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Anvil Range
MINING CORPORATION

DY PROJECT

April, 1996

Fritz F. Prugger, P.Eng.
Consulting Mining Engineer

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TABLE OF CONTENTS.

1. PREAMBLE.

Drill hole 76X21, the discovery hole of the Dy deposit, was drilled in 1976. An additional 85 holes have been drilled since, and much information has been generated and accumulated in the 20 years since that first discovery hole. This mass of information is based on data from the 56 drill holes (see Chapter 6, Ore Reserves, on page 9) that intersected ore. These holes are widely spaced within the deposit as it is defined today.

I first started looking at this information in November, 1995 and have, since then, spent most of my time collecting and reviewing available data to improve my knowledge of the Dy deposit for the preparation of this report and for future use.

Some of the information is repetitious, but there is much that needs to be digested and passed on to the people whose task it will be to eventually make and run the mine and who will need this background to help them in their work.

Reserve calculations make up a great portion of the available information, however, the bulk of the material was produced by Canadian Mine Development (CMD) and consists of proposals that deal with cost estimates for shafts, ramps and contract mining.

It is apparent that a push to get underground and start production was on at certain times, dropped, then reactivated again. Much effort was spent during the five years, 1988-92, to cost out shaft and ramp access options as well as production alternatives. However, not enough time seems to have been spent evaluating the merits of the locations selected for underground access, and the difficulties of developing mine design and production plans on the meagre information available from the existing drill data.

The criteria for selecting the location of underground access seems to have been the shortest distance (which was probably considered to be equal to the shortest time and lowest cost, an assumption not always true), which initially took the form of a shaft-ramp combination north of the orebody, and the Blind Creek valley ramp access.

A shaft location in the barren zone in the centre of the orebody gained favour when drilling pilot holes for a shaft north of the orebody revealed difficult ground conditions in this area, and the plan to sink a shaft there was rejected.

There is enough consistency in the distribution of mineralization in certain areas of the deposit to convince me, as it has others, that a mineable orebody exists at Dy. To go underground and prove that this is the case, is the next step.

It is premature to develop mine plans and select mining methods at this stage. One must first concentrate on a good approach to getting underground. This is why my efforts in the present report concentrate on two areas only, namely the reviewing of existing information, and how best to get underground.

The outline of the report, as shown under the main headings of *Table Of Contents*, was generated at the beginning of my work. I have not changed the format, it can be utilized as a frame during the development of the Dy.

The Dy has an existing infrastructure in the community of Faro, an operating organization, the Faro concentrator, power and water supply, and highway and communications systems all in place. Within 5 years, the concentrator will need feed from the Dy orebody.

The Dy, if economically feasible, must come on stream before open pit reserves at Grum are exhausted. The workload involved in preparing the Dy for production is great; time goes by quickly. In this context, it is necessary that one person assumes all responsibilities as soon as possible for completing this task.

The Dy is, unfortunately, not close enough to surface for open pit mining. Underground production is more costly and more difficult than open pit exploitation. A mine with a production of 1.2 million tpy from underground is a good size operation. This is, in my opinion, the output one can expect from Dy in due time.

Additional open pit ore might be found at Grum and a small amount of ore will come from Grum underground. Nevertheless, there will be a shortfall in mill feed once the Grum pit ceases production. Therefore, now is the time for Anvil Range management to think of the possible options for overcoming the difference between mill capacity and mill feed available.

2. INTRODUCTION.

Anvil Range Mining Corporation engaged the services of Fritz F. Prugger (FFP), P. Eng., Consulting Mining Engineer with Asesoria Minera Internacional, S.L. (AMINSA) of El Escorial, Spain, to review and comment on all available information about the Dy deposit.

Some reports were sent to Spain in November, 1995. FFP collected information at the Grum mine office between January 08 and 22, 1996. During that time, three field visits were made to the Blind Creek valley and the Vangorda Plateau for familiarization with the terrain and to look for a suitable location for underground access.

Core from the ore zone and a fault was observed in the Grum core shed. An inspection of the Grum ramp was not made because walls and floor were covered with ice.

Copies of reports were taken to Spain, where the review was carried out, resulting in this report.

The report consist of the 13 chapters shown in the *Table Of Contents*, but concentrates on Chapter *9. Underground Access*, and Chapter *13. Commentary on available Information.*

The information review establishes the necessity of underground exploration as the next step in the Dy development. Several access possibilities are proposed in the reports listed, but none of them satisfy completely, in the opinion of this writer, all requirements of today.

Chapter 9, therefore, presents many details that support the proposed selection of a ramp from Blind Creek for underground exploration and from the Vangorda pit for main access and conveyor haulage.

3. SUMMARY AND CONCLUSIONS.

The Dy lead-zinc-silver-gold deposit is the fourth of a number of mineral deposits in the Anvil District of Faro, Yukon. They start with the Faro deposit in the northwest and strike in a southeasterly trend over a distance of some 25 km as shown in Figure 1.

The Faro, Grum and Vangorda deposits are close enough to surface for open pit mining. For the Dy, 600 to 800 m below surface, underground mining must be contemplated.

Faro is mined out and Vangorda is almost mined out, all ore supply for the 13,000 tpd concentrator at Faro now comes from the Grum pit, which will be exhausted in about five years. Feed for the concentrator must then come from underground, unless further open pit reserves can be found.

Some underground production from Grum is expected, however, the bulk will have to come from the Dy.

The Dy deposit was discovered some 20 years ago. Much information has been amassed since that time. Surface exploration has provided sufficient data to verify the existence of mineral resources and warrant underground exploration as the next step towards production.

