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DY DEPOSIT

MINEABLE RESERVE ESTIMATE AND

UNDERGROUND MINE PLAN

FARO, YUKON

by

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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.

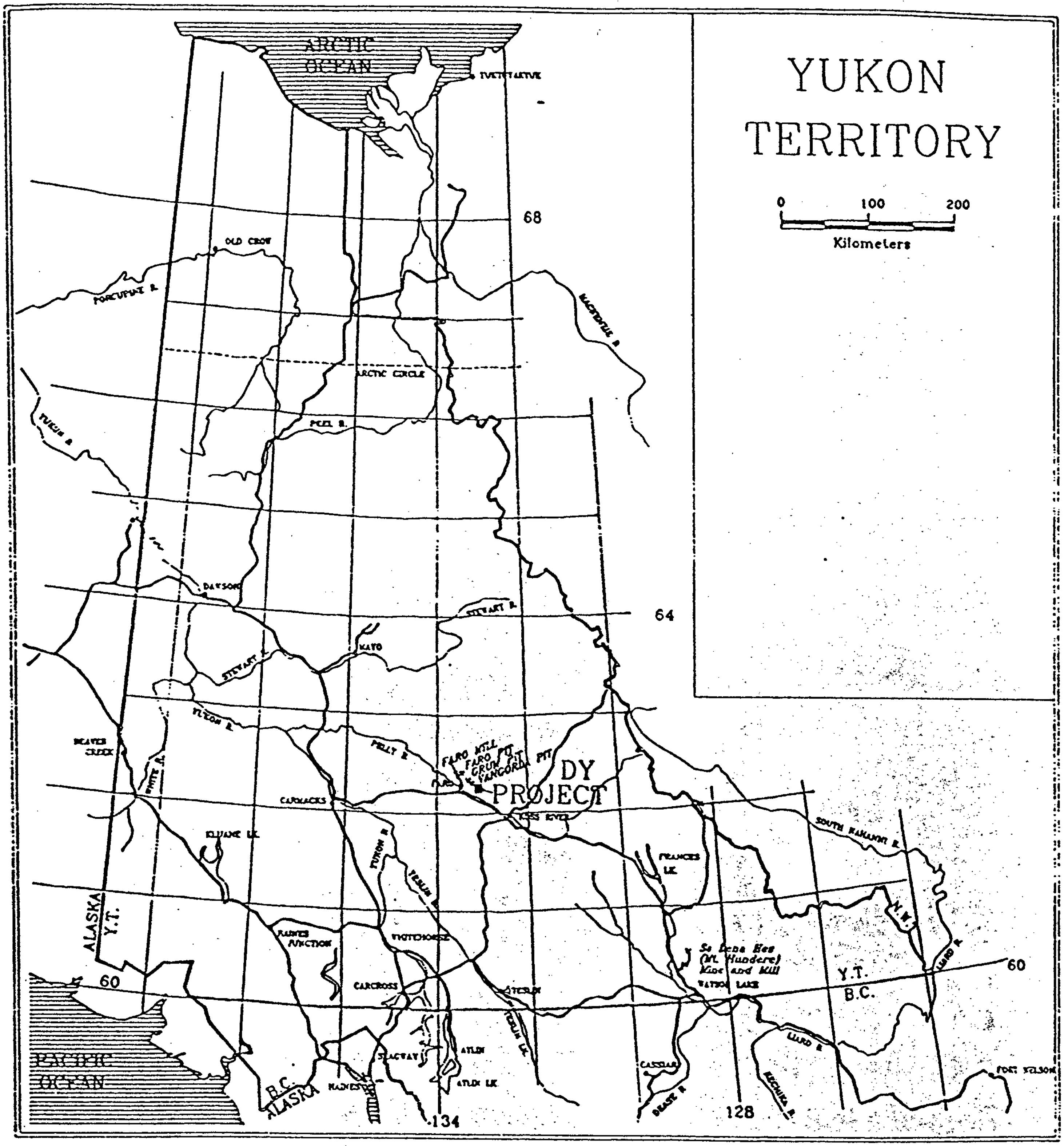


Figure 1: Map of the Yukon Territory showing the location of the Dy Project

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.

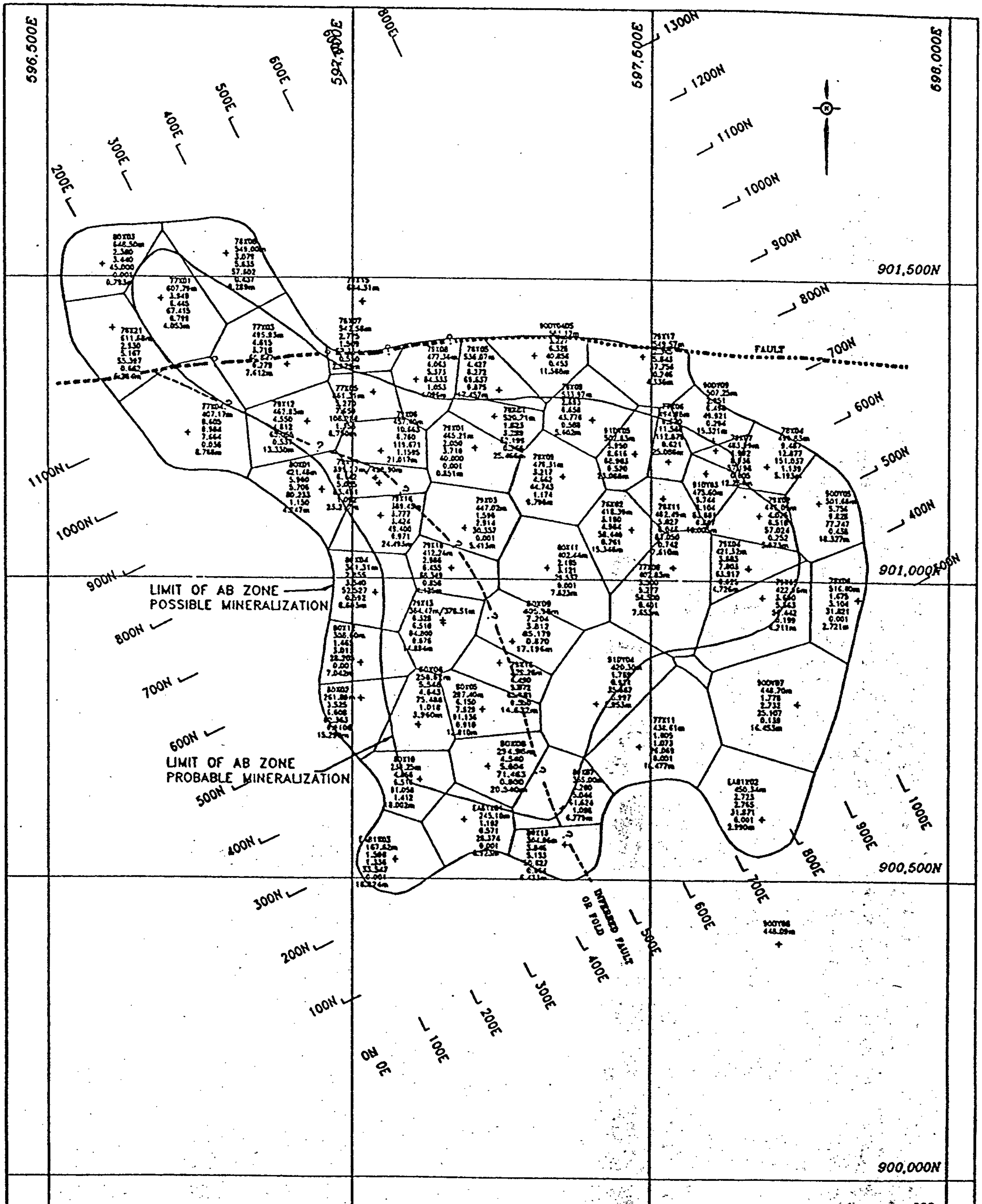
Drill hole spacing at Dy is considered too sparse to define a portion of the deposit as proven. Thus reserves are broken down into probable and possible categories.

