REPORT 60610

PRELIMINARY EVALUATION OF UNDERGROUND MINING AT THE FARO PIT

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Prepared for:

CURRAGH RESOURCES
117 Industrial Road
Whitehorse, Yukon Y1A 2T8

Prepared by:

STEFFEN ROBERTSON AND KIRSTEN (B.C.) INC.
801 - 1030 West Georgia Street
Vancouver, B.C. V6E 2Y3

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1.0 INTRODUCTION

The Curragh operations at Faro extract lead and zinc ore by open pit methods. The Faro pit is reaching the end of its economic life and will be completed by the third quarter of 1991. There is a possibility that extraction can be continued by following the ore underground from an access in the pit hanging wall. The ore thickness varies from a lower mineable limit of 7 feet up to a maximum thickness of approximately 30 feet. The dip is much too flat for gravity methods and therefore a room and pillar layout has been proposed.

Steffen Robertson and Kirsten have been engaged to carry out a preliminary evaluation of the geotechnical aspects of the mining method. In particular, pillar strength and room stability. It is anticipated that the geotechnical analysis will provide sufficient data for an updated cost and feasibility analysis prior to accessing the ore with a decline.

Room and pillar methods are very sensitive to the following:

- geometry; mechanized methods are limited to a maximum operating gradient and sudden changes in geometry are difficult to follow
- ore grade distribution; it is difficult to change the mining height and the pillar layout over short distances and the degree of flexibility is limited
- rock mass quality; determination of pillar size and roof support as operating costs are sensitive to the amount of roof support necessary and the practical maximum room sizes, whilst the percentage extraction will be sensitive to the pillar sizes

Room and pillar is unique to the extent that all extracted areas must remain relatively stable for the life of the mine unless regional pillars demarcate extracted areas. Ultimate pillar loads are only reached once large areas have been extracted. If the pillars are then found to be undersized it is difficult to strengthen and preserve their integrity without prior knowledge. Correct design is essential. It is not a method which can be easily adjusted to historical conditions.

The data available for this present study was limited to 11 boreholes as illustrated in Figure 1. Considerable use was made of a recent geological interpretation carried out by R. Morris (Beacon et al., 1988).

The recent core drilling had not been sampled or analyzed prior to the completion of this preliminary geotechnical study and therefore the ore intercepts were estimated. It will be noted from the spread of the core intersections that coverage of the deposit was limited. Core, on its own, is insufficient for the development of a detailed layout. The rock mass quality tends to be down-graded by the coring process and pessimistic values can be derived. On the other hand, unless very close spaced drilling is used the core intersections can result in an optimistic interpretation of the variation in geometry and ore grade distribution.

The results presented in this report are adequate for a detailed revision of the existing cost study but not for a detailed layout. That will require underground exposure and limited test mining if the percentage extraction is to be optimized.

The geotechnical study is based on the following steps:

- geotechnical environment:
  - description of the footwall, ore and immediate hanging wall horizons
  - geotechnical logging of the core with results presented in the form of rock mass classification values
  - index strength testing with confirmatory laboratory results
  - geometry in terms of footwall contours and core intersections relative to existing interpretation
- rock mass analysis:
  - support requirements at various room spans based on empirical relationships
  - pillar stability using intact strengths and rock mass quality values with empirical strength estimators and stress relationships for pillar loads
- mining analysis
  - comparison of layout options and extraction procedures relative to potential geometry
  - estimate of support costs.

The first subject which is addressed is the geotechnical environment.

2.0 GEOTECHNICAL ENVIRONMENT AND DATA GATHERING

The geotechnical data must be put into a form which can be used for evaluating room stability and pillar strength.

2.1 Geology

The faro deposit is hosted by late Precambrian to upper Paleozoic metasedimentary and metavolcanic rocks, intruded by a Cretaceous granitic plutonic suite.

The district has a complex deformational and metamorphic history. Two overlapping Mesozoic regional, metamorphic and folding events ranging from greenschist to amphibolite as well as subsequent non-penetrative folding and faulting have been recognized.

The underground development will extend from the SW corner of the pit. A minimum thickness of 7 ft has been used to outline the mineable areas. Geological interpretation in 1988 has identified three areas, Drawing 1. The areas are based on thickness of ore intercept and they include; the Northwest area containing three structural blocks with an average thickness of 13 ft, the Corridor area containing one structural block with an average thickness of 13 ft and the Southwest area containing a single structural block with numerous tight folds and an average thickness of 18 ft.

Figure 2 illustrates a generalized and simplified sequence through the hanging wall, mineralized and footwall rocks. This sequence shows the interval over which the data was gathered, about 60 ft. Unit 1D4, the white mica envelope (wme), will control roof stability when it occurs within the hanging wall.

Hanging wall rocks consist of schist/phyllite, the white mica envelope, graphitic quartzite, pyrite free quartzite and pyritic quartzite. The ore is found in massive sulfides which contain pyritic facies, buckshot facies, baritic facies and pyrrhotitic facies. Footwall rocks contain graphitic quartzite, the white mica envelope, and schist/phyllite.

2.2 Borehole Logging

A drilling program was performed in September and October 1988 to fill in gaps in existing borehole information. Boreholes 88F-01 to 88F-16 were available for geotechnical logging by SRK. The object was to log about 60 ft of each borehole including a 20 ft ore grade intersection in sulfides and 20 ft into the hanging wall and footwall rocks above and below. Boreholes 88F-01 and 88F-02 had already been logged for geology by Curragh. Assays were not available to confirm the ore intercept(s). Geological and assay data collection will be completed by Curragh in early 1989.