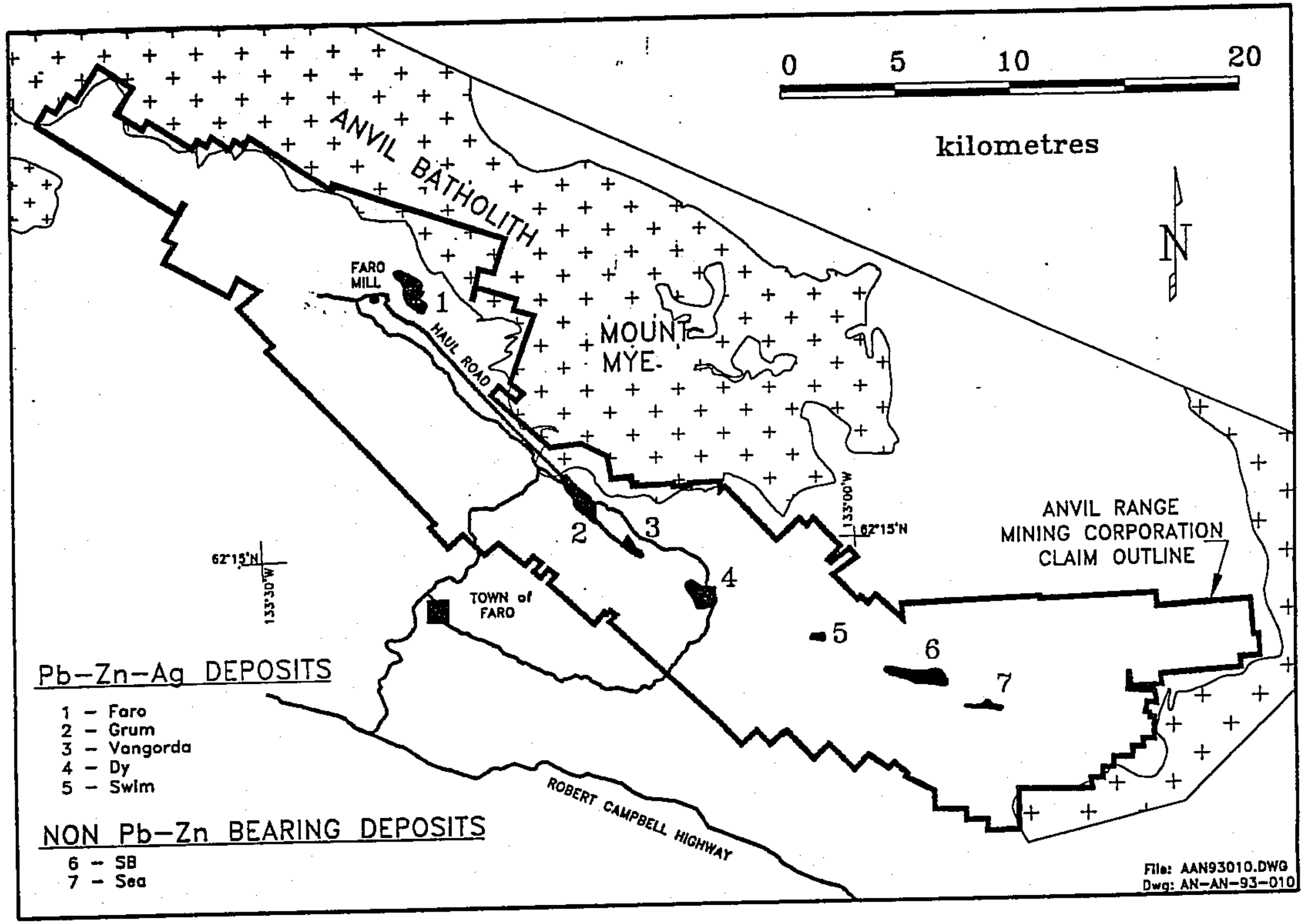
Examination of information available at Faro, and observation of the specific circumstances concerning Dy, has lead to the conclusion that the most desirable option for underground access for production to this orebody would be a ramp from the Vangorda pit.

A ramp from the Blind Creek valley to the southeast of the Dy, although shorter, is not favoured as a production ramp for environmental reasons. However, it serves well as access for underground exploration.

The Dy deposit is open in several directions. This makes the selection of a shaft location impossible without further surface drilling.

A review of drill information from the centre zone shows mineralization in all holes, and puts the existence of a barren zone in question. A shaft can not, therefore, be envisioned in this zone as proposed in previous studies.

A ramp from the Vangorda pit to the Dy creates the least hazard to the environment, is in line for ore transportation to the Faro concentrator and provides easy access for people from the existing Grum office and dry facilities. An access at Vangorda will defer the Vangorda closure plan and thus save up-front money. The possibility also exists of finding ore to the southeast of Vangorda towards Dy.


0 5 10 20
kilometres
N
ANVIL BATHOLITH
FARO MILL
1
HAUL ROAD
MOUNT MYE
133°00'W
62°15'N
2
3
4
5
6
7
ANVIL RANGE MINING CORPORATION CLAIM OUTLINE
62°15'N
133°30'W
TOWN of FARO
Pb–Zn–Ag DEPOSITS
1 – Faro
2 – Grum
3 – Vangorda
4 – Dy
5 – Swim
NON Pb–Zn BEARING DEPOSITS
6 – SB
7 – Sea
ROBERT CAMPBELL HIGHWAY
File: AAN93010.DWG
Dwg: AN-AN-93-010

4. RECOMMENDATIONS.

4.1 Select a competent Mining Engineer capable of managing all aspects of the Dy project and assign to him the responsibility of carrying out all tasks needed to bring the Dy into production. Add a Geologist to the team as soon as possible.

4.2 Decide to go underground and do underground exploration.

4.3 Provide financing for underground access and exploration and carry this out as Step I of the project.

4.4 Produce a bankable feasibility study.

4.5 Complete the mine infrastructure as Step II of the project and bring the mine into production.

5. ACTION PLAN.

See also Chapter 9 of this report on page 15.

- Design of underground access and preparation for advance.
- Advance of underground access.
- Design of underground exploration program.
- Carry out underground exploration program.
- Evaluation of underground exploration.
- Design and installation of mine infrastructure.
- Selection of mining methods and development of mine plans.
- Development of project and operating team.
- Selection and purchase of mining equipment.
- Development of safety manuals and standard procedures.
- Production.

6. ORE RESERVES.

Ore reserves have been calculated by various experts from different companies and can be accepted as they are now. Reserve summaries are shown in the enclosed Tables 2,3,4, and 5, taken from Curragh Resources Inc. (CRI) summary report, dated May 1993.

Table 6 gives a comparison of CRI 1991 calculations with those made by other experts in preceding years.

Ore reserves have been calculated with the data provided by surface drilling. Drill holes are some 150 metres apart. A total of 86 holes (85 in some reports) were drilled for the Dy project.

Drill hole purpose:

- 63 Testing main orebody
  (56 intersected mineralization)
- 14 Blind Creek ramp and portal
- 6 Pilot holes for shaft north of the orebody
- 3 Testing upper ore zone

86 Total

Table 7 shows mineable reserves using room and pillar mining with a recovery of 65% and a dilution factor of 10%, which, based on today's knowledge, are reasonable assumptions. In summary, these are:

Total Tonnes	% Pb+Zn	% Zn	% Pb	g/t Ag	g/t Au
14,578,000	**12.57**	**7.26**	**5.31**	**78.15**	**0.81**

The deposit is open in several directions, as shown in the sketch on the following page. The reserve picture can be expected to improve as more work is done within the existing outline of the deposit, and through step out exploration.

