.96	5.95	11.01	105.9	0.57	3.92	6228.8	177708.2	696616.2
77X09	3.27	12.10	5.01	7.09	78.7	0.98	3.92	26576.3	86904.5	340665.7
78X01	7.02	10.14	3.55	6.59	54.2	0.79	3.92	7785.2	54652.1	214236.2
78X02	16.79	9.72	4.02	5.70	64.6	0.70	3.92	13684.7	229765.4	900680.5
78X04	5.19	22.36	9.49	12.88	151.0	1.14	3.92	9840.3	51071.1	200198.5
78X05	13.35	14.28	4.97	9.31	77.9	1.04	3.92	9679.4	129219.6	506540.7
78X09	4.83	10.83	4.26	6.58	75.4	1.12	3.92	14022.6	67729.3	265498.9
78X11	9.42	12.30	4.72	7.58	77.3	0.73	3.92	7148.1	67335.5	263955.2
79X02	3.31	12.65	4.83	7.82	68.4	0.36	3.92	10190.9	33731.9	132229.0
79X04	4.73	11.49	3.68	7.80	63.9	0.63	3.92	10282.7	48637.1	190657.5
79X05	3.43	10.44	4.02	6.43	62.1	0.28	3.92	24052.7	82500.8	323403.3
79X07	8.75	13.97	4.79	9.18	66.9	0.69	3.92	2593.1	22689.5	88942.7
79X08	3.41	9.99	4.35	5.64	92.6	1.09	3.92	13751.6	46893.0	183820.4
79X09	3.98	9.28	3.85	5.43	48.8	0.98	3.92	17159.6	68295.2	267717.2
79X11	15.41	11.63	6.10	5.53	90.2	1.01	3.92	7605.5	117200.8	459427.3
80X02	10.32	13.59	4.55	9.05	79.7	1.35	3.92	13948.2	143945.7	564267.2
80X06	3.38	10.80	5.71	5.09	79.3	0.89	3.92	16254.0	54938.5	215358.7
80X07	3.45	10.04	3.79	6.25	48.5	0.8	3.92	17546.9	60536.8	237304.3
80X09	3.23	14.65	8.80	5.85	140.2	0.94	3.92	15331.8	49521.6	194124.6
80X13	4.51	9.68	3.97	5.71	52.6	0.7	3.92	11978.4	54022.8	211769.2
90DY04DS	9.09	11.13	3.87	7.26	48.3	0.53	3.92	7105.1	64585.8	253176.2
90DY05	17.83	15.98	5.89	10.09	79.7	0.45	3.92	15663.0	279271.1	1094742.6
90DY09	10.08	12.99	4.09	8.90	67.1	0.39	3.92	8419.4	84867.1	332678.9
91DY03	14.17	11.11	4.57	6.54	66.2	0.73	3.92	6527.3	92491.4	362566.2
91DY05	23.09	12.57	3.95	8.62	68.9	0.52	3.92	9366.0	216261.7	847745.8
		12.78	4.90	7.88	76.5	0.74		302,742	2,384,776	9,348,323

file: TABLE6-6.XLS

Note: Grades are insitu and undiluted.

TABLE 6.8
GRIZZLY DEPOSIT
UPPER-G HORIZON 6% Pb+Zn CUTOFF GRADE ORE BLOCKS

ORE BLOCK	DDH/ POLYGON	AREA (m^2)	VERT THICK (m)	TONNAGE	Pb+Zn (%)	Pb (%)	Zn (%)	Ag (g/t)	Au (g/t)	COMMENTS
A1	77X01	19 516	3 46	264,705	12 18	4.63	7.55	79 0	0 95	
	78X08	21 389	5 69	477 081	11 03	3.70	7.33	68 9	0 71	
SUB TOTAL		40 906	4.63	741 786	11 44	4.03	7.41	72 5	0.80	
10% DILUTION				**815,965**	**10.40**	**3.67**	**6.74**	**65.9**	**0.72**	
A2	77X03	23 843	7 61	711,256	8 33	4.62	3.72	60.8	0.39	
SUB TOTAL		23,843	7.61	711,256	8 33	4.62	3.72	60 8	0.39	
10% DILUTION				**782,381**	**7.57**	**4.20**	**3.38**	**55.3**	**0.35**	
A3	77X05	8 368	6 75	221 423	12 93	5.27	7.66	108.3	1.35	
SUB TOTAL		8 368	6.75	221,423	12 93	5.27	7.66	108.3	1.35	
10% DILUTION				**243,665**	**11.75**	**4.79**	**6.96**	**98.5**	**1.23**	
A4	79X06	10,343	33 42	1,354,930	12 24	7.09	5.15	93.0	1.09	
	79X11	6,994	15 70	430,420	10 92	5.82	5.10	70.7	0.91	
	79X12	15 615	13 33	815,927	9 36	4.55	4.81	65.1	0.58	
	79X14	12,149	14 12	672,471	8 84	4.64	4.19	59.8	1.04	
SUB TOTAL		45 100	18.52	3,273,748	10 65	5.79	4.86	76.3	0.93	
10% DILUTION				**3,601,123**	**9.68**	**5.26**	**4.42**	**69.4**	**0.85**	
A5	80X01	14,886	7 71	449,891	9 40	4.62	4.78	65.4	0.91	
SUB TOTAL		14,886	7.71	449,891	9 40	4.62	4.78	65.4	0.91	
10% DILUTION				**494,880**	**8.55**	**4.20**	**4.34**	**59.5**	**0.83**	
A6	79X18	11 821	6 53	302,578	8 59	2.87	5.72	72.1	1.14	
	80X04	13 835	3 31	179,509	11 74	5.20	6.54	81.5	1.54	
SUB TOTAL		25 655	4.79	482 086	9 76	3.74	6.03	75.6	1 29	
10% DILUTION				**530,295**	**8.88**	**3.40**	**5.48**	**68.7**	**1.17**	
A7	79X13	14 374	13 75	774,758	13 50	6 57	6.93	87.2	0.95	
	79X16	9 249	14 63	530,411	9 56	4 49	5.07	65 5	0 64	
	80X05	12 831	13 81	694 585	14 08	6 15	7 93	91 1	0 91	
	80X08	15 083	23 13	1,367,610	9 21	4 12	5 09	64 7	0 78	
	80X09	12 892	16 91	854 599	12 05	7 80	4.25	91 3	0 94	
	80X10	12 242	18 00	863 757	11 37	4.86	6 52	81 1	1.41	
SUB TOTAL		76 671	16.92	5,085 718	11 41	5.55	5.86	79 1	0 94	
10% DILUTION				**5,594,290**	**10.37**	**5.05**	**5.33**	**71.9**	**0.86**	
A8	80X02	15 061	21 49	1,268,750	9 48	3.97	5.51	51 2	0 49	
SUB TOTAL		15 061	21.49	1,268,750	9 48	3.97	5.51	51 2	0.49	
10% DILUTION				**1,395,625**	**8.62**	**3.61**	**5.01**	**46.5**	**0.45**	
A9	80X06	15,555	4 05	246,954	9 68	4.66	5.02	66 5	0.20	
SUB TOTAL		15,555	4.05	246,954	9 68	4.66	5.02	66.5	0.20	
10% DILUTION				**271,649**	**8.80**	**4.24**	**4.56**	**60.5**	**0.18**	
A10	80X07	16,938	10 21	677,894	7 10	3.22	3.88	46.2	0.80	
SUB TOTAL		16,938	10.21	677,894	7 10	3.22	3.88	46.2	0.80	
10% DILUTION				**745,684**	**6.45**	**2.93**	**3.53**	**42.0**	**0.73**	
"A ZONE" SUB TOTAL		282,982	11.86	13,159,506	10 51	5.08	5.43	72.3	0.85	
10% DILUTION				**14,475,457**	**9.56**	**4.62**	**4.94**	**65.7**	**0.78**	
B1	78X09	14 680	5 60	322,252	9 35	2.69	6.66	43.8	0.59	
SUB TOTAL		14 680	5.60	322,252	9 35	2.69	6.66	43.8	0.59	
10% DILUTION				**354,477**	**8.50**	**2.45**	**6.05**	**39.8**	**0.54**	
B2	78X04	10,363	3 34	135,683	11 04	3.38	7.66	57.8	0.46	
SUB TOTAL		10 363	3.34	135,683	11 04	3.38	7.66	57.8	0.46	
10% DILUTION				**149,251**	**10.04**	**3.07**	**6.96**	**52.5**	**0.42**	
"B ZONE" SUB TOTAL		25 043	4.66	457,935	9 85	2.89	6.96	47.9	0.55	
10% DILUTION				**503,728**	**8.96**	**2.63**	**6.32**	**43.6**	**0.50**	
Q1	77X09	24 583	18 73	1,804,948	7 20	2.42	4.78	36.8	0.30	
	78X02	13 605	11 41	608,517	6 76	2.22	4.54	29.1	0.20	
SUB TOTAL		38 188	16.12	2,413,465	7 09	2.37	4.72	34.9	0.28	
10% DILUTION				**2,664,811**	**6.44**	**2.16**	**4.29**	**31.7**	**0.25**	
Q2	79X04	9,949	3 75	146,244	6 53	3.16	3.38	43.2	0.20	
SUB TOTAL		9,949	3.75	146,244	6 53	3.16	3.38	43.2	0.20	
10% DILUTION				**160,868**	**5.94**	**2.87**	**3.07**	**39.3**	**0.18**	
Q3	77X06	6,070	6 91	164,424	7 78	2 78	5.00	53.9	0.20	
	78X11	8,784	5 81	200,048	7 58	2.62	4.96	42.3	0.47	
	91DY03	6,122	8 73	209,511	8 55	2.96	5.59	61.8	0.48	
SUB TOTAL		20,976	6.98	573,984	7 99	2.79	5.20	52.7	0.40	
10% DILUTION				**631,382**	**7.26**	**2.54**	**4.73**	**47.9**	**0.36**	
Q4	78X01	8 258	3 42	110,705	8 82	3.02	5.80	59.7	0.56	
	79X01	14,871	3 81	222,096	6 89	2.19	4.70	39.0	0.67	
	79X09	18,760	5 87	431,678	6 17	1.78	4.39	31.7	0.20	
SUB TOTAL		41,888	4.66	764,479	6 76	2.08	4.68	37.9	0.39	
10% DILUTION				**840,926**	**6.15**	**1.89**	**4.26**	**34.4**	**0.35**	
"Q ZONE" SUB TOTAL		111,001	8.96	3,898,171	7 14	2.41	4.73	38.4	0.31	
10% DILUTION				**4,287,988**	**6.49**	**2.19**	**4.30**	**34.9**	**0.29**	
UPPER-G 6% PB+ZN TOTAL		419,026	10.66	17,515,612	9 74	4.43	5.31	64.1	0.73	
10% DILUTION				**19,267,173**	**8.86**	**4.03**	**4.83**	**58.3**	**0.66**	

TABLE 6.9
GRIZZLY DEPOSIT
LOWER-G HORIZON 6% PB+ZN CUTOFF GRADE ORE BLOCKS

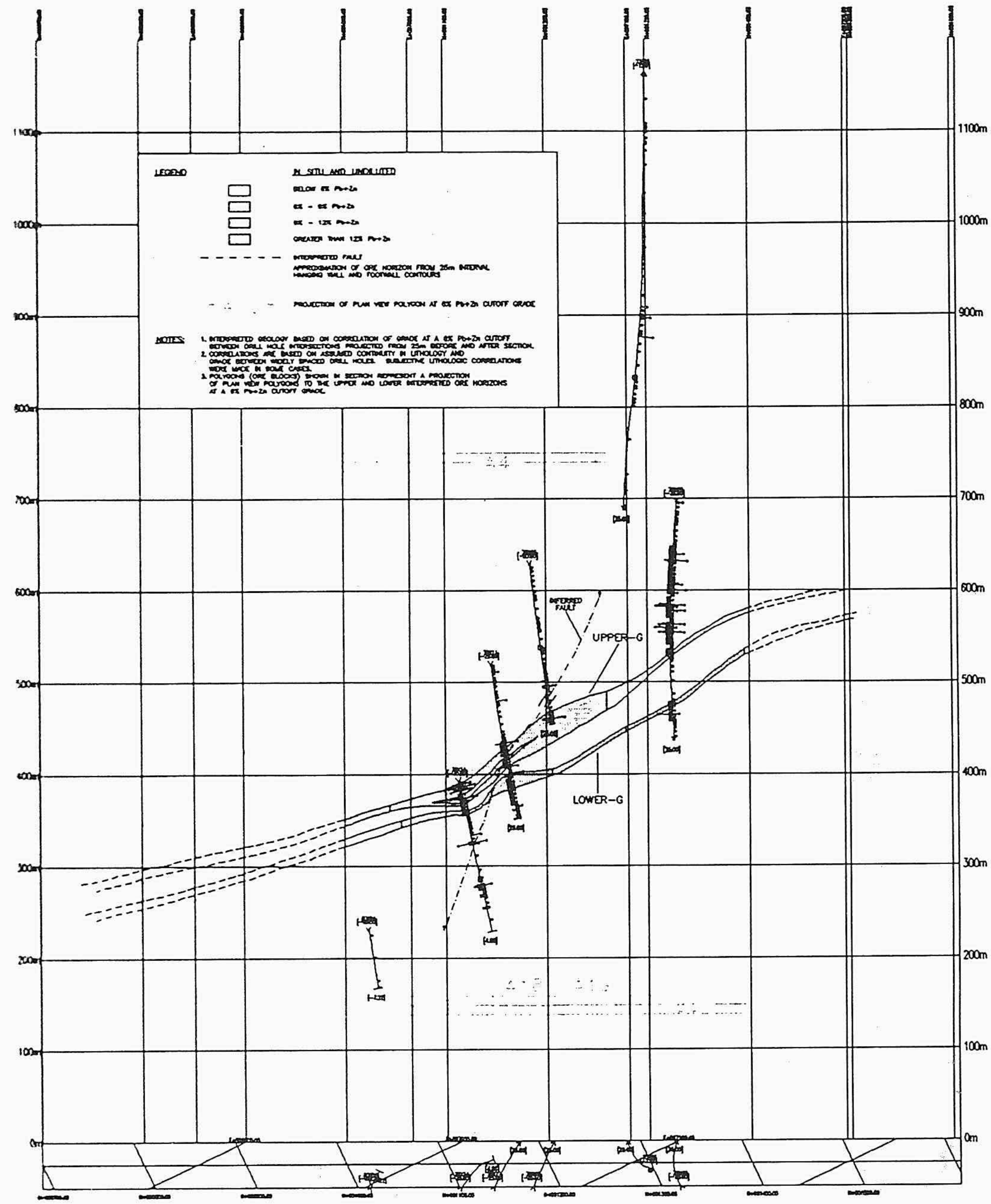
ORE BLOCK	DDH/ POLYGON	AREA (m^2)	VERT THICK (m)	TONNAGE	Pb+Zn (%)	Pb (%)	Zn (%)	Ag (g/t)	Au (g/t)	COMMENTS
A11	80X13	12 132	10.87	516 968	8.24	3.37	4.87	46.2	0.47	
SUB TOTAL		12 132	10.87	516 968	8.24	3.37	4.87	46.2	0.47	
10% DILUTION				568,665	7.49	3.06	4.43	42.0	0.43	
A12	80X07	17 407	4.04	275 673	9.88	3.74	6.14	48.9	0.79	
SUB TOTAL		17 407	4.04	275 673	9.88	3.74	6.14	48.9	0.79	
10% DILUTION				303,241	8.98	3.40	5.58	44.5	0.72	
A13	80X06	16 222	8.60	546 859	8.81	3.89	4.91	79.8	1.20	
	80X08	15 686	10.50	645 619	7.73	3.64	4.08	55.3	0.91	
SUB TOTAL		31 907	9.53	1 192 479	8.23	3.75	4.46	66.5	1.04	
10% DILUTION				1,311,726	7.48	3.41	4.06	60.5	0.95	
A14	80X02	13 977	15.20	832 815	11.24	3.88	7.36	67.7	0.98	
SUB TOTAL		13 977	15.20	832 815	11.24	3.88	7.36	67.7	0.98	
10% DILUTION				916,096	10.22	3.52	6.69	61.5	0.89	
	79X16	12 674	6.45	320,442	7.35	2.66	4.70	47.0	0.72	
A15	80X05	13 410	4.82	253,364	8.87	3.78	5.09	59.9	1.63	
SUB TOTAL		26 083	5.61	573,805	8.02	3.15	4.87	52.7	1.12	
10% DILUTION				631,186	7.29	2.87	4.43	47.9	1.02	
A16	80X09	15 332	3.23	194,125	14.65	8.80	5.85	140.2	0.94	
SUB TOTAL		15 332	3.23	194,125	14.65	8.80	5.85	140.2	0.94	
10% DILUTION				213,537	13.32	8.00	5.32	127.5	0.85	
A17	79X14	12,549	5.20	255,799	8.13	3.25	4.88	50.9	0.85	
SUB TOTAL		12 549	5.20	255,799	8.13	3.25	4.88	50.9	0.85	
10% DILUTION				281,378	7.39	2.95	4.44	46.3	0.77	
A18	79X11	7 606	15.41	459,427	11.63	6.10	5.53	90.2	1.01	
SUB TOTAL		7 606	15.41	459 427	11.63	6.10	5.53	90.2	1.01	
10% DILUTION				505,370	10.57	5.55	5.03	82.0	0.92	
A19	77X05	7 264	3.39	96,528	6.29	2.84	3.44	53.4	0.72	
	79X06	9 770	3.72	142,468	6.91	2.44	4.47	40.0	0.20	
SUB TOTAL		17 034	3.58	238,996	6.66	2.60	4.05	45.4	0.41	
10% DILUTION				262,895	6.05	2.37	3.69	41.3	0.37	
A20	76X21	16 703	6.28	411,176	8.10	2.93	5.17	55.4	0.66	
	77X01	19 269	3.45	260 595	6.75	2.62	4.13	49.5	0.20	
SUB TOTAL		35 972	4.76	671 770	7.58	2.81	4.77	53.1	0.48	
10% DILUTION				738,947	6.89	2.55	4.33	48.3	0.44	
"A ZONE" SUB TOTAL		189 998	7.00	5 211 856	9.15	3.86	5.29	63.6	0.85	
10% DILUTION				5,733,042	8.32	3.51	4.81	57.8	0.78	
B3	90DY05	15 663	17.83	1 094 743	15.98	5.89	10.09	79.7	0.45	
SUB TOTAL		15 663	17.83	1 094 743	15.98	5.89	10.09	79.7	0.45	
10% DILUTION				1,204,217	14.53	5.35	9.17	72.5	0.41	
B4	78X04	9 816	5.19	199 704	22.36	9.49	12.88	151.0	1.14	
	79X02	10 150	3.31	131 701	12.65	4.83	7.82	68.4	0.36	
SUB TOTAL		19 966	4.23	331 405	18.50	7.64	10.87	118.2	0.83	
10% DILUTION				364,546	16.82	6.94	9.88	107.4	0.75	
B5	77X06	6 263	31.28	767 939	15.96	5.61	10.35	99.8	0.54	
	78X11	7 164	9.42	264 530	12.30	4.72	7.58	77.3	0.73	
	79X07	2 722	12.25	130 713	12.03	3.99	8.04	57.2	0.61	
	90DY09	8 296	10.08	327 816	12.99	4.09	8.90	67.1	0.39	
	91DY03	6 558	19.13	491 808	9.87	3.96	5.91	56.7	0.62	
	91DY05	9 369	23.09	848 021	12.57	3.95	8.62	68.9	0.52	
SUB TOTAL		40 373	17.89	2 830 827	13.02	4.49	8.53	75.2	0.55	
10% DILUTION				3,113,910	11.84	4.08	7.75	68.4	0.50	
B6	78X09	14 010	4.83	265,251	10.83	4.26	6.58	75.4	1.12	
	79X17	13 947	4.34	237,272	8.19	2.35	5.84	37.8	0.25	
SUB TOTAL		27 956	4.59	502 523	9.58	3.36	6.23	57.6	0.71	
10% DILUTION				552,776	8.71	3.05	5.66	52.4	0.64	
B7	78X05	9 742	17.46	666,741	12.70	4.43	8.27	69.6	0.87	
	90DY04DS	7 106	11.59	322 856	9.55	3.23	6.33	40.9	0.45	
SUB TOTAL		16 848	14.98	989 597	11.67	4.04	7.64	60.2	0.73	
10% DILUTION				1,088,557	10.61	3.67	6.94	54.8	0.67	
B8	79X08	13 744	5.07	273,158	8.80	3.72	5.08	75.0	1.02	
SUB TOTAL		13 744	5.07	273,158	8.80	3.72	5.08	75.0	1.02	
10% DILUTION				300,474	8.00	3.38	4.62	68.2	0.93	
"B ZONE" SUB TOTAL		134 550	11.42	6 022,253	13.16	4.71	8.45	74.4	0.61	
10% DILUTION				6,624,479	11.96	4.29	7.68	67.7	0.56	
Q5	77X09	26 706	5.97	624,983	9.89	4.03	5.86	67.4	0.63	
	79X04	10 271	4.73	190,443	11.49	3.68	7.80	63.9	0.63	
	79X05	24 004	3.43	322,745	10.44	4.02	6.43	62.1	0.28	
SUB TOTAL		60 981	4.76	1 138 171	10.31	3.97	6.35	65.3	0.53	
10% DILUTION				1,251,988	9.38	3.61	5.77	59.4	0.48	
Q6	78X02	13 669	29.50	1 580 698	7.81	3.28	4.52	53.5	0.65	
SUB TOTAL		13 669	29.50	1 580 698	7.81	3.28	4.52	53.5	0.65	
10% DILUTION				1,738,768	7.10	2.98	4.11	48.6	0.59	
Q7	78X01	7 839	15.40	473,224	7.44	2.62	4.83	41.3	0.55	
	79X09	17 120	9.80	657 694	7.88	3.22	4.66	44.7	1.17	
SUB TOTAL		24 959	11.56	1 130 918	7.70	2.97	4.73	43.3	0.91	
10% DILUTION				1,244,010	7.00	2.70	4.30	39.3	0.83	
Q8	80X11	25 285	3.25	322 135	6.16	2.20	3.95	30.4	0.20	
SUB TOTAL		25 285	3.25	322 135	6.16	2.20	3.95	30.4	0.20	
10% DILUTION				354,349	5.60	2.00	3.59	27.6	0.18	
Q9	77X11	50 597	9.68	1 919 935	6.30	2.27	4.02	34.3	0.20	
SUB TOTAL		50 597	9.68	1 919 935	6.30	2.27	4.02	34.3	0.20	
10% DILUTION				2,111,929	5.73	2.06	3.65	31.2	0.18	
Q10	EA81X02	25 190	3.38	333 753	7.69	3.53	4.16	58.9	0.20	
	90DY07	35 251	3.79	523 709	8.76	3.66	5.10	55.8	0.39	
SUB TOTAL		60 440	3.62	857 462	8.34	3.61	4.73	57.0	0.32	
10% DILUTION				943,208	7.59	3.28	4.30	51.8	0.29	
"Q ZONE" SUB TOTAL		235 932	7.51	6,949 319	7.77	3.05	4.72	47.8	0.49	
10% DILUTION				7,644,251	7.07	2.78	4.29	43.5	0.44	
LOWER-G 6% PB+ZN TOTAL		560 480	8.28	18,183,429	9.95	3.84	6.11	61.2	0.63	
10% DILUTION				20,001,771	9.05	3.49	5.58	55.6	0.58	

TABLE 6.10
GRIZZLY DEPOSIT
UPPER-G HORIZON 9% PB+ZN CUTOFF GRADE ORE BLOCKS

ORE BLOCK #	DDH/ POLYGON	AREA (m^2)	VERT THICK (m)	TONNAGE	Pb+Zn (%)	Pb (%)	Zn (%)	Ag (g/t)	Au (g/t)	COMMENTS
A1	77x01	19,519	3.46	264,741	12.18	4.63	7.55	79.0	0.95	
	78x08	21,413	3.50	293,782	13.98	4.41	9.57	81.5	0.90	
SUB TOTAL		40,932	3.48	558,523	13.13	4.51	8.61	80.3	0.92	
10% DILUTION				**614,375**	**11.93**	**4.10**	**7.83**	**73.0**	**0.84**	
A2	77x03	23,884	3.42	320,199	11.07	6.07	5.00	81.0	0.62	
SUB TOTAL		23,884	3.42	320,199	11.07	6.07	5.00	81.0	0.62	
10% DILUTION				**352,219**	**10.06**	**5.52**	**4.55**	**73.6**	**0.56**	
A3	77x05	8,314	6.75	219,987	12.93	5.27	7.66	108.3	1.35	
SUB TOTAL		8,314	6.75	219,987	12.93	5.27	7.66	108.3	1.35	
10% DILUTION				**241,986**	**11.75**	**4.79**	**6.96**	**98.5**	**1.23**	
A4	79X06	10,272	20.49	825,061	15.19	9.24	5.95	119.7	1.18	
	79x11	7,197	13.82	389,882	11.40	5.73	5.67	75.4	0.79	
	79x12	15,584	10.17	621,258	10.36	5.03	5.33	71.6	0.67	
	79x14	12,223	10.63	509,306	9.58	5.18	4.40	67.9	1.32	
SUB TOTAL		45,275	13.22	2,345,507	12.06	6.66	5.40	88.3	1.01	
10% DILUTION				**2,580,058**	**10.97**	**6.05**	**4.91**	**80.3**	**0.92**	
A5	80x01	14,875	4.25	247,810	11.67	5.96	5.71	80.2	1.15	
SUB TOTAL		14,875	4.25	247,810	11.67	5.96	5.71	80.2	1.15	
10% DILUTION				**272,591**	**10.61**	**5.42**	**5.19**	**72.9**	**1.05**	
A6	79x18	11,773	3.36	155,067	9.87	2.96	6.91	60.9	0.93	
	80x04	13,712	3.31	177,915	11.74	5.20	6.54	81.5	1.54	
SUB TOTAL		25,485	3.33	332,982	10.87	4.16	6.71	71.9	1.26	
10% DILUTION				**366,280**	**9.88**	**3.78**	**6.10**	**65.4**	**1.14**	
A7	79x13	14,409	13.75	776,623	13.50	6.57	6.93	87.2	0.95	
	79x16	9,222	14.34	518,373	9.57	4.50	5.07	65.6	0.64	
	80X05	12,845	13.81	695,358	14.08	6.15	7.93	91.1	0.91	
	80x08	15,089	21.11	1,248,630	9.46	4.21	5.25	66.1	0.69	
	80x09	12,917	15.04	761,515	12.72	8.23	4.49	95.2	0.83	
	80x10	12,239	14.17	679,815	12.42	5.42	7.00	89.0	1.46	
SUB TOTAL		76,719	15.66	4,680,314	11.79	5.75	6.04	81.3	0.89	
10% DILUTION				**5,148,346**	**10.72**	**5.23**	**5.49**	**73.9**	**0.81**	
A8	80x02	15,150	5.83	346,220	18.77	8.85	9.92	101.7	1.21	
SUB TOTAL		15,150	5.83	346,220	18.77	8.85	9.92	101.7	1.21	
10% DILUTION				**380,842**	**17.06**	**8.05**	**9.02**	**92.5**	**1.10**	
A9	80X06	15,462	4.05	245,469	9.68	4.66	5.02	66.5	0.20	
SUB TOTAL		15,462	4.05	245,469	9.68	4.66	5.02	66.5	0.20	
10% DILUTION				**270,016**	**8.80**	**4.24**	**4.56**	**60.5**	**0.18**	
"A ZONE" SUB TOTAL		266,095	8.91	9,297,012	12.11	5.94	6.17	83.7	0.94	
10% DILUTION				**10,226,714**	**11.01**	**5.40**	**5.61**	**76.1**	**0.85**	
B1	78X09	14,680	5.60	322,252	9.35	2.69	6.66	43.8	0.59	
SUB TOTAL		14,680	5.60	322,252	9.35	2.69	6.66	43.8	0.59	
10% DILUTION				**354,477**	**8.50**	**2.45**	**6.05**	**39.8**	**0.54**	
B2	78x04	10,363	3.34	135,683	11.04	3.38	7.66	57.8	0.46	
SUB TOTAL		10,363	3.34	135,683	11.04	3.38	7.66	57.8	0.46	
10% DILUTION				**149,251**	**10.04**	**3.07**	**6.96**	**52.5**	**0.42**	
"B ZONE" SUB TOTAL		25,043	4.66	457,935	9.85	2.89	6.96	47.9	0.55	
10% DILUTION				**503,728**	**8.96**	**2.63**	**6.32**	**43.6**	**0.50**	
Q1	77X09	25,166	3.28	323,577	9.84	3.09	6.75	31.9	0.53	
SUB TOTAL		25,166	3.28	323,577	9.84	3.09	6.75	31.9	0.53	
10% DILUTION				**355,934**	**8.95**	**2.81**	**6.14**	**29.0**	**0.48**	
"Q ZONE" SUB TOTAL		25,166	3.28	323,577	9.84	3.09	6.75	31.9	0.53	
10% DILUTION				**355,934**	**8.95**	**2.81**	**6.14**	**29.0**	**0.48**	
UPPER-G 9% PB+ZN TOTAL		316,304	8.13	10,078,524	11.93	5.71	6.22	80.4	0.91	
10% DILUTION				**11,086,376**	**10.85**	**5.19**	**5.66**	**73.1**	**0.83**	

TABLE 6.11
GRIZZLY DEPOSIT
LOWER-G HORIZON 9% PB+ZN CUTOFF GRADE ORE BLOCKS

ORE BLOCK	DDH/ POLYGON	AREA (m^2)	VERT THICK (m)	TONNAGE	Pb+Zn (%)	Pb (%)	Zn (%)	Ag (g/t)	Au (g/t)	COMMENTS
A11	80X07	17.547	3.45	237,304	10.04	3.79	6.25	48.5	0.80	
	80X13	11.978	4.51	211,769	9.68	3.97	5.71	52.6	0.70	
SUB TOTAL		29.525	**3.88**	449,073	9.87	3.87	6.00	50.4	0.75	
10% DILUTION				**493,981**	**8.97**	**3.52**	**5.45**	**45.8**	**0.68**	
A13	80X06	16.254	3.38	215,359	10.80	5.71	5.09	79.3	0.89	
SUB TOTAL		16.254	**3.38**	215,359	10.80	5.71	5.09	79.3	0.89	
10% DILUTION				**236,895**	**9.82**	**5.19**	**4.63**	**72.1**	**0.81**	
A14	80X02	13.948	10.32	564,267	13.59	4.55	9.05	79.7	1.35	
SUB TOTAL		13.948	**10.32**	564,267	13.59	4.55	9.05	79.7	1.35	
10% DILUTION				**620,694**	**12.35**	**4.14**	**8.22**	**72.4**	**1.23**	
A16	80X09	15.332	3.23	194,125	14.65	8.80	5.85	140.2	0.94	
SUB TOTAL		15.332	**3.23**	194,125	14.65	8.80	5.85	140.2	0.94	
10% DILUTION				**213,537**	**13.32**	**8.00**	**5.32**	**127.5**	**0.85**	
A18	79X11	7.606	15.41	459,427	11.63	6.10	5.53	90.2	1.01	
SUB TOTAL		7.606	**15.41**	459,427	11.63	6.10	5.53	90.2	1.01	
10% DILUTION				**505,370**	**10.57**	**5.55**	**5.03**	**82.0**	**0.92**	
"A ZONE" SUB TOTAL		82.665	**5.81**	1,882,251	12.01	5.34	6.68	81.5	1.03	
10% DILUTION				**2,070,476**	**10.92**	**4.85**	**6.07**	**74.1**	**0.94**	
B3	90DY05	15.663	17.83	1,094,743	15.98	5.89	10.09	79.7	0.45	
SUB TOTAL		15.663	**17.83**	1,094,743	15.98	5.89	10.09	79.7	0.45	
10% DILUTION				**1,204,217**	**14.53**	**5.35**	**9.17**	**72.5**	**0.41**	
B4	78X04	9.840	5.19	200,199	22.36	9.49	12.88	151.0	1.14	
	79X02	10.191	3.31	132,229	12.65	4.83	7.82	68.4	0.36	
SUB TOTAL		20.031	**4.23**	332,428	18.50	7.64	10.87	118.1	0.83	
10% DILUTION				**365,670**	**16.82**	**6.94**	**9.88**	**107.4**	**0.75**	
B5	77X06	6.229	28.53	696,616	16.96	5.95	11.01	105.9	0.57	
	78X11	7.148	9.42	263,955	12.30	4.72	7.58	77.3	0.73	
	79X07	2.593	8.75	88,943	13.97	4.79	9.18	66.9	0.69	
	90DY09	8.419	10.08	332,679	12.99	4.09	8.90	67.1	0.39	
	91DY03	6.527	14.17	362,566	11.11	4.57	6.54	66.2	0.73	
	91DY05	9.366	23.09	847,746	12.57	3.95	8.62	68.9	0.52	
SUB TOTAL		40.283	**16.42**	2,592,505	13.62	4.70	8.92	79.0	0.57	
10% DILUTION				**2,851,756**	**12.38**	**4.27**	**8.11**	**71.8**	**0.52**	
B6	78X09	14.023	4.83	265,499	10.83	4.26	6.58	75.4	1.12	
SUB TOTAL		14.023	**4.83**	265,499	10.83	4.26	6.58	75.4	1.12	
10% DILUTION				**292,049**	**9.85**	**3.87**	**5.98**	**68.5**	**1.02**	
B7	78X05	9.679	13.35	506,541	14.28	4.97	9.31	77.9	1.04	
	90DY04DS	7.105	9.09	253,176	11.13	3.87	7.26	48.3	0.53	
SUB TOTAL		16.785	**11.55**	759,717	13.23	4.60	8.63	68.0	0.87	
10% DILUTION				**835,689**	**12.03**	**4.18**	**7.84**	**61.9**	**0.79**	
B8	79X08	13.752	3.41	183,820	9.99	4.35	5.64	92.6	1.09	
SUB TOTAL		13.752	**3.41**	183,820	9.99	4.35	5.64	92.6	1.09	
10% DILUTION				**202,202**	**9.08**	**3.95**	**5.13**	**84.2**	**0.99**	
"B ZONE" SUB TOTAL		120.536	11.07	5,228,711	14.10	5.09	9.01	80.3	0.65	
10% DILUTION				**5,751,582**	**12.82**	**4.62**	**8.19**	**73.0**	**0.59**	
Q5	77X09	26.576	3.27	340,666	12.10	5.01	7.09	78.7	0.98	
	79X04	10.283	4.73	190,658	11.49	3.68	7.80	63.9	0.63	
	79X05	24.053	3.43	323,403	10.44	4.02	6.43	62.1	0.28	
SUB TOTAL		60.912	**3.58**	854,727	11.34	4.34	7.00	69.1	0.64	
10% DILUTION				**940,199**	**10.31**	**3.94**	**6.36**	**62.8**	**0.58**	
Q6	78X02	13.685	16.79	900,681	9.72	4.02	5.70	64.6	0.70	
SUB TOTAL		13.685	**16.79**	900,681	9.72	4.02	5.70	64.6	0.70	
10% DILUTION				**990,749**	**8.84**	**3.65**	**5.19**	**58.7**	**0.64**	
Q7	78X01	7.785	7.02	214,236	10.14	3.55	6.59	54.2	0.79	
	79X09	17.160	3.98	267,717	9.28	3.85	5.43	48.8	0.98	
SUB TOTAL		24.945	**4.93**	481,953	9.66	3.72	5.95	51.2	0.90	
10% DILUTION				**530,149**	**8.78**	**3.38**	**5.41**	**46.5**	**0.81**	
"Q ZONE" SUB TOTAL		99.541	**5.73**	2,237,360	10.32	4.07	6.25	63.4	0.72	
10% DILUTION				**2,461,097**	**9.39**	**3.70**	**5.68**	**57.7**	**0.65**	
LOWER-G 9% PB+ZN TOTAL		302.742	7.88	9,348,323	12.78	4.90	7.88	76.5	0.74	
10% DILUTION				**10,283,155**	**11.61**	**4.45**	**7.16**	**69.6**	**0.68**	



