PITEAU ASSOCIATES
GEOTECHNICAL AND
HYDROGEOLOGICAL CONSULTANTS

215 - 260 WEST ESPLANADE
NORTH VANCOUVER, B.C.
CANADA V7M 3G7
TELEPHONE (604) 986-8551
FAX (604) 985-7286

ANVIL RANGE MINING COMPANY

CHAPTERS 6 AND 7 FOR

GRIZZLY PROJECT UNDERGROUND

PRE-FEASIBILITY STUDY

004918

PROJECT 1617 **NOVEMBER 1996**

PITEAU ASSOCIATES
GEOTECHNICAL AND
HYDROGEOLOGICAL CONSULTANTS

215 - 260 WEST ESPLANADE
NORTH VANCOUVER, B.C.
CANADA V7M 3G7
TELEPHONE (604) 986-8551
FAX (604) 985-7286

Our file: 1617

November 14, 1996

Mr. Fritz Prugger
Project Manager
Anvil Range Mining Corporation
117 Industrial Road
Whitehorse, Y.T.
Y1A 2T8

Dear Fritz:

Re: Draft of Chapters 6 and 7 on Geology, Ore Reserves and Geotechnical Considerations for the Anvil Range - Grizzly Pre-Feasibility Study Report

Enclosed is one (1) copy of our text, figures and tables for Chapters 6 and 7 of the Grizzly pre-feasibility report, which were also sent to Warren Yaaw at Proton International Engineering Corporation by E-mail and hard copy. Please note revisions to the titles of Tables 6.7 and 6.12 and the references to these tables in the text.

I trust the enclosed information is adequate for your present needs. Once you have finalized your report, we would be happy to review our contribution. If you have any questions, please contact me.

Yours very truly,

PITEAU ASSOCIATES ENGINEERING LTD.

Nick D. Rose, P.Eng.

NDR/ef

Enc.

CONTENTS

LIST OF FIGURES

LIST OF TABLES

6. GEOLOGY AND ORE RESERVES

6.1 REGIONAL GEOLOGY

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

6.2 DEPOSIT GEOLOGY

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

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

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

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

6.2.1 Structural Geology

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

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

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

6.2.2 Faults

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

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):

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

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

H = pillar height

Pillar stress is calculated by:

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

where:

σ_v = vertical stress field

σ_h = horizontal stress field

α = dip of ore (degrees)

e = extraction ratio

The Factor of Safety equals:

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

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

7.3.1 Proposed Mining of Thin Ore

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

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

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

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

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

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

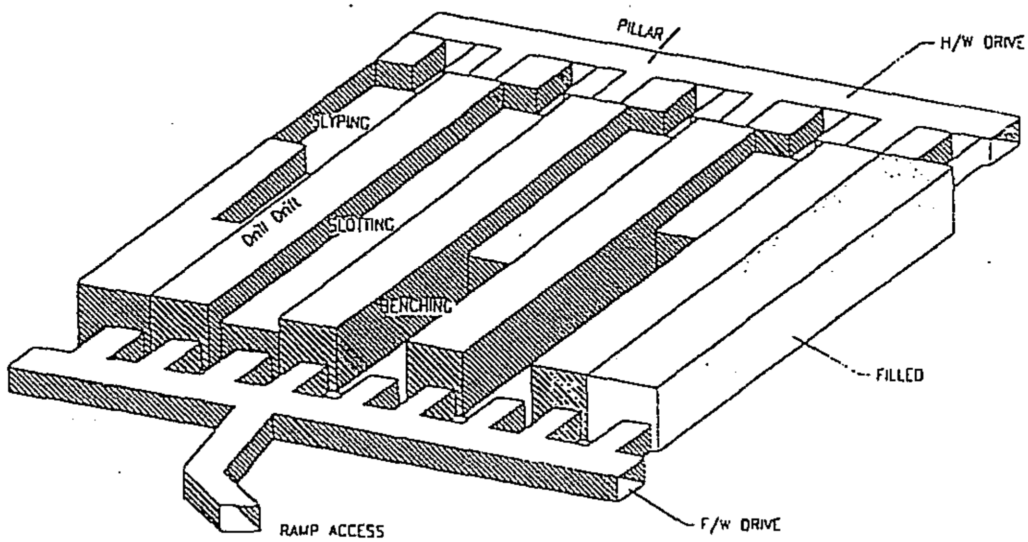

Figure 7.3-02 - Concrete Pillar Mining

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 6

TABLES