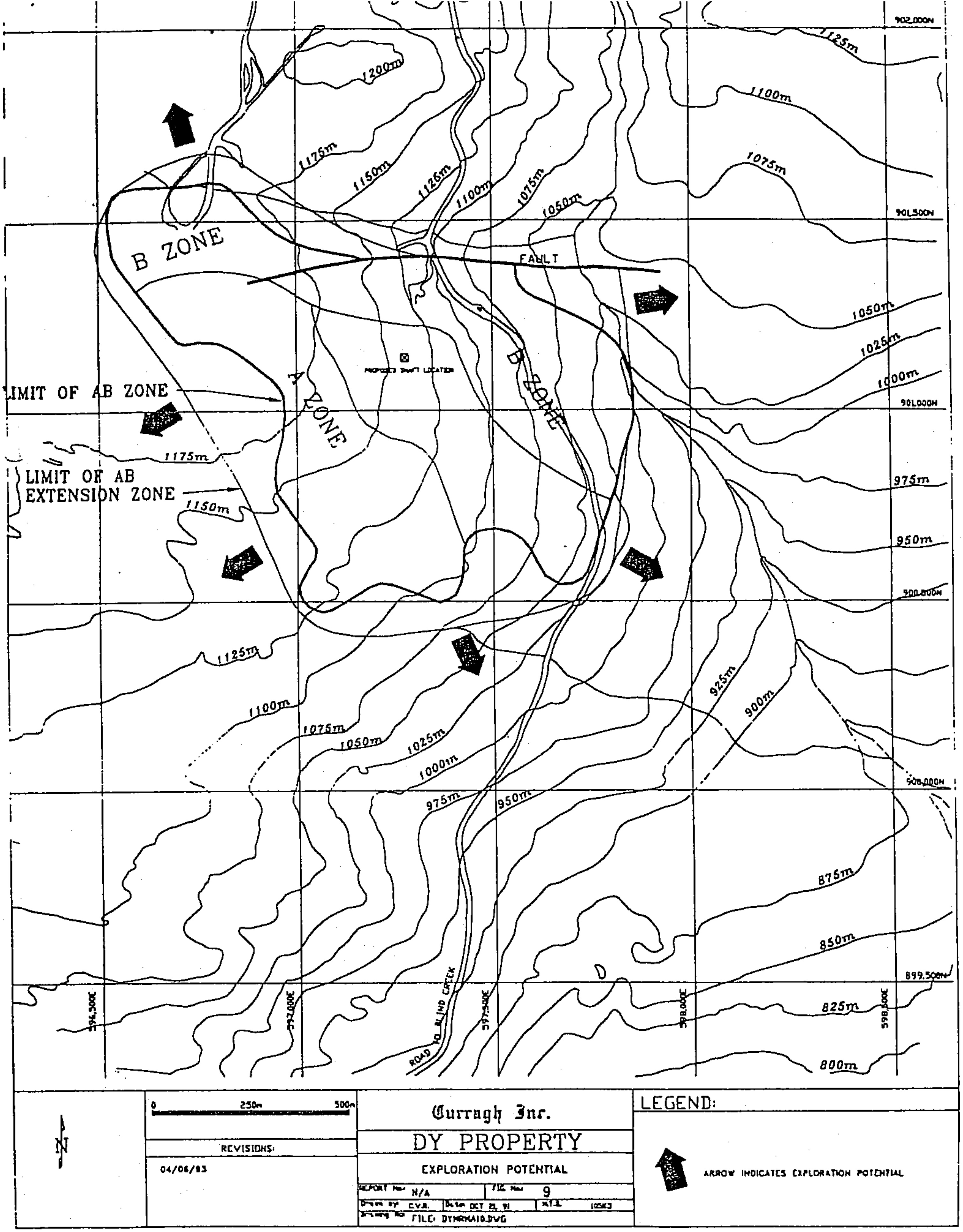

B ZONE
FAULT
A ZONE
B ZONE
LIMIT OF AB ZONE
LIMIT OF AB EXTENSION ZONE
1200m
1175m
1150m
1125m
1100m
1075m
1050m
1025m
1000m
975m
950m
925m
900m
875m
850m
825m
800m
902,000N
901,500N
901,000N
900,500N
900,000N
899,500N
596,500E
597,000E
597,500E
598,000E
598,500E
ROAD TO BLIND CREEK
N
0
250m
500m
REVISIONS:
04/06/93
Curragh Inc.
DY PROPERTY
EXPLORATION POTENTIAL
N/A
9
C.V.R.
OCT 21, 91
LEGEND:
ARROW INDICATES EXPLORATION POTENTIAL

TABLE 6. Comparison of 1991 Mineral Inventory to Previous Work

Calculation	Cutoff (%Pb+Zn)	Tonnes	% Pb	% Zn	Pb+Zn	Ag (g/mt)	Au (g/m
CRI 91	6	41,555,000	4.12	5.72	9.84	61.9	0.58
CRI 91	8	24,947,000	5.21	7.01	12.22	77.4	0.85
CRI 91	9	21,356,000	5.54	7.33	12.87	81.1	0.87
Cyprus Anvil Hall 81	9	21,334,127	5.68	6.95	12.63	81.6	n/a
" " Rollings 82	9	21,059,980	5.54	6.74	12.28	83.77	0.95
Kilborn 89	9	20,114,825	5.47	6.77	12.44	84.5	0.91

EXPLORATION POTENTIAL

There is considerable potential to extend the deposit by additional stepout drilling. The deposit is closed off by drilling only locally (Figure 9) and it is likely that additional mineralization will be found.

Exploration should proceed to the east of the deposit at least as far as the fault paralleling Blind Creek (not indicated on figures) since some of the peripheral intersections in that area are quite good and drilling in that direction was terminated by Cyprus Anvil because of concern over shallowly dipping dykes or sills interfering with the ore horizons.

Additional stepout drilling will be required to the west and south of the deposit. These areas are deep and larger drills than used to date will be required. This area is of lower priority as this part of the deposit could only be brought into production late in the likely mine life.

In order to quantify the potential additional mineralization that might be added to the Dy deposit, it has been hypothesized that along each of the south and east margins, a 200m further extension is possible along a 600m length of the deposit. No allowance has been made beyond the AB Extension Zone on the north or west because structural complications related to faults are expected in those directions. If average thickness is 5 to 6 metres, this would represent approximately an additional 5 million tonnes of potential mineralization. Approximately 12 holes 600 to 1100m deep (a total of 10,500m or approximately $1.6 million) would be needed to test this area.

There has been considerable speculation over a second deposit approximately one or two km to the north of the Dy beneath the thick sequence of mafic igneous rich Vangorda formation preserved there. These concepts should be evaluated by deep drilling in that area.

TABLE 2. Mineral Inventory for AB Zone, Dy Deposit In-situ - Undiluted

	Tonnes	%Pb+Zn	%Pb	%Zn	Ag (g/mt)	Au (g/mt)
6% Pb+Zn Cutoff						
Probable	24,949,000	9.70	4.21	5.49	63.0	0.67
Possible	10,348,000	10.43	4.01	6.42	61.3	0.62
Total	35,297,000	9.91	4.15	5.76	62.5	0.60
8% Pb+Zn Cutoff						
Probable	14,895,000	12.06	5.43	6.63	80.0	0.87
Possible	6,720,000	12.59	4.84	7.75	73.4	0.80
Total	21,705,000	12.23	5.25	6.98	78.0	0.84
9% Pb+Zn Cutoff						
Probable	13,133,000	12.58	5.71	6.87	83.1	0.86
Possible	5,389,000	13.62	5.26	8.36	78.2	0.85
Total	18,522,000	12.88	5.58	7.30	81.7	0.85

TABLE 3. Mineral Inventory for the AB Extension Zone, Dy Deposit In-situ, Undiluted, Possible Mineralization

	Tonnes	%Pb+Zn	%Pb	%Zn	Ag (g/mt)	Au (g/mt)
6% Pb+Zn Cutoff	3,746,000	10.30	4.07	6.23	61.9	0.63
8% Pb+Zn Cutoff	2,094,000	13.39	5.28	8.11	79.4	0.94
9% Pb+Zn Cutoff	1,904,000	13.92	5.52	8.40	82.0	0.98