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.



CURRAGH INC.

1:2000

DY DEPOSIT
9% Pb+Zn CUTOFF POLYGONS

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.

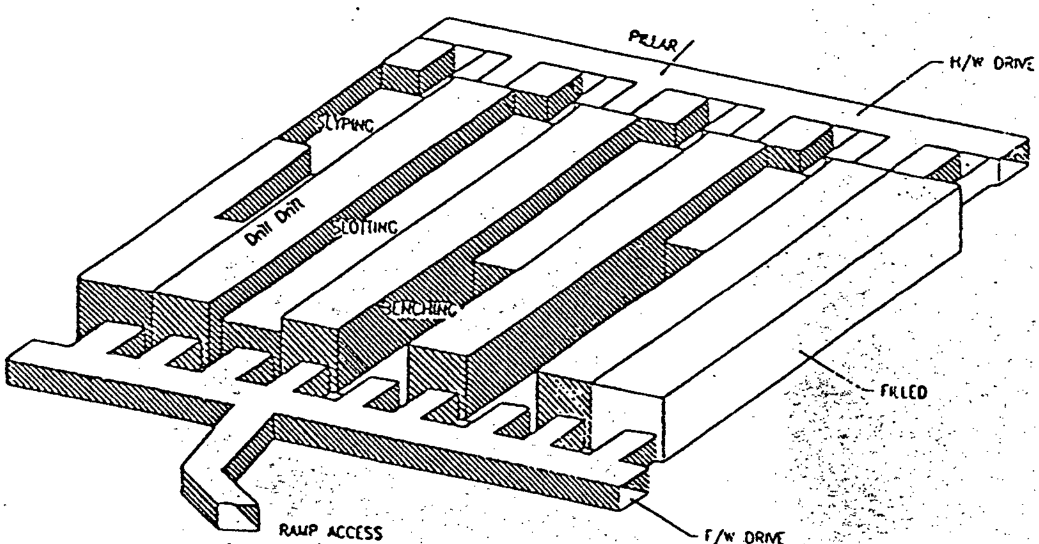


Figure 3 - Concrete Pillar Mining

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.

Table III
Dy Mineable Inventory at
85% Recovery and 10% Dilution

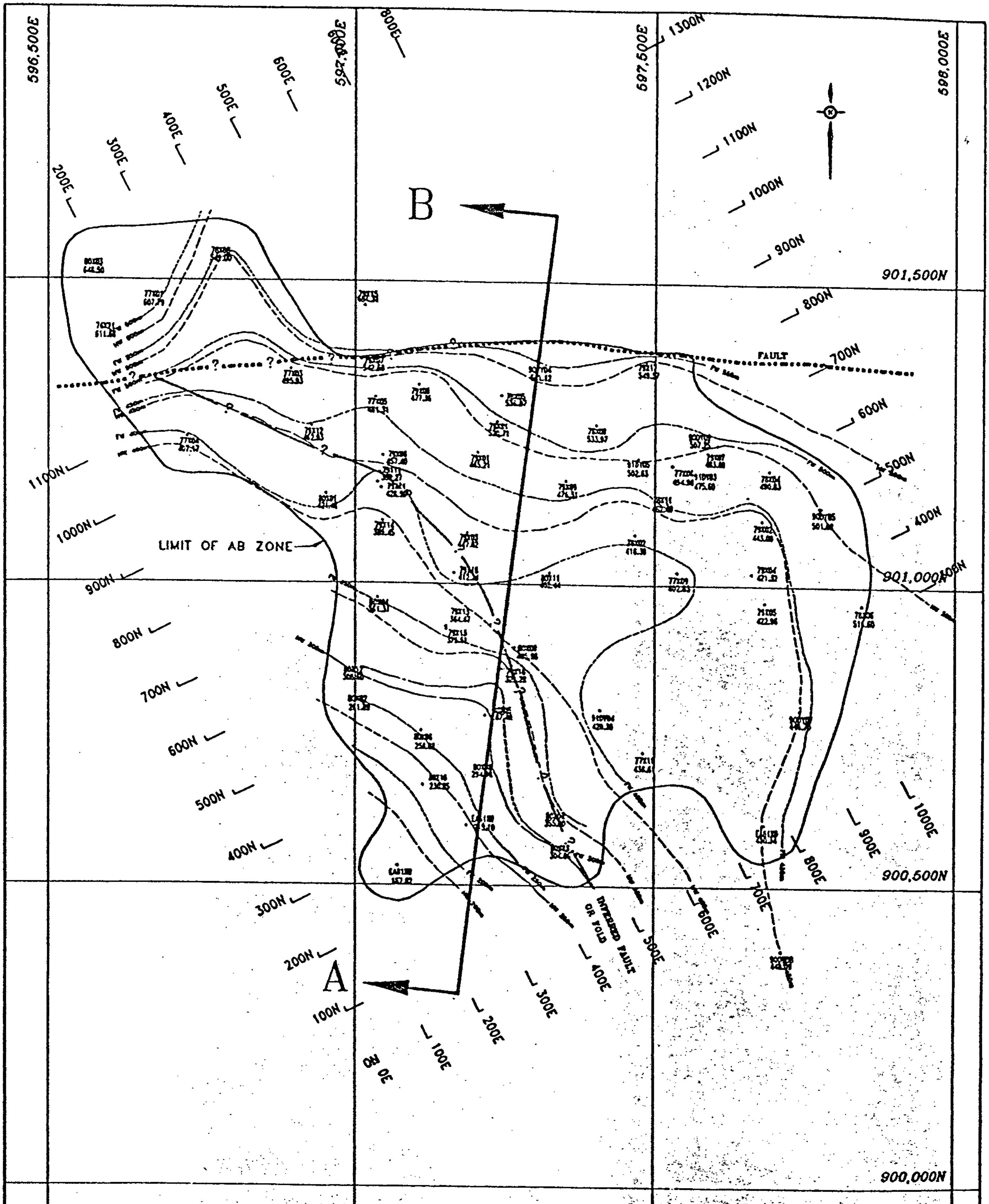
9% Lead+Zinc Cut-Off @ 85% Extraction	Tonnes	% Lead+Zinc	% Lead	% Zinc	Silver (g/t)	Gold (g/t)
Probable	9,251,382	12.19	5.70	6.48	81.7	0.834
Possible	4,039,501	11.99	4.31	7.68	67.3	0.702
Subtotal	13,290,883	12.13	5.28	6.85	77.3	0.794
10% Dilution	13,291,000	11.03	4.80	6.23	70.3	0.72