Geology and ore grade intercepts in boreholes 88F-03 to 88F-16 are based on interpretation by SRK. Holes 88F-06, 88F-09 and 88F-12 were stopped short of the sulfide zone because of difficult drilling. Boreholes 88F-03 and 88F-08 did not intersect ore grade sulfides of any appreciable thickness (less than 4 in.). In the remaining holes as many as four ore grade intersections were observed ranging in thickness from 2 to about 40 ft. The new and old boreholes are indicated in Drawing 1. Each borehole trace and ore grade intercept is shown in Figures 3 through 11 relative to the ore and low grade envelopes as presently interpreted.

2.3 Pit Mapping

The bottom of the current pit in the SW corner is close to the sulfide band. From here the band dips SW with the foliation. At this location hanging wall rock is exposed. Bench mapping was performed here. The white mica envelope was not visible so the sulfide band was assumed to be below the exposed rock on the bench face. Structural orientations were obtained on this bench along a 20 m exposure. The bench above was inaccessible and with poorly exposed structures. Other areas in the pit were considered too distant from the underground mine area. Previous pit mapping results were also incorporated.

2.4 Testing, Rock Strength

Point load index tests were performed as part of the core logging. Competent core samples were taken for unconfined compressive strength tests (UCS). The test results are presented in Table 1. Footwall and hanging wall rocks containing the dominant (S2) foliation have anisotropic strength that is weak parallel to the foliation. Where the foliation is not as pervasive, as in the graphitic quartzite (2A0), the rock is much stronger. The white mica envelope is very weak on the foliation. Most footwall and hanging wall core observed was broken on the foliation into short pieces. Much of the breakage, although probably drill induced, reflects the importance of this structure. The footwall and hanging wall rocks are much stronger normal to the foliation. The massive sulfides have moderate to high strength except when intersected by joints. Some of the joints are healed and were not obvious in the core.

2.5 Rock Mass Classification, MRMR

The Mining Rock Mass Rating, MRMR, logging system used by SRK makes estimates of intact rock strength, fracture frequency and fracture condition. The classes used, relative to the classification numbers, are shown below:

Class	1		2		3		4		5	
	A	B	A	B	A	B	A	B	A	B
MRMR	100-81		80-61		60-41		40-21		20-0	
Description	Very good		Good		Fair		Poor		Very poor	

The logging approach is shown in Table 2. Logged borehole intervals are shown in Figures 3 to 11. The phyllites receive MRMR ratings typically between 30 and 40 with the white mica envelope rating 10 to 20. The graphitic quartzites rate values between 40 to 50. The massive sulfides lack foliation but their higher rating values of 50 to 65 do not account for the hidden, healed fractures revealed by the point load testing. Broken or fractured sulfide zones rated at 30 to 50.

TABLE 1
TEST RESULTS

DDH	DEPTH (ft)	UNIT	FW AND HW ROCK PI* (MPa)	FW AND HW ROCK UCS (MPa)	SULFIDE PI* (MPa)	SULFIDE UCS (MPa)	COMMENT
F-01	51	2A0		144			Brittle Failure
	61	2A0	14				Failure on S2
	66	2F4			59		Failure on Joint
	69	2F4				98	Brittle Failure
	72	2F4			76		Failure on Joint
	76.5	2H4				167	Brittle Failure
	78.5	2E4			284		Failure at Platten Contact, Core Intact
	82.5	2C24	106				Failure on S2, Load // to Strike
F-04	308.5	2A0	83				Failure on S2, Load // to Strike
	310	2DH4				80	Brittle Failure
	310.5	2DH4			201		Brittle Failure
	329	2F4				102	Brittle Failure
	330.5	2H4			53		Failure on Joint
	333.5	2H4			175		Brittle Failure
F-05	183	N/A	133				Brittle Failure
	186	2A0	311				Brittle Failure on S2, S2 Normal to Core
	189	2A0	226				Brittle Failure on S2, S2 Normal to Core
	200	2H4			183		Brittle Failure
F-07	350.5	1C4		18.3			Failure on S2
	357	2H4			175		Brittle Failure
	358.5	2H4				101	Brittle Failure
	361	2H4			114		Brittle Failure
	367	2H4			187		Failure at Platten Contact, Core Intact
	371	2H4			47		Soft Break, Weak Matrix
	376.5	2H4			163		Brittle Failure
	382.5	2F4			59		Brittle Failure
F-10	322	2EH4			218		Brittle Failure
	326	2A4	201				Brittle Failure
	331	2A4	154				Brittle Failure
	337	2EF4			140		Brittle Failure

TABLE 1 cont'd.
TEST RESULTS

			FW AND HW ROCK		SULFIDE		
DDH	DEPTH (ft)	UNIT	PI* (MPa)	UCS (MPa)	PI* (MPa)	UCS (MPa)	COMMENT
F-11	400	2A0	47				Brittle Failure
	406	2A0		35			Brittle Failure
	406	2A0	70				Brittle Failure
	413	2A0	58				Brittle Failure on S2
	418	2EFH04			92		Brittle Failure
	421	2EFH04			88		No Failure, Plattens Penetrate Core
	426	2EFH04			135		Brittle Failure
F-13	270.5	2A4	33				Soft Break on S2
	272	2A4	83				Brittle on Joint
	285	2EFG4			194		Failure at Platten Contact, Core Intact
F-14	336.5	2A04	92				Brittle Failure
	341.5	2A04		193			Brittle Failure
	345.5	2F4			168		Failure at Platten Contact, S2 Vert in Core
	349	2A0	145				Failure at Platten Contact, Core Intact
	352	2F4			166		Failure at Platten Contact, Core Intact
	362	2A0	294				Brittle Failure, 98% Quartz
	376	2FH4				175	Brittle Failure
F-15	409	1C0		8			Failure on S2
	418.5	2A04	86				Brittle Failure on S2
	422	2A04	72				Brittle Failure
	424	2A04	141				Brittle Failure
	434	2F4			131		Brittle Failure
	436	2F4			82		Brittle Failure
	436	2F4			79		Brittle Failure
	441	2F4			95		Brittle Failure
F-16	680	2A0	57				Brittle Failure on S2
	684	2A0	0				Fell Apart on S2
	689	2A0		133			Brittle Failure
	693	2EFH4			21		Failure on Joint
	698	2EFH4			26		Failure on Joint
	701.5	2EFH4				66	Failure on Joint
	701.5	2EFH4				56	Brittle Failure, on Joint?
	705.5	2CO	144				Brittle Failure
	711.5	2A0	24				Brittle Failure on S2

*PI's are corrected values. Maximum value of 200 MPa used for determining roof an pillar support.
Unit, see Figure 2 for geologic legend.
UCS testing by Golder Associates.