TABLE 4. Mineral Inventory Above and Below the AB Zone, Dy Deposit In-situ, Undiluted Possible Mineralization

6% Cutoff	Tonnes	%Pb+Zn	%Pb	%Zn	Ag (g/mt)	Au (g/mt)
Above	1,828,800	7.77	3.81	3.96	50.6	0.36
Below	683,500	9.05	3.60	5.45	58.9	0.56
Total	2,512,300	8.12	3.75	4.37	52.9	0.41
8% Cutoff						
Above	606,500	10.27	4.96	5.31	65.2	0.74
Below	541,900	9.78	3.91	5.88	62.9	0.63
Total	1,148,400	10.04	4.46	5.58	64.1	0.69
9% Cutoff						
Above	606,500	10.27	4.96	5.31	65.2	0.74
Below	323,300	10.78	4.50	6.29	74.9	0.89
Total	929,800	10.45	4.80	5.65	68.6	0.79

TABLE 5. Dy Deposit Summary of Mineral Inventory for Entire Deposit
In-situ, Undiluted

	Category	Tonnes	%Pb+Zn	%Pb	%Zn	Ag (g/mt)	Au (g/mt)
6% Cutoff							
AB Zone	Probable	24,949,000	9.70	4.21	5.49	63.0	0.67
AB Zone	Possible	10,348,000	10.43	4.01	6.42	61.3	0.62
AB Extension	Possible	3,746,000	10.30	4.07	6.23	61.9	0.63
Above & below AB	Possible	2,512,000	8.12	3.75	4.37	52.9	0.41
Subtotal	Probable	24,949,000	9.70	4.21	5.49	63.0	0.67
Subtotal	Possible	16,606,000	10.05	3.98	6.07	60.2	0.59
Grand Total	Probable+Possible	41,555,000	9.84	4.12	5.72	61.9	0.65
8% Cutoff							
AB Zone	Probable	14,985,000	12.06	5.43	6.63	80.0	0.87
AB Zone	Possible	6,720,000	12.59	4.84	7.75	73.4	0.80
AB Extension	Possible	2,094,000	13.39	5.28	8.11	79.4	0.94
Above & below AB	Possible	1,148,000	10.04	4.46	5.58	64.1	0.69
Subtotal	Probable	14,985,000	12.06	5.43	6.63	80.0	0.87
Subtotal	Possible	9,962,000	12.47	4.89	7.58	73.6	0.82
Grand Total	Probable+Possible	24,947,000	12.22	5.21	7.01	77.4	0.85
9% Cutoff							
AB Zone	Probable	13,133,000	12.58	5.71	6.87	83.1	0.86
AB Zone	Possible	5,389,000	13.62	5.26	8.36	78.2	0.85
AB Extension	Possible	1,904,000	13.92	5.52	8.40	82.0	0.98
Above & below AB	Possible	929,800	10.45	4.80	5.65	68.6	0.79
Subtotal	Probable	13,133,000	12.58	5.71	6.87	83.1	0.86
Subtotal	Possible	8,223,000	13.33	5.27	8.06	78.0	0.87
Grand Total	Probable+Possible	21,356,000	12.87	5.54	7.33	81.1	0.87

Table 7
Estimate of Diluted Recoverable Mineralization
For the Dy Mineplan

Phase	Tonnes	Pb+Zn (%)	Zinc (%)	Lead (%)	Silver (g/t)	Gold (g/t)
1	4,830,584	13.11	8.43	4.68	73.75	0.61
2	1,449,697	12.94	6.97	5.97	88.93	0.87
3	8,298,161	12.20	6.64	5.56	78.84	0.91
Total	14,578,442	12.57	7.26	5.31	78.15	0.81

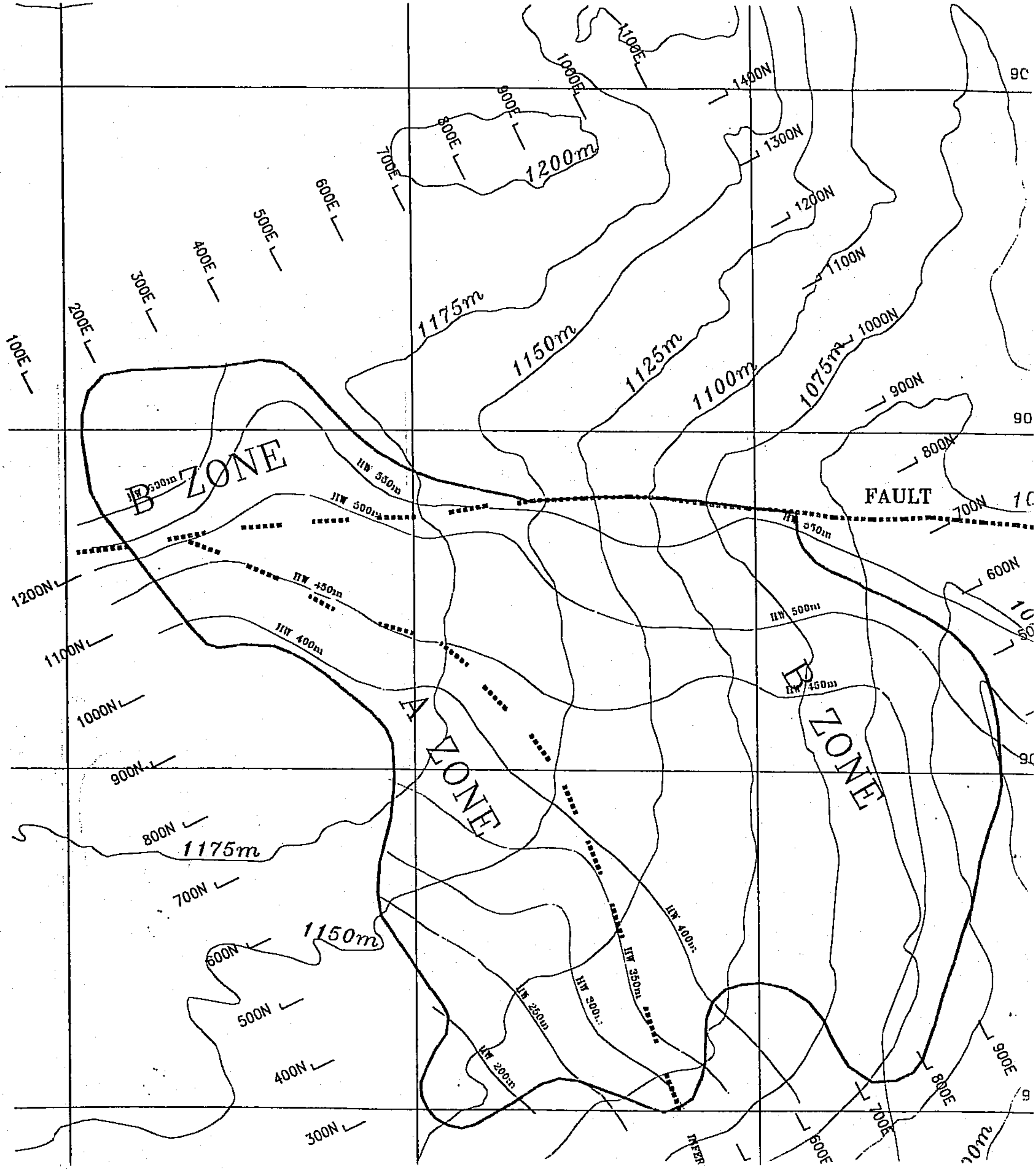
7. THE OREBODY

The outline of the Dy orebody, so far, has been characterized by the lack of drilling along its perimeter. The deposit is open in several directions. Continuity of mineralization exists throughout certain areas of the deposit as it is known today.