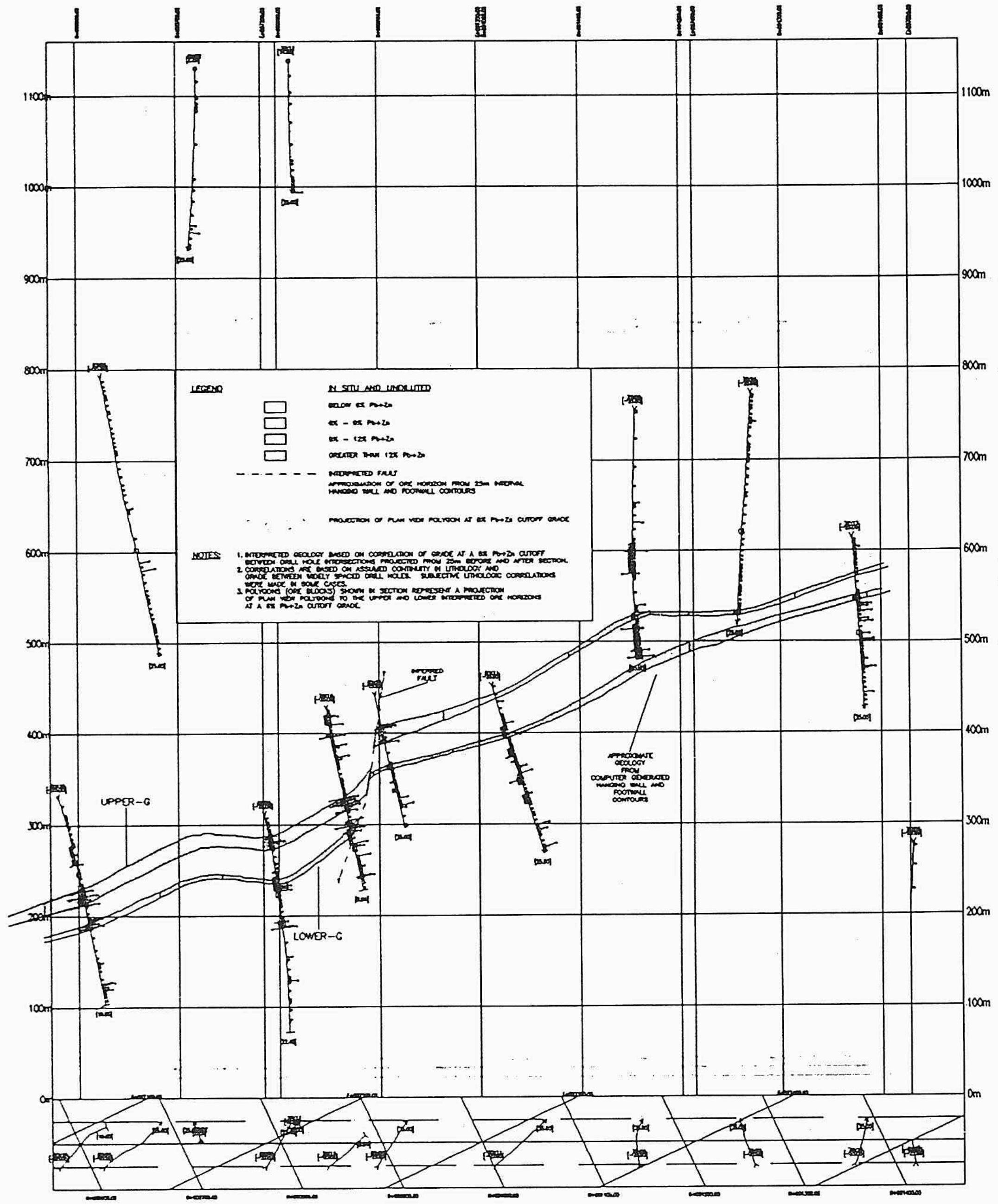
GRIZZLY PROJECT

PRE-FEASIBILITY STUDY

Fig. ~~6.2~~-02 Cross Section 300E
Interpreted Geology of Upper-G and Lower-G Ore Horizons

Anvil Range
MINING CORPORATION

SCALE	APPROVED	
DRAWING. 6202MOD.DWG	CHECKED	DATE 96/11



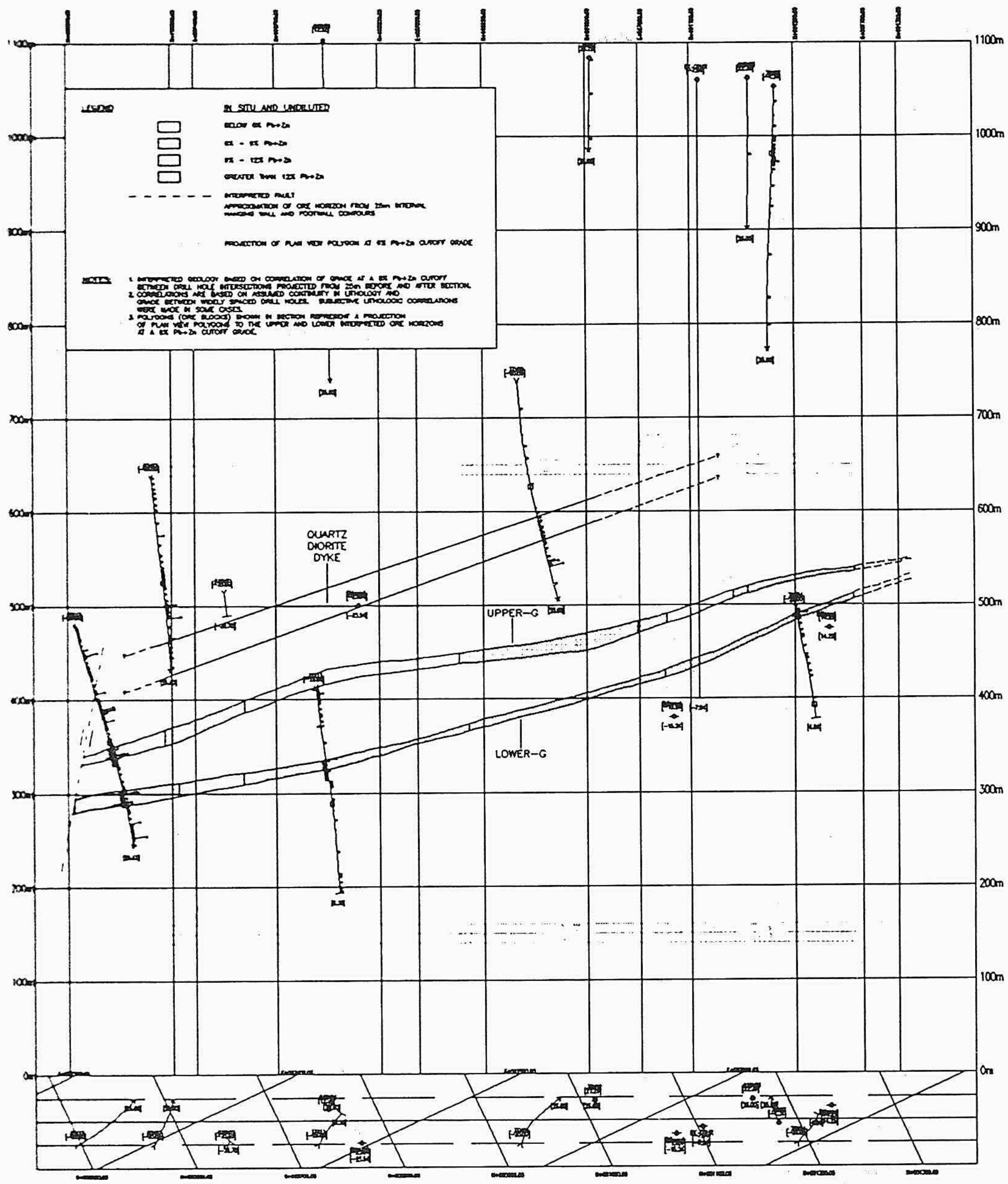
GRIZZLY PROJECT

PRE-FEASIBILITY STUDY

Fig. 6.2-03 Cross Section 600E
Interpreted Geology of Upper-G and Lower-G Ore Horizons

Anvil Range
MINING CORPORATION

SCALE	APPROVED	
DRAWING: 6203MOD.DWG	CHECKED	DATE 96/11



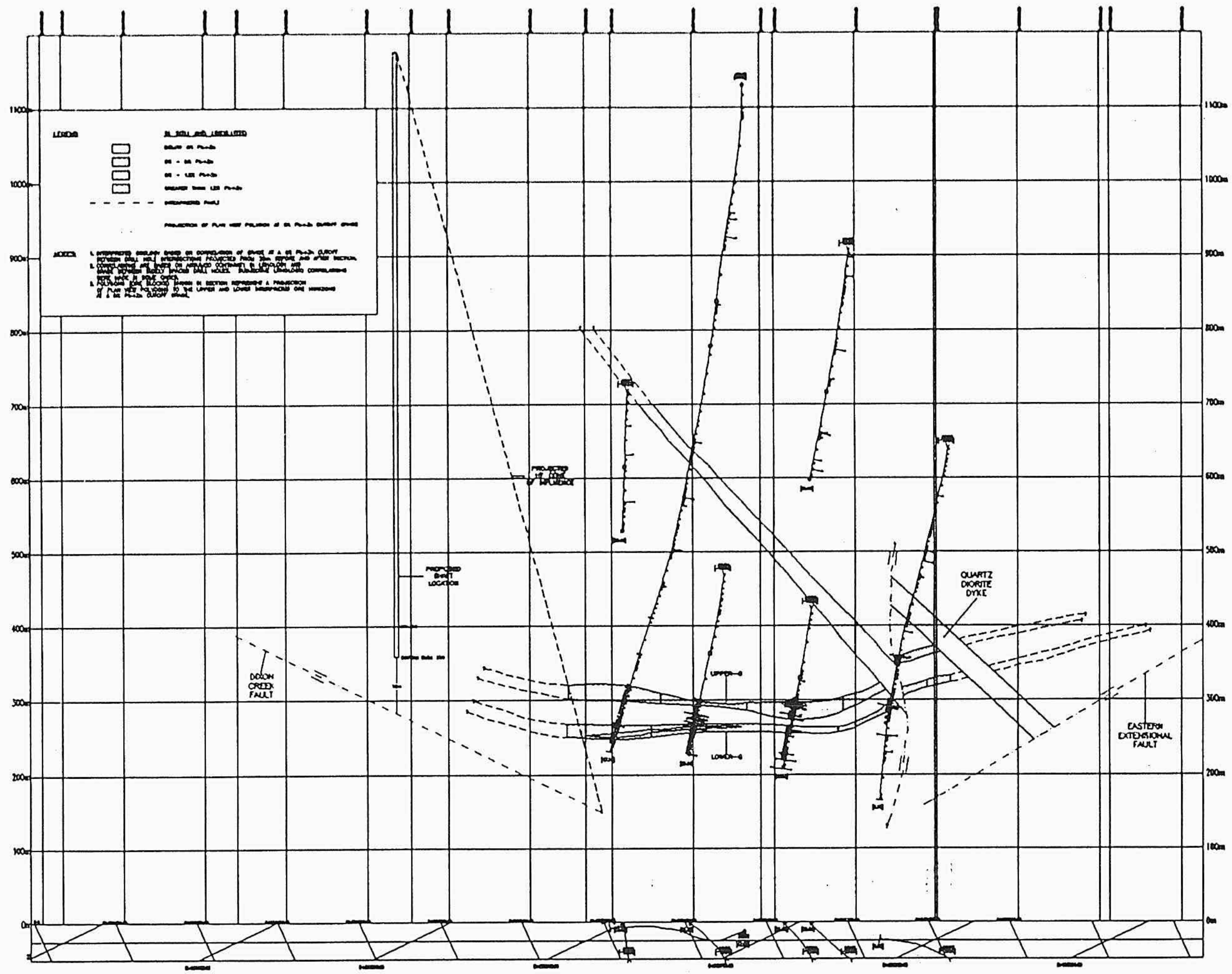
GRIZZLY PROJECT

PRE-FEASIBILITY STUDY

Fig. 6.2-04 Cross Section 850E
Interpreted Geology of Upper-G and Lower-G Ore Horizons

Anvil Range
MINING CORPORATION

SCALE	APPROVED	
DRAWING: 6204MOD.DWG	CHECKED	DATE 96/11



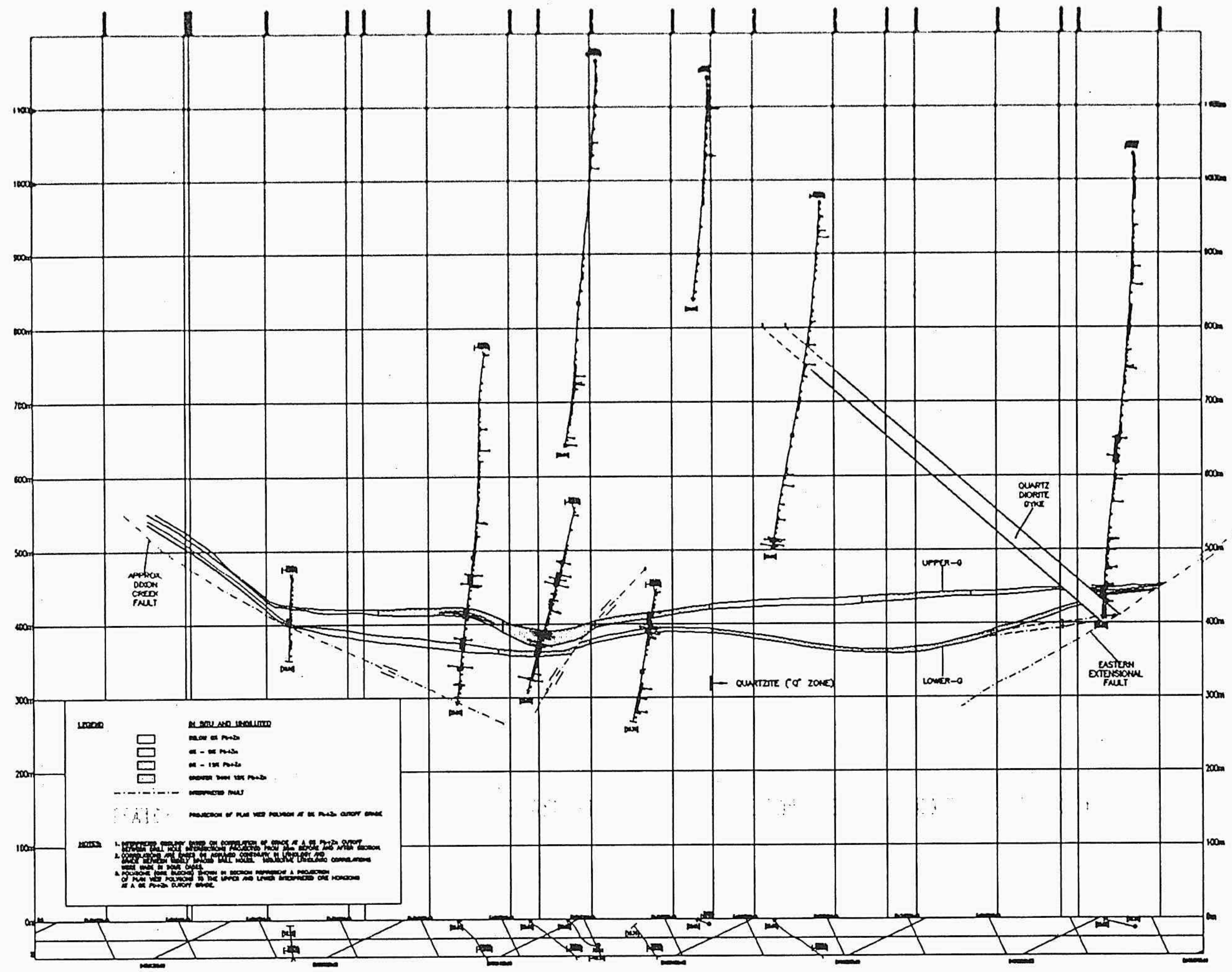
SCALE	
DRAWING:	6205MOD.DWG
APPROVED	
DATE	96/11

GRIZZLY PROJECT

PRE-FEASIBILITY STUDY

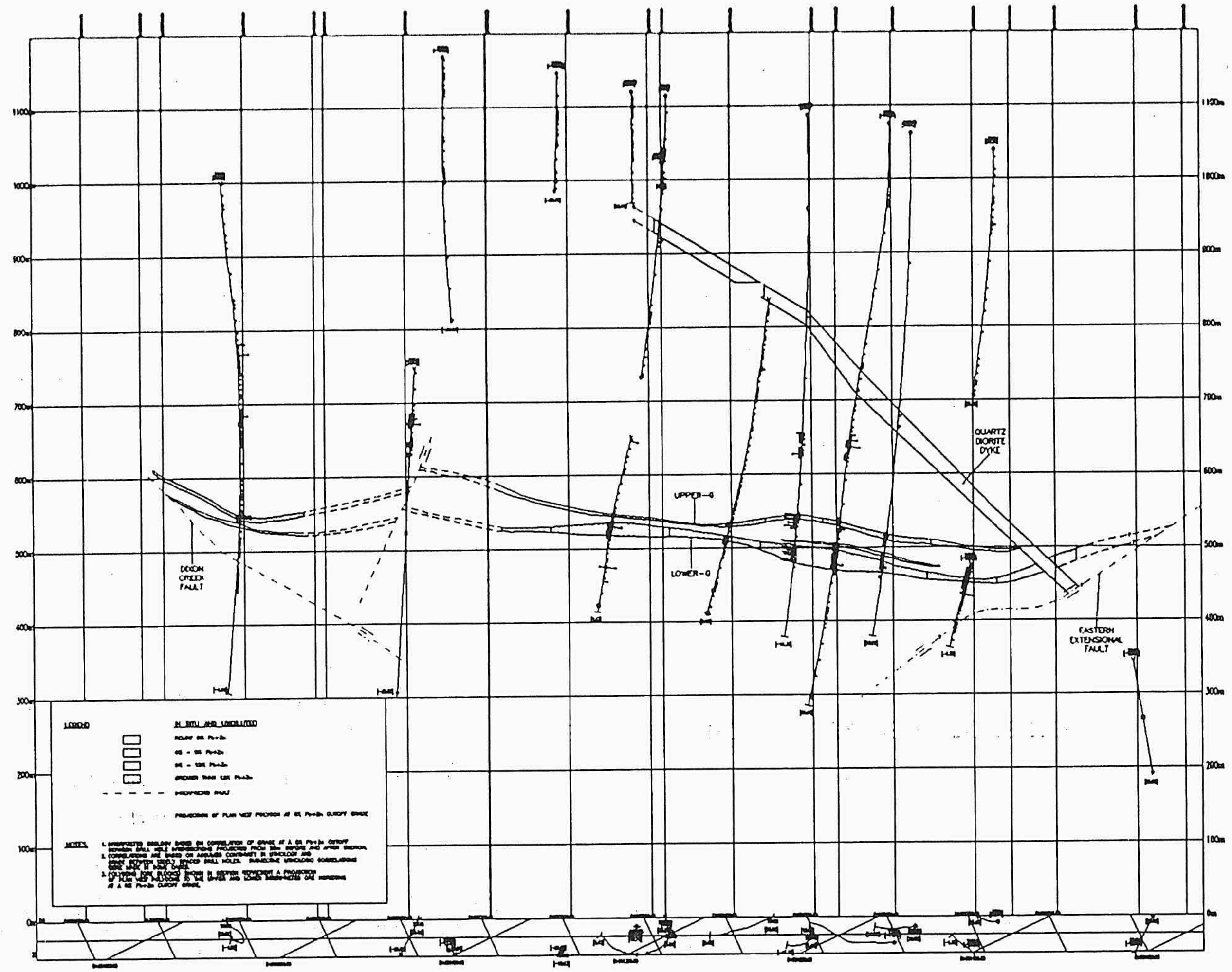
Fig. 6.2-05 Longitudinal Section 150N Interpreted Geology of Upper-G and Lower-G Ore Horizons

Anvil Range MINING CORPORATION



SCALE	**GRIZZLY PROJECT** PRE-FEASIBILITY STUDY
DRAWING: 6206MOD.DWG	
APPROVED	Fig. 6.2-06 Longitudinal Section 450N Interpreted Geology of Upper-G and Lower-G Ore Horizons
DATE 96/11	

Anvil Range
MINING CORPORATION



SCALE	
DRAWING:	6207MOD.DWG
APPROVED	
DATE	96/11

GRIZZLY PROJECT

PRE-FEASIBILITY STUDY

Fig. 6.2-07 Longitudinal Section 750N Interpreted Geology of Upper-G and Lower-G Ore Horizons

Anvil Range
MINING CORPORATION

APPENDIX L

J.S. REDPATH ESTIMATES

REDPATH

FACSIMILE

TO	Anvil Range Mining Corp. c/o Parwest Mining International	FROM	**J.S. Redpath Limited** 710 McKeown Avenue P.O. Box 810 North Bay, Ontario P1B 8K1
Name	Mr. Fritz Prugger, P. Eng.	Name	Larry Zuccherato for M. Medd
Fax	(604) 926 - 3446	Tel Fax	(705) 474-2461 (705) 474-9109
Date	18 October 1996	No. of Pages INCLUDING cover	3

COMMENTS

Fritz,

Please note that the enclosed prices are budgetary only ! They are based on a previous scope of work which has been modified to reflect changes in dimensions and concurrent activity.

Firm costs would have to be developed from first principles and take into account such things as whether or not any surface construction activities occur in winter months, etc.etc.

Hope this satisfies your immediate needs.

Regards,

L.J. Zuccherato

SUMMARY OF TARGET PRICE ESTIMATES - Budget

PROJECT: Anvil Range Mining Corporation	NO. 96053 Rev4	DATE:	18-Oct-96
AREA: Grizzly Deposit	ESTIMATED BY:	L.J.Z.	
BID ITEM: U/G Exploration Program - Ramp Option 4	CHECKED BY:		Page: 1 of 2

Item	Description	Units	Quantity	Unit Target Cost	Total Target Cost
1	Mobilization	1	Lump Sum	$257,713	$257,713
2	Surface Set up	1	Lump Sum	$702,925	$702,925
2.1	Surface preparation	1	Lump Sum	$18,554	
2.2	Setup Office Trailer	1	Lump Sum	$1,996	
2.3	Supply & Install Shop building	1	Lump Sum	$131,760	
2.4	Setup Wash/Dry Facility	1	Lump Sum	$4,015	
2.5	Septic System	1	Lump Sum	$64,800	
2.6	Settling Pond	1	Lump Sum	$64,049	
2.7	Setup Generator Facility	1	Lump Sum	$4,632	
2.8	Setup Compressor Facility	1	Lump Sum	$4,632	
2.9	Setup Explosives Facility	1	Lump Sum	$1,833	
2.10	Water Supply (Drilled Well)	1	Lump Sum	$15,817	
2.11	Main Fan, Mine Air Htr & Prop. Farm Inst'n	1	Lump Sum	$17,059	
2.12	Fuel Farm	1	Lump Sum	$8,412	
2.13	Site Services Distribution	1	Lump Sum	$33,685	
2.14	Electrical Distribution	1	Lump Sum	$331,671	
3	Secure Portal & Rehabilitation (12 metres)	1	Lump Sum	$84,575	$84,575
4	Excavate Main Decline 4.0 X 3.5 m @ -20%				
4.1.1	Excavate Decline	1,620	Metres	$1,409.58	$2,283,513
4.2	Decline Crosscuts/Muck Bays (10)	120	Equiv.Metre	$1,359.91	$163,189
4.3	Sumps (2)	24	Equiv.Metre	$1,359.91	$32,638
4.4	Transformer/Electrical Substations (3)	30	Equiv.Metre	$1,359.91	$40,797
4 5	Excav. Conveyor Decline to N.W. Exploration Drift 3.5 X 3.4 m @ -20 %	1,620	Metres	$1,042.15	$1,688,276
5	Excavate Level Access Declines 3.6 X 3.5 m @ -18%				
5.1	Exav. N.W Exploration Acc.Drift 3.8 X 4.25m @ 0 %	450	Metres	$1,780.59	$801,265
5.2	Exav. N.W Exploration Acc.Ramp 3.8 X 4.25m @ -18	450	Metres	$1,780.59	$801,265
5.3	Exav. N. Exploration Drift 3.8 X 4.25m @ 0%	250	Metres	$1,533.48	$383,369
5.4	Exav. S. Exploration Ramp 3.8 X 4.25m @ -18%	250	Metres	$1,533.48	$383,369
5.5	Muck bays	48	Equiv.Metre	$1,598.64	$76,735
5.6	Sumps	24	Equiv.Metre	$1,598.64	$38,367
5.7	Diamond Drill Stns(12)	65.5	Equiv.Metre	$1,598.64	$104,777
		1,538			
6	Safety Stations (2 cb.mtrs)	68	Each	$194.27	$13,211
7	Support Diamond Drilling Operations	0	Months	TBA	$0
				Subtotal Target Cost page 1	$7,855,984.10

SUMMARY OF TARGET PRICE ESTIMATES - Budget

PROJECT:	Anvil Range Mining Corporation	NO.	96053 Rev4	DATE:	18-Oct-96
AREA:	Grizzly Deposit	ESTIMATED BY:		L.J.Z.	
BID ITEM:	U/G Exploration Program - Ramp Option 4	CHECKED BY:			Page: 2 of 2

Item	Description	Units	Quantity	Unit Target Cost	Total Target Cost
8	Diamond Drilling Operations - BQ Core (Provisional Pricing)	13,000	Metres	$76.74	$997,560
9	Indirect Labour, Equipment & Plant Operating				
9.1	Phase 1	37	Cal. Day	$2,798.85	$103,557
9.2	Phase 2 (includes allow. for 15.4 & 15.14	235	Cal. Day	$7,212.03	$1,694,828
9 3	Phase 3 (includes allow. for 15.4 & 15.14	187	Cal. Day	$8,371.00	$1,565,377
		459			
10	Generated Power				
10.1	Phase 2 (Fuel By ARMC)	235	Eq.Cal. Day	$941.86	$221,337
10.2	Phase 3 (Fuel By ARMC)	187	Eq.Cal. Day	$1,080.00	$201,960
11	Mine Air Heat (Fuel By ARMC) (PERIOD OF OCTOBER THROUGH APRIL ONLY during 2 winter seasons)	324	Cal. Day	$0.00	By ARMC
12	Teardown	1	Lump Sum	$54,504	$54,504
13	Demobilization	1	Lump Sum	$244,388	$244,388
		Subtotal Target Cost Page 2			$5,083,511.41
			Estimated Target Cost		$12,939,495.51
		Contractor's Fee @		8.0%	$1,035,159.64
	36" /		Estimated Target Price		$13,974,655.15
NEW	Supply & install 30 inch conveyor (PROVISIONAL PRICING)	1,620	Metres	$1,250.00	$2,025,000.00
					$15,999,655.15
	Owner Supplied Items outside the target	1	Lump Sum	$2,926,228	$2,926,228.11
					$18,925,883.26
14	Suggested Contingency Items				
14.1	Grouting - Direct Crews (avg)	25	Days	$11,248.57	$0
14.2	Grouting - Indirect Crews (avg)	25	Days	$8,371.00	$0
14.3	Shotcrete - 50 mm @25% of total length	9,647	Sq.Mtr.	$75.46	$0
14.4	Shotcrete - Indirects (avg)	45	Days	$8,371.00	$0
14.5	Generated Power (avg)	70	Days	$1,080.00	$0
Note:	Contractor's fee to be added to items 14.1 to 14.5				

Anvil Range - 196591 Budget Cost/Tonne					
	Unit	Unit Cost	Estimated Quantity per Year	Yearly Cost	Cost/Tonne 5700 Tonnes/day
Direct Operating Costs					
Stope Access in waste					
Footwall Access	Lin m	$1,606.00	1,311	$2,105,996	$1.06
Hangingwall Access	Lin m	$1,606.00	1,311	$2,105,996	$1.06
Silling in Ore	Lin m	$1,606.00	4,196	$6,739,188	$3.38
Slashing in Ore	cu m	$55.00	53,402	$2,937,083	$1.47
Benching in Ore					
Drilling	Lin m	$29.00	164,416	$4,768,050	$2.39
Blasting	tonne	$1.51	1,430,415	$2,159,927	$1.08
Primary Mucking	tonne	$2.27	1,430,415	$3,247,042	$1.63
Tramming to dump					
900 metres	tonne	$2.26	1,995,000	$4,508,700	$2.26
Additional Ground Support Allowance	tonne	$1.00	1,995,000	$1,995,000	$1.00
Backfill	tonne	$8.00	1,995,000	$15,960,000	$8.00
Direct Operating Costs Subtotal				$46,526,983	$23.32
Indirect Personnel					
Supervision	day	$7,245.00	350	$2,535,750	$1.27
Mechanics/Electricians	day	$27,654.00	350	$9,678,900	$4.85
Hoisting/Materials Handling	day	$8,230.00	350	$2,880,500	$1.44
Crusher/Rockbreaker/Conveyor	day	$2,044.00	350	$715,400	$0.36
Safety/First Aid	day	$1,913.00	350	$669,550	$0.34
Misc. Ground support/UG Construction	day	$4,668.00	350	$1,633,800	$0.82
Engineering	day	$5,427.00	350	$1,899,450	$0.95
Geology	day	$3,723.00	350	$1,303,050	$0.65
Indirect Personnel Subtotal				$21,316,400	$10.68
Indirect Operating Costs					
Secondary Breaking	tonne	$0.025	1,995,000	$49,875	$0.03
Crushing/Rockbreaking	tonne	$0.080	1,995,000	$159,600	$0.08
Hoist and shaft mtce	tonne	$0.180	1,995,000	$359,100	$0.18
Material Handling underground	day	$2,960	350	$1,036,000	$0.52
Pumping	tonne	$0.065	1,995,000	$129,675	$0.07
Compressor Operating	tonne	$0.050	1,995,000	$99,750	$0.05
Power	day	$16,060	350	$5,621,000	$2.82
Small Tools	tonne	$0.050	1,995,000	$99,750	$0.05
Ongoing Misc. Ground support	tonne	$0.025	1,995,000	$49,875	$0.03
Miscellaneous shop Supplies	tonne	$0.050	1,995,000	$99,750	$0.05
Miscellaneous Construction Supplies	tonne	$0.050	1,995,000	$99,750	$0.05
Miscellaneous Ventilation Costs	tonne	$0.050	1,995,000	$99,750	$0.05
Room and Board Costs	day	$13,500.00	350	$4,725,000	$2.37
Mine Air Heating	day	$5,771.00	350	$2,019,850	$1.01
Indirect Operating Costs Subtotal				$14,648,725	$7.34
Equipment Rentals					
Rental from Contractor	month	$741,600.00	12	$8,899,200	$4.46
Operating Costs Subtotal				$91,391,308	$45.81
+ contingency @	10%			$9,139,131	$4.58
Total Estimated Operating Costs				$100,530,439	$50.39