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



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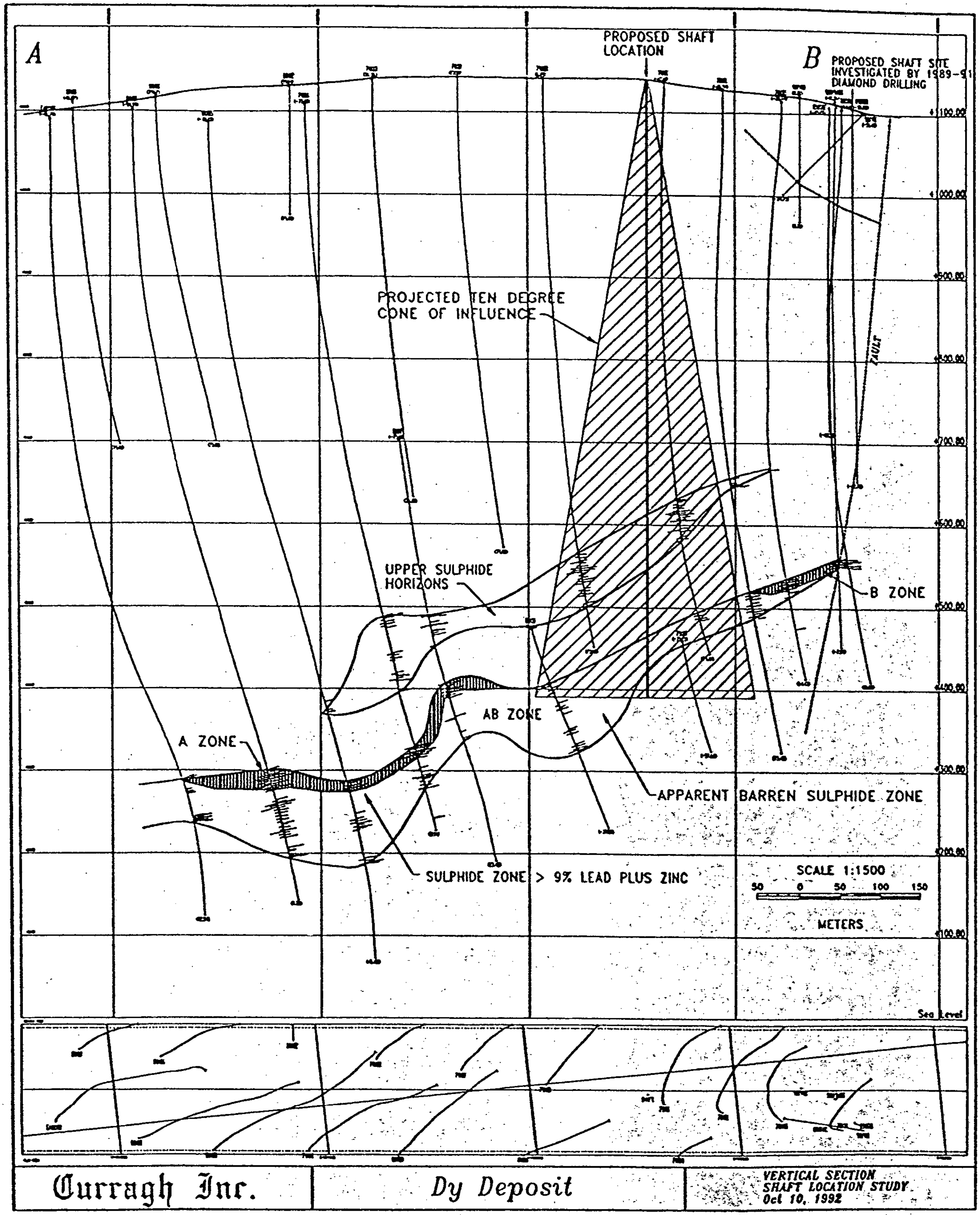
REVISED BY	K.D. ROSE, F.C.C.	DATE	20/10/92	1:2000
FILE	DYMPLAN	AUTOCAD V. 12		

DY DEPOSIT
MINING HW & FW CONTOURS


A
B
PROPOSED SHAFT
LOCATION
PROPOSED SHAFT SITE
INVESTIGATED BY 1989-91
DIAMOND DRILLING
PROJECTED TEN DEGREE
CONE OF INFLUENCE
FAULT
UPPER SULPHIDE
HORIZONS
B ZONE
AB ZONE
A ZONE
APPARENT BARREN SULPHIDE ZONE
SULPHIDE ZONE > 9% LEAD PLUS ZINC
SCALE 1:1500
50 0 50 100 150
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Sea Level
Curragh Inc.
Dy Deposit
VERTICAL SECTION
SHAFT LOCATION STUDY
Oct 10, 1992

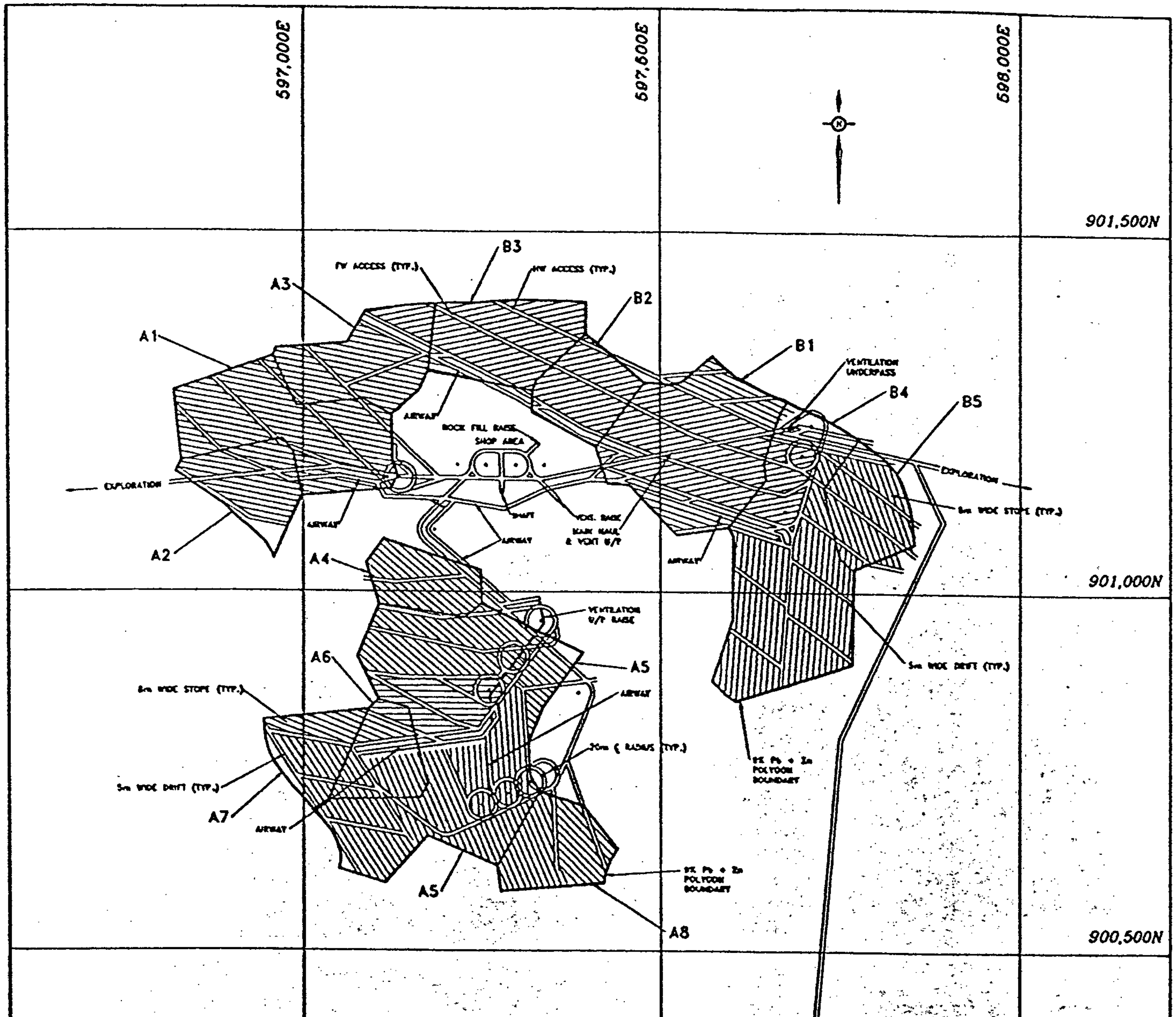
Figure 5

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.