The orebody is complex and cannot be judged on the information available from surface exploration only. Conditions at Faro, Grum, and Vangorda give an indication of the characteristics of Dy, but only an underground assessment will provide an adequate picture.

The orebody is located at a depth of 600 to 900 m below surface. The main body extends about 1,000 m from west to east and 850 m north to south and has a general dip of 20 to 35 degrees southeast. There is, in addition, the northwest corner, measuring 350 m from west to east and 230 m from north to south.

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. Phyllites and schists comprise the hanging wall. Strength of these rocks must be considered poor to very poor.



B ZONE
A ZONE
B ZONE
FAULT
1200m
1175m
1150m
1125m
1100m
1075m
1175m
1150m
100E
200E
300E
400E
500E
600E
700E
800E
900E
1000E
1100E
1400N
1300N
1200N
1100N
1000N
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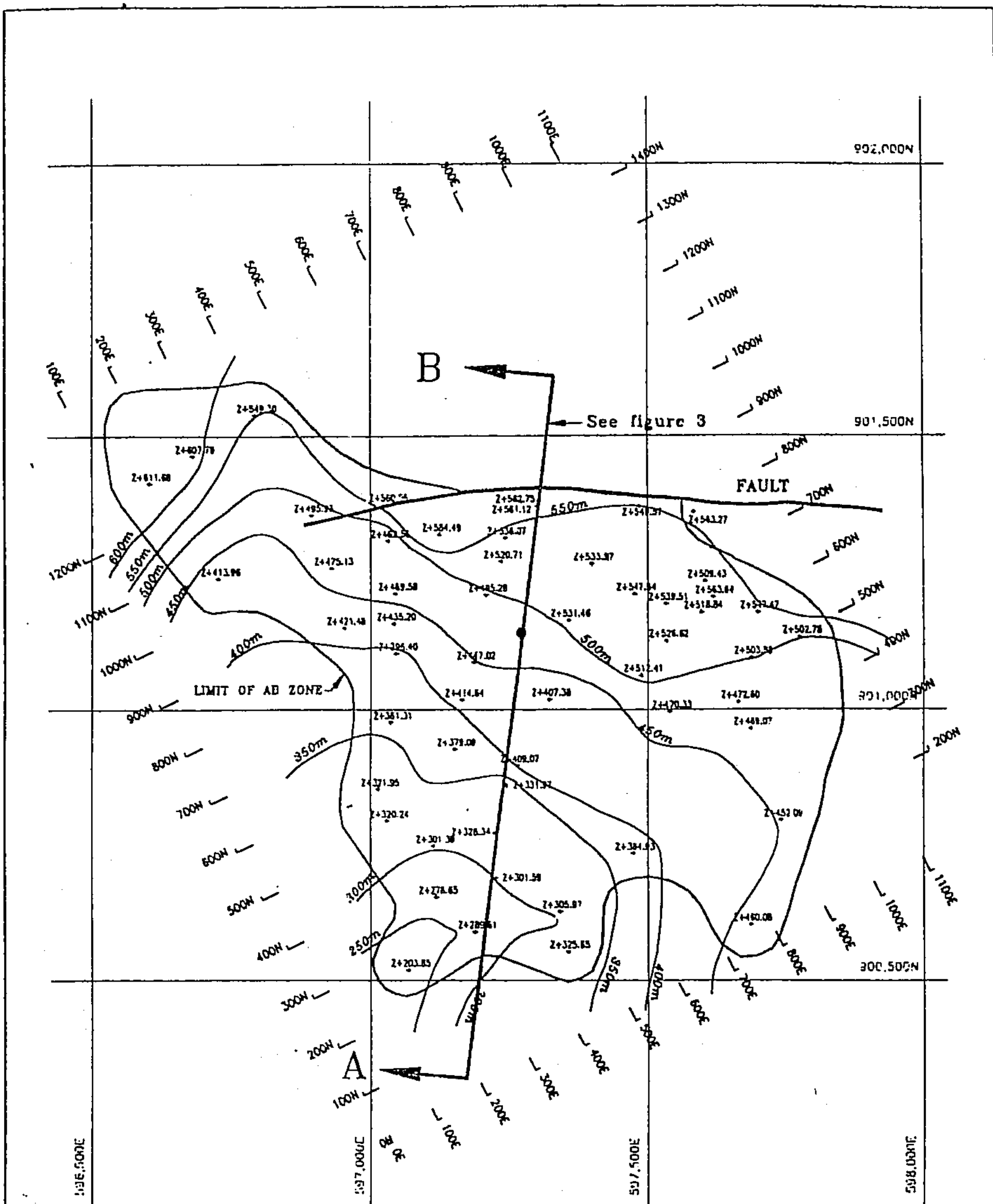
800N
700N
600N
500N
400N
300N
900E
800E
700E
600E
HW 550m
HW 500m
HW 450m
HW 400m
HW 250m
HW 200m
HW 350m
HW 400m
HW 500m
HW 450m
HW 550m

Dy Orebody



Curragh Inc.