Project: Anvil Range				By:	CAP	
Project No. 196596				Date:	11 Oct. 1996	
Equipment Type	Description	Engine	Est. Qty.	Unit HP	Total HP	
Development Jumbos	Minimatic 205D	Deutz F6L-912W	4	82	328	
Bolting Jumbos	ROBOLT	Deutz F6L-912W	2	82	164	
Production Drills	SOLO 1000 Sixty	Deutz F6L-912W	4	82	328	
LHD's - 6yd		Deutz F10L-413FW	6	231	1,386	
LHD's - 8yd		Deutz F12L-413FW	5	277	1,385	
U/G Trucks - 40t	TORO 40D	CAT 3408 PCTA	7	450	3,150	
Scissorlifts		Deutz F6L-912W	6	82	492	
Service Vehicles						
Supervision	*Allowance for vehicle*	*Deutz F6L-912W*	4	82	328	
Eng/Geology	*Allowance for vehicle*	*Deutz F6L-912W*	3	82	246	
Mech/Elec	*Allowance for vehicle*	*Deutz F6L-912W*	5	82	410	
Materials Handling Vehicles	TELEDYNE HIAB		2	185	370	
Lube Truck		Deutz F6L-912W	2	82	164	
Fuel Truck		Deutz F6L-912W	2	82	164	
Grader	CAT 12G Grader	CAT 3306	1	135	135	
Mobile shotcrete unit	*Allowance for shotcrete unit*	*Deutz F6L-912W*	2	82	164	
Anfo Loader	TELEDYNE ANFO	*Deutz F6L-912W*	4	82	328	
U/G Forklift	IT-28 Forklift	CAT3116T	1	125	125	
					9,667	
				For	75	cfm/HP
					725,025	cfm

PROJECT:
DESCR: Budget
SCHEDULE: 7 DAY/WEEK

DATE: 9-Oct-96
REVISED:

CLASS	DESCRIPTION	BASE WAGE PER MO/HR	BASE WAGE PER DAY	OVERTIME	BONUS SHARE	BONUS	BURDEN	DAILY ALLOWANCE	TOTAL MTHLY/HRLY	TOTAL PER DAY
STAFF										
	PROJ MANAGER	$8,500	$250 00		1.00	$144.00	$82.42	$38.00	$13,375.04	$514.42
	PROJ SUPER	$5,500	$211.54		1.00	$144.00	$74.38	$38.00	$12,165.84	$467.92
	Chief ENGINEER	$5,500	$211 54		0.75	$108.00	$66.85	$38.00	$11,034.03	$424.39
	ENGINEER	$5,000	$192.31		0.50	$72.00	$55.29	$38.00	$9,297.62	$357.60
	CLERK	$3,000	$115.38		0.00	$0.00	$24.14	$38.00	$4,615.60	$177.52
	Technician/Planners	$4,250	$163.46		0.33	$47.52	$44.14	$38.00	$7,621.09	$293.12
	Surveyors	$4,000	$153.85		0.25	$36.00	$39.72	$38.00	$6,956.61	$267.56
	Samplers	$3,500	$134 62		0.10	$14.40	$31.17	$38.00	$5,672.92	$218.19
	MECH/ELECT SUPER	$6,000	$230 77		1.00	$144.00	$78.40	$38.00	$12,770.44	$491.17
	CONST'N SUPER	$5,500	$211 54		1.00	$144.00	$74.38	$38.00	$12,165.84	$467.92
	PURCHASING AGENT	$4,000	$153 85		0.25	$36.00	$39.72	$38.00	$6,956.61	$267.56
	Shift Bosses	$5,000	$192.31		1.00	$144.00	$70.36	$38.00	$11,561.24	$444.66
HOURLY										
RAISE	Shaft Leader	$19 25		$2.75	1.00	$18.00	$5.09	$4.13	$49.22	$393.74
	Clam operator	$18.50		$2.64	1.00	$18.00	$4.98	$4.13	$48.25	$386.01
	SHAFTMAN	$18 00		$2.57	1.00	$18.00	$4.91	$4.13	$47.61	$380.85
	DECKMAN	$16.75		$2.39	0.50	$9.00	$3.58	$4.13	$35.85	$286.80
	SHAFT TRAINEE I	$18.00		$2.57	0.50	$9.00	$3.76	$4.13	$37.46	$299.69
	SHAFT TRAINEE II	$18.00		$2.57	0.00	$0.00	$2.62	$4.13	$27.32	$218.52
DEV	LEAD MINER	$18.75		$2.68	1.00	$18.00	$5.02	$4.13	$48.57	$388.58
	MINER	$16.75		$2.39	1.00	$18.00	$4.73	$4.13	$46.00	$367.97
	MINER II	$16.75		$2.39	0.75	$13.50	$4.16	$4.13	$40.92	$327.39
	TRUCK DRIVER	$17 00		$2.43	1.00	$18.00	$4.76	$4.13	$46.32	$370.55
	UNDERGROUND LABOURER	$16 00		$2.29	0.25	$4.50	$2.90	$4.13	$29.81	$238.49
	CAGE/SKIPTENDER	$17.00		$2.43	0.50	$9.00	$3.62	$4.13	$36.17	$289.38
	CONST'N FOREMAN	$18.00		$2.57	0.75	$13.50	$4.34	$4.13	$42.53	$340.27
	CONST'N LEADER	$18 00		$2.57	0.7	$12.60	$4.22	$4.13	$41.52	$332.15
	CONST'N MINER	$17 00		$2.43	0.7	$12.60	$4.08	$4.13	$40.23	$321.85
	CONST'N HELPER	$16.00		$2.29	0.25	$4.50	$2.90	$4.13	$29.81	$238.49
SUPPORT	LEAD/SHAFT MECH	$18 40		$2.63	0.75	$13.50	$4.40	$4.13	$43.05	$344.39
	MECHANIC	$17 00		$2.43	0.50	$9.00	$3.62	$4.13	$36.17	$289.38
	Crusher Operator/Conveyorman	$16 75		$2.39	0.50	$9.00	$3.58	$4.13	$35.85	$286.80
	Rigger/Shaft Inspection	$16 75		$2.39	0.75	$13.50	$4.16	$4.13	$40.92	$327.39
	LEAD ELECTRICIAN	$18.40		$2.63	0.75	$13.50	$4.40	$4.13	$43.05	$344.39
	ELECTRICIAN	$17 00		$2.43	0.50	$9 00	$3.62	$4.13	$36.17	$289.38
	Underground Labourer	$16 00		$2.29	0.25	$4 50	$2.90	$4.13	$29.81	$238.49
	HOISTMAN	$17 30		$2.47	0.50	$9 00	$3.66	$4.13	$36.56	$292.47
	SURFACE LABOURER	$12 00		$1.71	0.00	$0.00	$1.75	$4.13	$19.59	$156.68
	SURF EQUIP'T OPER	$16 00		$2.29	0.25	$4 50	$2.90	$4.13	$29.81	$238.49
	DRYMAN	$12.00		$1.71	0.00	$0.00	$1.75		$15.46	$123.68
	WATCHMAN	$12 00		$1.71	0.00	$0.00	$1.75		$15.46	$123.68

Project: Anvil Range				By:	CAP	
Project No. 196596				Date:	11 Oct. 1996	
Equipment Type	**Description**	**Engine**	**Est. Qty.**	**Unit HP**	**Total HP**	
Development Jumbos	Minimatic 205D	Deutz F6L-912W	4	82	328	
Bolting Jumbos	ROBOLT	Deutz F6L-912W	2	82	164	
Production Drills	SOLO 1000 Sixty	Deutz F6L-912W	4	82	328	
LHD's - 6yd		Deutz F10L-413FW	6	231	1,386	
LHD's - 8yd		Deutz F12L-413FW	5	277	1,385	
U/G Trucks - 40t	TORO 40D	CAT 3408 PCTA	7	450	3,150	
Scissorlifts		Deutz F6L-912W	6	82	492	
Service Vehicles						
Supervision	*Allowance for vehicle*	*Deutz F6L-912W*	4	82	328	
Eng Geology	*Allowance for vehicle*	*Deutz F6L-912W*	3	82	246	
Mech/Elec	*Allowance for vehicle*	*Deutz F6L-912W*	5	82	410	
Materials Handling Vehicles	TELEDYNE HIAB		2	185	370	
Lube Truck		Deutz F6L-912W	2	82	164	
Fuel Truck		Deutz F6L-912W	2	82	164	
Grader	CAT 12G Grader	CAT 3306	1	135	135	
Mobile shotcrete unit	*Allowance for shotcrete unit*	*Deutz F6L-912W*	2	82	164	
Anfo Loader	TELEDYNE ANFO	*Deutz F6L-912W*	4	82	328	
U/G Forklift	IT-28 Forklift	CAT3116T	1	125	125	
					9,667	
				For	75	cfm/HP
					725,025	cfm

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JS Redpath Ltd. 11-Oct-96

Anvil Range Grizzly Project

Order of Magnitude Estimate : Shaft 18' Diameter Concrete Lined Bald

Item #	Description	Estimated Units	Quantity	Unit Cost	Estimated Cost
1	Mobilization & Engineering	1	Lump Sum	$950,000	$950,000
2	Surface Setup	1	Lump Sum	$798,987	$798,987
3	Sinking Headframe & Collarhouse	1	Lump Sum	$595,000	$595,000
4	Sinking Hoist Foundations & Installation	1	Lump Sum	$650,000	$650,000
5	Setup & Install Sinking Equipment	1	Lump Sum	$765,000	$765,000
6	Excavate & Line Collar	120	Feet	$5,500	$660,000
7	Sink & Line 18' Diameter Shaft	2800	Feet	$3,145	$8,806,000
8	Station Excavation 1st Level	12500	Ft3	$11.25	$140,625
9	Station Excavation 2nd Level	12500	Ft3	$11.25	$140,625
10	Station Excavation 3rd Level	12500	Ft3	$11.25	$140,625
11	Station Construction	3	Per Station	$65,000.00	$195,000
12	Loading Pocket Excavation	11520	Ft3	$11.25	$129,600
13	Loading Pocket Supply & Install	1	Lump Sum	$625,000	$625,000
14	Remove Sinking Plant	1	Lump Sum	$250,000	$250,000
15	Supply & Install Shaft Bottom Steel	1	Lump Sum	$115,000	$115,000
16	Remove Sinking Hoist & Headframe	1	Lump Sum	$385,000	$385,000
17	Supply & Install Concrete Headframe	1	Lump Sum	$2,700,000	$2,700,000
18	Supply & Install Surface Bins & Collarhouse	1	Lump Sum	$2,500,000	$2,500,000
19	Friction Hoist Purchase	1	Lump Sum	$2,390,000	$2,390,000
20	Friction Hoist Installation	1	Lump Sum	$450,000	$450,000
21	Shaft Guide Ropes & Conveyances Supply	1	Lump Sum	$975,000	$975,000
22	Shaft Guide Ropes & Conveyances Install	1	Lump Sum	$328,000	$328,000
23	Demobilization	1	Lump Sum	$310,000	$310,000
	Total Cost				$24,999,462

JS Redpath Ltd. 11-Oct-96

Anvil Range Grizzly Project

Order of Magnitude Estimate : Bin Excavation & Lining

25' Diameter Bin by 100 ' High and Lined with Pattern Rebar, Concrete and Rail

Estimated 2400 Tonne Live Storage

Item #	Description	Estimated Units	Quantity	Unit Cost	Estimated Cost
1	Mobilization & Engineering	1	Lump Sum	$150,000	$150,000
2	Bin Pilot Raise Setup	1	Lump Sum	$15,000	$15,000
3	Bin Pilot Raise (8' diameter)	100	Feet	$430	$43,000
4	Strip Alimak Setup	1	Lump Sum	$15,000	$15,000
5	Setup on Top	1	Lump Sum	$45,000	$45,000
6	Slash Bin to 25' Diameter & Ground Support	100	Feet	$4,166	$416,600
7	Line Bin with Concrete & Rail	100	Feet	$9,765	$976,500
8	Mucking Raise & Slash	1	Lump Sum	$19,500	$19,500
9	Strip Bin Setup	1	Lump Sum	$25,000	$25,000
10	Supply Bin Bottom Chute	1	Lump Sum	$125,000	$125,000
11	Install Bin Bottom Chute	1	Lump Sum	$110,000	$110,000
12	Demobilize	1	Lump Sum	$25,000	$25,000
	Total Price				$1,965,600

Order of Magnitude Estimate : 10' Diameter Raisebore Hole Lining

Lined with Shotcrete to 2" thick from Top Down Alimak Setup

Item #	Description	Estimated Units	Quantity	Unit Cost	Estimated Cost
1	Mobilization & Engineering	1	Lump Sum	$75,000	$75,000
2	Setup	1	Lump Sum	$30,000	$30,000
3	Line Raise : Supply & Install Shotcrete	2200	Feet	$250	$550,000
4	Strip Raise	1	Lump Sum	$55,000	$55,000
5	Teardown & Demob	1	Lump Sum	$25,000	$25,000
	Total Cost				$735,000

REDPATH

MEMORANDUM

To: Roy Slack

Date: October 11, 1996

From: Robert E. Bettiol

Subject: ORDER OF MAGNITUDE - RAISEBORING
2000 ft. BY 8 ft. DIAMETER - YUKON

Item	Description	Est. Qty	Unit	Est. Unit Price	Est. Total Cost
1.	Mobilization	1	L.S.	$102,000.00	$102,000.00
*2.	Move, set up, tear down	14	DAY	$4,890.00	$68,460.00
3.	Drill 13.75" pilot	2000	FT.	$205.00	$410,000.00
4.	Ream to 8 ft. diameter	2000	FT.	$418.00	$836,000.00
5.	Demobilization	1	L.S.	$102,000.00	$102,000.00

Estimated Total Project Cost **$1,518,460.00**

*Estimated number of days required, actual days are invoiced.

Note: *These numbers are based on the following parameters:*

Ground hardness	-	*Up to 70 MPA*
Crews operating	-	*85R raisedrill*
Crews working	-	*2 X 12hr. shifts 7 days/week*

Schedule	Days
Mobilization on site	3
Move, set up, tear down	14
Drill 13.75" pilot	16
Ream to 8 ft. diameter	45
Reaming contingency	10
Demobilization off site	3
Estimated total days required	91

Anvil Range - 196591		
Manpower - Working		
Direct Labour		
Development Miners	48	
Production Drillers	12	
Load and Blast Crew	6	
Primary Mucking - Benced Muck	10	
Tramming	15	
Backfill	16	
Total Directs		**107**
Indirect Labour		
Supervision	14	
Mechanica/Electrical	78	
Hoist/Materials handling	26	
Crusher/Rockbreaker/Conveyor crew	6	
Safety/First aid/Training	4	
Misc. Underground construction	12	
Engineering	15	
Geology	11	
Total Indirects		**166**
Total Operating manpower		**273**

Equipment type	Unit cost	Estimated Quantity	Total Cost	Cost/Month Rental @ 2.90%
Development Jumbo's	$605,000	4	$2,420,000	$70,180
Bolting Jumbo's	$640,000	2	$1,280,000	$37,120
Production Drills	$350,000	4	$1,400,000	$40,600
LHD's - 6yd	$401,815	6	$2,410,890	$69,916
LHD's - 8yd	$758,000	5	$3,790,000	$109,910
Remotes for LHD's	$50,000	5	$250,000	$7,250
Underground trucks - 40t	$640,000	7	$4,480,000	$129,920
Scissorlifts	$185,000	6	$1,110,000	$32,190
Service Vehicles				
Supervision	$50,000	4	$200,000	$5,800
Eng/Geology	$50,000	3	$150,000	$4,350
Mechanical/Electrical	$50,000	5	$250,000	$7,250
Materials Handling Vehicles	$125,000	2	$250,000	$7,250
Lube Truck	$183,560	2	$367,120	$10,646
Fuel Truck	$172,600	2	$345,200	$10,011
Grader	$125,000	1	$125,000	$3,625
Mobile shotecrete unit	$250,000	2	$500,000	$14,500
Anfo Loaders	$250,000	4	$1,000,000	$29,000
Underground Forklift	$171,000	1	$171,000	$4,959
			$0	$0
			$0	$0
			$0	$0
			$0	$0
			$0	$0
			$0	$0
			$0	$0
Subtotal			$20,499,210	$594,477
+ Contingency @	5.00%			$29,724
Cost/Month				$624,201
+ Markup @	18.81%			$117,412
Price/month				$741,613

ANVIL RANGE - 18 FT DIAMETER SHAFT TO 2,920 FT

Set Up	6 months	180 days	
Shaft		300 days	
Stations	4 x 14 days	56 days	
LP Install	2 weeks	14 days	
Bottom Steel	1 week	7 days	
Remove Sinking Plant		30 days	
Friction Hoist		60 days	
Headframe		60 days	
Shaft Ropes		30 days	
		737 days	
	Cont	30 days	
		767 days	**25.6 months**

BIN LINING AND EXC.

Pilot Raise Setup		3 days	
Pilot 100 ft (8 ft dia.)		7 days	
Strip		3 days	
Setup on Top		7 days	
Slash & Support	100 ft to 25 ft dia.	25 days	
Conc. & Line (Rail & Conc)		25 days	
Mucking		Concurrent	7 days
Strip Bin Setup		7 days	
Install Bottom Chute		20 days	
		97 days	
	Cont. 10%	10 days	
		107 days	

3.5 months based on 3 shifts/day, 7 days/week

RAISE LINING - SHOTCRETE 2 INCHES - 10 FT DIAMETER

Setup		7 days
Line Raise	2,200 ft @ 50 ft/day	44 days
Strip Raise		7 days
Teardown		7 days
		65 days
	Cont. 20%	13 days
		78 days

Base on 3 shifts/day, 7 days/week

Project: Anvil Range				By:	CAP	
Project No. 196596				Date:	11 Oct. 1996	
Equipment Type	**Description**	**Engine**	**Est. Qty.**	**Unit HP**	**Total HP**	
Development Jumbos	Minimatic 205D	Deutz F6L-912W	4	82	328	
Bolting Jumbos	ROBOLT	Deutz F6L-912W	2	82	164	
Production Drills	SOLO 1000 Sixty	Deutz F6L-912W	4	82	328	
LHD's - 6yd		Deutz F10L-413FW	6	231	1,386	
LHD's - 8yd		Deutz F12L-413FW	5	277	1,385	
U/G Trucks - 40t	TORO 40D	CAT 3408 PCTA	7	450	3,150	
Scissorlifts		Deutz F6L-912W	6	82	492	
Service Vehicles						
Supervision	*Allowance for vehicle*	*Deutz F6L-912W*	4	82	328	
Eng/Geology	*Allowance for vehicle*	*Deutz F6L-912W*	3	82	246	
Mech/Elec	*Allowance for vehicle*	*Deutz F6L-912W*	5	82	410	
Materials Handling Vehicles	TELEDYNE HIAB		2	185	370	
Lube Truck		Deutz F6L-912W	2	82	164	
Fuel Truck		Deutz F6L-912W	2	82	164	
Grader	CAT 12G Grader	CAT 3306	1	135	135	
Mobile shotcrete unit	*Allowance for shotcrete unit*	*Deutz F6L-912W*	2	82	164	
Anfo Loader	TELEDYNE ANFO	*Deutz F6L-912W*	4	82	328	
U/G Forklift	IT-28 Forklift	CAT3116T	1	125	125	
					9,667	
				For	75	cfm/HP
					725,025	cfm

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APPENDIX M

STEFFEN, ROBERTSON
AND KIRSTEN (CANADA) INC.

REPORT DATED
AUGUST 6, 1992

1966 Mine - Underground Projects.

STEFFEN, ROBERTSON AND KIRSTEN (CANADA) INC. Consulting Engineers
Suite 800 580 Hornby Street, Vancouver B C Canada V6C 3B6
Phone (604) 681-4196 Fax (604) 687-5532

August 6, 1992
Project Number 60652

Curragh Resources
Box 1000
Faro, Yukon
Y0B 1K0

Attention: Leo Hwozdyk

Dear Leo:

RE: CONCEPTUAL MINING METHODS - DY DEPOSIT

1.0 INTRODUCTION

The Dy deposit is a high grade massive sulphide deposit located near Faro, Yukon Territory. It is located at a depth of 530m to 880m. The deposit ranges in thickness from 4m to 27m, with the thickness being locally quite variable. Folding has affected both the strike and the dip of the deposit. The strike changes direction by 90 degrees in the northern portion of the orebody as compared to the south. Dip can vary from 15 to 30 degrees, with the average being approximately 25 degrees.

Mining of the deposit is being contemplated. The purpose of this letter is to examine some of the potential mining methods which may be utilized in the deposit, detail the advantages and disadvantages of each method, as well as delineate any further problems which may need to be addressed in a full scale feasibility study.

2.0 GEOTECHNICAL BACKGROUND

The Dy orebody is genetically and structurally similar to the Faro deposit. Due to its depth, the only information available is drillhole derived. As such, the data may be biased both due to drill deviation as well as to forced interpretations based on relatively wide drillhole spacings. Vertical, or sub-vertical, faulting would not be apparent in vertical drillholes. Such faults do exist, as can be noted from geologic interpretations. Thus, what is described here may only be taken as an approximate description of what may be encountered.

Ore is hosted in a sequence of phyllites, schists, and quartzites. The phyllites and schists, which compose the hanging wall, have an average RMR of approximately 20-35. They are foliated with a very pronounced parting being imparted to the rock mass. The foliation is extremely weak and separates with only minor displacement. Estimated compressive strength of the materials, perpendicular to foliation is between 14 - 35 MPa. The ore appears to be similar to the Faro ore. However, the only core which could be examined during the March 5, 1991 site visit had been split. RMR for the sulphides appeared to be between 45 to 60. This equates to that of the Faro deposit. For that reason, a design rock mass strength of 54 MPa (Faro's estimated strength) will be used for the Dy massive sulphide.

In-situ stresses will be much higher at Dy than within the operational envelope of the Faro deposit. Vertical overburden stress within the ore should range from 13 MPa to 22 MPa, compared to a maximum of 7 MPa for the Faro underground operation. Horizontal stresses are unknown but could be up to twice the vertical stress.

3.0 MINING METHOD SELECTION

When choosing a mining method for a deposit, the physical constraints of the orebody and the required production rates become paramount. Of these two parameters, the physical constraints of the orebody will always control the eventual maximum sustained production rate of the mine.

Some of the physical parameters controlling the mining method are:

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

For Dy, the main constraints are the irregular, intermediate dip which complicates the method ultimately selected, the high ore strength with a weak hanging wall, moderate ore thickness, and relatively high vertical in-situ stresses as well as potential tectonic stresses.

For our intents, two broad categories of mining may be defined. These, partial extraction and full extraction of the orebody, are described below.

3.1 Partial Extraction

Partial extraction entails the removal of only a portion of the orebody, with some of the orebody being used as non-recoverable support. This is most readily typified by room and pillar mining, without pillar recovery, as practised at the Faro underground operation. Various stoping mining methods, such as open stoping with non-recoverable pillars are also included in this class.

Room and pillar mining without pillar recovery is possible in the Dy orebody. However, the weak hanging wall will likely require that ore be left in the back for support purposes. Recoveries would likely not exceed 50% of the mineable reserve. Table 3.1 illustrates the maximum theoretical recoveries for varying safety factors in certain zones of the orebody. Partial pillar recovery combined with numerical analysis and practical experience may allow one to approach or exceed these maximum theoretical values. For a feasibility study it would be prudent to utilize approximately 80%-90% of the theoretical value.

TABLE 3.1

Maximum theoretical recovery, standard room and pillar

Mining area	Pillar stress (MPa)	% Recovery @ SF=1.5	%Recovery @ SF=1.3
Southern zone	23	36	47
Northern zone	17	53	59
Western zone	21	42	49

Given the recoveries mentioned in the above table, it appears unlikely that room and pillar without pillar recovery is a viable alternative for mining.

3.2 Complete Extraction

Complete extraction of an orebody entails recovery of most of the delineated mining reserve. This can be typified by cut-and-fill, caving methods, or concrete pillar mining (a variant of cut-and-fill). All of these will be examined within this section.

3.2.1 Cut-and-Fill

Cut and fill could be operated in a post-pillar format using either a room and pillar layout or long rib pillars. Conventional post pillar cut and fill is usually operated within thicker, steeper dipping, deposits. Using this method at Dy would be a departure from current experience.

Cut and fill is usually operated with hydraulic fill (is fill available) which is both expensive and has a low productivity. Large areas would be open at any one time. This could require barrier pillars. A fairly low, overall recovery might be achieved.

Alternative fills could be used but this would not change the requirement for large, open areas.

Finally, the low dip might lead to considerable waste footwall development for access purposes.

We have not considered cut and fill in any detail as other methods seem more practical in this type of orebody.

3.2.2 Caving

Caving has been discussed as an alternative mining method for the Dy deposit. This has been discussed primarily in conjunction with the Cascade method (Mabson & Russell, 1981). A diagrammatic sketch describing this method, taken from the paper, is shown in Figure 3.1 overleaf. This method, although having possibilities, is not really suited for the conditions existing at Dy.

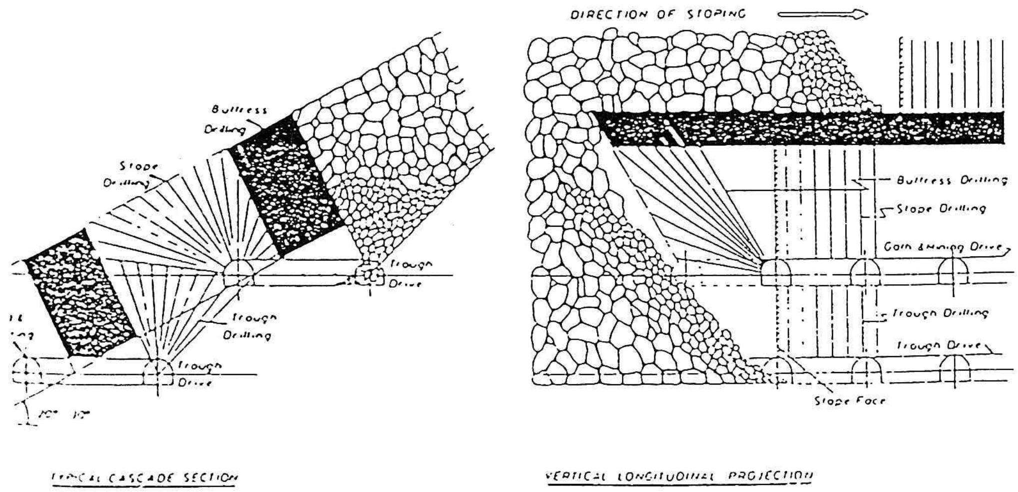

Figure 3.1 Cascade method (after Mabson & Russel, 1981)

The Cascade method was developed at the Mufilira mine for moderately dipping orebodies (20-40 degrees) with a thickness of 5m - 16m. This would be similar to Dy. However, the method works only if a large span can be maintained in the actual mining stope. Mufilira is characterized by extremely competent ground both in the orebody and the hanging wall. Given the apparent weak hanging wall and the high vertical stresses at Dy, the span which could be opened before substantial roof instability occurs is likely on the order of 10m to 15m. In addition, the buttress pillars will have to be quite large to provide protection against the live load of the caved muck as well as the overburden load. This results in relatively small rooms with large buttress pillars. Extraction during the primary phase would likely give about 35-45% recovery. Secondary mining of the buttress pillars would allow another 20-30% giving an overall extraction of between 55-70% with perhaps 15-40% dilution.

The Cascade method is development intensive, with all development conducted in the footwall. It could be a relatively high production method if worked properly. However, much depends on the actual ground conditions encountered in the immediate mining area. Recovery, although possibly better than room and pillar, would be offset by the large amounts of footwall development and very high dilution.

A method of caving using post-pillars is also possible. This is shown in Figure 3.2. The method would only work in relatively thin ore (5m-8m).

Main headings are driven on relatively wide spacings, leaving large strike pillars as buttresses. This are reduced to post pillars, with the remaining large pillars acting as the breaker line. Mucking is conducted under the protection of the post-pillars near the breaker line. These pillars will later collapse as full roof loads are applied. Recovery in this instance could be from 70-80%.

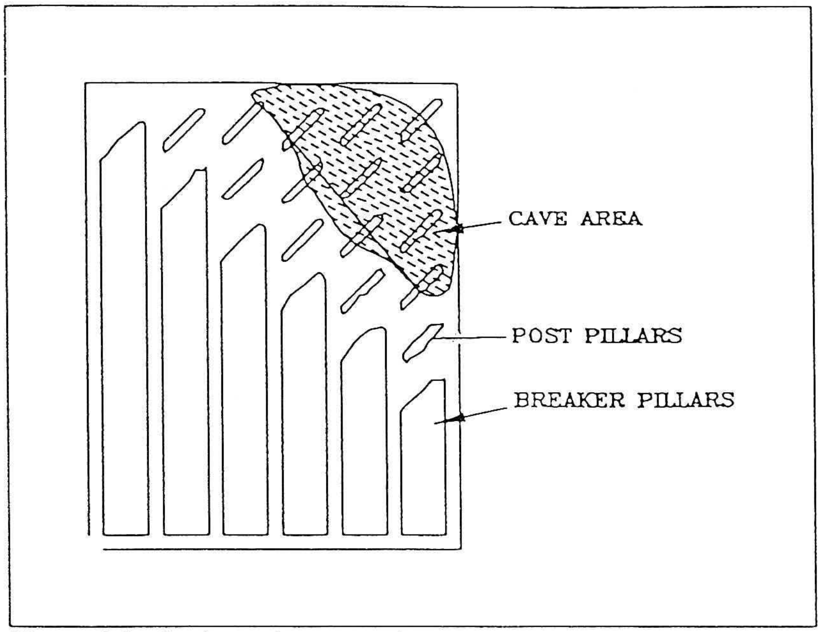

Figure 3.2 Caving using post pillar recovery

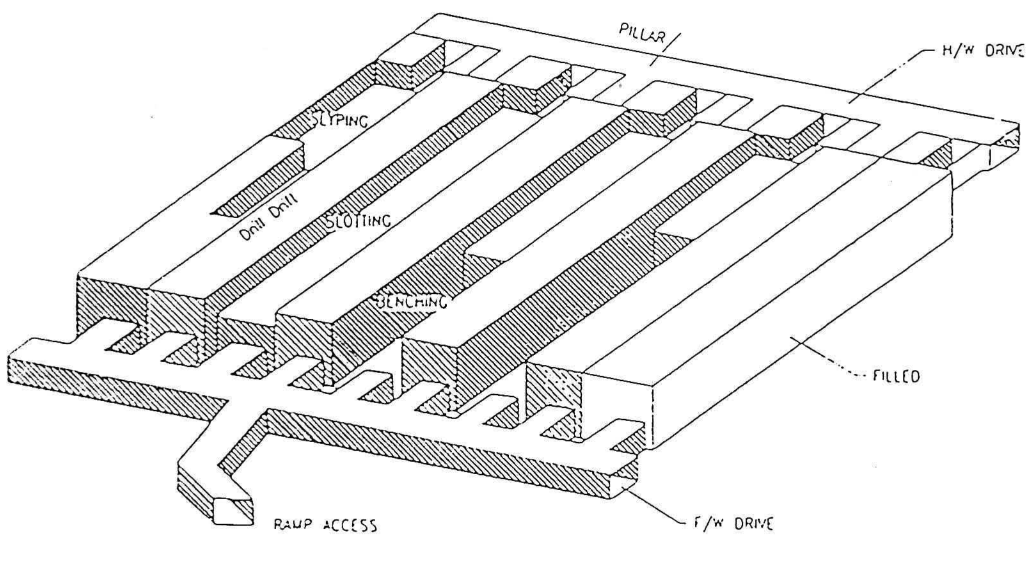

Figure 3.3 Concrete pillar mining

This method relies on a relatively weak hanging wall and strong ore. It may be applicable to the Dy but would be limited to the ore thicknesses mentioned above. It is also largely unproved except in copper mining in Poland and in coal mining.