B. BASIS OF THE CLASSIFICATION

1	IRS (MPa)	185	184-165	164-145	144-125	124-105	104-85	84-65	64-45	44-25	24-5	4-0
	RATING (= 0.1 x MPa)	20	18	16	14	12	10	8	6	4	2	0

2	FF/m **	0.2	0.3	0.4	0.5	0.7	1	1.5	2	3	5	7	10	15	20	30	40	60	80
	RATING 3 JOINT SETS	40	40	38	36	33	31	28	26	23	20	17	15	12	10	7	5	2	0
	2 JOINT SETS	40	40	40	38	36	33	31	28	26	23	20	17	15	12	10	7	5	2
	1 JOINT SET	40	40	40	40	38	36	33	31	28	26	23	20	17	15	12	10	7	4

3	JOINT CONDITION INCLUDING GRND.WATER	REFER C. BELOW
	RATING ($40 \times A \times B \times C \times D / 10^8$)	40 ———— 0

A. MEANING OF THE RATINGS

Class	1 A	1 B	2 A	2 B	3 A	3 B	4 A	4 B	5 A	5 B
Rating *	100 – 81		80 – 61		60 – 41		40 – 21		20 – 0	
Description	Very good		Good		Fair		Poor		Very poor	

Notes : * Sum of 1, 2, 3 of B.
** Measured fracture frequency divided by core recovery x 100

C. ASSESSMENT OF JOINT CONDITIONS

ACCUMULATIVE % ADJUSTMENT OF POSSIBLE RATING OF 40

Parameter	Description		Dry Cond	Wet Conditions: Moist	Mod Pressure 25 – 125 L/m	Sev Pressure >125 L/m
A Joint Expression (large scale irregularities)	Wavy	Multi-directional	100	100	85	90
		Uni-directional	90	90	90 85	80 75
	Curved		89 80	85 75	80 70	70 60
	Straight		79 70	74 65	60	55
B Joint Expression (small scale irregularities or roughness)	Very rough		100	100	85	80
	Striated or rough		89 85	99 85	80	70
	Smooth		84 60	80 55	60	50
	Polished		59 40	50 35	30	20
C Joint Wall Alteration Zone	Stronger than wall rock		100	100	100	100
	No alteration		100	100	100	100
	Weaker than wall rock		75	70	65	60

C. ASSESSMENT OF JOINT CONDITIONS (continued)

ACCUMULATIVE % ADJUSTMENT OF POSSIBLE RATING OF 40

Parameter	Description		Dry Cond	Wet Conditions: Moist	Mod Pressure 25 – 125 L/m	Sev Pressure >125 L/m
D Joint Filling	No fill – Surface Staining only		100	100	100	100
	Non Softening & sheared material (clay or talc free	Coarse Sheared	95	90	70	50
		Medium sheared	90	85	65	45
		Fine sheared	85	80	60	40
	Soft Sheared Material (eg Talc)	Coarse Sheared	60	55	40	20
		Medium Sheared	65	50	35	15
		Fine Sheared	50	45	30	10
	Gouge thickness <amplitude of irreg		40	30	10	
	Gouge thickness >amplitude of irreg		20	10	Flowing material 5	

Table 2 Modified Geomechanics Rock Mass Classification

2.6 Joint and Fault Orientation

Interpretations were made from a small joint and fault data base. Joint measurements by SRK in the southwest corner of the pit were combined with data already identified from the same area, Figure 12. Two prominent joint sets, A and B, are identified in Figure 13. The joint sets dip 078/90 and 168/50, respectively. A third joint set, C, dipping 005/90 is less frequent. Faults, as presently identified, appear to have a wide scatter of poles, Figure 14. Fault sets are shown in Figure 15. Fault sets A and B are the same as joint sets A and B with dips of 268/87 and 163/52, respectively. Fault sets C and D compliment joint set C with dips of 012/90 and 030/74. A fifth fault set, E, dips 138/84 and has no complimentary joint set.

Use of data from Piteau et al, 1975 was not attempted because their structural data was collected closer to the ground surface along the east wall of the original pit too far from the sulfide zone for current interests.

2.7 Geometry

The site is characterized by a dominant foliation (S2) dipping S in the Northwest area and SW in the Southwest area. The foliation is very apparent in the schists and phyllites but not in the sulfide horizon. The white mica envelope which surrounds the massive sulfide horizon follows foliation. The white mica envelope appears in the hanging wall in boreholes 88F-02, 88F-07 and 88F-10. In the remaining holes the white mica envelope is absent or above the hanging wall. The hanging wall is considered as the 10 ft. immediately above the ore, and similarly the footwall 10 ft. below. The sulfide footwall contours and foliation contours are probably sympathetic. Orebody footwall contours, dip directions, dips and slopes are indicated in Drawing 1. The dip and slope vary between 14°-42° and 25%-91% on short slope lengths. The dip direction changes from southerly in the Northwest area to westerly in the Southeast area. Average dip and slope in the Southwest area vary between 16°-22° and 29%-40%. The dips indicated in Figures 3 through 11 are apparent dips as the sections are not parallel to dip direction.