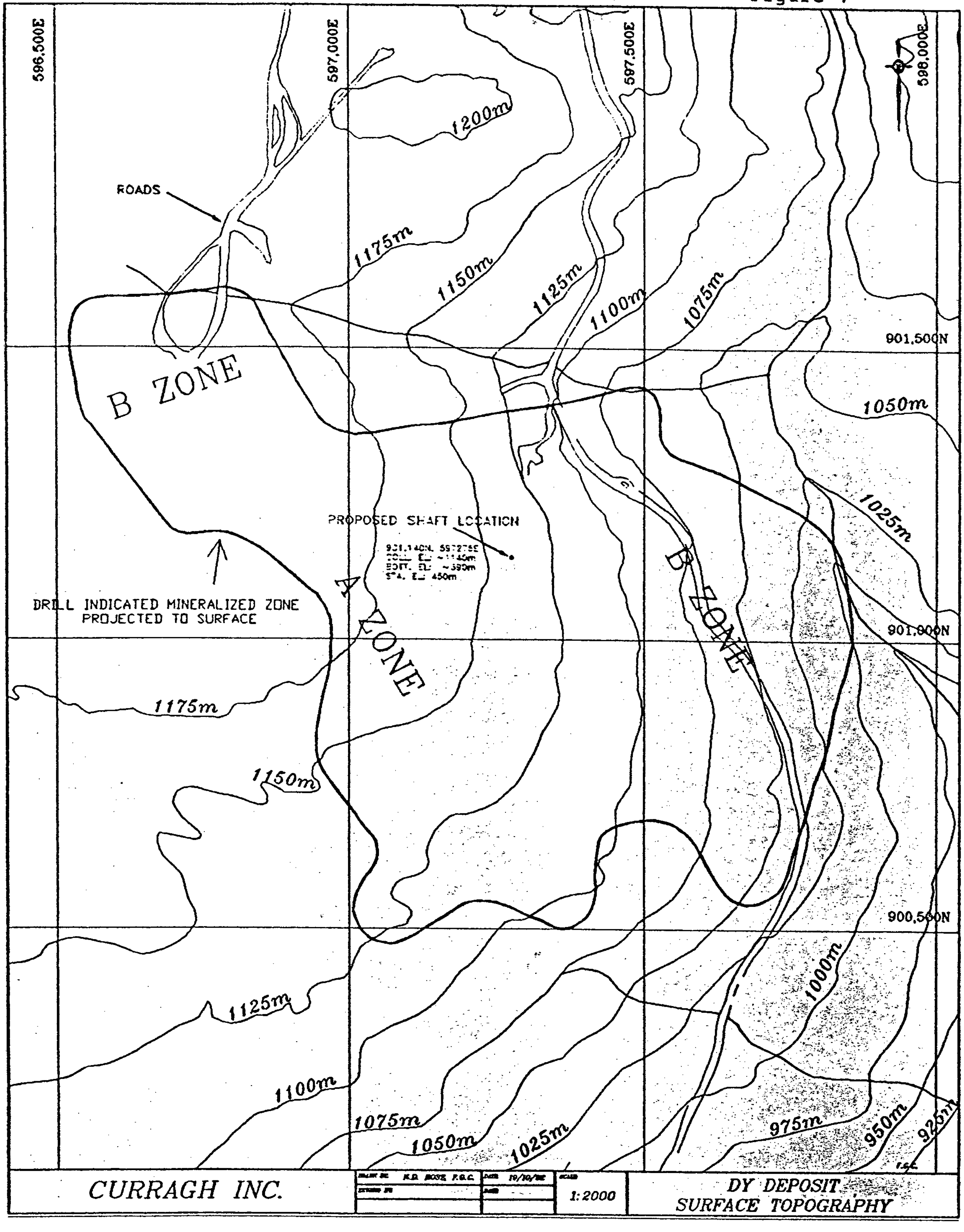
MINING INVENTORY

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	2.89	6.66	43.8	0.59
B3	14.52	1,205,995	3.91	7.43	57.2	0.69
B4	4.45	820,928	5.01	7.92	78.8	0.50
B5	18.33	1,116,889	5.76	9.83	77.7	0.44
A1	17.70	2,519,895	7.20	5.59	89.8	0.91
A2	4.25	245,848	5.98	5.71	80.2	1.15
A3	5.08	424,892	4.68	6.50	96.2	1.20
A4	4.13	190,979	2.99	8.48	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	0.81
A8	6.78	420,986	4.28	5.04	61.8	1.10
TOTAL	11.62	15,636,325	5.28	6.85	77.3	0.78
10 % DILUTED	11.62	15,636,000	4.80	6.23	70.3	0.72

F.C.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	PLAN DY-1

Figure 7


596,500E
597,000E
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ROADS
1200m
1175m
1150m
1125m
1100m
1075m
1050m
1025m
1000m
975m
950m
925m
901,500N
901,000N
900,500N
B ZONE
A ZONE
B ZONE
PROPOSED SHAFT LOCATION
DRILL INDICATED MINERALIZED ZONE
PROJECTED TO SURFACE
CURRAGH INC.
1:2000
DY DEPOSIT
SURFACE TOPOGRAPHY

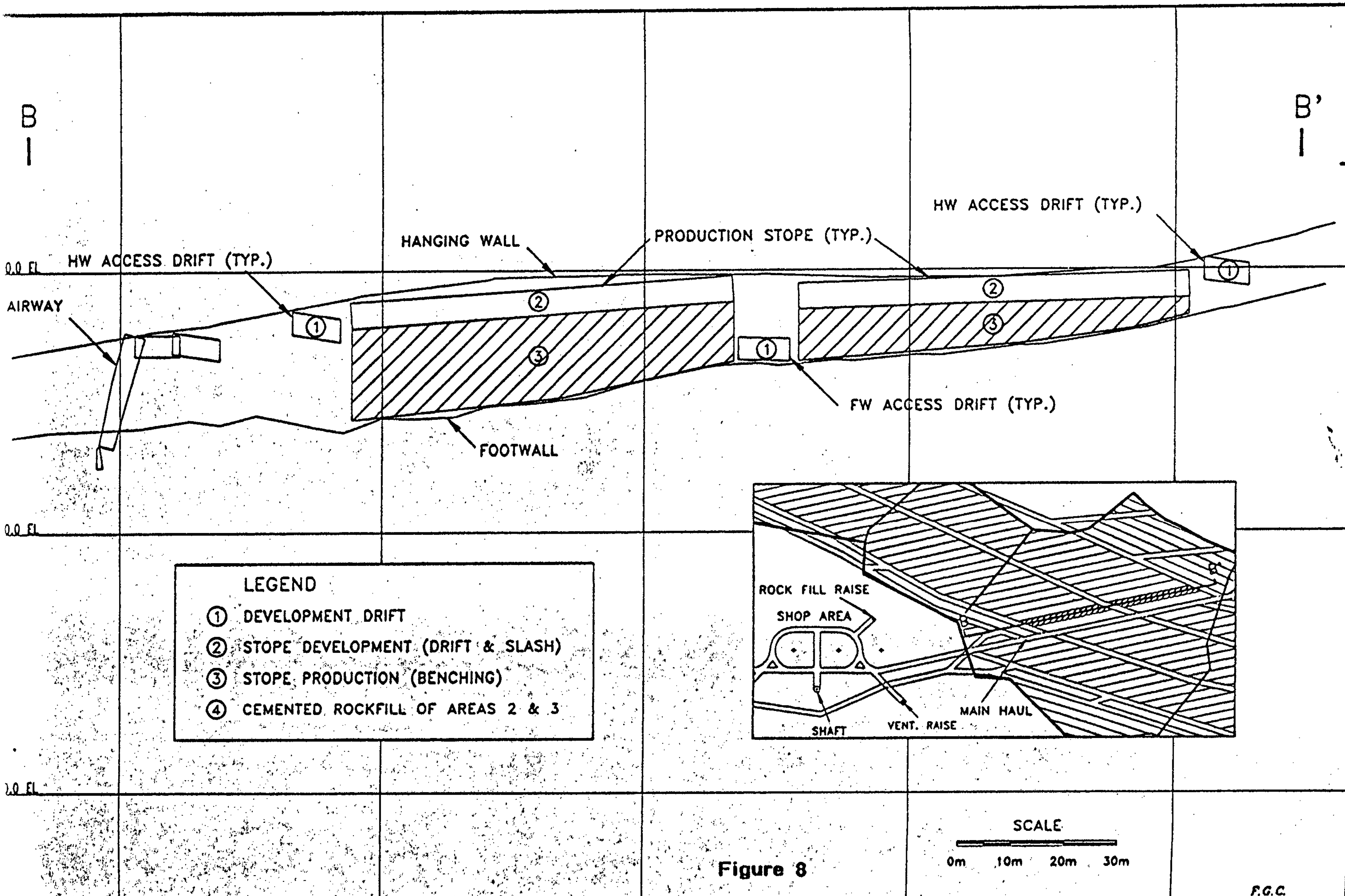

Figure 8

CURRAGH INC.	DESIGNED BY: N.D. ROSE F.G.C.	DATE: 17/08/92	SCALE: 1:1000	DY DEPOSIT TYPICAL B ZONE LONG-SECTION
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	FILE: STOPE1 AUTOCAD V. 12			