DY PROPERTY

AB ZONE HANGINGWALL CONTOURS

REVISIONS

04/06/93

N/A

4

OCT 21, 91

105K3

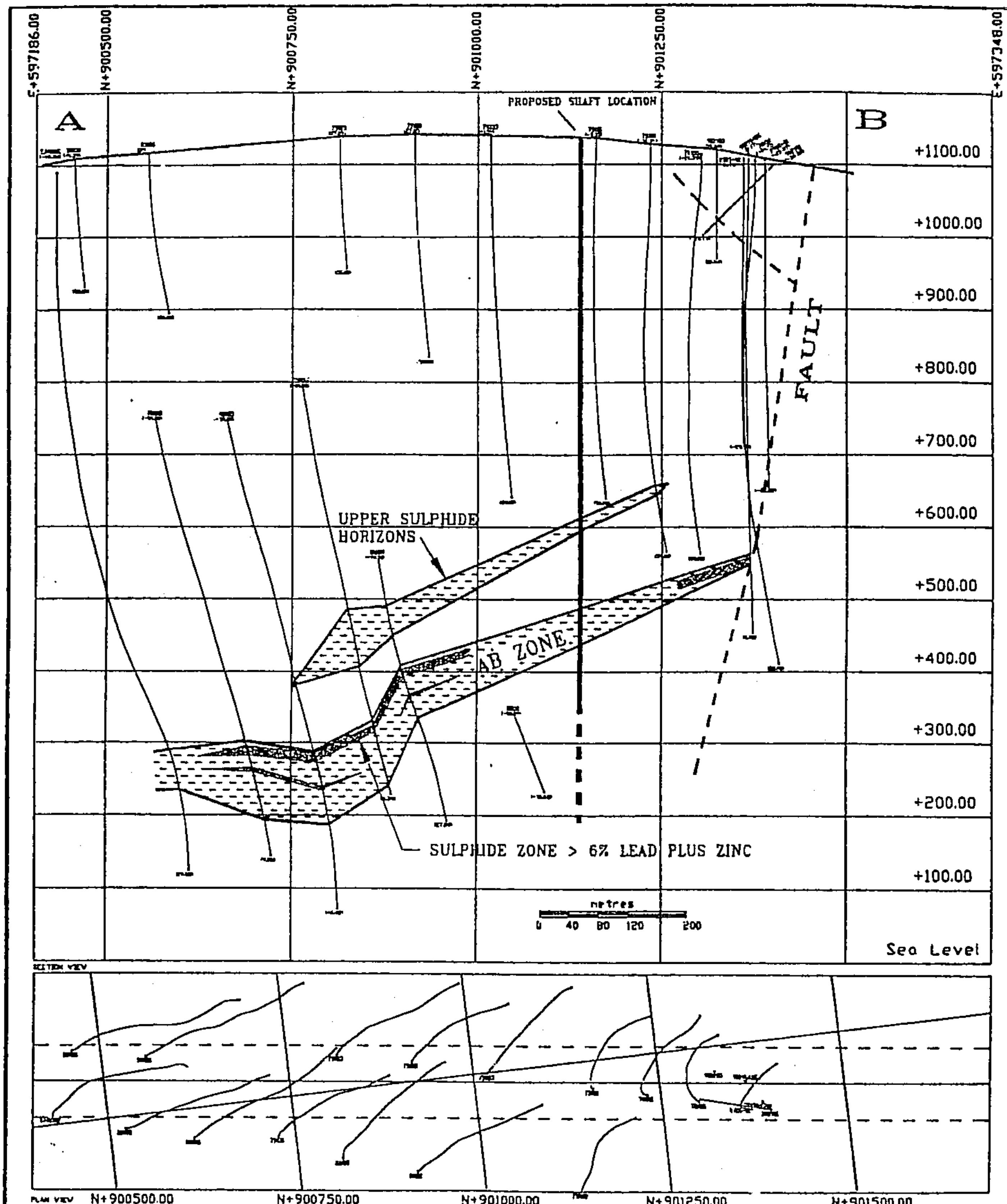
ABCONTSB.DWG

LEGEND:

Contour Interval = 50m

Contours are metres above sea level

● PROPOSED SHAFT LOCATION



Curragh Inc.
FIGURE 3

DY DEPOSIT
VERTICAL SECTION
SHAFT LOCATION STUDY

8. HYDROLOGY.

Data on hydrology is hardly existent and needs to be developed. It has been suggested that most of the rock mass will not make water.

Faults can be expected to be significant aquifers, which could discharge large amounts of water into mine openings until pressure is relieved. They will drain and are assumed not to recharge.

EBA Engineering has estimated that, based on double packer percolation tests, phyllites will be relatively impermeable (10^{-6} cm/sec), whereas fault zones will be significantly more permeable (10^{-4} cm/sec).

All Dy mineralization is below the level of Blind Creek and will not enter the drainage system to the creek by natural flow.

Information on the flow of groundwater needs to be collected as the deposit is developed.

9. UNDERGROUND ACCESS AND TRANSPORTATION.

9.1 Introduction.

Underground access, ore and waste hoisting and transportation of people, materials and equipment in and out of the mine, are discussed in this chapter.

Every producing underground mine must have at least two entries. These consist of vertical shafts, declines (ramps) of adits when the mine is located above the bottom of a valley.

Vertical shafts in combination with horizonal drifts have long been the accepted method of transporting people, ore and materials in mines. It can be automated, and usually provides low cost ore transportation to surface. This is especially true in the case of deep mines.

There are, however, the disadvantages of high up-front capital requirements and long lead times for the completion of shaft sinking and headframe and hoist installations. This system is not flexible and can not be changed easily for the purpose of future production increases.

The development of rubber tired diesel vehicles in the 1960's for transportation of men and materials and for ore haulage, made the need for a flexible system of movement between mine levels as well as from surface to underground apparent. Consequently, ramps between mine levels and from surface to underground have become an accepted feature in mining.

The ramp has opened new possibilities for mining. Up-front capital expenditure can be reduced, if the ramp can be driven along known or expected parts of an ore reserve, and revenue created by mining, if in the process of driving the ramp new parts of an orebody can be found.

In the 1980's, this concept of opening up an orebody was referred to in Sweden as "Cash Flow Mining". The Swedes made a mobile concentration plant part of this concept. Considering the high cost of money, this philosophy makes good sense.

The same practice was followed thousands of years ago by the first miners who exploited flint as well as metallic minerals from surface outcrops and with the depletion of the surface deposit continued mining in depth.

Most mining started on surface or close to surface and progressed in depth as this became necessary and was within the range of the miner's imagination and the resources available to him.

This author considers himself very fortunate in having been able to visit salt mines developed by the Celts some 3000 years ago in the Austrian Alps, mines developed by the Phoenicians in the Iberian peninsula, Roman gold mines in Spain, and iron and silver mines from Roman times in Austria.

Miners in ancient times showed great concern for economics: Exploration drifts in waste rock were advanced at the smallest section possible; the Romans brought chestnuts from Italy and planted them in Spain in order to feed thousands of slaves that worked underground in their Spanish gold mines.

The miners of old followed mineral occurrences to depth as exploitation advanced. It is our intention to learn not only from our forefathers in mining of centuries ago, but also from what has been done at the Dy so far.

A step by step advance to depth with the smallest necessary capital outlay up front, while generating cash through the production of ore as one drives a ramp, are attractive ideas in the setting of the Dy deposit.

There is a chance of finding ore between the Vangorda pit and the Dy orebody in spite of the fact that the few holes that have been drilled in this area so far have not come up with intersections of interest.

A ramp from the Vangorda pit to the Dy orebody would be in line of the general mineralization trend in the Faro area. The proposed portal at Vangorda would be in massive sulphides that could contain mineralization as one continues further to the southeast. The Dy orebody is open in the direction of the Vangorda pit to the northwest, from where the ramp would approach.

Six shaft locations as well as three locations for ramp portals have been considered by previous owners. The so-called barren zone in the centre of the orebody and the valley of the Blind Creek were the preferred locations for the shaft and possible ramp, respectively.

The latest study is dated December 1992, and shows a shaft with a diameter of 6 metres as the chosen means of access to the Dy orebody, with a bored raise for ventilation and emergency exit.

This chapter outlines the reasons why the Dy deposit should be accessed by a ramp and not by shaft.