The Northwest area contains three structural blocks with an average thickness of 13 ft. The throw between blocks A and B along fault NW2 is about 30 ft. The throw between blocks B and C along fault NW1 is about 35 ft. In block C a 5 ft. throw is indicated across fault NW3. The Corridor area contains one structural block with an average thickness of 13 ft. A throw of 15 ft is indiicated across fault Big Gulp. Recent interpretation suggests that the Southwest area contains a single structural block with numerous tight folds and an average thickness of 18 ft.

The foliation dip varied 90 degrees, from horizontal to vertical, within the logged intervals which is indicative of folding. Intercalations of the hanging wall, ore and footwall rocks were observed in boreholes 88F-10, 88F-11 and 88F-14. Over thickening was observed in boreholes 88F-04, 88F-07 and 88F-14. Intercalations and over thickening can be caused by folding. Ore intersections from recent drilling and plotted on Figures 3 to 11 do not coincide with the present interpretation. The ore and low grade envelope horizons shown in Figures 3 to 11 present a simpler geometry than probably exists.

3.0 ROCK MASS ANALYSIS

Room and pillar extraction, as the name implies, is a function of maximizing the room size whilst minimizing the pillar width.

The assumption has been made that the massive sulfide represents ore and that the hanging wall is always waste material either schist/phyllite or quartzite. It is noted from the recent geological interpretation that the 9% combined metal portion is sometimes within the massive zone as a grade cut-off. The case for massive sulphide as a roof material has not been considered.

The existing mining feasibility assessment assumes 30 foot rooms with 30 foot pillars and an approximate percentage extraction of 75%. The current geotechnical assessment has used this level of extraction to compare two different extraction geometries.

- 10 m rooms with 10 m pillars
- 5 m rooms with 5 m pillars

The previous section showed that there is a considerable range of rock mass quality and strength both in the roof material and in the orebody. A certain level of rock support will be necessary in some areas whatever the spans and extractions ratio. Room and pillar is a compromise between the practical advantages of large excavations and the economic impact of the rock support necessary to maintain stability. Large pillar widths mean strong pillars. If the spans are reduced to decrease the roof support then pillar width must also be reduced and support of a few of the weaker pillars may be required if the overall extraction ratio is to be maintained.

The two room and pillar layouts were compared to illustrate the difference in rock support that might be necessary and, finally, the cost per tonne of ore for the rock support.

Two empirical design methods were used for room design and pillar strength, as follows:

- span and support using the 'Q' system as illustrated in Figure 16. 'Q' rock mass values were estimated from the measured MRMR values using the relationship:

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

  The support values, although approximate, give a measure of the support intensity that may be necessary. A Modification Factor of three was used in the analysis, equivalent to a tunnel intersection within 'temporary mine' openings.

- pillar strength using the following relationship:

$$\text{pillar strength (in MPa)} = K \frac{(\text{pillar width . m})^{0.5}}{(\text{pillar height . m})^{0.75}}$$

where K is the rock mass strength which in turn is derived from:

$$\text{rock mass strength} = \text{intact strength} \frac{(\text{MRMR value - intact strength rating})}{100}$$

Each of the 11 drill hole intersections was independently evaluated for roof span and pillar strength as shown in Table 3.

It will be noted that the strength of long narrow pillars was also considered. The pillar size of 5.6 m by 22.4 m is equivalent to 75% extraction using 10 m rooms with 10 m wide pillar break-through every 32.4 m; that is pillars with a width to length ratio of 1:4. The object of considering this third form of pillar was to take into account the steep dips and the possibility that there will have to be a predominant mining direction. The pillar strength per unit area of a long pillar is greater than for a square pillar of the same width. This is because the long pillar has less face area per unit volume; in other words there is less freedom for the long pillar to deteriorate. This difference in shape can be taken into account by using the following relationship:

- $$\text{'effective' pillar width} = 4 \frac{(\text{pillar plan area})}{(\text{pillar plan perimeter})}$$

This was applied to the 'long' pillars to estimate pillar strength in Table 3.

A conventional approach to strength and stability would be to average the values. This might be justified in a predictable environment and where a large body of data is available. But it is evident that considerable spread exists and there are very few data points. Analysis of each core intersection as an independent portion of the deposit emphasizes the variability. Although the overall average might be satisfactory there may be significant areas that do not meet design parameters and require rock reinforcement.

3.1 Roof Stability

As stated previously, it is extremely difficult to accurately define roof stability and support requirements from a few core intersections. The rock when exposed in an underground excavation can behave in a markedly different manner. The 'Q' empirical method has been limited to identifying potential support difficulty using the following categories;

- no patterned support required, other than precautionary

TABLE 3

ROOM AND PILLAR DESIGN VALUES

Hole Number	Ore Rock Mass Strength	Roof Rock Mass Quality 'Q' values	Pillar Ht. Ore Vertical thickness m	Pillar Strength (MPa) Pillar Width 10 m	Pillar Width 5 m	Roof Support Difficulty Room Width 10 m	Room Width 5 m	Pillar Strength Long Pillars (5.6 m 22.4 m)
F-01	61	1	6.1	49.7	35.1	bolting	no support	47
F-02	40	0.04	5.2	36.8	26	intense	bolting +	34.9
F-04	46	3	11.3	23.6	16.7	bolting	no support	22.4
F-05	78	0.9	2.4	126	89	bolting	no support	120
F-07	52	0.05	10.4	28.5	20	intense	bolting +	27
F-10	76	0.05	2.4	123	87	intense	bolting +	117
F-11	26	30	4.6	26.3	18.6	bolting	25	
F-13	44	20	4.6	26.3	18.6	bolting +	bolting +	42.1
F-14	75	50	10.4	41.1	29	no support	no support	39
F-15	31	22	6.7	23.5	16.6	bolting +	bolting +	22.3
F-16	6	1	3.7	7.2	5.1	intense	bolting +	6.9

+ = plus mesh or strap support between bolts

intense = bolts plus mesh or strap support plus shotcrete

- patterned bolting required, one bolt per two square metres
- patterned bolting plus support between the bolts in the form of mesh or straps
- patterned bolting plus shotcrete plus support between the bolts in the form of mesh, or the use of fibre-crete

These four categories have been identified as "no support", "bolting", "bolting +" and "intense" in Table 3.