3.2.3 Concrete Pillar

The method with possibly the most potential for the Dy deposit is the concrete pillar method. This method, shown diagrammatically in Figure 3.3, utilizes fill pillars for support of hanging wall as secondary extraction is conducted. This results in complete recovery of thin to moderately thick orebodies.

The method is based on developing extraction panels within the orebody (Figure 3.3). Pillars are left between the panels for roof support. These pillars can be designed to support the entire overburden load or as post-pillars. The primary extraction panels are then filled with a high quality cemented fill (cemented rock fill). The pillars between the newly created concrete filled panels are then excavated. These, upon completion, can be backfilled with waste rock or fill. Extraction should approach 90-100%.

For feasibility analysis purposes, primary and secondary extraction panels have been assigned an equal width of 8m. This span should be stable if a reinforceable sulphide skin is left against the back and if the pillars are not subject to full overburden load.

Panel lengths have been restricted to a maximum of 100m due to the irregularity of the deposit (pinch and swell). In addition, if remote equipment is to be used, visual observation of the machine is recommended. This is difficult even at 100m.

Thin sections of the deposit (up to 7m) can be mined as one pass operations. A single heading would be driven at mining width, then slashed to 8m. Fill is placed tight against the back, allowed to cure, and the pillars extracted in a fashion similar to the primary room mining. Tight placement of high quality fill is a must for the concrete pillars to function properly. Their purpose is to prevent large roof displacements, as well as carrying some minor stress. If roof displacement is allowed to occur, it will be followed shortly by roof collapse.

For Dy, an 8m panel and pillar width have been assumed which equates to a 50% primary extraction ratio. From Table 3.1, it can be noticed that pillars with a 50% extraction ratio will not be stable for full overburden support. This means that the panel pillars (secondary recovery areas) will be, in most cases, behaving as post-pillars. As such, these pillars will be in a continual state of disintegration. Slabbing and spalling will be commonplace. Remedial pillar wall support to prevent falls on equipment, if desired, would consist of 8 ft (or longer) grouted rebar in conjunction with straps. This would have to be supplemented with mesh if men were to enter the working areas. Given the possible working conditions here, remote mining is to be recommended.

Only two to three primary stopes can be open at any one time. Fill must be kept current. If the fill cycle lags behind the primary cycle by too great a distance, full overburden weight will begin to be applied to the post-pillars, increasing their distress. This could cause loss of the mining area. Proper sequencing of fill and mining cycles will avoid these problems.

Thicker sections of the ore (over 7m) will require access from each end of the panel. This may be done by one of three methods: benching, breasting, or the Endako method. These are shown as Figures 3.4, 3.5, and 3.6 respectively.

Benching

Benching is conducted by driving a mining width excavation against the hanging wall followed by slashing to the required 8m width. This is followed by benching the ore with vertical blastholes. With care, even extremely thick ore may be taken.

As was stated earlier, the pillars will likely be in a yielding state. As such, they will be continually spalling and slabbing. For this reason, remote mucking is recommended for benching. However, if pillar support is desired, blasted muck can be left in the stope for machine access so that bolting can be conducted.

The advantages to this method is that the entire remaining ore thickness can be mined in one pass. Mucking can be ongoing concurrent with drilling. This reduces lag between blasts as well as increasing the tonnage of each blast, thereby increasing productivity. Pillar walls are kept relatively vertical by virtue of the vertical production blasts.

Disadvantages include the possibility of stope loss if a large failure occurs from the pillar walls during mining.

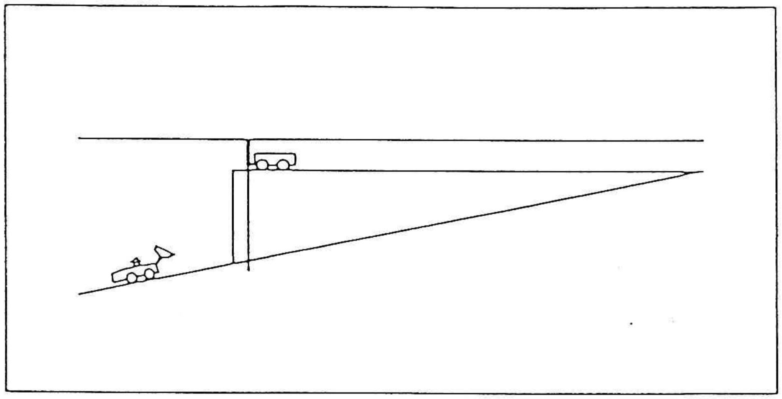

Figure 3.4 Bench blasting

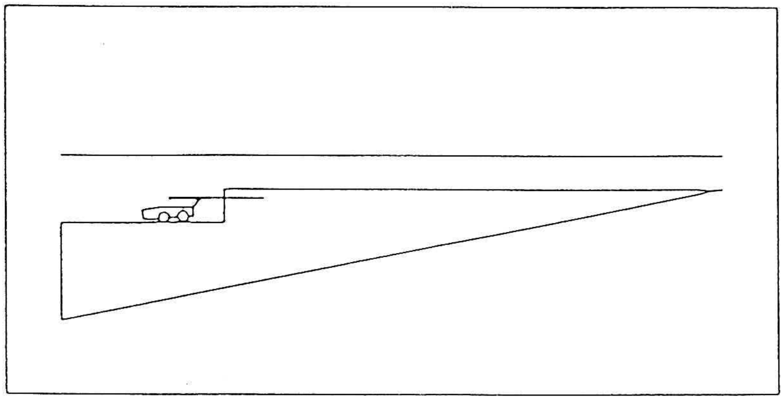

Figure 3.5 Breasting

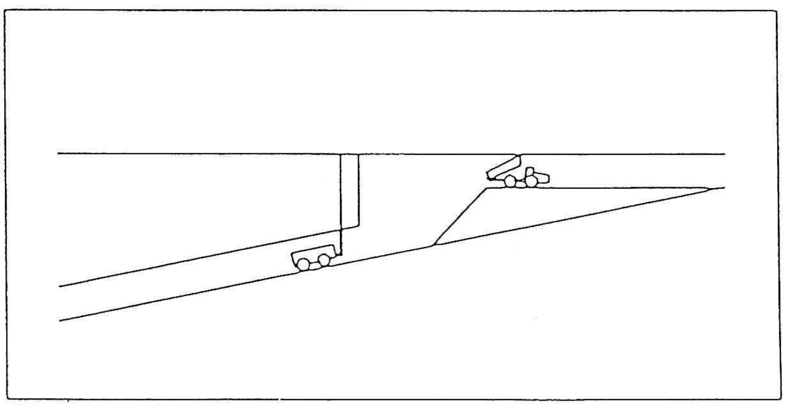

Figure 3.6 Endako method

Breasting

Breasting is depicted schematically in Figure 3.5. Here, as with benching, a primary excavation is opened near the hanging wall and slashed to final width. Breasting is then conducted on the ore in the sill of the excavation. If dual access is provided, this can be done from both ends of the panel simultaneously.

Advantages of this method are limited. Dual access to the panel is not required for mining and the pillar walls can be supported as the number of lifts progress into the sill.

The disadvantages of this excavation method generally outweigh the advantages. Multiple passes are required to access all the ore in the panel, resulting in relatively low production rates. Three distinct cycles are required during mining (drilling, support, mucking) without the opportunity for overlap as for standard benching. Pillar walls are not vertical but become thicker at depth due to the offset of each round from the pillar wall.

Endako method

This method differs from the other two in that primary access is along the footwall. After slashing to full panel width, vertical drillholes used for the production blasts as shown in Figure 3.6. Mucking is by remote loader from the footwall access drive.

Advantages to this method include the fact that fill can be placed while mining is taking place. This increases wall support and minimizes exposure time of the pillar wall, increasing stability. It also allows cycle overlap as the fill can be placed during the mining cycle which is not possible with the other two methods.

Disadvantages include the necessary cleanup, scaling, and support of a rough blasted roof while fill is being placed. Roof stability is not quite as certain as with the other two methods as the holes are drilled towards, not from, the hanging wall. The possibility of a hard-toe developing in the back is ever present due to the restriction on overdrilling into the hanging wall.

4.0 CONCLUSIONS AND RECOMMENDATIONS

At present, given the information available, the most promising method for mining the Dy deposit appears to be the concrete pillar method with standard benching. Undercut-and-fill as well as the caving methods described previously may rate some consideration but are risky for a feasibility level study.

More questions remained to be answered about the Dy deposit than have been asked to date. For example, we have assumed that the deposit is dry, that no abrupt breaks occur in the hanging or footwalls, that the ore is uniformly strong, that faults will have little or no impact, etc..

This may be allowable for a feasibility study for access to the area. However, what has been discussed up to this point is only conceptual in nature.

In order to complete this study for a true mine design access to the area, or additional detailed drilling information would be required. Backfill placement systems, fill sources, and backfill design will be critical to the success of the system. Allowable displacement, support required, and location of the production shaft should be addressed. Location of the main access headings, support requirements and excavation methodology should be examined. For example, should the main access be in cemented rock fill or in a large barrier type pillar? What loads can be expected on the pillars (secondary extraction panels)? What would happen if a mining area is lost (cost sensitivity)?

Some of these questions are not critical at this stage but should be addressed before a definitive mining decision is made.

Yours truly,

STEFFEN, ROBERTSON AND KIRSTEN (CANADA) INC.

Dr. James I. Mathis, P.E.

Dr. Chris H. Page, P.Eng.

JIM/067

APPENDIX N

N.D. ROSE,
FOX GEOLOGICAL CONSULTANTS LTD.

REPORT DATED
OCTOBER 19, 1992

DY DEPOSIT

MINEABLE RESERVE ESTIMATE AND

UNDERGROUND MINE PLAN

FARO, YUKON

by

N. D. Rose, B.A.Sc.
Fox Geological Consultants Ltd.
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for

Curragh Resources Inc.
Faro Mine
Faro, Yukon Y0B 1K0

October 19, 1992

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1.0 INTRODUCTION

An assessment of Dy deposit geology, mineable reserve inventory, and potential underground mine design was carried out in July and August, 1992.

The Dy deposit is a lead-zinc-silver-gold stratiform, syn-sedimentary, pyritic, massive sulphide deposit that occurs in a series of deposits located in the Anvil District, Faro, Yukon (Figure 1). This deposit consists of several exhalative massive sulphide horizons within a sequence of quartzites, phyllites, and schists. One main horizon, termed the AB Zone by Curragh Inc., hosts the majority of the sulphide mineralization and forms the most correlatable and continuous zone defined by surface drilling. With the exception of two diamond drill holes intersecting upper and lower sulphide zones, one massive sulphide horizon within the AB Zone was interpreted as "mineable" based on a 9% lead + zinc cut-off grade. The southerly A Zone (relatively lead-rich) and the northerly B Zone (relatively zinc-rich) are separated by an apparent barren massive sulphide zone which is comprised dominantly of disseminated sulphides in quartzite. The ore lies at an approximate depth of 530 to 880 metres (1,700 to 2,900 feet) and dips 20° to 35° to the southwest.

A polygonal reserve calculation of massive sulphide ore was carried out to determine a mineable reserve estimate and form the basis for a pre-feasibility mine design and production schedule. Details of calculations are included in Appendix I.

2.0 MINEABLE RESERVE CALCULATION - PREMISES AND METHODS

2.1 Drill Hole Database

All present drill hole data for Dy consisting of drilling from 1976 to 1991 was obtained using a Gemcom software PCXPLOR database. This database was set up by Curragh Inc. and contains all information used in the 1991 Dy Mineral Inventory. It should be noted that all premises and justifications for ore limits in the Curragh Inc. Mineral Inventory have been carried over to this investigation. Only the assay grade composites have been adjusted, where necessary, to reflect a mineable inventory of massive sulphide ore within the AB Zone.

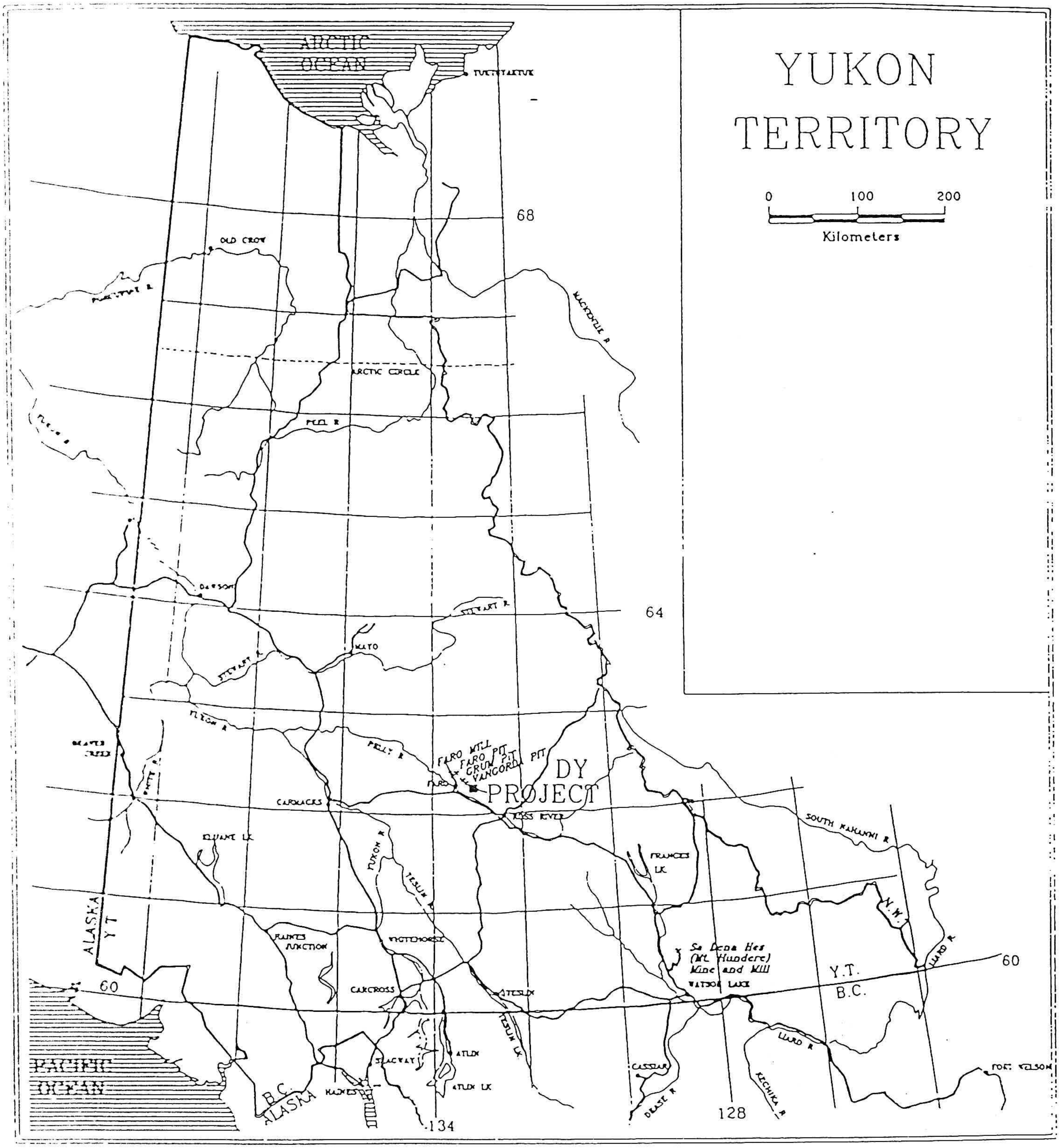


Figure 1: Map of the Yukon Territory showing the location of the Dy Project

2.2 Assumptions Defining Mineable Criteria

Based on experience mining the Faro underground and assumed similarities in structural complexity, ground behavior, and strengths of host rock units, the following assumptions have been applied to Dy:

1. Mining is to consist of "massive sulphide" ore.

2. A minimum mining height of four metres with a 9% lead+zinc cut-off grade (intersections meeting grade cut-off criteria were diluted to a minimum four-metre vertical thickness with footwall material).

3. A one-metre "skin" of ore would be required to provide adequate back support of a mineable block (see Section 3.1).

4. Grade differentiation within massive sulphide ore would not be possible as a mining parameter; mining of sulphides would occur on a visual basis. Thus sulphide intersections were weight-averaged over the whole sulphide intersection unless broken by intervals of waste defining possible hanging wall or footwall contacts.

2.3 Calculation Method

Cross sections, developed in PCXPLOR on an azimuth of 063° and a 100-metre grid spacing, were used to check above cut-off grade massive sulphide intersections for correlation and continuity along with simple geological interpretation. Assay intervals were composited over a minimum iterated four-metre vertical thickness to reflect a minimum four-metre mining height. Intervals of waste greater than five metres between sulphide intersections (two cases) were excluded from weight average composites. Intervals of less than approximately five metres were included.

A polygonal reserve calculation was done using Gemcom's GEOMODEL software. Polygons were generated by mid-point area projections between drill holes (to a maximum of 150 metres). At the edges of the deposit, the ore zone area of influence was arbitrarily defined as 60 metres beyond the most outboard drill holes.

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 drill hole from vertical at the location of each composite centre.

Polygon volumes are converted to tonnage using a density of 3.92 tonnes/cubic metre for all ore types. This value was derived by Curragh Inc. and is discussed in the 1991 Mineral Inventory Report.

2.4 Classification of Mineable Reserves

The classification of mineable reserves follows the premises and justification for ore limits in the Curragh Inc. 1991 Mineral Inventory (i.e, the same ore limits have been applied in this investigation). A more detailed explanation of ore limits is included in the 1991 Mineral Inventory Report.

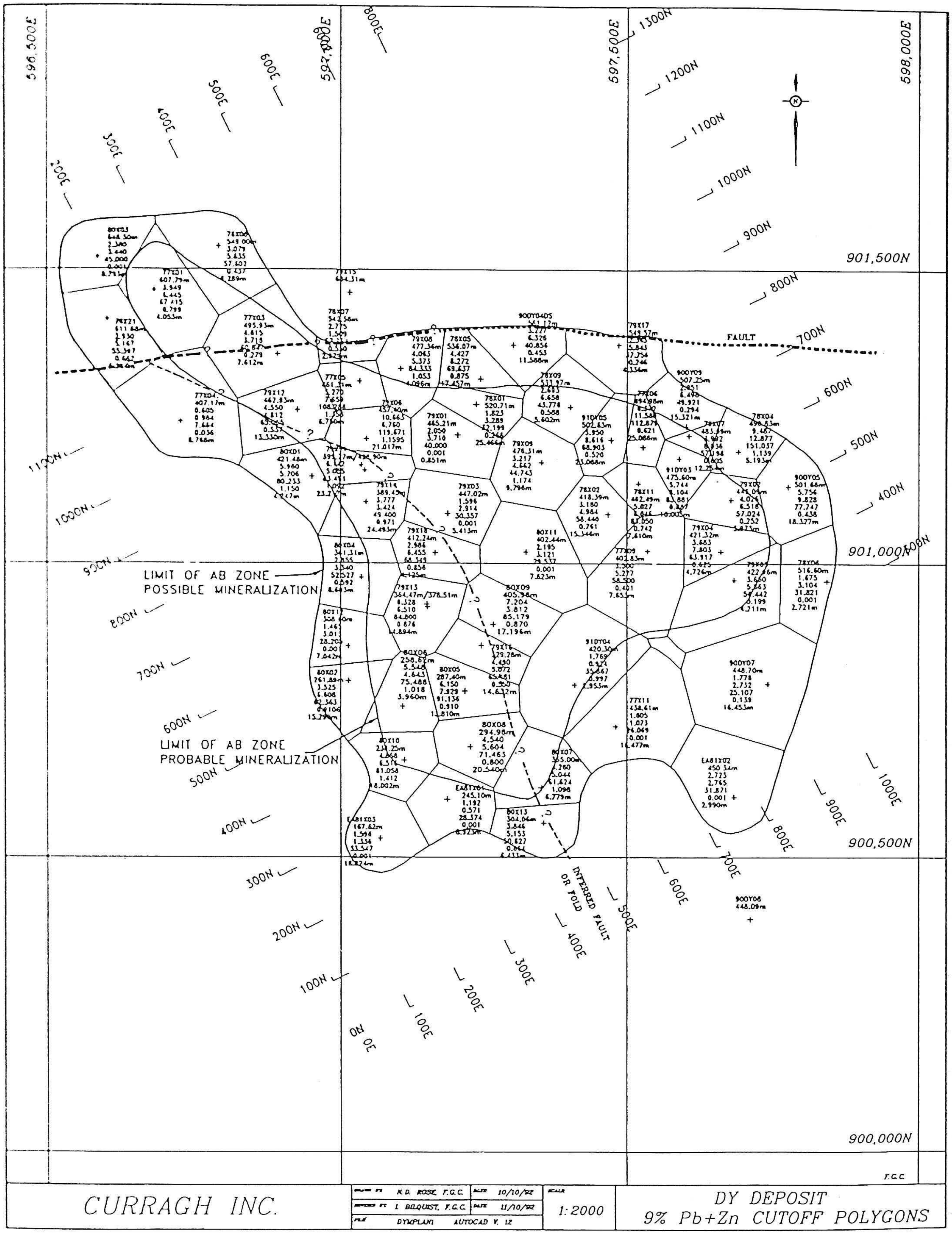
Drill hole spacing at Dy is considered too sparse to define a portion of the deposit as proven. Thus reserves are broken down into probable and possible categories.

2.4.1 Classification of Probable Mineralization

Based on continuity and nature of other deposits in the Anvil District, namely the Faro, Grum, and Vangorda deposits, the limit of probable mineralization at Dy is restricted to the area within the most outlying drill holes intersecting mineralization. That limit is shown in Figure 2.

2.4.2 Classification of Possible Mineralization

The classification of possible mineralization at Dy was restricted to a 60-metre radius of influence projection beyond the most outboard drill holes containing mineralization as seen in Figure 2. This category does not include possible mineralization above and below the AB Zone as well as the AB Extension Zone defined in the 1991 Curragh Inc. Mineral Inventory Report.


596,500E
597,000E
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598,000E
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FAULT
LIMIT OF AB ZONE
POSSIBLE MINERALIZATION
LIMIT OF AB ZONE
PROBABLE MINERALIZATION
INFERRED FAULT
OR FOLD
CURRAGH INC.
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10/10/92
L. BILQUIST, F.G.C.
11/10/92
DYMPLAN
AUTOCAD V. 12
1:2000
DY DEPOSIT
9% Pb+Zn CUTOFF POLYGONS

Figure 2

2.5 Results

The results of the Dy mineable reserve estimate at 9% and 10% lead+zinc cut-off grades are listed in Table I. A 10% dilution for all mining areas was introduced using waste material. Details of grade composite intervals and polygonal data are included in Appendix I.

Table I
Dy Mineable Inventory
10% Dilution

9% Lead + Zinc Cut-off	Tonnes	Lead + Zinc %	% Lead	% Zinc	Silver (g/t)	Gold (g/t)
Probable	10,883,979	12.19	5.70	6.48	81.7	0.834
Possible	4,752,354	11.99	4.31	7.68	67.3	0.702
Subtotal	15,636,333	12.13	5.28	6.85	77.3	0.794
10% Dilution	15,636,000	11.03	4.80	6.23	70.3	0.72
10% Lead + Zinc Cut-off	**Tonnes**	**Lead + Zinc %**	**% Lead**	**% Zinc**	**Silver (g/t)**	**Gold (g/t)**
Probable	8,858,897	12.82	6.08	6.74	86.1	0.855
Possible	2,856,180	13.67	4.84	8.83	75.9	0.815
Subtotal	11,715,077	13.03	5.78	7.25	83.6	0.868
10% Dilution	11,715,000	11.84	5.25	6.59	76.0	0.79

3.0 DY UNDERGROUND MINE PLAN

A preliminary underground mine design of the Dy deposit was carried out in July and August, 1992 by N. Rose of Fox Geological Consultants Ltd. under the general supervision of Leo Hwozdyk of Curragh Inc. Assistance defining rock mechanics constraints affecting mine design was provided by Dr. Chris Page and Dr. James Mathis of Steffen, Robertson and Kirsten (Vancouver) Inc. A detailed letter of recommendations is included in Appendix II.

3.1 Geotechnical Background

The Dy deposit 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. The phyllites and schists which comprise the hanging wall have an average rock mass rating (RMR) of 20 to 35 (poor to very poor) and an estimated compressive strength of 14 to 35 MPa perpendicular to foliation. Conversely the ore has a much higher RMR of 45 to 60 (moderate to good) and an estimated compressive strength of 54 MPa. At depths of 530 to 880 metres vertical overburden stresses should range from 13 to 22 MPa, or in the same order of magnitude as the strengths of the hanging wall material. Thus it will likely be critical to leave a reinforceable sulphide skin against the back to add support to open spans, as practised in the Faro underground operation.

3.2 Selection of a Mining Method

The depth and intermediate dip of the Dy deposit ultimately pose the greatest problems in choosing a suitable mining method. Conventional room and pillar mining at Dy is possible, but extraction will be limited due to its depth. Table II illustrates the maximum theoretical recoveries for varying safety factors in different portions of the ore body.

Table II
Maximum Theoretical Recoveries
Standard Room and Pillar

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

In order to increase the extractable reserve, a remnant of cut and fill mining, namely concrete pillar mining, was chosen. As illustrated in Figure 3, this method involves mining of primary and secondary panels with high quality cemented rock fill being placed in primary stopes to provide hanging wall support for extraction of secondary pillars. Equal panel widths of eight metres were chosen with an optimal panel length of 80 metres. Alternate hanging wall and footwall accesses allow development (drift and slash), production (longhole benching), and dumping of cemented rock fill from the hanging wall drive; mucking of ore is to occur from the footwall drive.

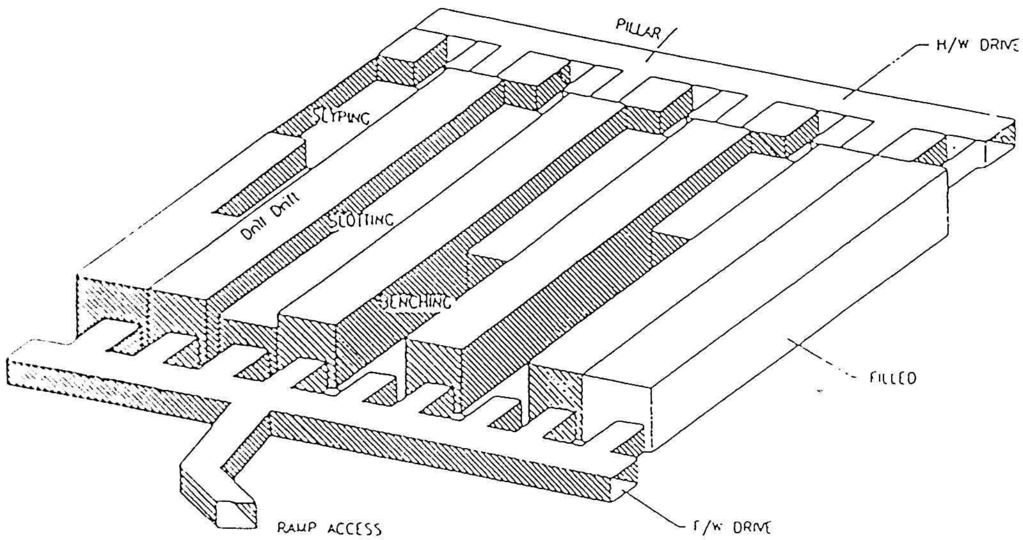

Figure 3 - Concrete Pillar Mining

In theory extraction with this method should approach 90% to 100%. With approximately 8% of the extractable reserve (approximately 1.25 MT) being left as a sulphide skin, a more appropriate recovery of 85% is chosen. The remainder of reserves are assumed to be tied up as pillar in roadways or as inaccessible due to the inferred complex nature of the deposit. Table III outlines the Dy mineable inventory at a 9% lead+zinc cut-off grade and an 85% extraction.

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Table III
Dy Mineable Inventory at
85% Recovery and 10% Dilution

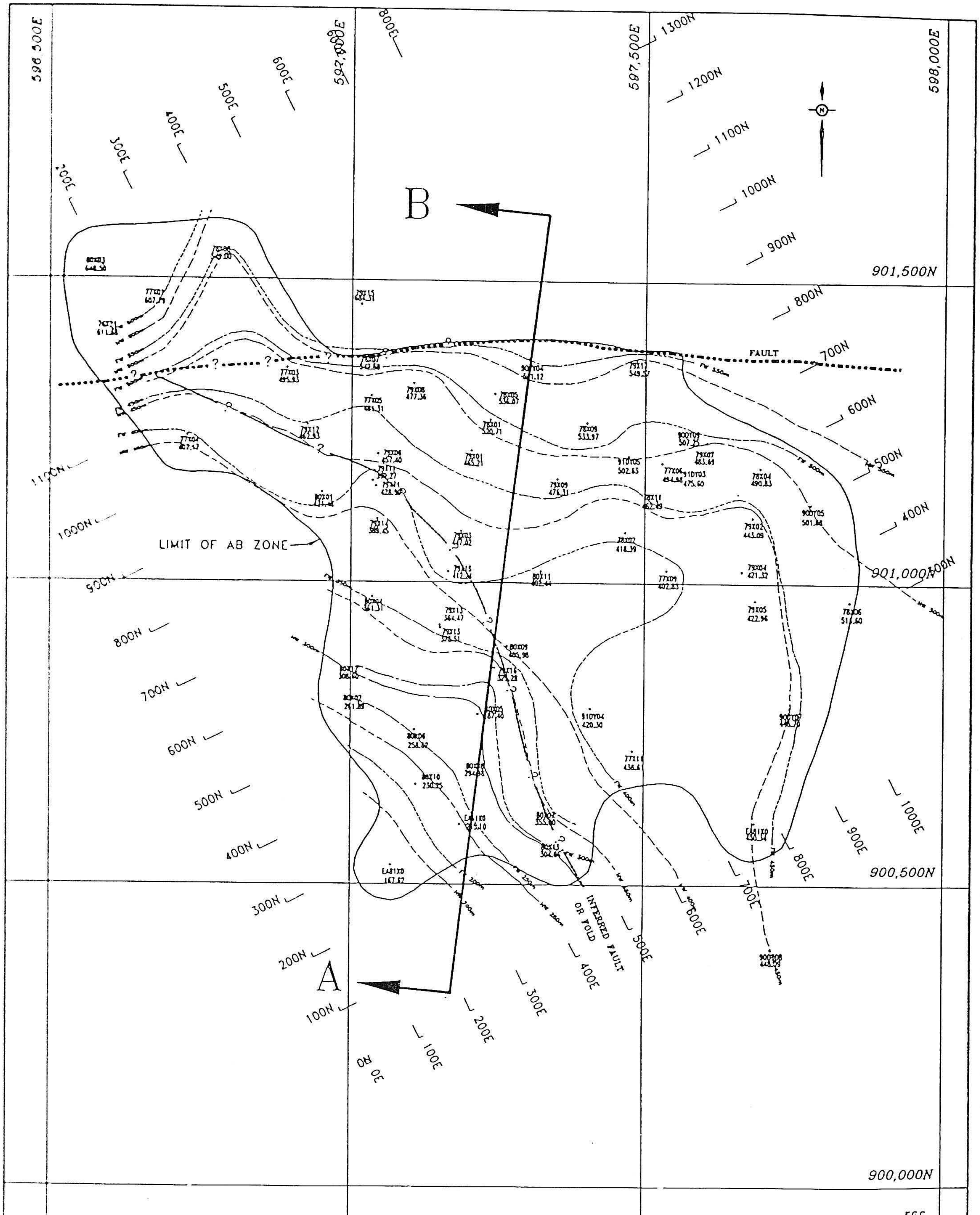
9% Lead + Zinc Cut-Off @ 85% Extraction	Tonnes	% Lead + Zinc	% Lead	% Zinc	Silver (g/t)	Gold (g/t)
Probable	9,251,382	12.19	5.70	6.48	81.7	0.834
Possible	4,039,501	11.99	4.31	7.68	67.3	0.702
Subtotal	13,290,883	12.13	5.28	6.85	77.3	0.794
10% Dilution	13,291,000	11.03	4.80	6.23	70.3	0.72

3.3 Underground Mine Design and Mining Sequence

Using the 9% lead+zinc cut-off grade polygon boundary as a limit for probable and possible material within the AB Zone, a preliminary mine design of Dy was conducted. A 10° cone of influence was deemed reasonable for a pre-feasibility mine design assuming negligible subsidence with a cemented fill method causing little potential for divergence in a shaft. It is recommended that this be investigated further with numerical analysis techniques in the early stages of a full feasibility study.