9.2 Summary and Comments.

A vertical shaft is rejected in this report as first access to the Dy deposit for the following reasons:

- High upfront capital

- Long construction time

- Lack of shaft location until the deposit has been explored further

    - The deposit is open in several directions

    - Bad ground conditions north of the deposit

    - Uncertainty about the barren zone in the centre of the orebody

    - Required size of shaft safety pillar under Dy conditions

    - Substantial loss of ore reserves with the shaft pillar within the orebody

- A shaft, if desired, can be constructed cheaper with an underground access available, using raise boring methods, and muck removal through a ramp

At present, the Blind Creek valley and the Vangorda pit are possible choices for a ramp portal. Other possible locations still need to be investigated.

The anticipated portal of the Blind Creek ramp is at elevation 835 m, the lowest possible location for the starting point of a ramp. Thus, it would be shorter and cheaper than any other underground access by decline. The entrance would be near Blind Creek, a salmon spawning river. The proximity of the ramp to the river would be an environmental hazard, especially after production start-up.

At the Vangorda pit, elevation 1094 m has been tentatively chosen as the starting point of a ramp. The main reason for this is the chance of finding ore in a southwesterly extension of an outcrop of massive sulphide at this elevation. Decommissioning of the Vangorda pit was a factor of consideration as well.

A starting point at a lower elevation in the pit might be possible, and would reduce the cost of the ramp considerably. This possibility will be investigated further by the writer.

9.3 Recommendations.

- Ramp access should be used to go underground for further exploration of the Dy deposit and preparation for production.
- Obtain realistic cost estimates for ramp advance.
- Drive the ramp.
- Do underground exploration.
- Develop concepts for Phase II.

9.4 Action Plan.

- Decision to accept ramp option for underground access.
- Investigate possibility of starting ramp at Vangorda Pit at a lower elevation.
- Investigate other possible locations.
- Decision on ramp location at Vangorda pit, Blind Creek, or other suitable location still to be found.
- Planning for ramp advance.
- Prepare tender documents.
- Obtain quotes.
- Award contract.
- Manage ramp advance.
- Prepare design and budget for underground exploration drilling.
- Preparation for production.

9.5 General Considerations.

This section provides some general information required when contemplating underground access options. The reason for making an underground access is often the need for further investigation of the orebody from underground. For this purpose, the opening could be very small in order to save on cost. This changes when hoisting of ore and waste, transportation of people, handling of supplies and equipment and ventilation requirements enter the picture.

Vertical shafts and ramps, or a combination thereof, are commonly used for underground access, transportation and ventilation.

Deep ore bodies are generally accessed by shafts.

A ramp is often driven as the initial entry. Once an underground passageway is secured, raise boring methods can be applied for shaft construction with muck removal through the ramp.

9.5.1 Vertical Shafts.

Vertical shafts for hardrock mining in good ground are traditionally rectangular and timbered, or lined with concrete sets for the installation of guides, manways etc.

In poor ground, shafts are usually round and concrete lined. Round shafts are either sunk by conventional means (drilling and blasting) or drilled from surface as blind shaft or constructed with raise boring methods and subsequent enlargement, if underground access is available.

Shafts in coal and potash mines are constructed with consideration for the special ground conditions that usually prevail in these deposits.

Size and diameter of a shaft depend on its function. This could include the transport of men, materials and equipment, production, backfill or ventilation. Diesel fuel, hydraulic oil and backfill are often brought into the mine through boreholes from surface.

Shaft sinking is normally carried out by companies specializing in this field. The work requires skilled people and particular equipment. At least one pilot hole is required to provide geological information for the shaft design.

Characteristics for shaft access:

- Capital Upfront
- Pilot hole
- Geological study
- Design by experts
- Civil Engineering design for
  - hoist foundations
  - headframe foundations
  - headframe
- Installation of
  - hoists and winches for shaft sinking
  - headframe
- Special know-how and skill for the execution of shaft sinking

9.5.2 Ramps.

A Ramp, Decline or Incline is an inclined access to a mine.
There are Straight Ramps, Spiral Ramps and Switchback Ramps. The gradient of ramps generally varies from 8 to 20%, depending on the purpose of the ramp.

Alternatives for ramp drivage include:

- **Conventional Drilling and Blasting** in combination with

  - Scooptrams
  - Scooptram and Diesel Truck(s)
  - Scooptram and Electric Truck(s)
  - Scooptram and Kiruna Container Haul
  - Scooptram and Feeder Breaker and Conveyor
  - In-line Loader with Diesel Truck
  - In-line Loader with Mobile Crusher and Conveyor

  In-line Loaders on the market are:
  - Häglund Loader, Sweden
  - Voest Alpine Loader, Austria
  - Oscillating Loader by Continuous Mining in Sudbury, Canada
  - ITC-Tunnel Loader, Switzerland
  - Gathering Arm Loaders, USA

- **Mechanized advance** with a full face Tunnel Bore Machine (TBM) or Roadheader Type Continuous Miner with a continuous haulage system or truck haulage for muck removal.

9.5.3 Transportation

Transportation includes the hoisting of ore and waste, transporting people, materials (supplies, backfill etc) and equipment in and out of the mine.

In case of vertical shaft access, the most common way is skip hoisting of ore and waste plus cage transportation. The mine infrastructure normally consists of ore and waste passes, horizontal transportation by rail car or truck, an underground crusher with loadout conveyor and loading pockets for ore and waste. Headframe bins or stock pile areas on surface are required for dumping of ore and waste on surface for further transportation to the concentrator or waste dump.
In case of ramp access various possibilities for ore and waste haulage exist:

- Haulage with diesel trucks

- Haulage with electric trucks

- Conveyor haulage
    - Standard conveyor, various flights
    - Standard conveyor with booster drives
    - Long flight with steel cord belt
    - Cable belt conveyor

Primary crushing underground is required for conveyor haulage and standard skip hoisting. Uncrushed ore can be brought to surface with diesel or electric trucks.

Inclined shaft hoisting is another method of hoisting uncrushed ore to surface that uses a skip travelling on rails. This was quite common in mines before ramp haulage with trucks or conveyors was introduced and is still regarded as a valid option in some new mines today. An example of this is the Kylylahti mine in eastern Finland, where hoisting of uncrushed ore with a skip on rails in a shaft with a 35 degree incline from the orebody at a depth of 500 to 700 metres has been selected as the option best suited to the specific conditions of this ore deposit.

Pumping is another possibility for bringing ore to surface. This method requires crushing and grinding of ore underground. It is economically feasible in mines of great depth where large quantities of water must be pumped to surface.

Each method has advantages and disadvantages, and must be considered with the specific local conditions in mind.

Transportation of people, materials and equipment in case of shaft access is via service cage. In case of ramp entry, man carriers and special diesel vehicles are utilized. Drillholes and bored raises are often used to bring diesel fuel, hydraulic oil or backfill into the mine or for ventilation and emergency exit.

9.6 Access Alternatives mentioned in the Information available at Faro.

An access possibility by shaft and ramp is shown in a drawing by Cyprus Anvil from 1982. Both shaft and ramp access to the Dy deposit were mentioned and costed out in numerous proposals presented by Canadian Mine Development between 1988 and 1992.

The Rescan report, dated March 1991, assumes a ramp from the Blind Creek valley as the accepted mine entry and describes in detail measures to prevent spillage of acidic water and acid generating waste rock and ore into Blind Creek.