The results in Table 3 can be summarized as follows:

Support Difficulty	5 m Room	10 m Room
∘ no support	36.5%	9.1%
∘ bolting	9.1%	36.4%
∘ bolting +	54.4%	18.1%
∘ intense	0%	36.4%

The difference in support requirements between 5 m and 10 m rooms is marked.

There is a further factor concerning roof stability that should be recognized. The immediate hanging wall material is strongly foliated. The steep jointing coupled with the foliation will result in unstable wedges being formed unless the roof plane follows the foliation plane. the practicality of following the foliation will depend on the short range variation in dip. There is some evidence that this short range variation could be significant in which case roof conditions could be worse than have been anticipated.

3.2 Support Costs

A direct cost can be put against the support difficulty based on the following very approximate costs derived from contractual quotes:

- Support Cost:
  - bolting \$22.50/m^2 of roof
  - bolting plus mesh or straps \$50.5/m^2 of roof
  - bolting plus shotcrete and mesh \$113/m^2 of roof

The average orebody thickness is approximately 6 m which, together with a 3.9 S.G. for the ore, translates to 24 tonnes of ore per one square metre of exposed roof.

Roof support costs per tonne of ore become:

- bolting = \$0.94/tonne
- bolting plus mesh and steps = \$2.1/tonne
- bolting plus mesh and shotcrete = \$4.71/tonne

The cumulative support costs for 10 m and 5 m rooms are as follows:

5 m rooms	-	36.5%	at	\$0
	-	9.1%	at	\$0.94/tonne
	-	54.5%	at	\$2.10/tonne
	-	0%	at	\$4.71/tonne
		Average cost		\$1.23/tonne of ore
10 m rooms	-	9.1%	at	\$0/tonne
	-	36.4%	at	\$0.94/tonne
	-	18.2%	at	\$2.10/tonne
	-	36.4%	at	\$4.71/tonne
		Average cost		\$2.44/tonne of ore

Obviously the 5 m rooms cost substantially less but the disadvantage is that pillars may be less stable.

3.3 Pillar Stability

Pillar stability is a function of the pillar strength, the load and the appropriate factor of safety. The pillar loads are best calculated by using numerical analysis techniques. The alternative is to use "tributary area" theory which is much more conservative. This option has been used for the preliminary evaluation. Overburden load varies across the deposit dependent on dip, depth and the proximity of the open pit. The deposit was divided approximately into two halves with average depths of 128 m and 213 m respectively. The overburden stress would be 3.2 and 5.33 MPa at these depths and the pillar stresses would be 12.8 and 21.3 MPa at 75% extraction.

An appropriate factor of safety for pillared workings is 1.5, however values down to 1.3 have been used in Elliot Lake but after many years of experience.

The pillar strengths shown in Table 3 can be viewed as representing a spread of conditions through the deposit. The pillar strength necessary to accept the overburden load can be evaluated in terms of the factor of safety as follows:

Average Depth Below Surface	FOS	10 m Pillar Width	5 m Pillar Width
128 m	+1.5	91%	73%
	1.0-1.5	0%	18%
	<1.0	9%	9%
213 m	+1.5	55%	36%
	1.0-1.5	36%	18%
	<1.0	9%	45%

The average strength is sufficient for total load but certain pillars may show signs of distress. The significance of the FOS lies in the condition of the pillar walls and is an indication of the amount of pillar support that may be necessary.

To reiterate, the 'tributary area' approach is very conservative and does not take into account the load that would be shed into the abutments and the load shed by the weak pillars into short strong pillars. But without a detailed layout the use of numerical analysis is not justified.

One can conclude that the 10 m pillar layout would be reasonably stable with only a small proportion of pillar walls requiring support. The 5 m pillar layout would be far less table with 63% of the pillar walls requiring some degree of rock support and 45% requiring fairly intensive support.

The problem with the 5 m pillars is that there are a number of areas where the pillar height is significantly greater than the pillar width. This not only reduces the pillar strength but greatly increases the probability of structures 'daylighting' on both sides of a pillar. Tall thin pillars can very easily fall apart due to structure regardless of stress.

The degree of support difficulty for the pillar walls has been evaluated as follows:

- FOS greater than 1.5; no support necessary
- FOS between 1.0 and 1.5; bolting and straps
- FOS less than 1.0; bolting, mesh and shotcrete

3.4 Pillar Support Costs

The support costs per square metre of wall will be similar to those for roof support. The tonnes of ore per square metre of wall will be 29 tonnes for the 10 m pillars and 15 tonnes for the 5 m pillars.

The cumulative support costs for 10 m and 5 m pillar sizes are as follows:

∘	5 m pillars;	at 128 m depth	- 73% with no support
			- 18% with bolts and straps, \$3.37/tonne
			- 9% with intensive support, \$7.53/tonne
		at 215 m depth	- 36% with no support
			- 18% with bolts and straps, \$3.37/tonne
			- 45% with intensive support, \$7.53/tonne
			average cost \$2.64/tonne of ore
∘	10 m pillars;	at 128 m depth	- 91% with no support
			- 0% with bolts and straps
			- 9% with intensive support, \$3.90/tonne
		at 215 m depth	- 55% with no support
			- 36% with bolts and straps, \$1.74/tonne
			- 9% with intensive support, \$3.90/tonne
			average cost \$0.66/tonne of ore

The cumulative support costs for the two layouts would be approximately \$2.64/tonne of ore for 5 m pillars and \$0.66/tonne for 10 m pillars.