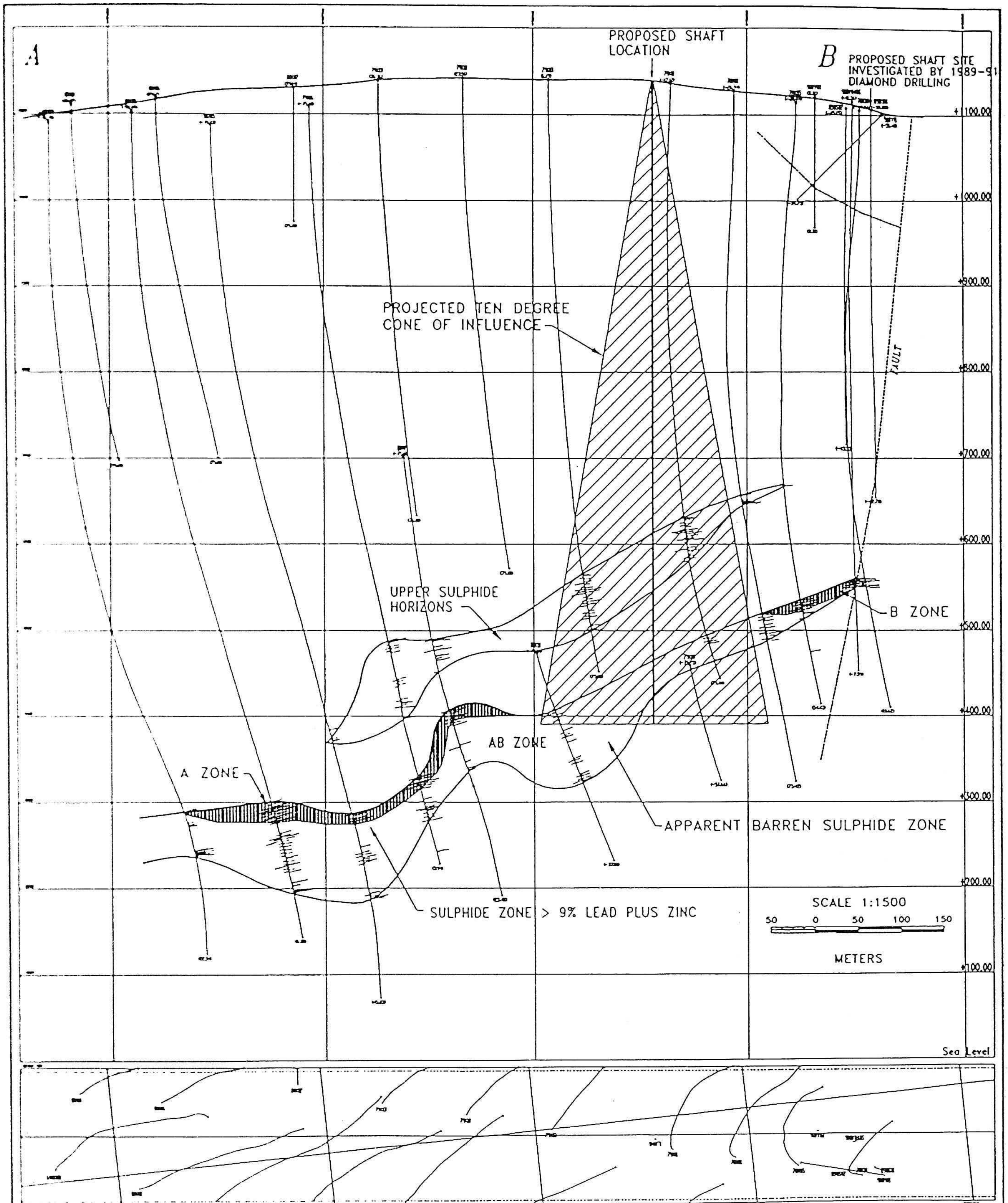
A 10° cone of influence allows a central location of the shaft in the apparent barren massive sulphide zone that separates the A and B Zones. Polygons 78X01 and 79X09 local to the shaft contain above 9% lead+zinc cut-off grade quartzites. An accurate inventory of mineralized material can be obtained from the Curragh Inc. 1991 Mineral Inventory Report. With further confirmation of a barren massive sulphide zone, these quartzites may allow placement of shaft facilities and initial development in a pay zone. Also this shaft location strategically allows early mining of the zinc-rich B Zone.

Hanging wall and footwall contours of the AB Zone massive sulphides are included in Figure 4 along with cross section A-B shown in Figure 5. Contours were hand created from hanging wall and footwall pierce point elevations generated in PCXPLOR. These contours were used to generate a three-dimensional model of Dy reserves using



CURRAGH INC.	DRAWN BY K.D. ROSE, F.G.C.	DATE 10/10/92	SCALE 1:2000	DY DEPOSIT MINING HW & FW CONTOURS
	REVISED BY K.D. ROSE, F.G.C.	DATE 10/10/92		
	FILE DYPLAN	AUTOCAD V. 12		

Figure 4



Curragh Inc.	Dy Deposit	VERTICAL SECTION SHAFT LOCATION STUDY Oct 10, 1992

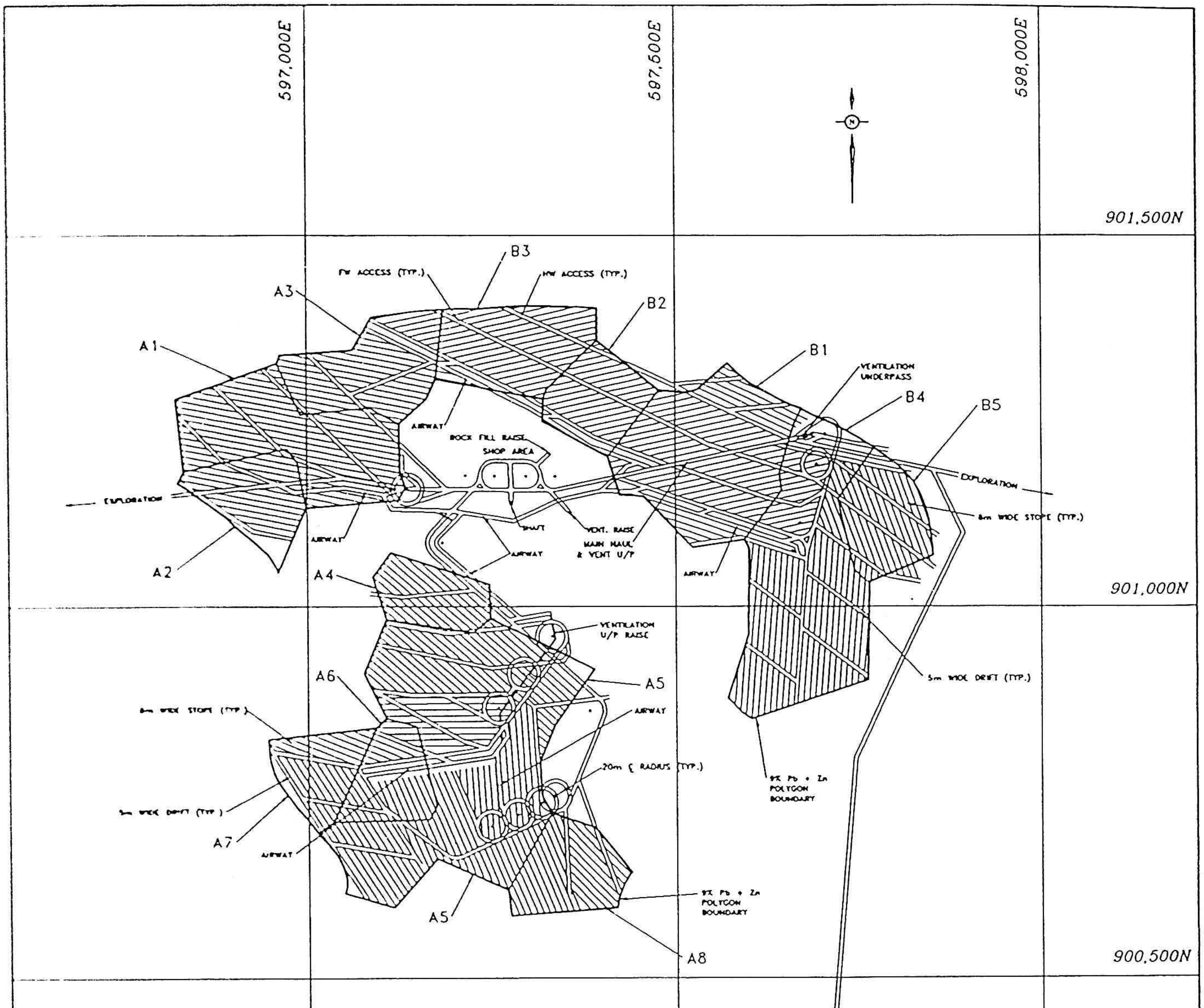
Figure 5

Gemcom's GEMSOLID software. Mine design was later added to the 3-D model to confirm and illustrate design.

The over-all mine design is illustrated in Figure 6. Development consists of 4.25-metre high by five-metre wide drifts in ore and five-metre high by 4.25-metre wide arched drifts in waste. Maximum gradients of 18% were assigned to development and 15% to sublevel footwall spiral access ramps designed in areas of steep dip (25° to 35°) and varying strike. Production stopes consist of five-metre high by eight-metre wide drift and slash development and longhole production benching; again maximum gradients of 18%. A ventilation and rock fill raise are located close to the shaft to allow early installation and central service. The shaft location and surface topography are illustrated in Figure 7.

The mining sequence is illustrated by typical B Zone longitudinal and cross sections as shown in Figures 8 and 9. Alternate hanging wall (HW) and footwall (FW) drifts allow complete access to the stopes. Development of eight-metre wide by 80-metre long panels is to occur from the hanging wall drives; every second stope being developed as part of the primary mining cycle. Once developed on hanging wall and ground is secured, longhole production benching follows with mucking of stopes from the footwall access. Securing of stope walls can occur working off the muck from the FW access drives or in high stopes, remote control mucking may decrease support requirements and allow increased productivity. Cemented rock fill is then dumped by truck from the HW drive until the stope is full and rock fill is jammed tight against the back. Fill is allowed seven days to cure before mining continues.

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



MINING INVENTORY

MINING BLOCK	AVG. VERT. THICK. (m)	TONNES	% Pb	% Zn	Ag (g/t)	Au (g/t)
B1	16.04	2,567,144	4.58	8.73	77.2	0.56
B2	5.60	309,443	2.69	6.66	43.8	0.59
B3	14.32	1,203,995	3.91	7.43	57.2	0.69
B4	4.45	920,928	5.01	7.92	78.9	0.50
B5	18.33	1,116,889	5.76	9.83	77.7	0.44
A1	17.70	2,319,895	7.20	5.59	89.8	0.91
A2	4.25	245,846	5.96	5.71	80.2	1.15
A3	5.08	424,692	4.68	6.50	96.2	1.20
A4	4.13	190,979	2.99	6.48	68.3	0.85
A5	16.65	5,016,219	5.61	5.85	80.1	0.93
A6	3.96	194,099	5.55	4.64	75.5	1.02
A7	15.29	705,210	3.53	6.61	62.4	0.91
A8	6.78	470,986	4.26	5.04	61.6	1.10
TOTAL	11.62	15,636,325	5.28	6.85	77.3	0.79
10 % DILUTED	11.62	15,634,000	4.80	6.23	70.3	0.72

F.G.C.

CURRAGH INC.

DESIGNED BY K.D. ROSE, F.G.C. DATE 17/06/92
DRAWN BY R.P. PLUT, S.R.K. DATE 17/06/92
FILE DYMPLAN3 AUTOCAD V. 12

SCALE 1:2000

DY DEPOSIT
11 YEAR MINE PLAN

PLATE DY-1

Figure 6

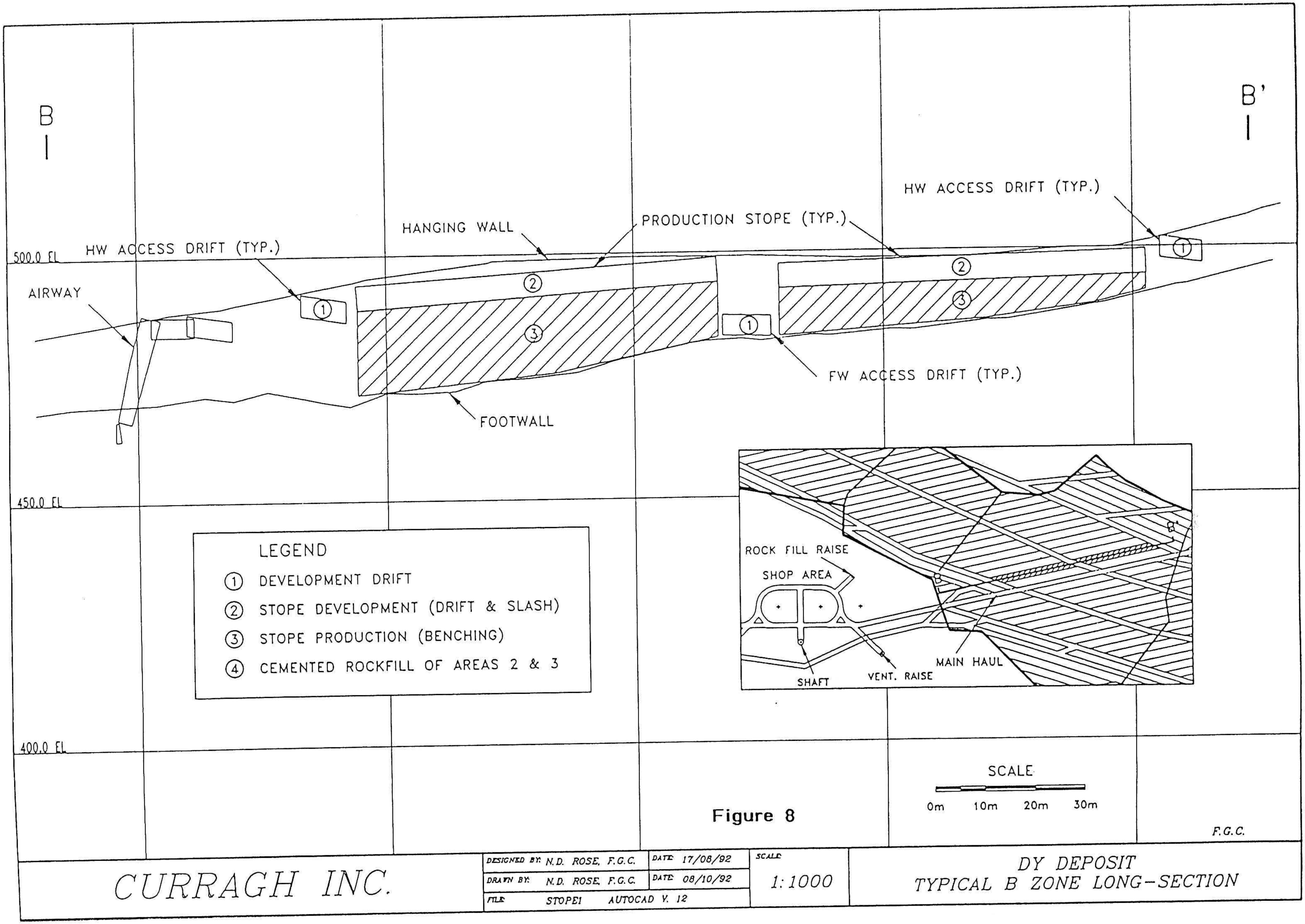
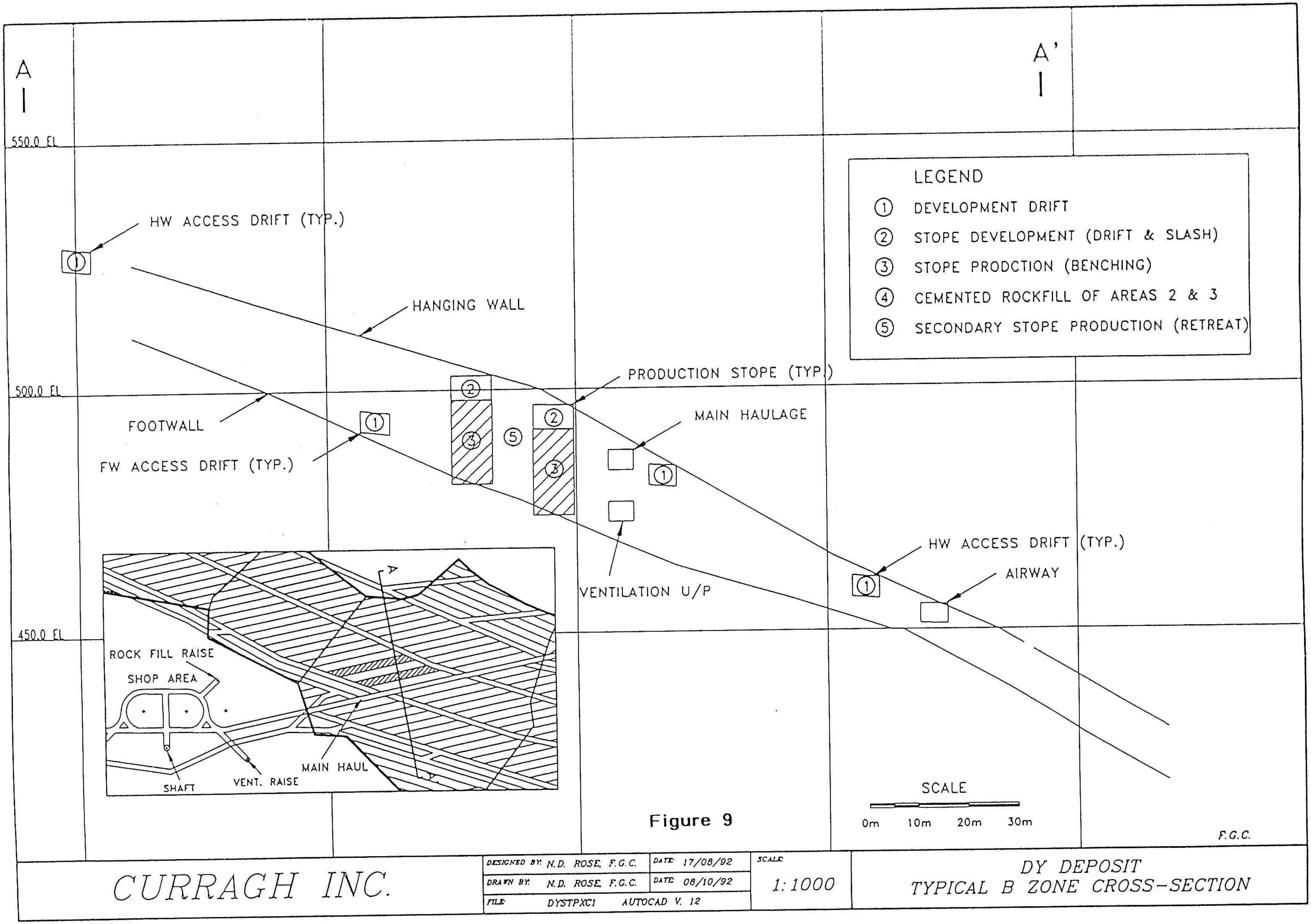

B
B'
500.0 EL
450.0 EL
400.0 EL
HW ACCESS DRIFT (TYP.)
HANGING WALL
PRODUCTION STOPE (TYP.)
HW ACCESS DRIFT (TYP.)
AIRWAY
FW ACCESS DRIFT (TYP.)
FOOTWALL
1
2
3
LEGEND
1 DEVELOPMENT DRIFT
2 STOPE DEVELOPMENT (DRIFT & SLASH)
3 STOPE PRODUCTION (BENCHING)
4 CEMENTED ROCKFILL OF AREAS 2 & 3
ROCK FILL RAISE
SHOP AREA
SHAFT
VENT. RAISE
MAIN HAUL
SCALE
0m 10m 20m 30m
F.G.C.
CURRAGH INC.
DESIGNED BY: N.D. ROSE, F.G.C.
DATE: 17/08/92
DRAWN BY: N.D. ROSE, F.G.C.
DATE: 08/10/92
FILE: STOPE1 AUTOCAD V. 12
SCALE: 1:1000
DY DEPOSIT
TYPICAL B ZONE LONG-SECTION

Figure 8


A
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LEGEND
① DEVELOPMENT DRIFT
② STOPE DEVELOPMENT (DRIFT & SLASH)
③ STOPE PRODCTION (BENCHING)
④ CEMENTED ROCKFILL OF AREAS 2 & 3
⑤ SECONDARY STOPE PRODUCTION (RETREAT)
HW ACCESS DRIFT (TYP.)
HANGING WALL
PRODUCTION STOPE (TYP.)
FOOTWALL
FW ACCESS DRIFT (TYP.)
MAIN HAULAGE
VENTILATION U/P
AIRWAY
ROCK FILL RAISE
SHOP AREA
SHAFT
VENT. RAISE
MAIN HAUL
SCALE
0m 10m 20m 30m
F.G.C.
CURRAGH INC.
DESIGNED BY: N.D. ROSE, F.G.C.
DATE: 17/08/92
DRAWN BY: N.D. ROSE, F.G.C.
DATE: 08/10/92
FILE: DYSTPXC1 AUTOCAD V. 12
SCALE: 1:1000
DY DEPOSIT
TYPICAL B ZONE CROSS-SECTION

Figure 9

4.0 DY ELEVEN-YEAR DEVELOPMENT/PRODUCTION SCHEDULE

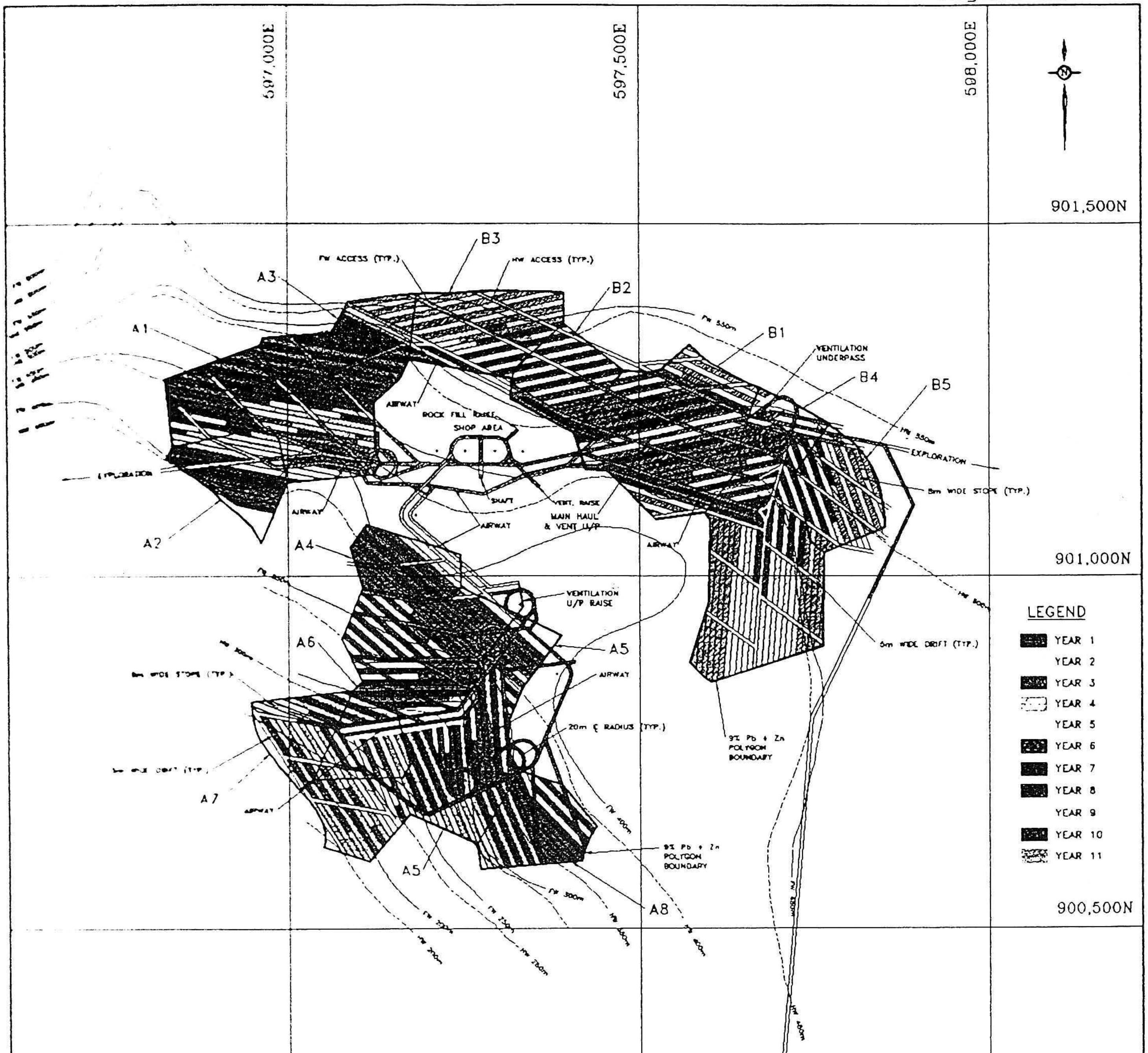
An 11-year development/production schedule of the Dy Mineable Inventory was developed in August, 1992. A summary of production is listed in Table IV. A more detailed production schedule is included in Appendix III along with a 1:2000 mine plan showing a yearly breakdown of mining.

Unfortunately due to time constraints, the underground mine design could not be adjusted to allow alternate means of haulage and/or conveyance of broken ore. Ultimately the incorporation of cemented rock fill to the mining cycle added constriction to a single haulage system which in turn limited production capacity and yearly output.

Table IV
Dy Underground Development/Production
Schedule Summary

			ASSAY GRADES			
YEAR	ADV (m)	TONNES	%Pb	%Zn	Ag(g/t)	Au(g/t)
YEAR 1	4743	687,308	4.52	8.55	75.5	0.55
YEAR 2	5995	1,000,296	4.68	8.45	71.8	0.54
YEAR 3	5655	1,006,333	4.99	7.83	73.0	0.65
YEAR 4	5310	1,012,742	5.26	7.63	74.6	0.69
YEAR 5	5595	1,051,506	5.17	7.24	73.7	0.72
YEAR 6	5600	1,060,924	5.29	6.76	80.9	0.84
YEAR 7	5900	1,112,349	5.39	6.66	81.4	0.86
YEAR 8	5875	1,148,790	5.56	5.94	79.7	0.94
YEAR 9	5455	1,120,267	5.77	5.80	79.9	0.94
YEAR 10	4495	1,042,250	5.75	5.82	80.4	0.93
YEAR 11	4455	1,006,813	5.08	5.92	75.0	0.94
SUB TOTAL	59078	11,249,578	5.26	6.89	77.1	0.79
10% DILUTION	59078	11,249,578	4.78	6.26	70.1	0.72

Figure 10



MINING INVENTORY

MINING BLOCK	AVG. VERT. THICK. (m)	TONNES	% Pb	% Zn	Ag (g/t)	Au (g/t)
B1	16.04	2,567,144	4.58	8.73	77.2	0.568
B2	5.60	308,443	2.68	6.66	43.8	0.588
B3	14.32	1,203,993	3.91	7.43	57.2	0.683
B4	4.45	820,928	5.01	7.92	78.9	0.802
B5	18.33	1,118,880	5.76	8.83	77.7	0.438
A1	17.70	2,318,895	7.20	5.59	89.8	0.912
A2	4.25	245,844	5.96	5.71	80.2	1.150
A3	5.08	424,692	4.66	6.50	94.2	1.201
A4	4.13	190,979	2.99	4.44	68.5	0.854
A5	16.65	5,016,219	6.81	6.85	80.1	0.825
A6	3.94	184,099	5.35	4.84	73.5	1.018
A7	13.23	705,210	3.33	6.61	62.4	0.811
A8	6.78	420,986	4.26	5.04	61.6	1.098
TOTAL	11.62	15,636,323	5.28	6.85	77.3	0.794
10 % DILUTED	11.62	15,634,000	4.80	6.23	70.3	0.77

F.G.C.

CURRAGH INC.	DESIGNED BY N.D. ROSE, F.G.C.	DATE 17/08/92	SCALE 1:2000	DY DEPOSIT 11 YEAR MINE PLAN	PLATE DY-1
	DRAWN BY Y.P. PLUT, S.R.K.	DATE 17/08/92			
	APPROVED BY	DATE			

5.0 CONCLUSIONS AND RECOMMENDATIONS

The Dy deposit appears to be a complex, but poorly defined orebody. Due to its depth, the 30 to 40 drill holes that define the bulk of its mineralization, only provide an indication of its character. The orebody appears to be open-ended in many directions and has much potential for further exploration. Grade variation and zonation of massive sulphides is evident and may cause problems in mining.

A more detailed investigation of geology of the deposit is warranted. Due to the limited time frame of this study, a revisal of the mineable inventory was not possible. Adjustments in the grade composites for polygons 80X04 and 80X13 would have led to their inclusion into the mineable inventory at a 9% lead+zinc cut-off grade. Also an inventory of mineralized quartzites was not included in the investigation of massive sulphide ore.

Particular emphasis should be given to polygons 78X01 and 79X09 which contain above 9% lead+zinc cut-off grade quartzites in the apparent massive sulphide zone local to the shaft. Possibly with re-logging of drill core to a common standard, these polygons and others may by included under a separate category and not excluded from the mineable inventory. Ultimately the rock qualities of these mineralized quartzites will determine the likelihood of their mineability. Due to oxidation of drill core, this information may now be difficult to ascertain.

Much more work is necessary to adequately define Dy reserves and potential structure encountered in mining. An extensive drilling program, preferably underground, should precede the details of a mine design. Obviously given the present information, a true mine design can only be conceptual in nature.

If the proposed shaft location is chosen, confirmation of a barren massive sulphide zone should accompany the investigation of a 10° cone of influence. It is recommended that the shaft pillar be confirmed using numerical analysis techniques.

Information critical in the stages of a full feasibility study will be the collection of hydrological data which at present appears to be limited or non-existent. Also potential rock fill sources and placement systems will ultimately determine the success of a fill method if it is used.

Prepared by:

FOX GEOLOGICAL CONSULTANTS LTD.

Nick Rose

N. D. Rose, B.A. Sc.
October 19, 1992

APPENDIX I

Fox Geological Consultants Ltd.
1992 Polygonal Mineable Inventory
Calculation Tables

Fox Geological Consultants Ltd. 1409 - 409 Granville Street, Vancouver, B.C. V6C 1T8
Telephone (604) 669-5736 Fax (604) 681-3920

CURRAGH INC.
DY 9% LEAD PLUS ZINC CUT-OFF
ASSAY COMPOSITES

file DY9%ASCO
Date 19-Oct-92

HOLE ID	FROM	TO	INT. (m)	VERT. TH. (m)	%Pb+Zn	%Pb	%Zn	Ag (g/t)	Au (g/t)
77X05	709.0	716.0	7.0	6.75	12.93	5.27	7.66	108.3	1.35
77X06	586.5	612.1	25.6	25.09	17.95	6.37	11.58	112.9	0.62
78X04	556.6	562.0	5.4	5.19	22.36	9.49	12.88	151.0	1.14
78X05	586.3	604.2	17.9	17.46	11.67	4.43	8.27	69.6	0.88
78X09	556.3	562.1	5.8	5.60	9.35	2.69	6.66	43.8	0.59
78X11	617.2	625.2	8.0	7.61	13.07	5.03	8.05	83.1	0.74
79X02	600.0	604.2	4.2	3.97	10.54	4.03	6.52	57.0	0.25
79X04	625.8	630.6	4.8	4.73	11.49	3.68	7.80	63.9	0.63
79X05	632.6	636.9	4.3	4.21	9.51	3.65	5.86	56.4	0.20
79X06	718.2	739.8	21.6	21.02	17.42	10.66	6.76	119.7	1.16
79X07	574.2	586.8	12.6	12.25	12.03	3.99	8.04	57.2	0.61
79X08	676.3	680.5	4.2	4.10	9.44	4.06	5.37	84.3	1.05
79X11	747.7	755.4	7.7	7.25	10.79	6.57	4.22	72.9	1.36
79X11	779.2	796.2	17.0	15.97	11.38	5.95	5.43	88.3	0.97
Hole Total			24.7	23.22	11.20	6.14	5.06	83.5	1.09
79X12	723.8	737.7	13.9	13.33	9.36	4.55	4.81	65.1	0.54
79X13	771.3	781.3	10.0	9.56	12.99	6.33	6.66	86.6	1.11
79X13	786.0	791.6	5.6	5.33	12.57	6.32	6.25	81.6	0.45
Hole Total			15.6	14.89	12.39	6.33	6.51	84.8	0.88
79X16	805.0	820.2	15.2	14.63	9.56	4.49	5.07	65.5	0.55
79X18	740.5	744.8	4.3	4.13	9.44	2.99	6.46	68.3	0.86
80X01	757.3	761.6	4.3	4.25	11.67	5.96	5.71	80.2	1.15
80X02	888.9	904.9	16.0	15.29	10.13	3.53	6.61	62.4	0.91
80X05	846.5	861.2	14.7	13.81	14.08	6.15	7.93	91.1	0.91
80X06	885.4	889.5	4.1	3.96	10.19	5.55	4.64	75.5	1.02
80X07	746.5	753.4	6.9	6.78	9.30	4.26	5.04	61.6	1.10
80X08	829.2	850.6	21.4	20.54	10.14	4.54	5.60	71.5	0.80
80X09	726.8	745.2	18.4	17.20	11.02	7.20	3.81	85.2	0.87
80X10	909.8	928.6	18.8	18.00	11.37	4.86	6.52	81.1	1.41
90DY04DS	554.3	565.9	11.6	11.59	9.55	3.23	6.33	40.9	0.45
90DY05	516.2	534.6	18.4	18.33	15.58	5.76	9.83	77.8	0.44
90DY09	551.2	566.7	15.5	15.32	9.45	2.95	6.50	49.9	0.29
91DY03	588.3	598.4	10.1	10.00	13.85	5.74	8.10	83.9	0.89
91DY05	584.9	608.1	23.2	23.09	12.57	3.95	8.62	68.9	0.52
TOTAL					12.13	5.28	6.85	77.3	0.79

FOX GEOLOGICAL CONSULTANTS LTD.