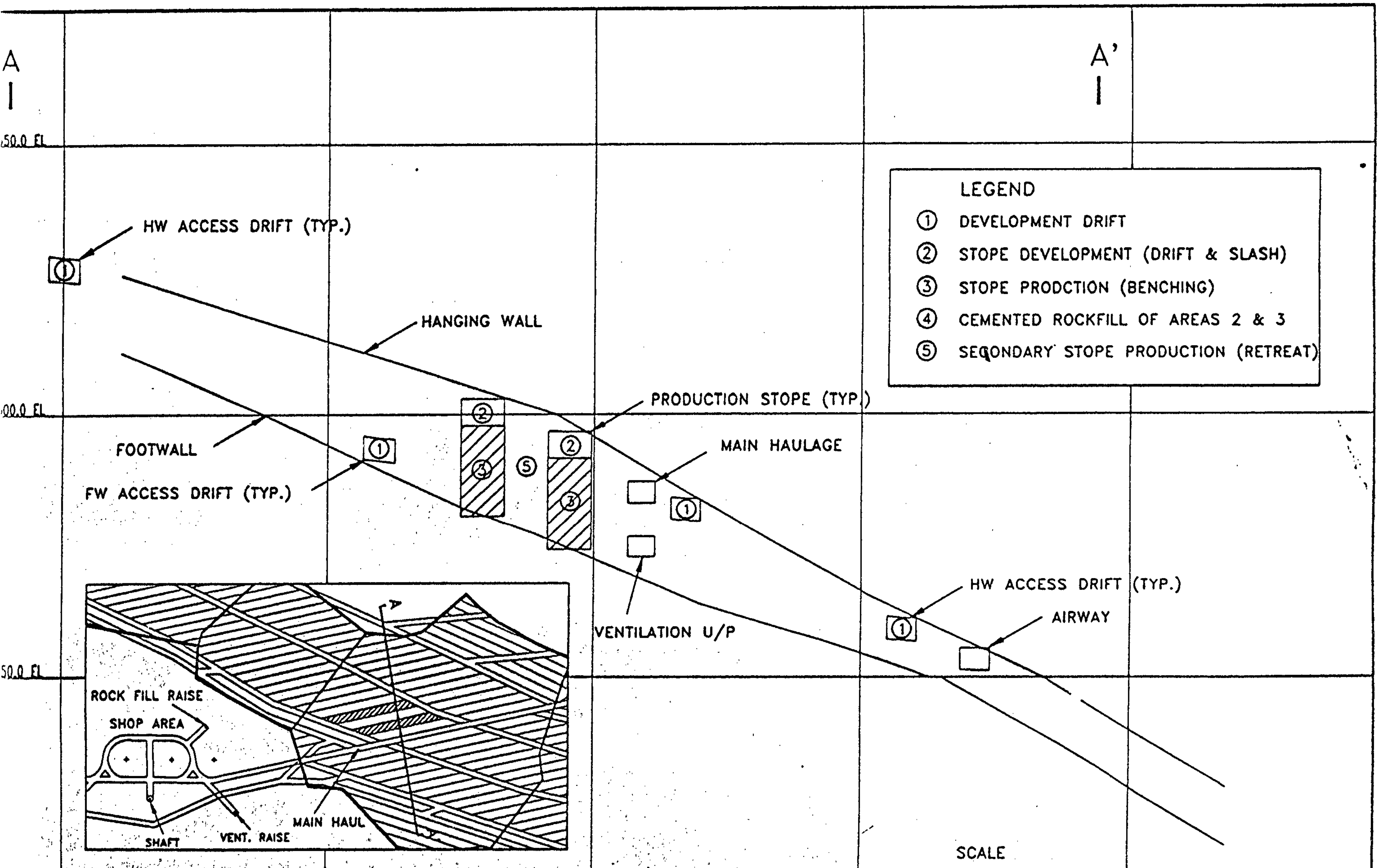


Figure 9

CURRAGH INC.	DESIGNED BY: N.D. ROSE, F.G.C.	DATE 17/08/92	SCALE 1:1000	DY DEPOSIT TYPICAL B ZONE CROSS-SECTION
	DRAWN BY: N.D. ROSE, F.G.C.	DATE 08/10/92		
	FILE DYSTPXC1 AUTOCAD V. 12			

4.0 DY ELEVEN-YEAR DEVELOPMENT/PRODUCTION SCHEDULE

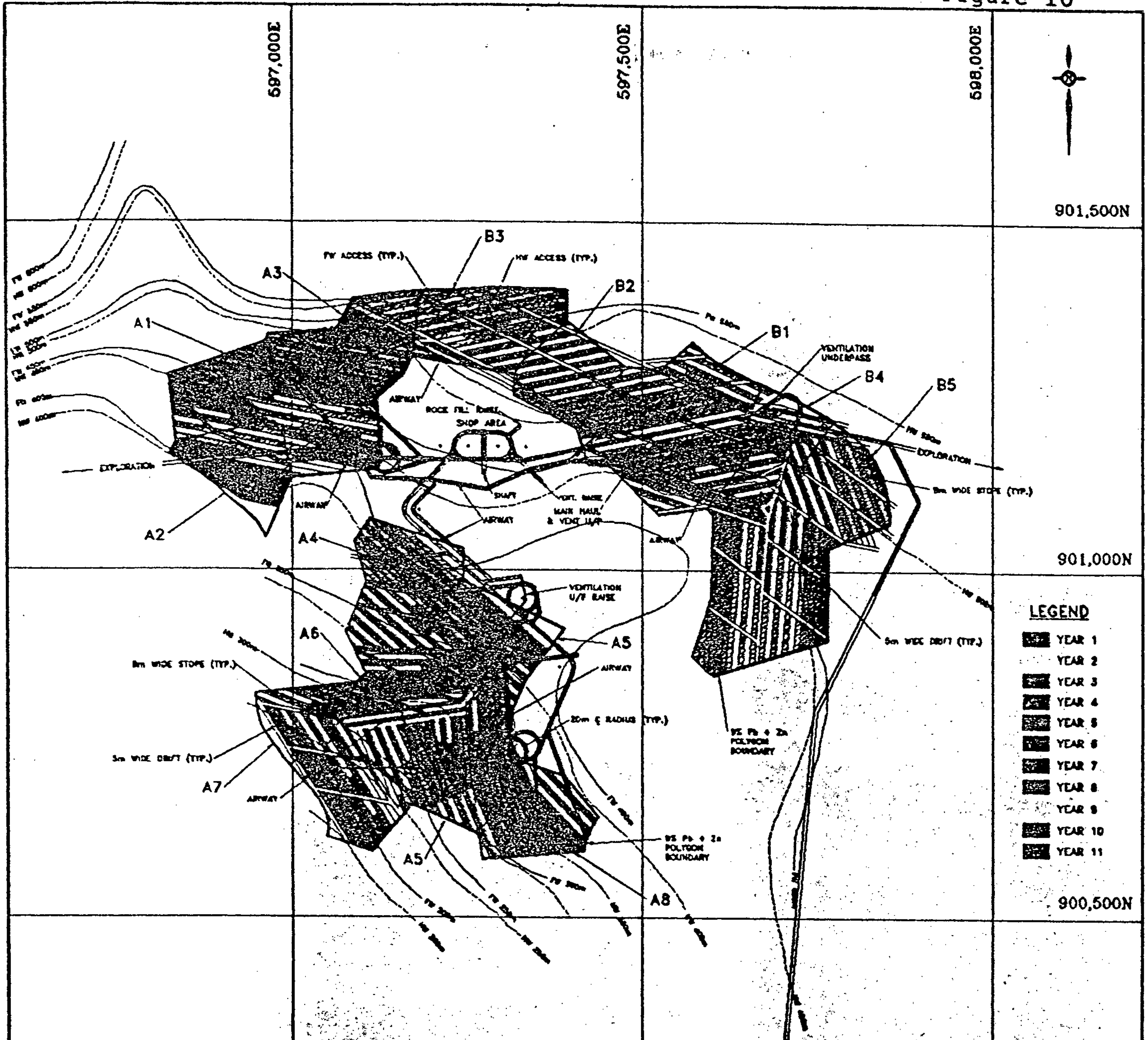
An 11-year development/production schedule of the Dy Mineable Inventory was developed in August, 1992. A summary of production is listed in Table IV. A more detailed production schedule is included in Appendix III along with a 1:2000 mine plan showing a yearly breakdown of mining.