It should be noted that there is only one very weak pillar in both layouts but the cost difference between 10 m and 5 m pillars is considerable.

3.5 Foundation Strength

Pillars are designed to accept high loads. The immediate hanging wall and footwall material constitutes the foundation for the pillar and must be strong enough to accept these loads. The maximum design load is 21.3 MPa. The foundation strength can be approximated from the following conservative relationship:

- foundation strength (MPa) = 13 x rock mass strength + pillar width.

The lowest rock mass strength is 2 MPa which gives a foundation strength well in excess of the value required. It should be noted that where low strength, low quality material appears in the immediate hanging wall provision has been made for fairly intensive support.

3.6 Discussion

The total support cost for both pillars and roof is \$3.9/tonne of ore for the 5 m layout and \$3.1/tonne of ore for the 10 m layout. The difference is within the degree of accuracy of the analysis procedures.

The method of calculating pillar stability is far more conservative that the analysis for roof stability because of the process for estimating pillar load. In addition, it is likely that there will be areas of unpayable ore either too thin, too contorted or missing from the succession. Drill holes 88F-03, and 88-F-10 illustrate this point. These unpayable areas could potentially be regional pillars. The areas between large pillars could shed load and reduce the incidence of pillar distress.

The geotechnical work was carried out on core which had not been geologically logged or assayed. For the purpose of this study 'ore' was delineated by the boundaries of the massive sulphide. This may be incorrect. In the recent geological interpretation it is noted that the 9% combined metal does not always coincide with the boundary of the massive sulphide. In a certain number of areas massive sulphide may be left in the roof. If this massive sulphide is thick enough then the roof behaviour may be improved. However, practical mining constraints may dictate that the mining interval be controlled by visual or structural boundaries in which case it may be difficult to maintain the interval within the massive sulphide and leave low grade ore in the roof.

It is concluded that for preliminary cost estimation the roof instability should be taken as the most critical factor. Further, that for the purposes of productivity estimation a room size of 5 to 7 m should be used together with a cost of $3.50 per tonne of ore for support difficulty. The support cost is directly proportional to the orebody thickness and a value of 6.1 m (20 feet) has been used. It should be noted that the above support costs assume some mechanization.

4.0 LAYOUT AND EXTRACTION OPTIONS

Room and pillar methods are very sensitive to geometry. The dip can restrict the application of mechanized methods. Rapid variations in geometry are very difficult to follow with a regular room and pillar layout.

The costs will be very sensitive to labour productivity and the degree of mechanization that can be achieved. Rubber tired equipment is generally restricted to gradients of 20 to 25%. A conventional square layout with the two operating directions running at 45° to the dip direction of the orebody would be limited to a dip of between 16° and 19.5° for 20% and 25% operating gradients. This ignores the problems and costs associated with mucking at steep positive gradients.

It is evident from Drawing 1 that very little of the deposit will lie at dips at which conventional room and pillar layouts will be possible. Diamond shaped pillars could be used with the mining direction at 60° to the dip resulting in a mineable gradient of 20% at a dip 23°. But this would significantly affect pillar strength.

There are further complicating factors. The geometry is likely to be much more variable than had first been considered. The orebody intersections are markedly difference from the recent geological interpretation as illustrated in Figures 3 to 11. These sections already suggest significant folding and thickening and the recent drilling adds to this variability.

Finally, the cut-off grade constraint must be considered. It is very difficult to maintain grade boundaries unless they are visual. The alternative is to use definition drilling from waste development and extract purely on line and grade. Although there is reference in the recent geological interpretation to a grade cut-off within the massive sulphide it is understood from discussions with Curragh personnel that the majority of the ore grade material will be the massive sulphide interval. In which case visual control will be possible.

4.1 Steep Dip Room and Pillar

Conventional room and pillar is unlikely to be possible because of the steep dips. Alternative methods which have been used are:

- slusher stoping - with haulages along strike and extraction with jacklegs and slushing down dip
- strike mining - a single major extraction direction using long pillars and inclined haulages

These two methods are illustrated in Figure 17. A degree of mechanization may be possible with the slusher method but with no precedents, as far as is known. That is to use tracked drill jumbos and dozers to operate at much steeper gradients. This combination is only possible where sufficient thickness includes two cycles of extraction; horizontal drilling followed by benching to enable use of the dozer.

There will be a significant cost difference between jackleg/slusher and jumbo/LHD extraction. Figures from elsewhere suggest a productivity range as follows:

- jackleg/slusher/in stope productivity 20-50 tons/man shift
- jumbo/LHD/in stope productivity 80-90 tons/man shift

The indexed cost comparison would be approximately 100 and 60 for the jackleg and jumbo combinations respectively.

<u>Slusher Method</u>

The slusher method, as conventionally applied, is based on wide stopes. The existing feasibility, based on the practices at Elliot Lake, has assumed 30 foot wide stopes with 13 foot wide pillars. The support cost for 10 m rooms calculated in Section 3.2 would be applicable. The 13 foot pillars are much narrower than the 5.6 m assessed in Section 3.3 but the method, as outlined, has short, squat pillars around the haulage and these pillars would be extremely strong. The layout, as envisaged, should be stable as long as the geometry is not extremely variable.

The percentage extraction will be sensitive to the stope dip length. This, in turn, will be sensitive to variation in geometry. Stopes will have to terminate at the boundaries of faults and folds. The irregularity in the dip of the footwall will influence the effectiveness of the slusher operation.

The slusher method is feasible but there could be very little opportunity for mechanization. The installation of roof and wall support in the weaker areas may be particularly onerous at steep dips when all operations have to be manual and materials moved with a tugger-hoist.

The alternative to the slusher stoping is a form of strike extraction as described below.

Strike Extraction

The objective of an alternative method would be a significantly increased level of mechanization. Extraction would be predominantly from parallel drifts. These would probably be driven at a gradient that minimized the potential for 'ponding' at low points. Figure 18 illustrates the concept for minimizing the amount of footwall waste that has to be extracted to provide a level running surface.