CURRAGH INC.
DY MINEABLE INVENTORY
CLASSIFICATION: PROBABLE + POSSIBLE
CUTOFF = 9% LEAD PLUS ZINC

file DYPOPB9%
Date 19-Oct-92

HOLE ID	VERT. TH. (m)	%Pb+Zn	%Pb	%Zn	Ag (g/t)	Au (g/t)	S.G.	AREA	VOLUME	TONNAGE
77X05	6.75	12.93	5.27	7.66	108.3	1.35	3.92	7941.4	53604.3	210128.9
77X06	25.09	17.95	6.37	11.58	112.9	0.62	3.92	5944.6	149127.4	584579.4
78X04	5.19	22.36	9.49	12.88	151.0	1.14	3.92	9973.0	51789.7	203015.5
78X05	17.46	11.67	4.43	8.27	69.6	0.88	3.92	9994.9	174481.8	683968.5
78X09	5.60	9.35	2.69	6.66	43.8	0.59	3.92	14091.3	78939.7	309443.6
78X11	7.61	13.07	5.03	8.05	83.1	0.74	3.92	7563.7	57560.1	225635.5
79X02	3.97	10.54	4.03	6.52	57.0	0.25	3.92	10266.6	40789.1	159893.3
79X04	4.73	11.49	3.68	7.80	63.9	0.63	3.92	10109.6	47777.8	187288.9
79X05	4.21	9.51	3.65	5.86	56.4	0.20	3.92	22459.0	94574.7	370732.8
79X06	21.02	17.42	10.66	6.76	119.7	1.16	3.92	10238.3	215178.4	843499.1
79X07	12.25	12.03	3.99	8.04	57.2	0.61	3.92	2775.8	34015.1	133339.1
79X08	4.10	9.44	4.06	5.37	84.3	1.05	3.92	13363.1	54735.3	214562.4
79X11	23.22	11.20	6.14	5.06	83.5	1.09	3.92	6815.1	158226.5	620247.7
79X12	13.33	9.36	4.55	4.81	65.1	0.54	3.92	16384.5	218405.4	856149.1
79X13	14.89	12.39	6.33	6.51	84.8	0.88	3.92	14281.7	212711.4	833828.8
79X16	14.63	9.56	4.49	5.07	65.5	0.55	3.92	9092.0	133033.4	521491.0
79X18	4.13	9.44	2.99	6.46	68.3	0.86	3.92	11810.7	48719.1	190978.8
80X01	4.25	11.67	5.96	5.71	80.2	1.15	3.92	14767.1	62715.7	245845.6
80X02	15.29	10.13	3.53	6.61	62.4	0.91	3.92	11765.9	179900.6	705210.4
80X05	13.81	14.08	6.15	7.93	91.1	0.91	3.92	12725.6	175740.4	688902.4
80X06	3.96	10.19	5.55	4.64	75.5	1.02	3.92	12503.8	49515.0	194098.8
80X07	6.78	9.30	4.26	5.04	61.6	1.10	3.92	15842.2	107394.5	420986.6
80X08	20.54	10.14	4.54	5.60	71.5	0.80	3.92	14002.5	287611.5	1127437.0
80X09	17.20	11.02	7.20	3.81	85.2	0.87	3.92	13931.3	239563.0	939086.9
80X10	18.00	11.37	4.86	6.52	81.1	1.41	3.92	12831.2	230986.5	905467.1
90DY04DS	11.59	9.55	3.23	6.33	40.9	0.45	3.92	11448.1	132660.5	520029.0
90DY05	18.33	15.58	5.76	9.83	77.8	0.44	3.92	15546.5	284920.4	1116887.8
90DY09	15.32	9.45	2.95	6.50	49.9	0.29	3.92	8606.3	131857.7	516882.1
91DY03	10.00	13.85	5.74	8.10	83.9	0.89	3.92	6542.3	65455.8	256586.8
91DY05	23.09	12.57	3.95	8.62	68.9	0.52	3.92	9393.2	216870.0	850130.3
TOTAL		12.13	5.28	6.85	77.3	0.79		343011	3988861	15636333

FOX GEOLOGICAL CONSULTANTS LTD.

CURRAGH INC.
DY MINEABLE INVENTORY
CLASSIFICATION: PROBABLE
CUTOFF = 9% LEAD PLUS ZINC

file: DYPROB9%
Date 19-Oct-92

HOLE ID	VERT. TH. (m)	%Pb+Zn	%Pb	%Zn	Ag (g/t)	Au (g/t)	S.G.	AREA	VOLUME	TONNAGE
77X05	6.75	12.93	5.27	7.66	108.3	1.35	3.92	9897.9	66810.8	261898.4
77X06	25.09	17.95	6.37	11.58	112.9	0.62	3.92	5635.0	141358.4	554124.8
78X09	5.60	9.35	2.69	6.66	43.8	0.59	3.92	9837.0	55107.1	216019.8
78X11	7.61	13.07	5.03	8.05	83.1	0.74	3.92	7563.7	57560.1	225635.5
79X02	3.97	10.54	4.03	6.52	57.0	0.25	3.92	9667.4	38408.4	150561.0
79X04	4.73	11.49	3.68	7.80	63.9	0.63	3.92	10065.2	47567.9	186466.2
79X05	4.21	9.51	3.65	5.86	56.4	0.20	3.92	11264.0	47432.7	185936.1
79X06	21.02	17.42	10.66	6.76	119.7	1.16	3.92	10918.9	229482.5	899571.5
79X07	12.25	12.03	3.99	8.04	57.2	0.61	3.92	2872.0	35193.8	137959.8
79X11	23.22	11.20	6.14	5.06	83.5	1.09	3.92	8595.2	199555.6	782258.0
79X12	13.33	9.36	4.55	4.81	65.1	0.54	3.92	12652.6	168659.0	661143.2
79X13	14.89	12.39	6.33	6.51	84.8	0.88	3.92	19726.6	293808.0	1151727.4
79X16	14.63	9.56	4.49	5.07	65.5	0.55	3.92	9092.0	133033.4	521491.0
79X18	4.13	9.44	2.99	6.46	68.3	0.86	3.92	13292.2	54830.4	214935.1
80X05	13.81	14.08	6.15	7.93	91.1	0.91	3.92	12725.6	175740.4	688902.4
80X06	3.96	10.19	5.55	4.64	75.5	1.02	3.92	13996.3	55425.5	217268.0
80X07	6.78	9.30	4.26	5.04	61.6	1.10	3.92	8488.0	57540.0	225556.7
80X08	20.54	10.14	4.54	5.60	71.5	0.80	3.92	15192.6	312056.6	1223261.8
80X09	17.20	11.02	7.20	3.81	85.2	0.87	3.92	13931.3	239563.0	939086.9
80X10	18.00	11.37	4.86	6.52	81.1	1.41	3.92	5398.6	97186.2	380969.7
91DY03	10.00	13.85	5.74	8.10	83.9	0.89	3.92	6734.8	67382.0	264137.6
91DY05	23.09	12.57	3.95	8.62	68.9	0.52	3.92	8784.8	202823.4	795067.7
TOTAL		12.19	5.70	6.48	81.7	0.83		226,332	2,776,525	10,883,979

FOX GEOLOGICAL CONSULTANTS LTD.

CURRAGH INC.
DY MINEABLE INVENTORY
CLASSIFICATION: PROBABLE + POSSIBLE
CUTOFF = 10% LEAD PLUS ZINC

file: DYPOPB10
Date 19-Oct-92

HOLE ID	VERT. TH. (m)	%Pb+Zn	%Pb	%Zn	Ag (g/t)	Au (g/t)	S.G.	AREA	VOLUME	TONNAGE
77X05	6.75	12.93	5.27	7.66	108.3	1.35	3.92	7941.4	53604.3	210128.9
77X06	25.09	17.95	6.37	11.58	112.9	0.62	3.92	5944.6	149127.4	584579.4
78X04	5.19	22.36	9.49	12.88	151.0	1.14	3.92	9973.0	51789.7	203015.5
78X05	17.46	11.67	4.43	8.27	69.6	0.88	3.92	9994.9	174481.8	683968.5
78X11	7.61	13.07	5.03	8.05	83.1	0.74	3.92	7563.7	57560.1	225635.5
79X02	3.97	10.54	4.03	6.52	57.0	0.25	3.92	10266.6	40789.1	159893.3
79X04	4.73	11.49	3.68	7.80	63.9	0.63	3.92	10109.6	47777.8	187288.9
79X06	21.02	17.42	10.66	6.76	119.7	1.16	3.92	10238.3	215178.4	843499.1
79X07	12.25	12.03	3.99	8.04	57.2	0.61	3.92	2775.8	34015.1	133339.1
79X11	23.22	11.20	6.14	5.06	83.5	1.09	3.92	6815.1	158226.5	620247.7
79X13	14.89	12.39	6.33	6.51	84.8	0.88	3.92	14281.7	212711.4	833828.8
80X01	4.25	11.67	5.96	5.71	80.2	1.15	3.92	14767.1	62715.7	245845.6
80X02	15.29	10.13	3.53	6.61	62.4	0.91	3.92	11765.9	179900.6	705210.4
80X05	13.81	14.08	6.15	7.93	91.1	0.91	3.92	12725.6	175740.4	688902.4
80X06	3.96	10.19	5.55	4.64	75.5	1.02	3.92	12503.8	49515.0	194098.8
80X08	20.54	10.14	4.54	5.60	71.5	0.80	3.92	14002.5	287611.5	1127437.0
80X09	17.20	11.02	7.20	3.81	85.2	0.87	3.92	13931.3	239563.0	939086.9
80X10	18.00	11.37	4.86	6.52	81.1	1.41	3.92	12831.2	230986.5	905467.1
90DY05	18.33	15.58	5.76	9.83	77.8	0.44	3.92	15546.5	284920.4	1116887.8
91DY03	10.00	13.85	5.74	8.10	83.9	0.89	3.92	6542.3	65455.8	256586.8
91DY05	23.09	12.57	3.95	8.62	68.9	0.52	3.92	9393.2	216870.0	850130.3
TOTAL		13.03	5.78	7.25	83.6	0.87		219914	2988541	11715078

FOX GEOLOGICAL CONSULTANTS LTD.

CURRAGH INC.
DY MINEABLE INVENTORY
CLASSIFICATION: PROBABLE
CUTOFF = 10% LEAD PLUS ZINC

file: DYPROB10
Date: 19-Oct-92

HOLE ID	VERT. TH. (m)	%Pb+Zn	%Pb	%Zn	Ag (g/t)	Au (g/t)	S.G.	AREA	VOLUME	TONNAGE
77X05	6.75	12.93	5.27	7.66	108.3	1.35	3.92	9897.9	66810.8	261898.4
77X06	25.09	17.95	6.37	11.58	112.9	0.62	3.92	5635.0	141358.4	554124.8
78X11	7.61	13.07	5.03	8.05	83.1	0.74	3.92	7563.7	57560.1	225635.5
79X02	3.97	10.54	4.03	6.52	57.0	0.25	3.92	9667.4	38408.4	150561.0
79X04	4.73	11.49	3.68	7.80	63.9	0.63	3.92	10065.2	47567.9	186466.2
79X06	21.02	17.42	10.66	6.76	119.7	1.16	3.92	10918.9	229482.5	899571.5
79X07	12.25	12.03	3.99	8.04	57.2	0.61	3.92	2872.0	35193.8	137959.8
79X11	23.22	11.20	6.14	5.06	83.5	1.09	3.92	8595.2	199555.6	782258.0
79X13	14.89	12.39	6.33	6.51	84.8	0.88	3.92	19726.6	293808.0	1151727.4
80X05	13.81	14.08	6.15	7.93	91.1	0.91	3.92	12725.6	175740.4	688902.4
80X06	3.96	10.19	5.55	4.64	75.5	1.02	3.92	13996.3	55425.5	217268.0
80X08	20.54	10.14	4.54	5.60	71.5	0.80	3.92	15192.6	312056.6	1223261.8
80X09	17.20	11.02	7.20	3.81	85.2	0.87	3.92	13931.3	239563.0	939086.9
80X10	18.00	11.37	4.86	6.52	81.1	1.41	3.92	5398.6	97186.2	380969.7
91DY03	10.00	13.85	5.74	8.10	83.9	0.89	3.92	6734.8	67382.0	264137.6
91DY05	23.09	12.57	3.95	8.62	68.9	0.52	3.92	8784.8	202823.4	795067.7
TOTAL		12.82	6.08	6.74	86.1	0.89		161,706	2,259,923	8,858,897

FOX GEOLOGICAL CONSULTANTS LTD.

APPENDIX II

Steffen, Robertson & Kirsten (Vancouver) Ltd.
Letter of Recommendations
Mining and Rock Mechanics

STEFFEN, ROBERTSON AND KIRSTEN (CANADA) INC. Consulting Engineers
Suite 800, 580 Hornby Street, Vancouver, B.C. Canada V6C 3B6
Phone (604) 681-4196 Fax (604) 687-5532

August 6, 1992
Project Number 60652

Curragh Resources
Box 1000
Faro, Yukon
Y0B 1K0

Attention: Leo Hwozdyk

Dear Leo:

RE: CONCEPTUAL MINING METHODS - DY DEPOSIT

1.0 INTRODUCTION

The Dy deposit is a high grade massive sulphide deposit located near Faro, Yukon Territory. It is located at a depth of 530m to 880m. The deposit ranges in thickness from 4m to 27m, with the thickness being locally quite variable. Folding has affected both the strike and the dip of the deposit. The strike changes direction by 90 degrees in the northern portion of the orebody as compared to the south. Dip can vary from 15 to 30 degrees, with the average being approximately 25 degrees.

Mining of the deposit is being contemplated. The purpose of this letter is to examine some of the potential mining methods which may be utilized in the deposit, detail the advantages and disadvantages of each method, as well as delineate any further problems which may need to be addressed in a full scale feasibility study.

2.0 GEOTECHNICAL BACKGROUND

The Dy orebody is genetically and structurally similar to the Faro deposit. Due to its depth, the only information available is drillhole derived. As such, the data may be biased both due to drill deviation as well as to forced interpretations based on relatively wide drillhole spacings. Vertical, or sub-vertical, faulting would not be apparent in vertical drillholes. Such faults do exist, as can be noted from geologic interpretations. Thus, what is described here may only be taken as an approximate description of what may be encountered.

Ore is hosted in a sequence of phyllites, schists, and quartzites. The phyllites and schists, which compose the hanging wall, have an average RMR of approximately 20-35. They are foliated with a very pronounced parting being imparted to the rock mass. The foliation is extremely weak and separates with only minor displacement. Estimated compressive strength of the materials, perpendicular to foliation is between 14 - 35 MPa. The ore appears to be similar to the Faro ore. However, the only core which could be examined during the March 5, 1991 site visit had been split. RMR for the sulphides appeared to be between 45 to 60. This equates to that of the Faro deposit. For that reason, a design rock mass strength of 54 MPa (Faro's estimated strength) will be used for the Dy massive sulphide.

In-situ stresses will be much higher at Dy than within the operational envelope of the Faro deposit. Vertical overburden stress within the ore should range from 13 MPa to 22 MPa, compared to a maximum of 7 MPa for the Faro underground operation. Horizontal stresses are unknown but could be up to twice the vertical stress.

3.0 MINING METHOD SELECTION

When choosing a mining method for a deposit, the physical constraints of the orebody and the required production rates become paramount. Of these two parameters, the physical constraints of the orebody will always control the eventual maximum sustained production rate of the mine.

Some of the physical parameters controlling the mining method are:

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

For Dy, the main constraints are the irregular, intermediate dip which complicates the method ultimately selected, the high ore strength with a weak hanging wall, moderate ore thickness, and relatively high vertical in-situ stresses as well as potential tectonic stresses.

For our intents, two broad categories of mining may be defined. These, partial extraction and full extraction of the orebody, are described below.

3.1 Partial Extraction

Partial extraction entails the removal of only a portion of the orebody, with some of the orebody being used as non-recoverable support. This is most readily typified by room and pillar mining, without pillar recovery, as practised at the Faro underground operation. Various stoping mining methods, such as open stoping with non-recoverable pillars are also included in this class.

Room and pillar mining without pillar recovery is possible in the Dy orebody. However, the weak hanging wall will likely require that ore be left in the back for support purposes. Recoveries would likely not exceed 50% of the mineable reserve. Table 3.1 illustrates the maximum theoretical recoveries for varying safety factors in certain zones of the orebody. Partial pillar recovery combined with numerical analysis and practical experience may allow one to approach or exceed these maximum theoretical values. For a feasibility study it would be prudent to utilize approximately 80%-90% of the theoretical value.

TABLE 3.1

Maximum theoretical recovery, standard room and pillar

Mining area	Pillar stress (MPa)	% Recovery @ SF=1.5	%Recovery @ SF=1.3
Southern zone	23	36	47
Northern zone	17	53	59
Western zone	21	42	49

Given the recoveries mentioned in the above table, it appears unlikely that room and pillar without pillar recovery is a viable alternative for mining.

3.2 Complete Extraction

Complete extraction of an orebody entails recovery of most of the delineated mining reserve. This can be typified by cut-and-fill, caving methods, or concrete pillar mining (a variant of cut-and-fill). All of these will be examined within this section.

3.2.1 Cut-and-Fill

Cut and fill could be operated in a post-pillar format using either a room and pillar layout or long rib pillars. Conventional post pillar cut and fill is usually operated within thicker, steeper dipping, deposits. Using this method at Dy would be a departure from current experience.

Cut and fill is usually operated with hydraulic fill (is fill available) which is both expensive and has a low productivity. Large areas would be open at any one time. This could require barrier pillars. A fairly low, overall recovery might be achieved.

Alternative fills could be used but this would not change the requirement for large, open areas.

Finally, the low dip might lead to considerable waste footwall development for access purposes.

We have not considered cut and fill in any detail as other methods seem more practical in this type of orebody.

3.2.2 Caving

Caving has been discussed as an alternative mining method for the Dy deposit. This has been discussed primarily in conjunction with the Cascade method (Mabson & Russell, 1981). A diagrammatic sketch describing this method, taken from the paper, is shown in Figure 3.1 overleaf. This method, although having possibilities, is not really suited for the conditions existing at Dy.

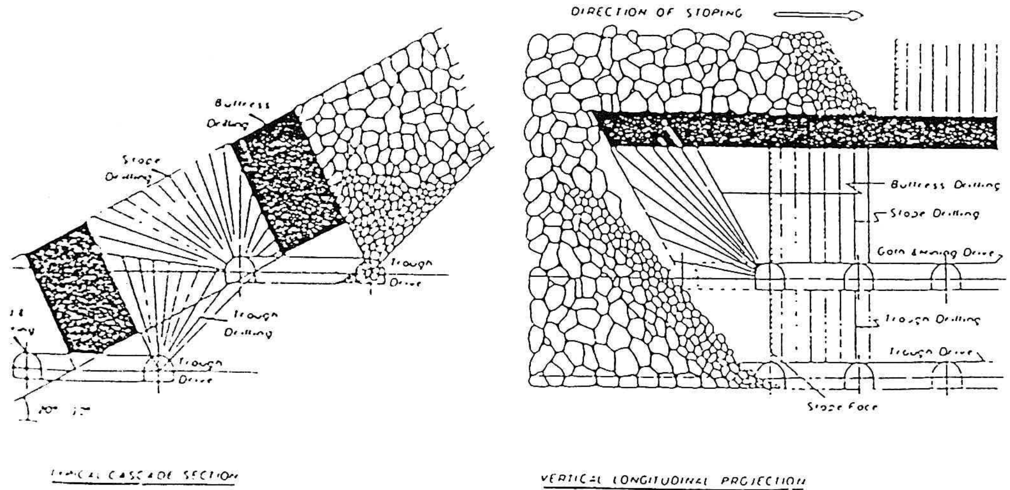


Figure 3.1 Cascade method (after Mabson & Russel, 1981)

The Cascade method was developed at the Mufilira mine for moderately dipping orebodies (20-40 degrees) with a thickness of 5m - 16m. This would be similar to Dy. However, the method works only if a large span can be maintained in the actual mining stope. Mufilira is characterized by extremely competent ground both in the orebody and the hanging wall. Given the apparent weak hanging wall and the high vertical stresses at Dy, the span which could be opened before substantial roof instability occurs is likely on the order of 10m to 15m. In addition, the buttress pillars will have to be quite large to provide protection against the live load of the caved muck as well as the overburden load. This results in relatively small rooms with large buttress pillars. Extraction during the primary phase would likely give about 35-45% recovery. Secondary mining of the buttress pillars would allow another 20-30% giving an overall extraction of between 55-70% with perhaps 15-40% dilution.

The Cascade method is development intensive, with all development conducted in the footwall. It could be a relatively high production method if worked properly. However, much depends on the actual ground conditions encountered in the immediate mining area. Recovery, although possibly better than room and pillar, would be offset by the large amounts of footwall development and very high dilution.

A method of caving using post-pillars is also possible. This is shown in Figure 3.2. The method would only work in relatively thin ore (5m-8m).

Main headings are driven on relatively wide spacings, leaving large strike pillars as buttresses. This are reduced to post pillars, with the remaining large pillars acting as the breaker line. Mucking is conducted under the protection of the post-pillars near the breaker line. These pillars will later collapse as full roof loads are applied. Recovery in this instance could be from 70-80%.

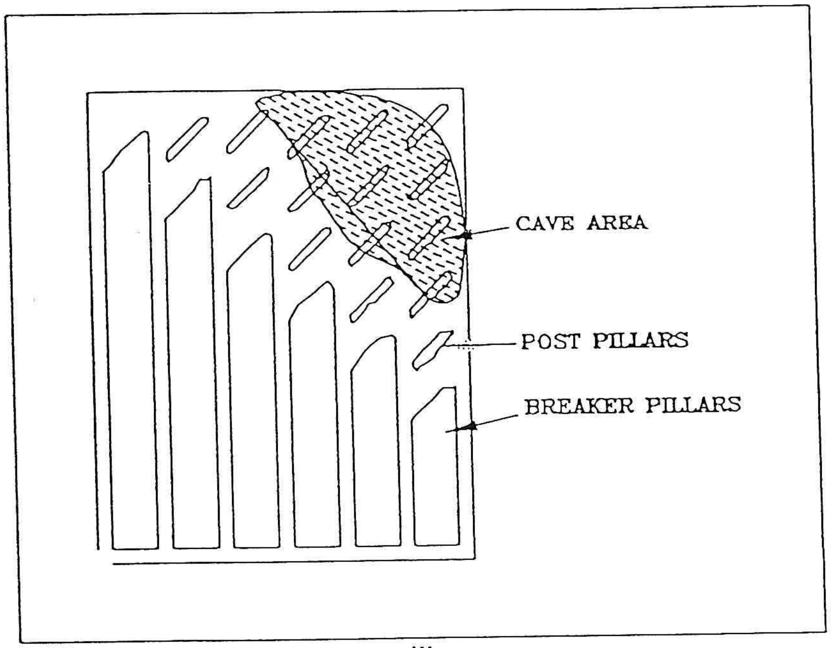


Figure 3.2 Caving using post pillar recovery

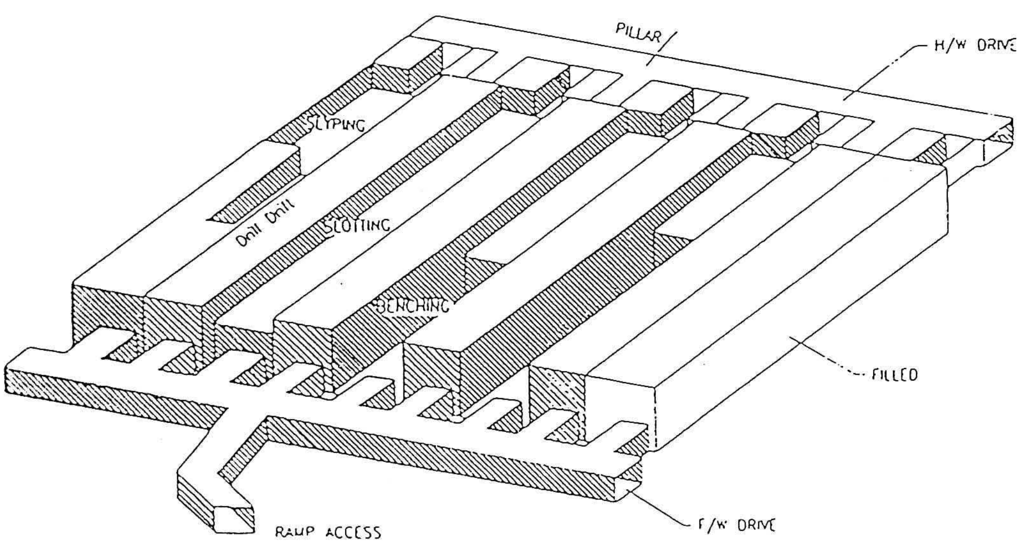


Figure 3.3 Concrete pillar mining

This method relies on a relatively weak hanging wall and strong ore. It may be applicable to the Dy but would be limited to the ore thicknesses mentioned above. It is also largely unproved except in copper mining in Poland and in coal mining.

3.2.3 Concrete Pillar

The method with possibly the most potential for the Dy deposit is the concrete pillar method. This method, shown diagrammatically in Figure 3.3, utilizes fill pillars for support of hanging wall as secondary extraction is conducted. This results in complete recovery of thin to moderately thick orebodies.

The method is based on developing extraction panels within the orebody (Figure 3.3). Pillars are left between the panels for roof support. These pillars can be designed to support the entire overburden load or as post-pillars. The primary extraction panels are then filled with a high quality cemented fill (cemented rock fill). The pillars between the newly created concrete filled panels are then excavated. These, upon completion, can be backfilled with waste rock or fill. Extraction should approach 90-100%.

For feasibility analysis purposes, primary and secondary extraction panels have been assigned an equal width of 8m. This span should be stable if a reinforceable sulphide skin is left against the back and if the pillars are not subject to full overburden load.

Panel lengths have been restricted to a maximum of 100m due to the irregularity of the deposit (pinch and swell). In addition, if remote equipment is to be used, visual observation of the machine is recommended. This is difficult even at 100m.

Thin sections of the deposit (up to 7m) can be mined as one pass operations. A single heading would be driven at mining width, then slashed to 8m. Fill is placed tight against the back, allowed to cure, and the pillars extracted in a fashion similar to the primary room mining. Tight placement of high quality fill is a must for the concrete pillars to function properly. Their purpose is to prevent large roof displacements, as well as carrying some minor stress. If roof displacement is allowed to occur, it will be followed shortly by roof collapse.

For Dy, an 8m panel and pillar width have been assumed which equates to a 50% primary extraction ratio. From Table 3.1, it can be noticed that pillars with a 50% extraction ratio will not be stable for full overburden support. This means that the panel pillars (secondary recovery areas) will be, in most cases, behaving as post-pillars. As such, these pillars will be in a continual state of disintegration. Slabbing and spalling will be commonplace. Remedial pillar wall support to prevent falls on equipment, if desired, would consist of 8 ft (or longer) grouted rebar in conjunction with straps. This would have to be supplemented with mesh if men were to enter the working areas. Given the possible working conditions here, remote mining is to be recommended.

Only two to three primary stopes can be open at any one time. Fill must be kept current. If the fill cycle lags behind the primary cycle by too great a distance, full overburden weight will begin to be applied to the post-pillars, increasing their distress. This could cause loss of the mining area. Proper sequencing of fill and mining cycles will avoid these problems.

Thicker sections of the ore (over 7m) will require access from each end of the panel. This may be done by one of three methods: benching, breasting, or the Endako method. These are shown as Figures 3.4, 3.5, and 3.6 respectively.

Benching

Benching is conducted by driving a mining width excavation against the hanging wall followed by slashing to the required 8m width. This is followed by benching the ore with vertical blastholes. With care, even extremely thick ore may be taken.

As was stated earlier, the pillars will likely be in a yielding state. As such, they will be continually spalling and slabbing. For this reason, remote mucking is recommended for benching. However, if pillar support is desired, blasted muck can be left in the stope for machine access so that bolting can be conducted.

The advantages to this method is that the entire remaining ore thickness can be mined in one pass. Mucking can be ongoing concurrent with drilling. This reduces lag between blasts as well as increasing the tonnage of each blast, thereby increasing productivity. Pillar walls are kept relatively vertical by virtue of the vertical production blasts.

Disadvantages include the possibility of stope loss if a large failure occurs from the pillar walls during mining.

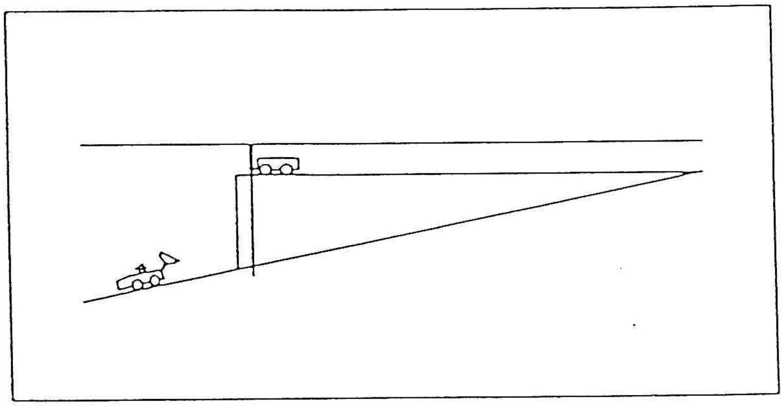


Figure 3.4 Bench blasting

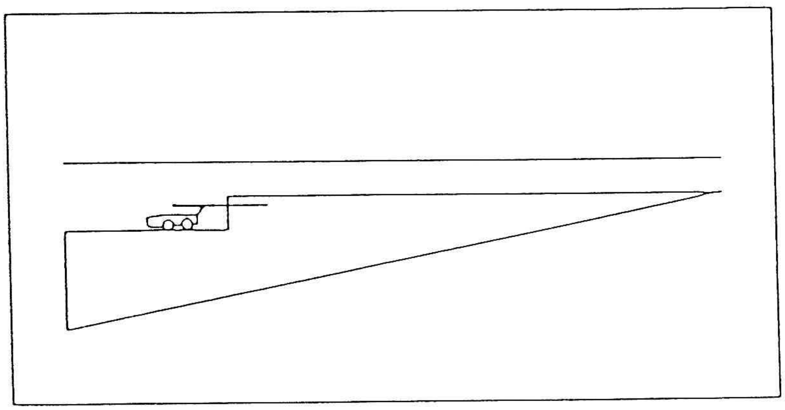


Figure 3.5 Breasting

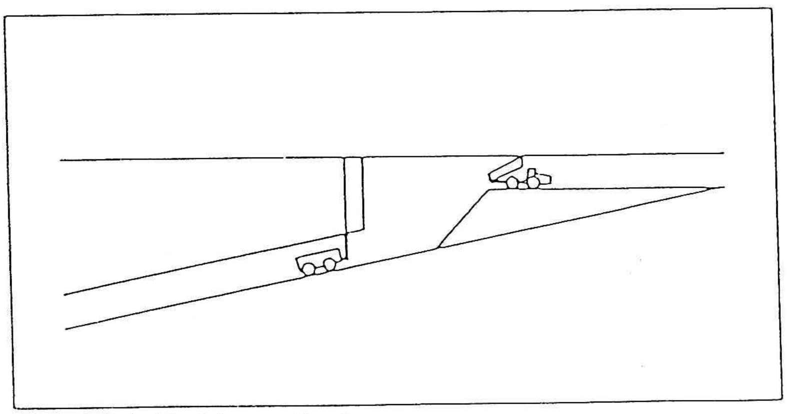


Figure 3.6 Endako method

Breasting

Breasting is depicted schematically in Figure 3.5. Here, as with benching, a primary excavation is opened near the hanging wall and slashed to final width. Breasting is then conducted on the ore in the sill of the excavation. If dual access is provided, this can be done from both ends of the panel simultaneously.