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

YEAR	ADV (m)	TONNES	ASSAY GRADES				
			%Pb	%Zn	Ag(g/t)	Au(g/t)	[illegible]
YEAR 1	4743	687,308	4.52	8.55	75.5	0.55	[illegible]
YEAR 2	5995	1,000,296	4.68	8.45	71.8	0.54	2740
YEAR 3	5655	1,006,333	4.99	7.83	73.0	0.65	2757
YEAR 4	5310	1,012,742	5.26	7.63	74.6	0.69	2774
YEAR 5	5595	1,051,506	5.17	7.24	73.7	0.72	2855
YEAR 6	5600	1,060,924	5.29	6.76	80.9	0.84	2907
YEAR 7	5900	1,112,349	5.39	6.66	81.4	0.86	3047
YEAR 8	5875	1,148,790	5.56	5.94	79.7	0.94	3157
YEAR 9	5455	1,120,267	5.77	5.80	79.9	0.94	3049
YEAR 10	4495	1,042,250	5.75	5.82	80.4	0.93	2855
YEAR 11	4455	1,006,813	5.08	5.92	75.0	0.94	2757
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	15.84	2,367,144	4.58	8.73	77.2	0.308
B2	3.60	308,443	2.69	6.06	43.8	0.580
B3	14.32	1,203,993	3.91	7.43	57.2	0.683
B4	4.45	520,526	8.01	7.93	78.9	0.302
B5	16.33	1,116,880	6.76	8.83	77.7	0.438
A1	17.76	2,313,260	7.28	5.50	80.8	0.912
A2	4.25	245,846	5.96	5.71	80.2	1.150
A3	5.08	424,630	4.66	6.30	96.2	1.201
A4	4.13	190,879	2.89	4.46	68.5	0.656
A5	15.65	5,018,219	6.81	5.85	80.1	0.928
A6	3.96	194,080	5.33	4.84	73.5	1.018
A7	13.29	705,210	3.53	6.61	62.4	0.811
A8	6.78	430,906	4.25	8.04	61.6	1.090
TOTAL	11.62	15,634,355	5.35	6.85	77.3	0.794
90 % DILUTED	11.62	15,634,000	4.80	6.23	70.3	0.72

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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.

30° HT. WHITEHORSE UPPER IN DIORITE/SKARN

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.

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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
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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

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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.

IF 60° CAVE OF SUBSIDENCE
SURFACE SHAFT IS AT RISK
CAVING METHODS USED

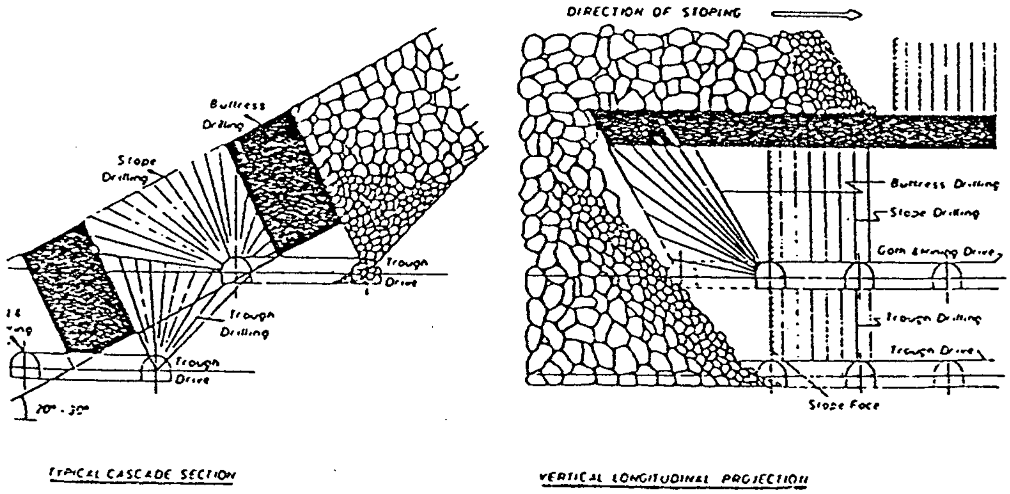


Figure 3.1 Cascade method (after Mabson & Russel, 1981)

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%.

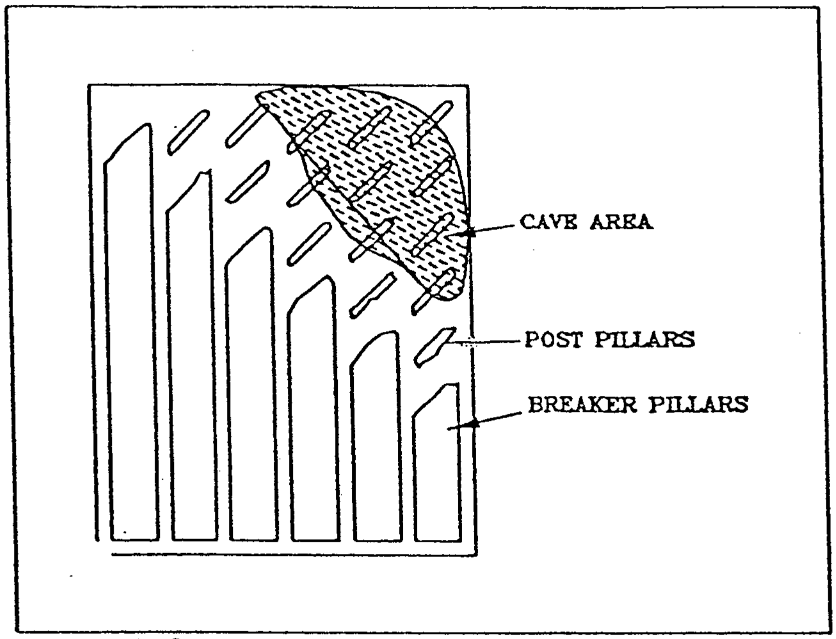


Figure 3.2 Caving using post pillar recovery

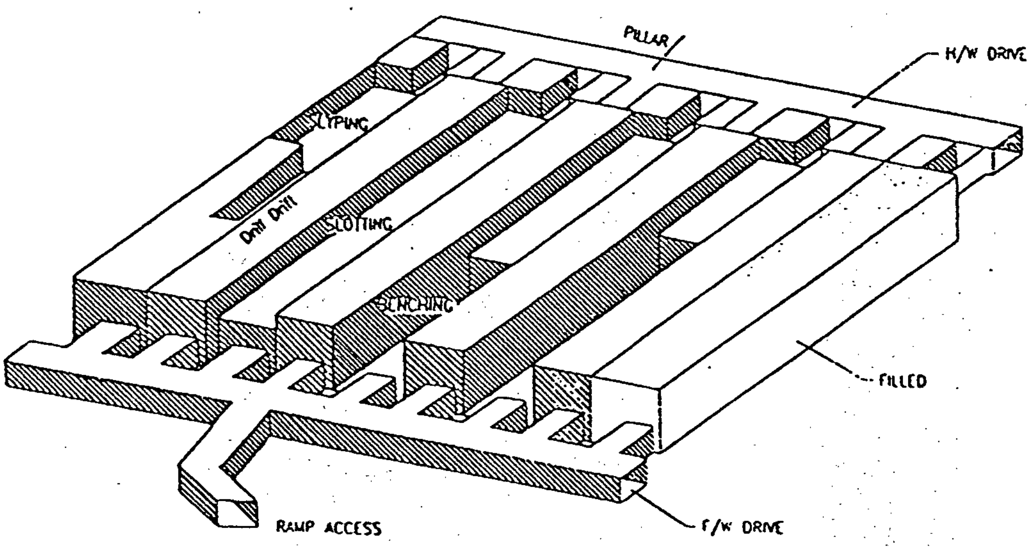


Figure 3.3 Concrete pillar mining

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%.