Semi-strike drifts would be separated by long pillars. Pillar cut-throughs between drifts could be extracted using the drill jumbo and extension steels where the length is excessive. Some broken ore would be unrecoverable because of the LHD would not be able to access all of the cut-through but the loss should be small.

The pillar width will vary to some extent due to variation in dip and thickness. The hanging wall would be extracted first but constraints on gradient might result in some variation in the distance between drifts. The roof of the cut-throughs could be supported using a roof bolting rig with a basket on a separate boom for handling mesh and straps where necessary.

The method would be sensitive to changes in geometry and the following could be the most signifiant:

- large dips would result in a very steeply sloping roof which could impose constraints on equipment and working procedures, Figure 18.

- rapid changes in geometry could result in ponding in the floor of an extraction drift. This would affect the conditions of the floor which must be excellent if good productivity and low operating costs are to be achieved with mechanized equipment. If the floor gradient is maintained at a positive value then the width of the pillar between extraction drifts could vary considerably, Figure 18.

The significance of these problems cannot be evaluated without a more detailed orebody interpretation. The probable result is that the percentage extraction would be decreased to ensure high productivity and low cost extraction.

The above discussion is based on very limited data regarding the geometry of the deposit but it serves to illustrate the shortcomings of the room and pillar method with a variable geometry.

5.0 CONCLUSIONS AND RECOMMENDATIONS

The objective of this study was to characterize the deposit in terms of rock mass quality and geometry and to evaluate the influence on room and pillar design and operation.

The geotechnical study has been carried out on core alone. These holes had not been interpreted at the time of the study. Their interpretation may significantly alter the existing orebody interpretation.

The overall conclusion is that some form of room and pillar would be possible but it would not be a conventional layout. There is no justification, at this stage, for taking the geotechnical work any further until a more detailed orebody interpretation has been carried out and approximate extraction layouts are available. The overall conclusion is based on the following:

<u>Rock Mass Quality</u>

- hanging wall - much of the material at the immediate contact of the massive sulphide could be relatively unstable due to the frequency of the foliation and weakness of the rock material. Both 10 m and 5 m room widths would require considerable roof support.

- ore body - the massive sulphide is usually strong and competent. There are a few instances of weak or broken bands which would necessitate pillar wall support. The massive sulphide does not pose any significant stability problem and a percentage extraction in excess of 75% should be achievable.

- foot wall - no significant footwall problems are foreseen.

- support costs - the cumulative cost of roof and wall support could be in the region of $3 to $4 per tonne of ore.

The characterization of the rock mass has been from core alone. The process of coring damages the core and any subsequent analysis tends to result in conservative conclusions. There is some potential for better ground conditions than have been identified in this report.

Geometry

Geometry is by far the most critical aspect. The dips are too steep for conventional, trackless room and pillar operations. Alternative methods would be either much more expensive, jackleg/slusher option, or have a lower recovery, jumbo/LHD with strike drifts.

There is some evidence that there will be a significant number of folds and faults. The influence of these can only be assessed from more detailed development and extraction layouts, based on the layout concept presented in this report, which would identify the extraction efficiency and unit productivity.

Recommendations

The most immediate concern is the geometrical variation as represented by the presence of folds and faults and the short range variation in dip and thickness. Once the degree of variation has been identified it should be possible to outline an extraction layout based on an estimated orebody which includes this variation. An extraction layout is essential for a feasibility as productivity, cost and percentage extraction will be signicantly influenced by the possible layout. The layout will place a constraint on the type of equipment, the procedures used and the shape of the pillars and attitude of the roof.

There is doubt concerning the most appropriate mining method and layout. It would be advisable to complete a more detailed mine feasibility prior to initial decline development and trial stoping.

When more detailed layouts are available the geotechnical study should be updated to ensure optimization of support costs and percentage extraction.