Advantages of this method are limited. Dual access to the panel is not required for mining and the pillar walls can be supported as the number of lifts progress into the sill.

The disadvantages of this excavation method generally outweigh the advantages. Multiple passes are required to access all the ore in the panel, resulting in relatively low production rates. Three distinct cycles are required during mining (drilling, support, mucking) without the opportunity for overlap as for standard benching. Pillar walls are not vertical but become thicker at depth due to the offset of each round from the pillar wall.

Endako method

This method differs from the other two in that primary access is along the footwall. After slashing to full panel width, vertical drillholes used for the production blasts as shown in Figure 3.6. Mucking is by remote loader from the footwall access drive.

Advantages to this method include the fact that fill can be placed while mining is taking place. This increases wall support and minimizes exposure time of the pillar wall, increasing stability. It also allows cycle overlap as the fill can be placed during the mining cycle which is not possible with the other two methods.

Disadvantages include the necessary cleanup, scaling, and support of a rough blasted roof while fill is being placed. Roof stability is not quite as certain as with the other two methods as the holes are drilled towards, not from, the hanging wall. The possibility of a hard-toe developing in the back is ever present due to the restriction on overdrilling into the hanging wall.

4.0 CONCLUSIONS AND RECOMMENDATIONS

At present, given the information available, the most promising method for mining the Dy deposit appears to be the concrete pillar method with standard benching. Undercut-and-fill as well as the caving methods described previously may rate some consideration but are risky for a feasibility level study.

More questions remained to be answered about the Dy deposit than have been asked to date. For example, we have assumed that the deposit is dry, that no abrupt breaks occur in the hanging or footwalls, that the ore is uniformly strong, that faults will have little or no impact, etc..

This may be allowable for a feasibility study for access to the area. However, what has been discussed up to this point is only conceptual in nature.

In order to complete this study for a true mine design access to the area, or additional detailed drilling information would be required. Backfill placement systems, fill sources, and backfill design will be critical to the success of the system. Allowable displacement, support required, and location of the production shaft should be addressed. Location of the main access headings, support requirements and excavation methodology should be examined. For example, should the main access be in cemented rock fill or in a large barrier type pillar? What loads can be expected on the pillars (secondary extraction panels)? What would happen if a mining area is lost (cost sensitivity)?

Some of these questions are not critical at this stage but should be addressed before a definitive mining decision is made.

Yours truly,

STEFFEN, ROBERTSON AND KIRSTEN (CANADA) INC.

Dr. James I. Mathis, P.E.

Dr. Chris H. Page, P.Eng.
JIM/067

APPENDIX III

Fox Geological Consultants Ltd.
Eleven-Year Development and Production Schedule

Fox Geological Consultants Ltd. 1409 - 409 Granville Street, Vancouver, B.C. V6C 1T8
Telephone (604) 669-5736 Fax (604) 681-3920

CURRAGH INC.
DY UNDERGROUND MINING BLOCKS
file: DYMBLOCK.WR1
Date 11-Oct-92

MINING BLOCK	DDH POLYGON	AREA	VRT TH	TONNES	ASSAY GRADES %Pb	%Zn	Ag(g/t)	Au(g/t)	COMMENTS
B1	77X06	5944.6	25.09	584574.8	6.37	11.58	112.9	0.621	
	78X11	7563.7	7.61	225634.2	5.03	8.05	83.1	0.742	
	79X07	2775.8	12.25	133337.4	3.99	8.04	57.2	0.605	
	80DY09	8606.3	15.32	516879.9	2.95	6.50	49.9	0.294	
	91DY03	6542.3	10.01	256586.4	5.74	8.10	83.9	0.887	
	91DY05	9393.2	23.09	850131.2	3.95	8.62	68.9	0.520	
SUB TOTAL				2,567,144.0	4.58	8.73	77.2	0.558	
B2	78X09	14091.3	5.60	309442.7	2.69	6.66	43.8	0.588	
				0.0					
SUB TOTAL				309,442.7	2.69	6.66	43.8	0.588	
B3	78X05*	9994.9	17.46	683965.4	4.43	8.27	69.6	0.875	* BASE METAL RICH PYRITIC QUARTZITES
	80DY04	11448.1	11.59	520029.5	3.23	6.33	40.9	0.453	
SUB TOTAL				1,203,994.9	3.91	7.43	57.2	0.693	
B4	78X04	9973	5.19	203016.0	9.49	12.88	151.0	1.139	
	79X02	10266.3	3.97	159889.0	4.03	6.52	57.0	0.252	
	79X04	10109.6	4.73	187289.6	3.68	7.80	63.9	0.625	
	79X05*	22459	4.21	370733.4	3.65	5.86	56.4	0.199	* BASE METAL RICH PYRITIC QUARTZITES
SUB TOTAL				920,928.0	5.01	7.92	78.9	0.502	
B5	90DY05	15546.5	18.33	1116889.2	5.76	9.83	77.7	0.438	
				0.0					
SUB TOTAL				1,116,889.2	5.76	9.83	77.7	0.438	
"B" ZONE SUB TOTAL				6,118,398.8	4.63	8.45	71.9	0.556	
A1	79X06	10238.3	21.02	843499.1	10.66	6.76	119.7	1.160	
	79X11	6815.1	23.22	620246.6	6.14	5.06	83.5	1.092	
	79X12	16384.5	13.33	856149.1	4.55	4.81	65.1	0.537	
SUB TOTAL				2,319,894.9	7.20	5.59	89.8	0.912	
A2	80X01	14767.1	4.25	245846.2	5.96	5.71	80.2	1.150	
				0.0					
SUB TOTAL				245,846.2	5.96	5.71	80.2	1.150	
A3	77X05	7941.4	6.75	210129.4	5.27	7.66	108.3	1.353	
	79X08	13363.1	4.10	214562.2	4.06	5.37	84.3	1.053	
SUB TOTAL				424,691.7	4.66	6.50	96.2	1.201	
A4	79X18	11810.7	4.13	190979.0	2.99	6.46	68.3	0.856	
				0.0					
SUB TOTAL				190,979.0	2.99	6.46	68.3	0.856	
A5	79X13	14281.7	14.89	833829.6	6.33	6.51	84.8	0.876	
	79X16	9092	14.63	521493.8	4.49	5.07	65.5	0.550	
	80X05	12725.6	13.81	688902.9	6.15	7.93	91.1	0.910	
	80X08	14002.5	20.54	1127436.5	4.54	5.60	71.5	0.800	
	80X09	13931.3	17.20	939085.5	7.20	3.81	85.2	0.870	
	80X10	12831.2	18.00	905470.1	4.86	6.52	81.1	1.412	
SUB TOTAL				5,016,218.5	5.61	5.85	80.1	0.925	
A6	80X06	12503.8	3.96	194099.0	5.55	4.64	75.5	1.018	
				0.0					
SUB TOTAL				194,099.0	5.55	4.64	75.5	1.018	
A7	80X02	11765.9	15.29	705210.4	3.53	6.61	62.4	0.911	
				0.0					
SUB TOTAL				705,210.4	3.53	6.61	62.4	0.911	
A8	80X07	15842.2	6.78	420985.6	4.26	5.04	61.6	1.098	
				0.0					
SUB TOTAL				420,985.6	4.26	5.04	61.6	1.098	
"A" ZONE SUB TOTAL				9,517,925	5.70	5.82	80.7	0.947	
GRAND TOT				15,636,324	5.28	6.85	77.3	0.794	
10% DILUTION				15,636,000	4.80	6.22	70.3	0.72	

FOX GEOLOGICAL CONSULTANTS LTD.

* NOTE: INCLUSION OF POLYGONS BASED ON ASSUMED ROCK QUALITIES FOR BASE METAL RICH PYRITIC QUARTZITES MEETING GRADE CUTOFF CRITERIA AND HAVING MODRATELY HIGH S.G. VALUES.

file DYMPBLOCK

ASSUMPTIONS DEFINING "MINEABLE" CRITERIA:

- MINING OF "MASSIVE SULPHIDE" ORE.

- A 4m MINIMUM MINING HEIGHT WITH A 9% Pb+Zn GRADE CUTOFF. (INTERSECTIONS OF LESS THAN 4m THICKNESS MEETING GRADE CUTOFF CRITERIA WERE DILUTED TO A 4m THICKNESS WITH FOOTWALL MATERIAL).

- A 1m "SKIN" OF ORE WOULD BE REQUIRED TO PROVIDE ADEQUATE BACK SUPPORT OF A MINEABLE BLOCK.

- GRADE DIFFERENTIATION WITHIN MASSIVE SULPHIDE ORE WOULD NOT BE POSSIBLE AS A MINING PARAMETER (ie. MINING OF SULPHIDES WOULD OCCUR ON A VISUAL BASIS). THUS SULPHIDE INTERSECTIONS WERE WEIGHT AVERAGED OVER THE WHOLE SULPHIDE INTERSECTION UNLESS BROKEN BY INTERVALS OF WASTE DEFINING POSSIBLE "HANGINGWALL" OR "FOOTWALL" CONTACTS.

- WASTE INTERVALS OF 5m OR GREATER BETWEEN SULPHIDE INTERSECTIONS WERE EXCLUDED FROM WEIGHT AVERAGE GRADE COMPOSITES.

POLYGONS INCLUDED IN MINEABLE CATEGORY NOT MEETING ABOVE CRITERIA

MINING	DDH				ASSAY GRADES				
BLOCK	POLYGON	AREA	VRT TH	TONNES	%Pb	%Zn	Ag(g/t)	Au(g/t)	COMMENTS
B3	78X05*	9994.9	17.46	683965.4	4.43	8.27	69.6	0.875	* BASE METAL RICH PYRITIC QUARTZITES
B4	79X05*	22459	4.21	370733.4	3.65	5.86	56.4	0.199	* BASE METAL RICH PYRITIC QUARTZITES
SUB TOTAL				1,054,698.8	4.15	7.43	65.0	0.637	
10% DILUTED				1,054,699	3.78	6.75	59.1	0.58	

* NOTE INCLUSION OF THESE POLYGONS WAS BASED ON ASSUMED ROCK QUALITIES FOR BASE METAL RICH PYRITIC QUARTZITES MEETING GRADE CUTOFF CRITERIA AND HAVING MODERATELY HIGH S.G. VALUES. THESE POLYGONS SHOULD LIKELY BE REASSESSED AS PART OF THE MINEABLE RESERVE INVENTORY.

POLYGONS NOT INCLUDED IN MINEABLE RESERVE CATEGORY

77X01 - 4m OF 10.39% Pb+Zn BOUNDED BY DRILLHOLES OF BELOW CUTOFF GRADE AND THICKNESS.

80X02 - AN UPPER SULPHIDE ZONE OF 12.6m @ 12.43% Pb+Zn

79X14 - 17.5m OF 8.23% Pb+Zn MASSIVE SULPHIDES (THE UPPER 8.5m GRADING 10.73% Pb+Zn).

80X04 - 3.4m OF 12.09% Pb+Zn (DILUTED TO 4.0m THIS POLYGON WOULD MEET THE 9% Pb+Zn CUTOFF)

CURRAGH INC.

DY UNDERGROUND DEVELOPMENT/PRODUCTION SCHEDULE SUMMARY

			ASSAY GRADES			
YEAR	ADV (m)	TONNES	%Pb	%Zn	Ag(g/t)	Au(g/t)
YEAR 1	4743	687,308	4.52	8.55	75.5	0.55
YEAR 2	5995	1,000,296	4.68	8.45	71.8	0.54
YEAR 3	5655	1,006,333	4.99	7.83	73.0	0.65
YEAR 4	5310	1,012,742	5.26	7.63	74.6	0.69
YEAR 5	5595	1,051,506	5.17	7.24	73.7	0.72
YEAR 6	5600	1,060,924	5.29	6.76	80.9	0.84
YEAR 7	5900	1,112,349	5.39	6.66	81.4	0.86
YEAR 8	5875	1,148,790	5.56	5.94	79.7	0.94
YEAR 9	5455	1,120,267	5.77	5.80	79.9	0.94
YEAR 10	4495	1,042,250	5.75	5.82	80.4	0.93
YEAR 11	4455	1,006,813	5.08	5.92	75.0	0.94
SUB TOTAL	59078	11,249,578	5.26	6.89	77.1	0.79
10% DILUTION	59078	11,249,578	4.78	6.26	70.1	0.72

Date: 11-Oct-92
file: DYSCHED.WK1

CURRAGH RESOURCES INC
DY UNDERGROUND DEVELOPMENT/PRODUCTION SCHEDULE
File DYYR1SCH.WR1
Date 11-Oct-92

YEAR 1

DEVELOPMENT

ORE DEVELOPMENT			1ST QTR	# DAYS= 90					2ND QTR	# DAYS= 91					3RD QTR	# DAYS= 92				
				SG=3.92	ASSAY GRADES						ASSAY GRADES						ASSAY GRADES			
HEADING	HT (m)	W (m)	ADV (m)	TONNES	%Pb	%Zn	Ag(g/t)	Au(g/t)	ADV (m)	TONNES	%Pb	%Zn	Ag(g/t)	Au(g/t)	ADV (m)	TONNES	%Pb	%Zn	Ag(g/t)	Au(g/t)
N1	4.25	5.00	110	9163.0	4.58	8.73	77.2	0.558	200	16660.0	4.58	8.73	77.2	0.558	70	5831.0	5.01	7.92	78.9	0.502
N1 U/P	4.00	4.00	110	6899.2	4.58	8.73	77.2	0.558	155	9721.6	4.58	8.73	77.2	0.558		0.0				
N2	4.25	5.00	80	6664.0	4.58	8.73	77.2	0.558	50	4165.0	4.58	8.73	77.2	0.558	70	5831.0	4.58	8.73	77.2	0.558
N4	4.25	5.00	80	6664.0	4.58	8.73	77.2	0.558	50	4165.0	4.58	8.73	77.2	0.558	60	4998.0	4.58	8.73	77.2	0.558
N5	4.25	5.00	50	4165.0	4.58	8.73	77.2	0.558	50	4165.0	4.58	8.73	77.2	0.558	50	4165.0	4.58	8.73	77.2	0.558
N6	4.25	5.00	30	2499.0	4.58	8.73	77.2	0.558	50	4165.0	4.58	8.73	77.2	0.558	50	4165.0	4.58	8.73	77.2	0.558
N7	4.25	5.00	30	2499.0	4.58	8.73	77.2	0.558	50	4165.0	4.58	8.73	77.2	0.558	50	4165.0	2.69	6.66	43.8	0.588
N8	4.25	5.00		0.0					50	4165.0	4.58	8.73	77.2	0.558	60	4998.0	5.01	7.92	78.9	0.502
N9	4.25	5.00		0.0					50	4165.0	4.58	8.73	77.2	0.558	50	4165.0	4.58	8.73	77.2	0.558
N10	4.25	5.00		0.0					50	4165.0	5.01	7.92	78.9	0.502	50	4165.0	5.01	7.92	78.9	0.502
N11	4.25	5.00		0.0					50	4165.0	4.58	8.73	77.2	0.558	50	4165.0	4.58	8.73	77.2	0.558
N12	4.25	5.00		0.0						0.0						0.0				
N14	4.25	5.00		0.0						0.0						0.0				
N15	4.25	5.00		0.0						0.0					30	2499.0	5.01	7.92	78.9	0.502
N17	4.25	5.00		0.0						0.0						0.0				
				0.0						0.0						0.0				
SUB TOTAL			490	38553	4.58	8.73	77.2	0.558	805	63867	4.61	8.68	77.3	0.564	590	49147	4.57	8.27	75.0	0.541

WASTE DEVELOPMENT			1ST QTR		2ND QTR		3RD QTR	
				SG=2.90 [5.0x4.25m ARCH=17.183sq m]				
HEADING	HT (m)	W (m)	ADV (m)	TONNES	ADV (m)	TONNES	ADV (m)	TONNES
SHOP	6.50	9.00	30	4306.5		0.0		0.0
N1	5.00	4.25	140	6976.3		0.0		0.0
N1 U/P	5.00	4.25	20	996.6	30	1494.9		0.0
N2	5.00	4.25	140	6976.3		0.0		0.0
N2 V RSE	2.50	DIAM	665	9466.5		0.0		0.0
N3	5.00	4.25	125	6228.8		0.0		0.0
N3 R RSE	1.50	DIAM		0.0	663	3397.7		0.0
N13	5.00	4.25		0.0	50	2491.5	100	4983.1
DECLINE	5.00	4.25		0.0	200	9966.1	500	24915.4
W1	5.00	4.25	20	996.6		0.0		0.0
SUB TOTAL			1140	35948	943	17350.3	600	29898

PRODUCTION

STOPE DEVELOPMENT (DRIFT & SLASH)			1ST QTR						2ND QTR						3RD QTR					
				SG=3.92	ASSAY GRADES						ASSAY GRADES						ASSAY GRADES			
HEADING	HT (m)	W (m)	ADV (m)	TONNES	%Pb	%Zn	Ag(g/t)	Au(g/t)	ADV (m)	TONNES	%Pb	%Zn	Ag(g/t)	Au(g/t)	ADV (m)	TONNES	%Pb	%Zn	Ag(g/t)	Au(g/t)
N4 - 5	5.00	8.00		0.0					140	21952.0	4.58	8.73	77.2	0.558	70	10976.0	4.58	8.73	77.2	0.558
N5 - 6	5.00	8.00		0.0					75	11760.0	4.58	8.73	77.2	0.558	113	17640.0	4.58	8.73	77.2	0.558
N7 - 8	5.00	8.00		0.0					150	23520.0	4.58	8.73	77.2	0.558	75	11760.0	4.58	8.73	77.2	0.558
N7 - 9	5.00	8.00		0.0						0.0					38	5880.0	2.69	6.66	43.8	0.588
N8 - 10	5.00	8.00		0.0					50	7840.0	4.58	8.73	77.2	0.558	75	11760.0	5.01	7.92	78.9	0.502
N9 - 11	5.00	8.00		0.0					75	11760.0	4.58	8.73	77.2	0.558	113	17640.0	4.58	8.73	77.2	0.558
N10 - 12	5.00	8.00		0.0						0.0						0.0				
N12 - 14	5.00	8.00		0.0						0.0						0.0				
N17	5.00	8.00		0.0						0.0						0.0				
				0.0						0.0						0.0				
SUB TOTAL			0	0	0.00	0.00	0.0	0.000	490	76832	4.58	8.73	77.2	0.558	483	75656	4.50	8.44	74.9	0.562

STOPE PRODUCTION (BENCHING)			1ST QTR						2ND QTR						3RD QTR					
				SG=3.92	ASSAY GRADES						ASSAY GRADES						ASSAY GRADES			
HEADING	HT (m)	W (m)	ADV (m)	TONNES	%Pb	%Zn	Ag(g/t)	Au(g/t)	ADV (m)	TONNES	%Pb	%Zn	Ag(g/t)	Au(g/t)	ADV (m)	TONNES	%Pb	%Zn	Ag(g/t)	Au(g/t)
N4 - 5	10.00	8.00		0.0					70	21952.0	4.58	8.73	77.2	0.558	70	21952.0	4.58	8.73	77.2	0.558
N5 - 6	10.00	8.00		0.0					35	10976.0	4.58	8.73	77.2	0.558	70	21952.0	4.58	8.73	77.2	0.558
N7 - 8	10.00	8.00		0.0					75	23520.0	4.58	8.73	77.2	0.558	75	23520.0	4.58	8.73	77.2	0.558
N8 - 10	10.00	8.00		0.0						0.0					50	15680.0	4.58	8.73	77.2	0.558
N9 - 11	10.00	8.00		0.0						0.0					75	23520.0	4.58	8.73	77.2	0.558
				0.0						0.0						0.0				
				0.0						0.0						0.0				
SUB TOTAL			0	0	0.00	0.00	0.0	0.000	180	56448	4.58	8.73	77.2	0.558	340	106624	4.58	8.73	77.2	0.558
PROD TOT			0	0	0.00	0.00	0.0	0.000	670	133280	4.58	8.73	77.2	0.558	823	182280	4.55	8.61	76.2	0.555
TOTAL ORE			490	38553	4.58	8.73	77.2	0.558	1475	197147	4.59	8.71	77.2	0.557	1413	231427	4.55	8.54	76.0	0.552
10% DILUTION			490	38,553	4.16	7.94	70.2	0.51	1,475	197,147	4.17	7.92	70.2	0.51	1,413	231,427	4.14	7.76	69.1	0.50

CEMENTED ROCKFILL		1ST QTR	2ND QTR	3RD QTR
		SG=2.36		
LOCATION		TONNES	TONNES	TONNES
		0.0	0.0	102557.0
		0.0	0.0	0.0
SUB TOTAL		0	0	102557

FOX GEOLOGICAL CONSULTANTS LTD.

1ST QTR		2ND QTR		3RD QTR	
#TONNES/DAY ORE:	428.4	#TONNES/DAY ORE:	2166.4	#TONNES/DAY ORE:	2515.5
#TONNES/DAY WASTE:	399.4	#TONNES/DAY WASTE	190.7	#TONNES/DAY WASTE:	325.0
#TONNES/DAY MINED.	827.8	#TONNES/DAY MINED:	2357.1	#TONNES/DAY MINED:	2840.5
#TONNES/DAY FILLED	0	#TONNES/DAY FILLED	0	#TONNES/DAY FILLED:	1114.75
%DEVELOPMENT:	100.0%	%DEVELOPMENT:	37.9%	%DEVELOPMENT:	30.2%
%PRODUCTION.	0.0%	%PRODUCTION:	62.1%	%PRODUCTION:	69.8%
%BENCH PRODUCTION	0.0%	%BENCH PRODUCTIO	26.3%	%BENCH PRODUCTION:	40.8%

CURRAGH RESOURCES INC.
DY UNDERGROUND DEVELOPMENT/PRODUCTION SCHEDULE
File DYYR1SCH WR1
Date 11 - Oct - 92

YEAR 1

DEVELOPMENT

ORE DEVELOPMENT			4TH QTR	# DAYS= 77					TOTAL	# DAYS= 350				
					ASSAY GRADES						ASSAY GRADES			
HEADING	HT (m)	W (m)	ADV (m)	TONNES	%Pb	%Zn	Ag(g/t)	Au(g/t)	ADV (m)	TONNES	%Pb	%Zn	Ag(g/t)	Au(g/t)
N 1	4.25	5.00		0.0					380	31654	4.66	8.58	77.5	0.548
N 1 U/P	4.00	4.00		0.0					265	16621	4.58	8.73	77.2	0.558
N 2	4.25	5.00	65	5414.5	5.01	7.92	78.9	0.502	265	22075	4.69	8.53	77.6	0.544
N 4	4.25	5.00	60	4998.0	5.01	7.92	78.9	0.502	250	20825	4.68	8.54	77.6	0.546
N 5	4.25	5.00	50	4165.0	2.69	6.66	43.8	0.588	200	16660	4.11	8.21	68.9	0.566
N 6	4.25	5.00	30	2499.0	2.69	6.66	43.8	0.588	160	13328	4.23	8.34	70.9	0.564
N 7	4.25	5.00	60	4998.0	2.69	6.66	43.8	0.588	190	15827	3.49	7.53	57.9	0.576
N 8	4.25	5.00		0.0					110	9163	4.81	8.29	78.1	0.527
N 9	4.25	5.00	50	4165.0	2.69	6.66	43.8	0.588	150	12495	3.95	8.04	66.1	0.568
N 10	4.25	5.00	50	4165.0	5.76	9.83	77.7	0.438	150	12495	5.26	8.56	78.5	0.481
N 11	4.25	5.00	50	4165.0	4.58	8.73	77.2	0.558	150	12495	4.58	8.73	77.2	0.558
N 12	4.25	5.00	50	4165.0	5.01	7.92	78.9	0.502	50	4165	5.01	7.92	78.9	0.502
N 14	4.25	5.00	50	4165.0	5.01	7.92	78.9	0.502	50	4165	5.01	7.92	78.9	0.502
N 16	4.25	5.00	50	4165.0	5.01	7.92	78.9	0.502	80	6664	5.01	7.92	78.9	0.502
N 17	4.25	5.00	50	4165.0	4.58	8.73	77.2	0.558	50	4165	4.58	8.73	77.2	0.558
				0.0					0	0	0.00	0.00	0.0	0.000
SUB TOTAL			615	61230	4.28	7.82	67.7	0.532	2500	202796	4.51	8.37	74.3	0.546

WASTE DEVELOPMENT			4TH QTR						TOTAL					
HEADING	HT (m)	W (m)	ADV (m)	TONNES					ADV (m)	TONNES				
SHOP	6.50	9.00		0.0					30	4307				
N 1	5.00	4.25		0.0					140	6976				
N 1 U/P	5.00	4.25		0.0					50	2492				
N 2	5.00	4.25		0.0					140	6976				
N 2 V RSE	2.60	DIAM		0.0					665	9467				
N 3	5.00	4.25		0.0					125	6229				
N 3 R RSE	1.50	DIAM		0.0					663	3398				
N 13	5.00	4.25	100	4983.1					250	12458				
DECLINE	5.00	4.25	400	19932.3					1100	54814				
W 1	5.00	4.25		0.0					20	997				
SUB TOTAL			500	24915					3183	108112				

PRODUCTION

STOPE DEVELOPMENT (DRIFT & SLASH)			4TH QTR						TOTAL					
					ASSAY GRADES						ASSAY GRADES			
HEADING	HT (m)	W (m)	ADV (m)	TONNES	%Pb	%Zn	Ag(g/t)	Au(g/t)	ADV (m)	TONNES	%Pb	%Zn	Ag(g/t)	Au(g/t)
N4 - 6	5.00	8.00	70	10976.0	4.58	8.73	77.2	0.558	280	43904	4.58	8.73	77.2	0.558
N6 - 8	5.00	8.00	75	11760.0	4.58	8.73	77.2	0.558	263	41160	4.58	8.73	77.2	0.558
N7 - 9	5.00	8.00	38	5880.0	4.58	8.73	77.2	0.558	263	41160	4.58	8.73	77.2	0.558
N7 - 9	5.00	8.00	75	11760.0	2.69	6.66	43.8	0.588	113	17640	2.69	6.66	43.8	0.588
N8 - 10	5.00	8.00		0.0					125	19600	4.84	8.24	78.2	0.524
N9 - 11	5.00	8.00	75	11760.0	4.58	8.73	77.2	0.558	263	41160	4.58	8.73	77.2	0.558
N10 - 12	5.00	8.00	30	4704.0	5.01	7.92	78.9	0.502	30	4704	5.01	7.92	78.9	0.502
N12 - 14	5.00	8.00		0.0					0	0	0.00	0.00	0.0	0.000
N17	5.00	8.00	60	9408.0	4.58	8.73	77.2	0.558	60	9408	4.58	8.73	77.2	0.558
				0.0					0	0	0.00	0.00	0.0	0.000
SUB TOTAL			423	66248	4.28	8.31	71.4	0.559	1395	218736	4.46	8.50	74.6	0.556

STOPE PRODUCTION (BENCHING)			4TH QTR						TOTAL					
					ASSAY GRADES						ASSAY GRADES			
HEADING	HT (m)	W (m)	ADV (m)	TONNES	%Pb	%Zn	Ag(g/t)	Au(g/t)	ADV (m)	TONNES	%Pb	%Zn	Ag(g/t)	Au(g/t)
N4 - 6	10.00	8.00	70	21952.0	4.58	8.73	77.2	0.558	210	65856	4.58	8.73	77.2	0.558
N6 - 8	10.00	8.00	70	21952.0	4.58	8.73	77.2	0.558	175	54880	4.58	8.73	77.2	0.558
N7 - 9	10.00	8.00	113	35280.0	4.58	8.73	77.2	0.558	263	82320	4.58	8.73	77.2	0.558
N8 - 10	10.00	8.00		0.0					50	15680	4.58	8.73	77.2	0.558
N9 - 11	10.00	8.00	75	23520.0	4.58	8.73	77.2	0.558	150	47040	4.58	8.73	77.2	0.558
				0.0					0	0	0.00	0.00	0.0	0.000
				0.0					0	0	0.00	0.00	0.0	0.000
SUB TOTAL			328	102704	4.58	8.73	77.2	0.558	848	265776	4.58	8.73	77.2	0.558
PROD TOT			750	168952	4.46	8.56	74.9	0.559	2243	484512	4.53	8.63	76.0	0.557
TOTAL ORE			1365	220182	4.42	8.39	73.2	0.552	4743	687308	4.52	8.55	75.5	0.554
10% DILUTION			1,365	220,182	4.02	7.63	66.6	0.50	4,743	687,308	4.11	7.77	68.7	0.50

CEMENTED ROCKFILL		4TH QTR			TOTAL		
LOCATION			TONNES			TONNES	
			105369.6			207927	
			0.0			0.0	
SUB TOTAL			105370			207927	

FOX GEOLOGICAL CONSULTANTS LTD.