[illegible] 6000 [illegible] THE CEMENTED [illegible]

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.

DRIVE DOWNHILL TO FACILITATE FILLING.

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.

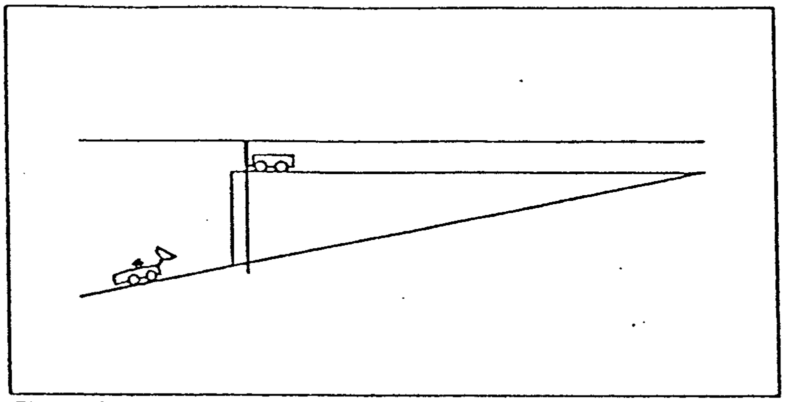
BACK SUPPORT [illegible]

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.

BUT BACK SUPPORT [illegible] WOULD BE DIFFICULT

Disadvantages include the possibility of stope loss if a large failure occurs from the pillar walls during mining. — MORE PROBABLY BACKS WOULD FAIL.



INACCESSIBLE
BACK SURFACE
TOO LARGE

Figure 3.4 Bench blasting VERTICAL HOLES.

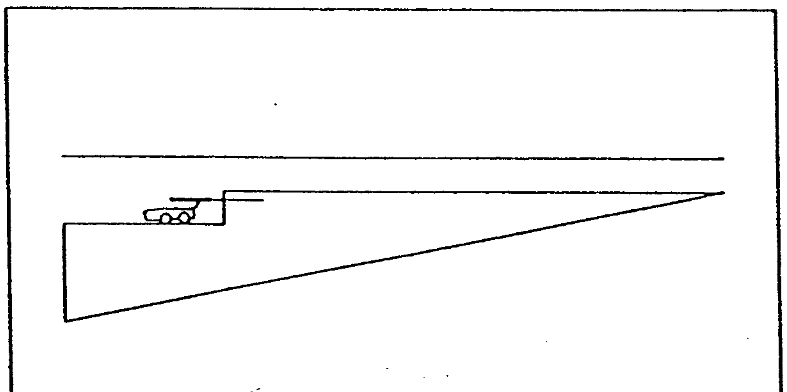
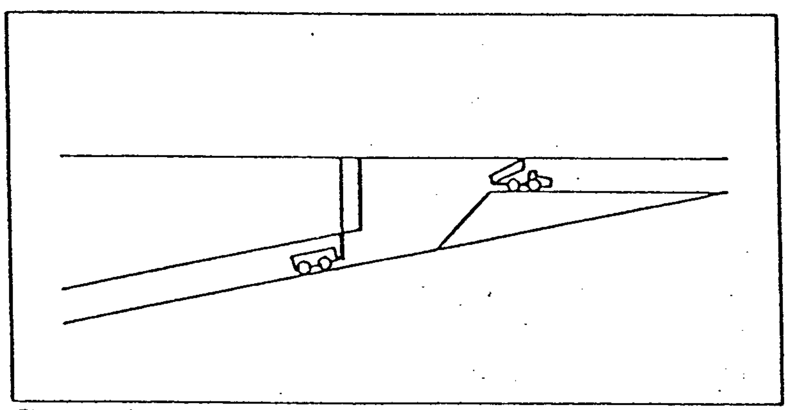


Figure 3.5 ~~Breasting~~ ? BENCHING - HORIZONTAL HOLES.



AREA OF INACCESSIBLE
BACK IS LIMITED -
CAN ALSO HALT BLASTING
AND FILL UP TO
RETREATING FACE
IF NECESSARY FOR
BOLTING BACK.

Figure 3.6 Endako method BREASTING - VERTICAL HOLES.

AVOCA 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

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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	
	90DY09	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 0.0	2.69	6.66	43.8	0.588	
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
	90DY04	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 0.0	5.76	9.83	77.7	0.438	
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 0.0	5.96	5.71	80.2	1.150	
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 0.0	2.99	6.46	68.3	0.856	
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 0.0	5.55	4.64	75.5	1.018	
SUB TOTAL				194,099.0	5.55	4.64	75.5	1.018	
A7	80X02	11765.9	15.29	705210.4 0.0	3.53	6.61	62.4	0.911	
SUB TOTAL				705,210.4	3.53	6.61	62.4	0.911	
A8	80X07	15842.2	6.78	420985.6 0.0	4.26	5.04	61.6	1.098	
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 BLOCK	DDH POLYGON	AREA	VRT TH	TONNES	ASSAY GRADES %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

YEAR	ADV (m)	TONNES	ASSAY GRADES			
			%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	0.554	590	49147	4.67	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	5.50	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)
N4-6	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
N6-8	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
N7-9	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	4.60	8.44	74.9	0.552

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)
N4-6	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
N6-8	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
N7-9	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	4.56	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	2166.4	2515.5
#TONNES/DAY WASTE:	399.4	190.7	325.0
#TONNES/DAY MINED:	827.8	2357.1	2840.5
#TONNES/DAY FILLED:	0	0	1114.75
%DEVELOPMENT:	100.0%	37.9%	30.2%
%PRODUCTION:	0.0%	62.1%	69.8%
%BENCH PRODUCTION:	0.0%	26.3%	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)
N1	4.25	5.00		0.0					380	31654	4.66	8.58	77.5	0.548
N1 U/P	4.00	4.00		0.0					265	16621	4.58	8.73	77.2	0.558
N2	4.25	5.00	65	5414.5	5.01	7.92	78.9	0.502	265	22075	4.69	8.63	77.8	0.544
N4	4.25	5.00	60	4998.0	5.01	7.92	78.9	0.502	250	20825	4.64	8.54	77.6	0.545
N5	4.25	5.00	50	4165.0	2.89	6.66	43.8	0.588	200	16660	4.11	8.21	68.9	0.566
N6	4.25	5.00	30	2499.0	2.89	6.66	43.8	0.588	160	13328	4.23	8.34	70.9	0.564
N7	4.25	5.00	60	4998.0	2.89	6.66	43.8	0.588	190	15827	3.49	7.53	57.9	0.575
N8	4.25	5.00		0.0					110	9163	4.81	8.29	78.1	0.527
N9	4.25	5.00	50	4165.0	2.89	6.66	43.8	0.588	150	12495	3.96	8.04	66.1	0.568
N10	4.25	5.00	50	4165.0	5.76	9.83	77.7	0.438	150	12495	5.26	8.86	78.5	0.481
N11	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
N12	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
N14	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
N16	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
N17	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	51230	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				
N1	5.00	4.25		0.0					140	6976				
N1 U/P	5.00	4.25		0.0					50	2492				
N2	5.00	4.25		0.0					140	6976				
N2 V.RSE	2.50	DIAM		0.0					665	9467				
N3	5.00	4.25		0.0					125	6229				
N3 R.RSE	1.50	DIAM		0.0					663	3396				
N13	5.00	4.25	100	4983.1					250	12458				
DECLINE	5.00	4.25	400	19932.3					1100	54814				
W1	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	36	5880.0	4.58	8.73	77.2	0.558	263	41160	4.58	8.73	77.2	0.558
N7-8	5.00	8.00	75	11760.0	2.89	6.66	43.8	0.588	113	17640	2.89	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	0.558