6.0 REFERENCES

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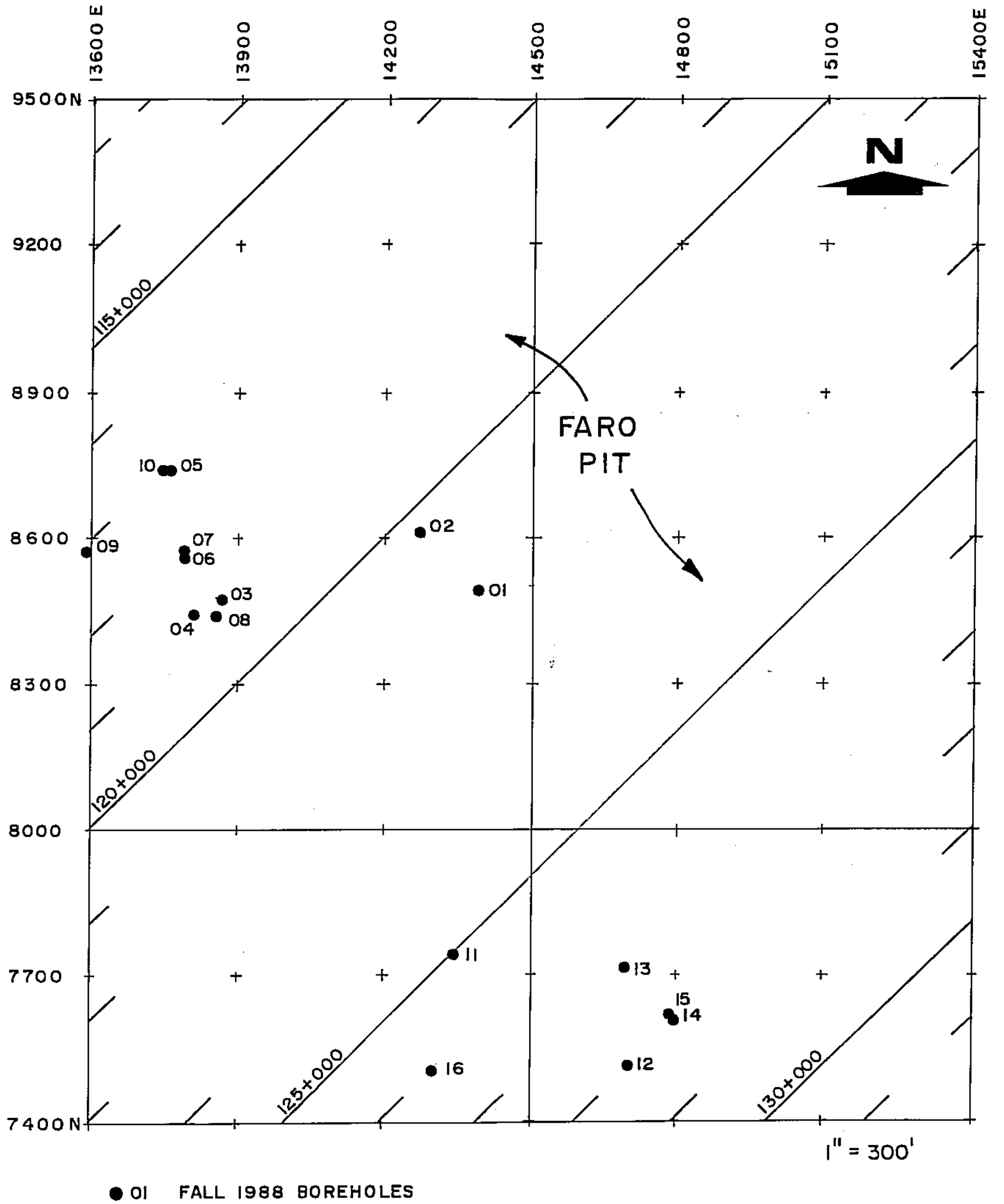
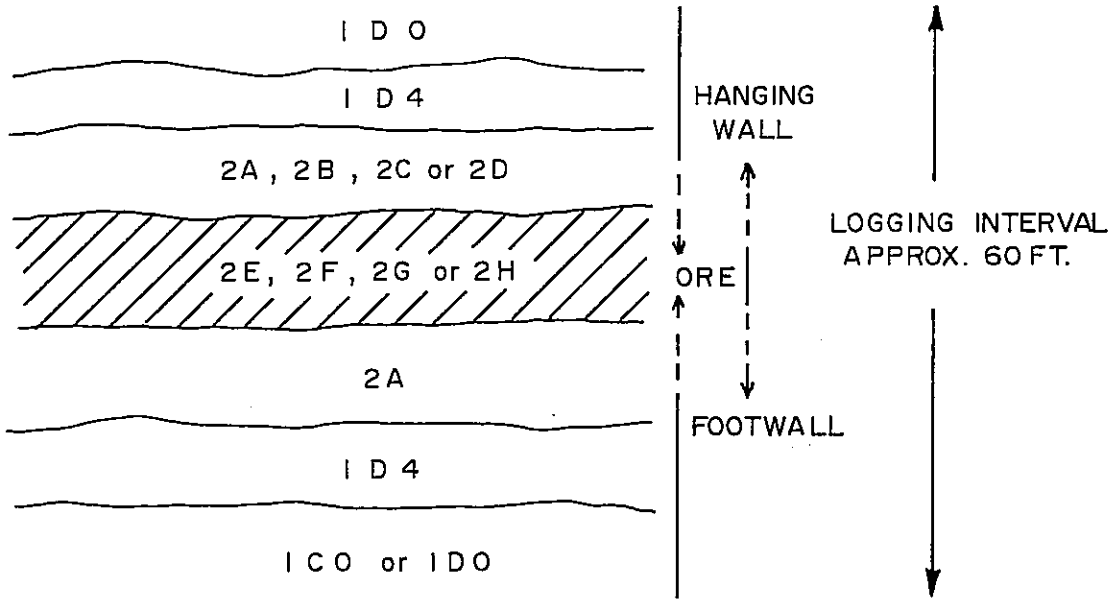
Piteau Gadsby MacLeod Ltd., 1975. Slope Stability Analysis and Design of the Open Pit Slopes, report to Cyprus Anvil Mining Corporation, North Vancouver.

FIGURES



CURRAGH RESOURCES INC.	FARO UNDERGROUND	DATE DEC. 1988
		PROJ. NO. 60610
1988 BOREHOLE LOCATIONS		APPROVED
		NO. 1
STEFFEN ROBERTSON & KIRSTEN, Consulting Engineers		



GEOLOGIC LEGEND

Faro Pit Mineralized Rocks

Unit 2
- A Sulfide-bearing, ribbon-banded, graphitic quartzite
- B Pyrite-free quartzite (may contain base metal sulfides)
- C Base metal-poor, pyritic quartzite
- D Base metal-bearing, pyritic quartzite
- E Massive pyritic sulfides
- F Buckshot facies, massive sulfides
- G Baritic facies, massive sulfides/sulfates (>10%:$BaSO_4$)
- H Pyrrhotitic facies, massive sulfides
  - 2 Pyrite-bearing
  - 4 ZnS and/or PbS-bearing
  - 0 Normal

Faro Waste Rocks

Unit 1
- C Quartzo-feldspathic, biotite - muscovite gneiss/schist
- D Carbonaceous biotite-muscovite-andalusite schist
  - 4 Altered, pyritic (wme)*
  - 0 Normal

*(wme) white mica envelope

CURRAGH RESOURCES INC. FARO UNDERGROUND

DATE NOV. 1988
PROJ. NO. 60610
APPROVED

SIMPLIFIED SECTION THROUGH
ORE ZONE

NO. 2

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