4TH QTR		TOTAL	
# TONNES/DAY ORE:	2859.5	# TONNES/DAY ORE:	1963.7
# TONNES/DAY WASTE:	323.6	# TONNES/DAY WASTE:	308.9
# TONNES/DAY MINED:	3183.1	# TONNES/DAY MINED:	2272.6
# TONNES/DAY FILLED:	1368.4	# TONNES/DAY FILLED:	594.076
%DEVELOPMENT:	31.1%	%DEVELOPMENT:	39.1%
%PRODUCTION:	68.9%	%PRODUCTION:	60.9%
%BENCH PRODUCTION:	41.9%	%BENCH PRODUCTION:	33.4%

[illegible] RESOURCES INC

[illegible] UNDERGROUND DEVELOPMENT/PRODUCTION SCHEDULE

[illegible]

[illegible] 19-OCT-92

DEVELOPMENT

YEAR 2 — YEAR 3 — YEAR 4

ORE DEVELOPMENT			YEAR 2	# DAYS= 350					YEAR 3	# DAYS= 350					YEAR 4	# DAYS 350				
MINING				SG=3.92	ASSAY GRADES						ASSAY GRADES						ASSAY GRADES			
BLOCK	HT (m)	W (m)	ADV (m)	TONNES	%Pb	%Zn	Ag(g/t)	Au(g/t)	ADV (m)	TONNES	%Pb	%Zn	Ag(g/t)	Au(g/t)	ADV (m)	TONNES	%Pb	%Zn	Ag(g/t)	Au(g/t)
B1	4.25	5.00	100.0	8330.0	4.58	8.73	77.2	0.558	80	6664.0	4.58	8.73	77.2	0.558		0.0				
B2	4.25	5.00	130.0	10829.0	2.69	6.66	43.8	0.588		0.0						0.0				
B3	4.25	5.00	380.0	31654.0	3.91	7.43	57.2	0.693	145	12078.5	3.91	7.43	57.2	0.693		0.0				
B4	4.25	5.00	630.0	52479.0	5.01	7.92	78.9	0.502		0.0						0.0				
B5	4.25	5.00	310.0	25823.0	5.76	9.83	77.7	0.438		0.0						0.0				
A1	4.25	5.00		0.0					735	61225.5	7.20	5.59	89.8	0.912	235	19575.5	7.20	5.59	89.8	0.912
A2	4.25	5.00		0.0						0.0					210	17493.0	5.96	5.71	80.2	1.150
A3	4.25	5.00		0.0					190	15827.0	4.66	6.50	96.2	1.201	30	2499.0	4.66	6.50	96.2	1.201
A4	4.25	5.00		0.0						0.0					155	12911.5	2.99	6.46	68.3	0.856
A5	4.25	5.00		0.0						0.0					100	8330.0	5.61	5.85	80.1	0.925
A6	4.25	5.00		0.0						0.0						0.0				
	4.25	5.00		0.0						0.0						0.0				
				0.0						0.0						0.0				
SUB TOTAL			1560	129,115	4.67	8.13	70.3	0.547	1150	95,795	6.18	6.19	85.9	0.908	730	60,809	5.63	5.88	81.4	0.982

WASTE DEVELOPMENT			YEAR 2						YEAR 3						YEAR 4					
				SG=2.90	(5.0x4.25m ARCH = 17.183sq m)															
HEADING	HT (m)	W (m)	ADV (m)	TONNES					ADV (m)	TONNES					ADV (m)	TONNES				
N5	5.00	4.25	80	3986.5						0.0						0.0				
N7	5.00	4.25	35	1744.1						0.0						0.0				
N11	5.00	4.25	90	4484.8						0.0						0.0				
N13	5.00	4.25	140	6976.3						0.0						0.0				
DECLINE	5.00	4.25	770	38369.6						0.0						0.0				
W1	5.00	4.25		0.0					120	5979.7						0.0				
W2	5.00	4.25		0.0					220	10962.8						0.0				
W3	5.00	4.25		0.0					60	2989.8						0.0				
W4	5.00	4.25		0.0					80	3986.5						0.0				
W5	5.00	4.25		0.0					145	7225.5					125	6228.8				
W6	5.00	4.25		0.0					40	1993.2						0.0				
S1	5.00	4.25		0.0					50	2491.5					180	8969.5				
S2	5.00	4.25		0.0						0.0					230	11461.1				
S2 CAP RSE	2.40	2.40		0.0						0.0						0.0				
S4	5.00	4.25		0.0						0.0						0.0				
				0.0						0.0						0.0				
				0.0						0.0						0.0				
SUB TOTAL			1115	55,561					715	35,629					536	26,659				

PRODUCTION

STOPE DEVELOPMENT (DRIFT & SLASH)			YEAR 2						YEAR 3						YEAR 4					
MINING				SG=3.92	ASSAY GRADES						ASSAY GRADES						ASSAY GRADES			
BLOCK	HT (m)	W (m)	ADV (m)	TONNES	%Pb	%Zn	Ag(g/t)	Au(g/t)	ADV (m)	TONNES	%Pb	%Zn	Ag(g/t)	Au(g/t)	ADV (m)	TONNES	%Pb	%Zn	Ag(g/t)	Au(g/t)
B1	5.00	8.00	530	83104.0	4.58	8.73	77.2	0.558	315	49392.0	4.58	8.73	77.2	0.558		0.0				
B2	5.00	8.00	550	86240.0	2.69	6.66	43.8	0.588		0.0						0.0				
B3	5.00	8.00	325	50960.0	3.91	7.43	57.2	0.693	800	125440.0	3.91	7.43	57.2	0.693	670	105056.0	3.91	7.43	57.2	0.693
B4	4.00	8.00	1285	161190.4	5.01	7.92	78.9	0.502	1385	173734.4	5.01	7.92	78.9	0.502	955	119795.2	5.01	7.92	78.9	0.502
B5	5.00	8.00	570	89376.0	5.76	9.83	77.7	0.438	235	36848.0	5.76	9.83	77.7	0.438	445	69776.0	5.76	9.83	77.7	0.438
A1	5.00	8.00		0.0					260	40768.0	7.20	5.59	89.8	0.912	470	73696.0	7.20	5.59	89.8	0.912
A2	4.00	8.00		0.0						0.0					240	30105.6	5.96	5.71	80.2	1.150
A3	5.00	8.00		0.0					150	23520.0	4.66	6.50	96.2	1.201	220	34496.0	4.66	6.50	96.2	1.201
A4	4.00	8.00		0.0						0.0					100	12544.0	2.99	6.46	68.3	0.856
A5	5.00	8.00		0.0						0.0						0.0				
				0.0						0.0						0.0				
SUB TOTAL			3260	470,870	4.53	8.14	69.6	0.536	3145	449,702	4.90	7.74	74.5	0.630	3100	445,469	5.21	7.42	76.5	0.713

STOPE PRODUCTION (BENCHING)			YEAR 2						YEAR 3						YEAR 4					
MINING				SG=3.92	ASSAY GRADES						ASSAY GRADES						ASSAY GRADES			
BLOCK	HT (m)	W (m)	ADV (m)	TONNES	%Pb	%Zn	Ag(g/t)	Au(g/t)	ADV (m)	TONNES	%Pb	%Zn	Ag(g/t)	Au(g/t)	ADV (m)	TONNES	%Pb	%Zn	Ag(g/t)	Au(g/t)
B1	10.00	8.00	730	228928.0	4.58	8.73	77.2	0.558	195	61152.0	4.58	8.73	77.2	0.558	105	32928.0	4.58	8.73	77.2	0.558
B3	10.00	8.00	150	47040.0	3.91	7.43	57.2	0.693	695	217952.0	3.91	7.43	57.2	0.693	615	192864.0	3.91	7.43	57.2	0.693
B5	13.00	8.00	305	124342.4	5.76	9.83	77.7	0.438	365	148803.2	5.76	9.83	77.7	0.438	450	183456.0	5.76	9.83	77.7	0.438
A1	10.00	8.00		0.0					105	32928.0	7.20	5.59	89.8	0.912	310	97216.0	7.20	5.59	89.8	0.912
A3	10.00	8.00		0.0						0.0						0.0				
A7	10.00	8.00		0.0						0.0						0.0				
				0.0						0.0						0.0				
SUB TOTAL			1185	400,310	4.87	8.92	75.0	0.537	1360	460,835	4.83	8.25	68.8	0.608	1480	506,464	5.26	8.03	72.2	0.634
PROD TOT			4445	871,181	4.69	8.50	72.1	0.536	4505	910,538	4.86	8.00	71.6	0.619	4580	951,933	5.23	7.74	74.2	0.671
TOTAL ORE			5995	1,000,296	4.68	8.45	71.8	0.538	5655	1,006,333	4.99	7.83	73.0	0.646	5310	1,012,742	5.26	7.63	74.6	0.689
10% DILUTION			5995	1,000,296	4.26	7.68	65.3	0.49	5655	1,006,333	4.54	7.11	66.3	0.59	5310	1,012,742	4.78	6.94	67.9	0.63

CEMENTED ROCKFILL		YEAR 2		YEAR 3		YEAR 4	
		SG=2.35					
LOCATION		TONNES		TONNES		TONNES	
		522708.5		546322.6		571159.7	
		0.0		0.0		0.0	
SUB TOTAL		522,708		546,323		571,160	

FOX GEOLOGICAL CONSULTANTS LTD.

	YEAR 2		YEAR 3		YEAR 4
# TONNES/DAY ORE	2858.0	# TONNES/DAY ORE	2875.2	# TONNES/DAY ORE.	2893.5
# TONNES/DAY WASTE	158.7	# TONNES/DAY WASTE	101.8	# TONNES/DAY WASTE	76.2
# TONNES/DAY MINED	3016.7	# TONNES/DAY MINED	2977.0	# TONNES/DAY MINED	2969.7
# TONNES/DAY FILLED	1493.5	# TONNES/DAY FILLED	1560.9	# TONNES/DAY FILLED	1631.9
%DEVELOPMENT	17.5%	%DEVELOPMENT	12.6%	%DEVELOPMENT:	8.4%
%PRODUCTION	82.5%	%PRODUCTION	87.4%	%PRODUCTION	91.6%
%BENCH PRODUCTION	37.9%	%BENCH PRODUCTION	44.2%	%BENCH PRODUCTION	48.7%

CURRAGH RESOURCES INC
DY UNDERGROUND DEVELOPMENT/PRODUCTION SCHEDULE
[illegible] DYSCH2-6
[illegible] 19-Oct-92

DEVELOPMENT — YEAR 5

ORE DEVELOPMENT			YEAR 5	# DAYS= 350					TOTAL	# DAYS= 1400				
MINING					ASSAY GRADES						ASSAY GRADES			
BLOCK	HT (m)	W (m)	ADV (m)	TONNES	%Pb	%Zn	Ag(g/t)	Au(g/t)	ADV (m)	TONNES	%Pb	%Zn	Ag(g/t)	Au(g/t)
B1	4.25	5.00		0.0					180	14994	4.58	8.73	77.2	0.558
B2	4.25	5.00		0.0					130	10829	2.69	6.66	43.8	0.588
B3	4.25	5.00		0.0					525	43733	3.91	7.43	57.2	0.693
B4	4.25	5.00		0.0					630	52479	5.01	7.92	78.9	0.502
B5	4.25	5.00		0.0					310	25823	5.76	9.83	77.7	0.438
A1	4.25	5.00	90	7497.0	7.20	5.59	89.8	0.912	1060	88298	7.20	5.59	89.8	0.912
A2	4.25	5.00	145	12078.5	5.96	5.71	80.2	1.150	355	29572	5.96	5.71	80.2	1.150
A3	4.25	5.00	100	8330.0	4.66	6.50	96.2	1.201	320	26656	4.66	6.50	96.2	1.201
A4	4.25	5.00	60	4998.0	2.99	6.46	68.3	0.856	215	17910	2.99	6.46	68.3	0.856
A5	4.25	5.00	230	19159.0	5.61	5.85	80.1	0.925	330	27489	5.61	5.85	80.1	0.925
A6	4.25	5.00		0.0					0	0	0.00	0.00	0.0	0.000
	4.25	5.00		0.0					0	0	0.00	0.00	0.0	0.000
				0.0					0	0	0.00	0.00	0.0	0.000
SUB TOTAL			625	52,063	5.52	5.94	83.0	1.013	4055	337,782	5.40	6.84	78.7	0.799

WASTE DEVELOPMENT			YEAR 5						TOTAL					
HEADING	HT (m)	W (m)	ADV (m)	TONNES					ADV (m)	TONNES				
N5	5.00	4.25		0.0					80	3986				
N7	5.00	4.25		0.0					35	1744				
N11	5.00	4.25		0.0					90	4485				
N13	5.00	4.25		0.0					140	6976				
DECLINE	5.00	4.25		0.0					770	38370				
W1	5.00	4.25		0.0					120	5980				
W2	5.00	4.25		0.0					220	10963				
W3	5.00	4.25		0.0					60	2990				
W4	5.00	4.25		0.0					80	3986				
W5	5.00	4.25		0.0					270	13454				
W6	5.00	4.25		0.0					40	1993				
S1	5.00	4.25	330	16444.1					560	27905				
S2	5.00	4.25		0.0					230	11461				
S2 U/P RISE	2.40	2.40	105	1753.9					105	1754				
S4	5.00	4.25	70	3488.1					70	3488				
				0.0					0	0				
				0.0					0	0				
SUB TOTAL			505	21,686					2870	139,536				

PRODUCTION

STOPE DEVELOPMENT (DRIFT & SLASH)			YEAR 5						TOTAL					
MINING					ASSAY GRADES						ASSAY GRADES			
BLOCK	HT (m)	W (m)	ADV (m)	TONNES	%Pb	%Zn	Ag(g/t)	Au(g/t)	ADV (m)	TONNES	%Pb	%Zn	Ag(g/t)	Au(g/t)
B1	5.00	8.00		0.0					845	132496	4.58	8.73	77.2	0.558
B2	5.00	8.00	420	76832.0	2.69	6.66	43.8	0.588	1040	163072	2.69	6.66	43.8	0.588
B3	5.00	8.00	355	55664.0	3.91	7.43	57.2	0.693	2150	337120	3.91	7.43	57.2	0.693
B4	4.00	8.00	900	112896.0	5.01	7.92	78.9	0.502	4525	567616	5.01	7.92	78.9	0.502
B5	5.00	8.00	245	38416.0	5.76	9.83	77.7	0.438	1495	234416	5.76	9.83	77.7	0.438
A1	5.00	8.00	460	72128.0	7.20	5.59	89.8	0.912	1190	186592	7.20	5.59	89.8	0.912
A2	4.00	8.00	275	34496.0	5.96	5.71	80.2	1.150	515	64602	5.96	5.71	80.2	1.150
A3	5.00	8.00	140	21952.0	4.66	6.50	96.2	1.201	510	79968	4.66	6.50	96.2	1.201
A4	4.00	8.00	385	48294.4	2.99	6.46	68.3	0.856	485	60838	2.99	6.46	68.3	0.856
A5	5.00	8.00	250	39200.0	5.61	5.85	80.1	0.925	250	39200	5.61	5.85	80.1	0.925
				0.0					0	0	0.00	0.00	0.0	0.000
SUB TOTAL			3500	499,878	4.81	6.96	72.5	0.734	13005	1,865,920	4.86	7.56	73.2	0.654

STOPE PRODUCTION (BENCHING)			YEAR 5						TOTAL					
MINING					ASSAY GRADES						ASSAY GRADES			
BLOCK	HT (m)	W (m)	ADV (m)	TONNES	%Pb	%Zn	Ag(g/t)	Au(g/t)	ADV (m)	TONNES	%Pb	%Zn	Ag(g/t)	Au(g/t)
B1	10.00	8.00		0.0					1030	323008	4.58	8.73	77.2	0.558
B3	10.00	8.00	535	167776.0	3.91	7.43	57.2	0.693	1995	625632	3.91	7.43	57.2	0.693
B5	13.00	8.00	410	167148.8	5.76	9.83	77.7	0.438	1530	623750	5.76	9.83	77.7	0.438
A1	10.00	8.00	400	125440.0	7.20	5.59	89.8	0.912	815	255584	7.20	5.59	89.8	0.912
A5	10.00	8.00	125	39200.0	5.61	5.85	80.1	0.925	125	39200	5.61	5.85	80.1	0.925
A7	10.00	8.00		0.0					0	0	0.00	0.00	0.0	0.000
				0.0					0	0	0.00	0.00	0.0	0.000
SUB TOTAL			1470	499,565	5.49	7.65	74.0	0.681	5495	1,867,174	5.13	8.17	72.5	0.619
PROD TOT			4970	999,443	5.15	7.31	73.3	0.707	18500	3,733,094	4.99	7.86	72.8	0.637
TOTAL ORE			5595	1,051,506	5.17	7.24	73.7	0.722	22555	4,070,876	5.03	7.78	73.3	0.650
10% DILUTION			5595	1,051,506	4.7	6.58	67.0	0.66	22555	4,070,876	4.57	7.07	66.6	0.59

CEMENTED ROCKFILL		YEAR 5		TOTAL	
LOCATION		TONNES		TONNES	
		599665.9		2239856.6	
		0.0		0.0	
SUB TOTAL		599,666		2,239,857	

FOX GEOLOGICAL CONSULTANTS LTD.

YEAR 5

# TONNES/DAY ORE	3004.3
# TONNES/DAY WASTE	62.0
# TONNES/DAY MINED	3066.3
# TONNES/DAY FILLED	1713.3
%DEVELOPMENT	6.9%
%PRODUCTION	93.1%
%BENCH PRODUCTION	46.5%

TOTAL

# TONNES/DAY ORE	2907.8
# TONNES/DAY WASTE	99.7
# TONNES/DAY MINED	3007.4
# TONNES/DAY FILLED	1599.9
%DEVELOPMENT	11.3%
%PRODUCTION	88.7%
%BENCH PRODUCTION	44.3%

CURRAGH RESOURCES INC
DY UNDERGROUND DEVELOPMENT/PRODUCTION SCHEDULE
file DYSCH6-8
Date 19-Oct-92

DEVELOPMENT

YEAR 6 YEAR 7

ORE DEVELOPMENT			YEAR 6	# DAYS = 350					YEAR 7	# DAYS = 350				
MINING				SG = 3.92	ASSAY GRADES						ASSAY GRADES			
BLOCK	HT (m)	W (m)	ADV (m)	TONNES	%Pb	%Zn	Ag(g/t)	Au(g/t)	ADV (m)	TONNES	%Pb	%Zn	Ag(g/t)	Au(g/t)
A3	4.25	5.00	120	9996.0	4.66	6.50	96.2	1.201		0.0				
A5	4.25	5.00	475	39567.5	5.61	5.85	80.1	0.925	725	60392.5	5.61	5.85	80.1	0.925
A6	4.25	5.00		0.0						0.0				
A7	4.25	5.00		0.0						0.0				
A8	4.25	5.00		0.0					205	17076.5	4.26	5.04	61.6	1.098
				0.0						0.0				
				0.0						0.0				
SUB TOTAL			595	49,564	5.42	5.98	83.3	0.981	930	77,469	5.31	5.67	76.0	0.963

WASTE DEVELOPMENT			YEAR 6						YEAR 7					
				SG = 2.90	(5.0*4.25m ARCH = 17.183sq m)									
HEADING	HT (m)	W (m)	ADV (m)	TONNES					ADV (m)	TONNES				
S1	5.00	4.25	300	14949.2						0.0				
S2 U/P RSE	2.40	2.40	105	1753.9						0.0				
S4	5.00	4.25	165	8222.1					370	18437.4				
S5	5.00	4.25	60	2989.8						0.0				
				0.0						0.0				
				0.0						0.0				
SUB TOTAL			630	27,915					370	18,437				

NOTE WASTE DEVELOPMENT IS CONSIDERED CONSERVATIVE FOR MINING OF STEEP ORE IN SOUTHERN A ZONE.

PRODUCTION

STOPE DEVELOPMENT (DRIFT & SLASH)				YEAR 6					YEAR 7					
MINING				SG = 3.92	ASSAY GRADES						ASSAY GRADES			
BLOCK	HT (m)	W (m)	ADV (m)	TONNES	%Pb	%Zn	Ag(g/t)	Au(g/t)	ADV (m)	TONNES	%Pb	%Zn	Ag(g/t)	Au(g/t)
B1	5.00	8.00	850	133280.0	4.58	8.73	77.2	0.558	585	91728.0	4.58	8.73	77.2	0.558
B2	5.00	8.00	180	28224.0	2.69	6.66	43.8	0.588	90	14112.0	2.69	6.66	43.8	0.588
B3	5.00	8.00	40	6272.0	3.91	7.43	57.2	0.693		0.0				
A1	5.00	8.00	320	50176.0	7.20	5.59	89.8	0.912	510	79968.0	7.20	5.59	89.8	0.912
A2	4.00	8.00	240	30105.6	5.96	5.71	80.2	1.150	150	18816.0	5.96	5.71	80.2	1.150
A3	5.00	8.00	600	94080.0	4.66	6.50	96.2	1.201	845	132496.0	4.66	6.50	96.2	1.201
A4	4.00	8.00	635	79654.4	2.99	6.46	68.3	0.856		0.0				
A5	5.00	8.00	520	81536.0	5.61	5.85	80.1	0.925	810	127008.0	5.61	5.85	80.1	0.925
A6	4.00	8.00		0.0						0.0				
A7	5.00	8.00		0.0						0.0				
A8	5.00	8.00		0.0					320	50176.0	4.26	5.04	61.6	1.098
				0.0						0.0				
SUB TOTAL			3385	503,328	4.74	6.86	79.1	0.859	3310	514,304	5.23	6.43	82.4	0.944

STOPE PRODUCTION (BENCHING)				YEAR 6					YEAR 7					
MINING				SG = 3.92	ASSAY GRADES						ASSAY GRADES			
BLOCK	HT (m)	W (m)	ADV (m)	TONNES	%Pb	%Zn	Ag(g/t)	Au(g/t)	ADV (m)	TONNES	%Pb	%Zn	Ag(g/t)	Au(g/t)
B1	10.00	8.00	545	170912.0	4.58	8.73	77.2	0.558	720	225792.0	4.58	8.73	77.2	0.558
A1	10.00	8.00	565	177184.0	7.20	5.59	89.8	0.912	410	128576.0	7.20	5.59	89.8	0.912
A5	10.00	8.00	510	159936.0	5.61	5.85	80.1	0.925	530	166208.0	5.61	5.85	80.1	0.925
A7	10.00	8.00		0.0						0.0				
				0.0						0.0				
				0.0						0.0				
SUB TOTAL			1620	508,032	5.82	6.73	82.5	0.797	1660	520,576	5.56	7.03	81.2	0.763
PROD TOT			5005	1,011,360	5.28	6.79	80.8	0.828	4970	1,034,880	5.39	6.73	81.8	0.853
TOTAL ORE			5600	1,060,924	5.29	6.76	80.9	0.835	5900	1,112,349	5.39	6.66	81.4	0.861
10% DILUTION			5600	1,060,924	4.81	6.14	73.6	0.76	5900	1,112,349	4.9	6.05	74.0	0.78

CEMENTED ROCKFILL			YEAR 6		YEAR 7	
LOCATION			SG = 2.35 TONNES			TONNES
			606816.0			620928.0
			0.0			0.0
SUB TOTAL			606,816			620,928

FOX GEOLOGICAL CONSULTANTS LTD.

YEAR 6		YEAR 7	
# TONNES/DAY ORE:	3031.2	# TONNES/DAY ORE:	3178.1
# TONNES/DAY WASTE	79.8	# TONNES/DAY WASTE:	52.7
# TONNES/DAY MINED	3111.0	# TONNES/DAY MINED	3230.8
# TONNES/DAY FILLED	1733.8	# TONNES/DAY FILLED	1774.1
% DEVELOPMENT:	7.1%	% DEVELOPMENT:	8.5%
% PRODUCTION:	92.9%	% PRODUCTION:	91.5%
% BENCH PRODUCTION	46.7%	% BENCH PRODUCTION	46.0%

CURRAGH RESOURCES INC.
DY UNDERGROUND DEVELOPMENT/PRODUCTION SCHEDULE
file DYSCH6-8
Date 19-Oct-92

DEVELOPMENT

YEAR 8

ORE DEVELOPMENT			YEAR 8		# DAYS =			350	TOTAL		# DAYS =			1050
MINING					ASSAY GRADES						ASSAY GRADES			
BLOCK	HT (m)	W (m)	ADV (m)	TONNES	%Pb	%Zn	Ag(g/t)	Au(g/t)	ADV (m)	TONNES	%Pb	%Zn	Ag(g/t)	Au(g/t)
A3	4.25	5.00		0.0					120	9996	4.66	6.50	96.2	1.201
A5	4.25	5.00	285	23740.5	5.61	5.85	80.1	0.925	1485	123701	5.61	5.85	80.1	0.925
A6	4.25	5.00	230	19159.0	5.55	4.64	75.5	1.018	230	19159	5.55	4.64	75.5	1.018
A7	4.25	5.00	60	4998.0	3.53	6.61	62.4	0.911	60	4998	3.53	6.61	62.4	0.911
A8	4.25	5.00		0.0					205	17077	4.26	5.04	61.6	1.098
				0.0					0	0	0.00	0.00	0.0	0.000
				0.0					0	0	0.00	0.00	0.0	0.000
SUB TOTAL			575	47,898	5.37	5.45	76.4	0.961	2100	174,930	5.36	5.70	78.2	0.967

WASTE DEVELOPMENT			YEAR 8						TOTAL					
HEADING	HT (m)	W (m)	ADV (m)	TONNES					ADV (m)	TONNES				
S1	5.00	4.25	60	2989.8					360	17939				
S2 U/P RSE	2.40	2.40		0.0					105	1754				
S4	5.00	4.25	375	18686.5					910	45346				
S5	5.00	4.25		0.0					60	2990				
				0.0					0	0				
				0.0					0	0				
SUB TOTAL			435	21,676					1435	68,029				

NOTE: WASTE DEVELOPMENT IS CONSIDERED CONSERVATIVE FOR MINING OF STEEP ORE IN SOUTHERN A ZONE.

PRODUCTION

STOPE DEVELOPMENT (DRIFT & SLASH)					YEAR 8				TOTAL					
MINING					ASSAY GRADES						ASSAY GRADES			
BLOCK	HT (m)	W (m)	ADV (m)	TONNES	%Pb	%Zn	Ag(g/t)	Au(g/t)	ADV (m)	TONNES	%Pb	%Zn	Ag(g/t)	Au(g/t)
B1	5.00	8.00	165	25872.0	4.58	8.73	77.2	0.558	1600	250880	4.58	8.73	77.2	0.558
B2	5.00	8.00		0.0					270	42336	2.69	6.66	43.8	0.588
B3	5.00	8.00		0.0					40	6272	3.91	7.43	57.2	0.693
A1	5.00	8.00	575	90160.0	7.20	5.59	89.8	0.912	1405	220304	7.20	5.59	89.8	0.912
A2	4.00	8.00	155	19443.2	5.96	5.71	80.2	1.150	545	68365	5.96	5.71	80.2	1.150
A3	5.00	8.00	380	59584.0	4.66	6.50	96.2	1.201	1825	286160	4.66	6.50	96.2	1.201
A4	4.00	8.00		0.0					635	79654	2.99	6.46	68.3	0.856
A5	5.00	8.00	1105	173264.0	5.61	5.85	80.1	0.925	2435	381808	5.61	5.85	80.1	0.925
A6	4.00	8.00	190	23833.6	5.55	4.64	75.5	1.018	190	23834	5.55	4.64	75.5	1.018
A7	5.00	8.00	220	34496.0	3.53	6.61	62.4	0.911	220	34496	3.53	6.61	62.4	0.911
A8	5.00	8.00	720.0	112896.0	4.26	5.04	61.6	1.098	1040	163072	4.26	5.04	61.6	1.098
				0.0					0	0	0.00	0.00	0.0	0.000
SUB TOTAL			3510	539,549	5.32	5.84	78.2	0.983	10205	1,557,181	5.10	6.36	79.9	0.930

STOPE PRODUCTION (BENCHING)					YEAR 8				TOTAL					
MINING					ASSAY GRADES						ASSAY GRADES			
BLOCK	HT (m)	W (m)	ADV (m)	TONNES	%Pb	%Zn	Ag(g/t)	Au(g/t)	ADV (m)	TONNES	%Pb	%Zn	Ag(g/t)	Au(g/t)
B1	10.00	8.00	165	51744.0	4.58	8.73	77.2	0.558	1430	448448	4.58	8.73	77.2	0.558
A1	10.00	8.00	470	147392.0	7.20	5.59	89.8	0.912	1445	453152	7.20	5.59	89.8	0.912
A5	10.00	8.00	1055	330848.0	5.61	5.85	80.1	0.925	2095	656992	5.61	5.85	80.1	0.925
A7	10.00	8.00	100	31360.0	3.53	6.61	62.4	0.911	100	31360	3.53	6.61	62.4	0.911
				0.0					0	0	0.00	0.00	0.0	0.000
				0.0					0	0	0.00	0.00	0.0	0.000
SUB TOTAL			1790	561,344	5.82	6.09	81.4	0.887	5070	1,589,952	5.73	6.60	81.7	0.818
PROD TOT			5300	1,100,893	5.57	5.97	79.8	0.934	15275	3,147,133	5.42	6.48	80.8	0.873
TOTAL ORE			5875	1,148,790	5.56	5.94	79.7	0.935	17375	3,322,063	5.42	6.44	80.7	0.878
10% DILUTION			5875	1,148,790	5.06	5.4	72.4	0.85	17375	3,322,063	4.92	5.86	73.3	0.80

CEMENTED ROCKFILL		YEAR 8		TOTAL	
LOCATION		TONNES		TONNES	
		660535.7		1888279.7	
		0.0		0.0	
SUB TOTAL		660,536		1,888,280	

FOX GEOLOGICAL CONSULTANTS LTD.

YEAR 8		TOTAL	
# TONNES/DAY ORE	3282.3	# TONNES/DAY ORE:	3163.9
# TONNES/DAY WASTE	61.9	# TONNES/DAY WASTE:	64.8
# TONNES/DAY MINED	3344.2	# TONNES/DAY MINED:	3228.7
# TONNES/DAY FILLED	1887.2	# TONNES/DAY FILLED:	1798.4
% DEVELOPMENT	5.9%	% DEVELOPMENT:	7.2%
% PRODUCTION	94.1%	% PRODUCTION:	92.8%
% BENCH PRODUCTION	48.0%	% BENCH PRODUCTION:	46.9%

CURRAGH RESOURCES INC.
DY UNDERGROUND DEVELOPMENT/PRODUCTION SCHEDULE
file DYSC9-11
Date 19-Oct-92

DEVELOPMENT

YEAR 9 YEAR 10

ORE DEVELOPMENT				YEAR 9	# DAYS=	350				YEAR 10	# DAYS=	350		
MINING				SG=3.92	ASSAY GRADES						ASSAY GRADES			
BLOCK	HT (m)	W (m)	ADV (m)	TONNES	%Pb	%Zn	Ag(g/t)	Au(g/t)	ADV (m)	TONNES	%Pb	%Zn	Ag(g/t)	Au(g/t)
A3	4.25	5.00		0.0						0.0				
A5	4.25	5.00	275	22907.5	5.61	5.85	80.1	0.925		0.0				
A6	4.25	5.00	20	1666.0	5.55	4.64	75.5	1.018		0.0				
A7	4.25	5.00	235	19575.5	3.53	6.61	62.4	0.911		0.0				
A8	4.25	5.00		0.0						0.0				
				0.0						0.0				
				0.0						0.0				
SUB TOTAL			530	44,149	4.69	6.14	72.1	0.922	0	0	0.00	0.00	0.0	0.000

WASTE DEVELOPMENT				YEAR 9					YEAR 10					
				SG=2.90	(5.0*4.25m ARCH=17.183sq m)									
HEADING	HT (m)	W (m)	ADV (m)	TONNES					ADV (m)	TONNES				
S1	5.00	4.25	90	4484.8					90	4484.8				
S2 U/P RSE	2.40	2.40		0.0						0.0				
S4	5.00	4.25	90	4484.8					90	4484.8				
S5	5.00	4.25		0.0						0.0				
				0.0						0.0				
SUB TOTAL			180	8,970					180	8,970				

NOTE WASTE DEVELOPMENT IS CONSIDERED CONSERVATIVE FOR MINING OF STEEP ORE IN SOUTHERN A ZONE.

PRODUCTION

STOPE DEVELOPMENT (DRIFT & SLASH)					YEAR 9				YEAR 10					
MINING				SG=3.92	ASSAY GRADES						ASSAY GRADES			
BLOCK	HT (m)	W (m)	ADV (m)	TONNES	%Pb	%Zn	Ag(g/t)	Au(g/t)	ADV (m)	TONNES	%Pb	%Zn	Ag(g/t)	Au(g/t)
A1	5.00	8.00	690	108192.0	7.20	5.59	89.8	0.912	265	41552.0	7.20	5.59	89.8	0.912
A2	4.00	8.00	260	32614.4	5.96	5.71	80.2	1.150		0.0				
A5	5.00	8.00	1220	191296.0	5.61	5.85	80.1	0.925	1610	252448.0	5.61	5.85	80.1	0.925
A6	4.00	8.00	200	25088.0	5.55	4.64	75.5	1.018	290	36377.6	5.55	4.64	75.5	1.018
A7	5.00	8.00	225	35280.0	3.53	6.61	62.4	0.911	120	18816.0	3.53	6.61	62.4	0.911
A8	5.00	8.00	300	47040.0	4.26	5.04	61.6	1.098		0.0				
				0.0						0.0				
SUB TOTAL			2895	439,510	5.71	5.68	78.8	0.961	2285	349,194	5.68	5.73	79.8	0.932

STOPE PRODUCTION (BENCHING)					YEAR 9				YEAR 10					
MINING				SG=3.92	ASSAY GRADES						ASSAY GRADES			
BLOCK	HT (m)	W (m)	ADV (m)	TONNES	%Pb	%Zn	Ag(g/t)	Au(g/t)	ADV (m)	TONNES	%Pb	%Zn	Ag(g/t)	Au(g/t)
A1	10.00	8.00	690	216384.0	7.20	5.59	89.8	0.912	480	150528.0	7.20	5.59	89.8	0.912
A5	10.00	8.00	1085	340256.0	5.61	5.85	80.1	0.925	1545	484512.0	5.61	5.85	80.1	0.925
A7	10.00	8.00	255	79968.0	3.53	6.61	62.4	0.911	185	58016.0	3.53	6.61	62.4	0.911
				0.0						0.0				
SUB TOTAL			2030	636,608	5.89	5.86	81.2	0.919	2210	693,056	5.78	5.86	80.7	0.921
PROD TOT			4925	1,076,118	5.82	5.79	80.2	0.936	4495	1,042,250	5.75	5.82	80.4	0.925
TOTAL ORE			5455	1,120,267	5.77	5.80	79.9	0.936	4495	1,042,250	5.75	5.82	80.4	0.925
10% DILUTION			5455	1,120,267	5.25	5.27	72.6	0.85	4495	1,042,250	5.23	5.29	73.1	0.84

CEMENTED ROCKFILL				YEAR 9			YEAR 10	
LOCATION				SG=2.36 TONNES			TONNES	
				645671.0			625349.8	
				0.0			0.0	
SUB TOTAL				645,671			625,350	

FOX GEOLOGICAL CONSULTANTS LTD.

YEAR 9		YEAR 10	
#TONNES/DAY ORE:	3200.8	#TONNES/DAY ORE:	2977.9
#TONNES/DAY WASTE:	25.6	#TONNES/DAY WASTE:	25.6
#TONNES/DAY MINED:	3226.4	#TONNES/DAY MINED:	3003.5
#TONNES/DAY FILLED:	1844.8	#TONNES/DAY FILLED:	1786.7
%DEVELOPMENT:	4.7%	%DEVELOPMENT:	0.9%
%PRODUCTION:	95.3%	%PRODUCTION:	99.1%
%BENCH PRODUCTION:	56.4%	%BENCH PRODUCTION:	65.9%