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	66.5	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:	594.078
%DEVELOPMENT:	31.1%	%DEVELOPMENT:	39.1%
%PRODUCTION:	68.9%	%PRODUCTION:	60.9%
%BENCH PRODUCTION:	41.9%	%BENCH PRODUCTION:	33.4%

CURRAGH RESOURCES INC.
DY UNDERGROUND DEVELOPMENT/PRODUCTION SCHEDULE
file: DYSCH2-6
Date: 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	4.56	8.73	77.2	0.558	80	6654.0	4.56	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	15627.0	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	0.906	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	11481.1				
S2 U/P 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					535	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	4.56	8.73	77.2	0.558	315	49392.0	4.56	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	78.5	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	4.56	8.73	77.2	0.558	195	61152.0	4.56	8.73	77.2	0.558	105	32928.0	4.56	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	146803.2	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
A5	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	0.606	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	7.83	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	2875.2	2893.5
# TONNES/DAY WASTE:	158.7	101.8	76.2
# TONNES/DAY MINED:	3016.7	2977.0	2969.7
# TONNES/DAY FILLED:	1493.5	1560.9	1631.9
%DEVELOPMENT:	17.5%	12.6%	8.4%
%PRODUCTION:	82.5%	87.4%	91.6%
%BENCH PRODUCTION:	37.9%	44.2%	48.7%

CURRAGH RESOURCES INC.
DY UNDERGROUND DEVELOPMENT/PRODUCTION SCHEDULE
file: DYSCH2-6
Date: 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	0.586
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 RSE	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	490	76832.0	2.69	6.66	43.8	0.568	1040	163072	2.69	6.66	43.8	0.583
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	0.858
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	167146.8	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		TOTAL	
#TONNES/DAY ORE:	3004.3	#TONNES/DAY ORE:	2907.8
#TONNES/DAY WASTE:	62.0	#TONNES/DAY WASTE:	99.7
#TONNES/DAY MINED:	3066.3	#TONNES/DAY MINED:	3007.4
#TONNES/DAY FILLED:	1713.3	#TONNES/DAY FILLED:	1599.9
%DEVELOPMENT:	6.9%	%DEVELOPMENT:	11.3%
%PRODUCTION:	93.1%	%PRODUCTION:	88.7%
%BENCH PRODUCTION:	46.5%	%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	4.28	5.04	61.8	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	4.28	5.04	61.8	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	73.8	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%

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	61.8	1.096
				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	2.89	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	6.00	720.0	112896.0	4.26	5.04	61.8	1.096	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

#TONNES/DAY ORE:	3282.3
#TONNES/DAY WASTE:	61.9
#TONNES/DAY MINED:	3344.2
#TONNES/DAY FILLED:	1887.2
%DEVELOPMENT:	5.9%
%PRODUCTION:	94.1%
%BENCH PRODUCTION:	[illegible]

TOTAL

#TONNES/DAY ORE:	3163.9
#TONNES/DAY WASTE:	64.8
#TONNES/DAY MINED:	3228.7
#TONNES/DAY FILLED:	1798.4
%DEVELOPMENT:	7.2%
%PRODUCTION:	92.8%
%BENCH PRODUCTION:	[illegible]

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=2.35 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%

CURRAGH RESOURCES INC.
DY UNDERGROUND DEVELOPMENT/PRODUCTION SCHEDULE
file: DYSC9-11
Date: 19-Oct-92

DEVELOPMENT

YEAR 11

ORE DEVELOPMENT				YEAR 11	# 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					0	0	0.00	0.00	0.0	0.000
A5	4.25	5.00		0.0					275	22908	5.61	5.85	80.1	0.925
A6	4.25	5.00		0.0					20	1666	5.55	4.64	75.5	1.018
A7	4.25	5.00		0.0					235	19576	3.53	6.61	62.4	0.911
A8	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
				0.0					0	0	0.00	0.00	0.0	0.000
SUB TOTAL			0	0	0.00	0.00	0.0	0.000	530	44,149	4.69	6.14	72.1	0.922

WASTE DEVELOPMENT				YEAR 11					TOTAL					
HEADING	HT (m)	W (m)	ADV (m)	TONNES					ADV (m)	TONNES				
S1	5.00	4.25	90	4484.8					270	13454				
S2 U/P RSE	2.40	2.40		0.0					0	0				
S4	5.00	4.25	90	4484.8					270	13454				
S5	5.00	4.25		0.0					0	0				
				0.0					0	0				
SUB TOTAL			180	8,970					540	26,909				

NOTE: WASTE DEVELOPMENT IS CONSIDERED CONSERVATIVE FOR MINING OF STEEP ORE IN SOUTHERN A ZONE.

PRODUCTION

STOPE DEVELOPMENT (DRIFT & SLASH)				YEAR 11					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)
A1	5.00	8.00		0.0					955	149744	7.20	5.59	89.8	0.912
A2	4.00	8.00		0.0					260	32614	5.96	5.71	80.2	1.150
A5	5.00	8.00	1295	203056.0	5.61	5.85	80.1	0.925	4125	646800	5.61	5.85	80.1	0.925
A6	4.00	8.00	295	37004.8	5.55	4.64	75.5	1.018	785	98470	5.55	4.64	75.5	1.018
A7	5.00	8.00	460	72128.0	3.53	6.61	62.4	0.911	805	126224	3.53	6.61	62.4	0.911
A8	5.00	8.00	380.0	59584.0	4.26	5.04	61.6	1.098	680	106624	4.26	5.04	61.6	1.098
				0.0					0	0	0.00	0.00	0.0	0.000
SUB TOTAL			2430	371,773	4.98	5.75	73.2	0.959	7610	1,160,477	5.47	5.72	77.3	0.952

STOPE PRODUCTION (BENCHING)				YEAR 11					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)
A1	10.00	8.00		0.0					1170	366912	7.20	5.59	89.8	0.912
A5	10.00	8.00	1565	490784.0	5.61	5.85	80.1	0.925	4195	1315552	5.61	5.85	80.1	0.925
A7	10.00	8.00	460	144256.0	3.53	6.61	62.4	0.911	900	282240	3.53	6.61	62.4	0.911
				0.0					0	0	0.00	0.00	0.0	0.000
SUB TOTAL			2025	635,040	5.14	6.02	76.1	0.922	6265	1,964,704	5.61	5.91	79.4	0.921
PROD TOT			4455	1,006,813	5.08	5.92	75.0	0.936	13875	3,125,181	5.56	5.84	78.6	0.932
TOTAL ORE			4455	1,006,813	5.08	5.92	75.0	0.936	14405	3,169,330	5.54	5.84	78.5	0.932
10% DILUTION			4455	1,006,813	4.62	5.38	68.2	0.85	14405	3,169,330	5.04	5.31	71.4	0.85

CEMENTED ROCKFILL				YEAR 11		TOTAL	
LOCATION				TONNES		TONNES	
				604087.7		1875108.5	
				0.0		0.0	
SUB TOTAL				604,088		1,875,108	

FOX GEOLOGICAL CONSULTANTS LTD.

YEAR 11		TOTAL	
#TONNES/DAY ORE:	2876.6	#TONNES/DAY ORE:	3018.4
#TONNES/DAY WASTE:	25.6	#TONNES/DAY WASTE:	25.6
#TONNES/DAY MINED:	2902.2	#TONNES/DAY MINED:	3044.0
#TONNES/DAY FILLED:	1726.0	#TONNES/DAY FILLED:	1785.8
%DEVELOPMENT:	0.9%	%DEVELOPMENT:	2.2%
%PRODUCTION:	99.1%	%PRODUCTION:	97.8%
%BENCH PRODUCTION:	62.5%	%BENCH PRODUCTION:	61.5%