The report by Fox Geological Consultants, October 1992, shows the Blind Creek ramp, a production shaft in the centre of the orebody and ventilation and backfill raises close to the shaft location. A 10-degree cone is shown as safety pillar for the shaft. The report points out the need for further investigation of the apparently barren zone in the centre of the orebody. A study of the size of the shaft safety pillar is also suggested.

The drawing by Cyprus Anvil Mining Corporation dated April 1982, shows a rectangular three-compartment vertical shaft as well as a ramp. Both are located on the Vangorda Plateau to the northwest of the Dy deposit at an elevation of some 1,200 metres above sea level. The shaft was meant to reach the orebody at an elevation of about 400 metres.

The shaft was outlined to be 800 metres deep. The ramp is shown to have a total length of some 5,400 metres.

There are seven proposals from CMD (Canadian Mine Development) on file at Faro. These proposals were developed between June 1988 and December 1992 and not only consider the access to the orebody, but include underground exploration drilling and mining as well. Section 13.3 of this report contains details of the proposals by CMD.

The CMD data reflect the thinking and the objectives of the property owners at the time. The emphasis is often on getting into production as quickly as possible without sufficient consideration of other factors.

Rectangular shafts, circular shafts of different diameters and declines with combinations thereof are discussed.

Among several possible shaft locations, a point north of the deposit at an elevation of around 1100 m was first selected. Depth of this shaft was planned to be 660 m with the shaft bottom at elevation 440 m. Drilling of shaft pilot holes proved difficult, the holes entered a major fault to the north of the orebody.

A location in the centre of the orebody, in the zone of presumably barren massive sulphides, was then favoured as shaft site. Pilot holes have not been drilled at this location.

In one proposal, the ramp portal was to be located to the north of the orebody, at elevation 925 metres. In this case, the ramp would have had to measure some 2,600 metres at a gradient of 20% to reach the orebody. It was intended for underground exploration and had a profile of 4.3 metres x 4.60 metres. The ramp was designed to intersect the future shaft at a point in midshaft to allow shaftslashing into a pilot hole and muck removal from underground through the ramp.

A point at the low elevation of 835 metres was then selected for the portal of the ramp in the valley of Blind Creek. This ramp would need to be 1,730 metres long at a gradient of 20% to reach the orebody at an elevation of 490 metres. The location of the Blind Creek portal is to the southeast of the orebody in the direction opposite to the Faro concentrator.

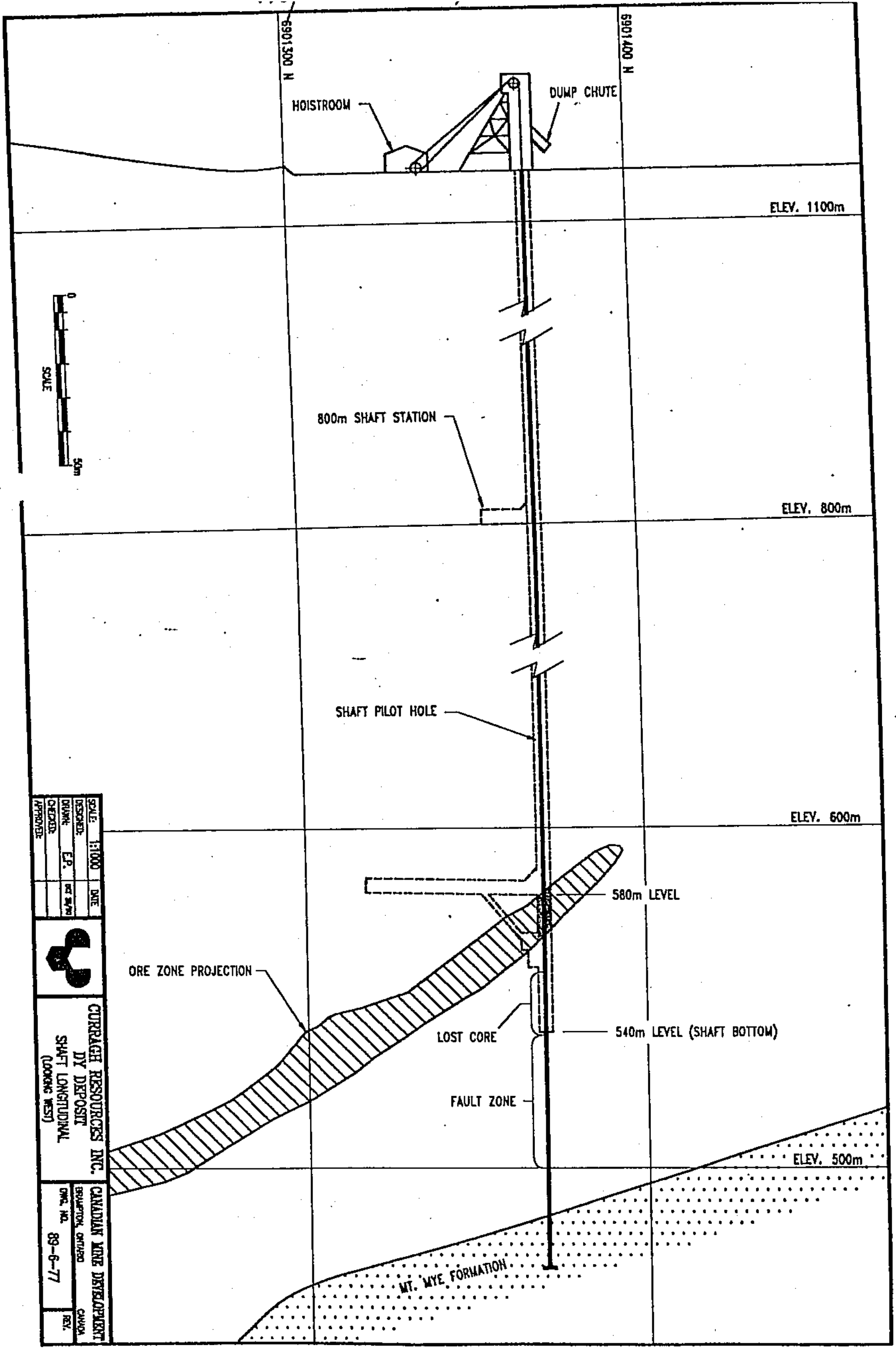
The alternatives presented by CMD include several combinations for getting underground. In some the ramp is driven first, in others shaftsinking is done before driving the ramp, others consider shaft access only.

Cost estimates for different shaft sizes and for ramp drivage have been made. Costs vary in each case and depend on a number of details. A metre of shaft is generally shown to cost ten times more than a metre of ramp. Approximate figures are $ 12,000 per metre of shaft and $ 1,200 per metre of ramp, for excavation only.

A few examples of proposals are shown here to give the reader a feel for the cost estimates presented in the CMD offers. The shaft cost estimates shown are taken from the proposals of October 1990-February 91, and December 1992. Estimates for circular concrete shafts with a diameter of 4.0, 4.5 and 4.75 metres and a depth of 563 metres are included in the October 1990-February 91 offer. The December 1992 proposal includes an estimate for a circular concrete shaft of 6 m diameter and a depth of 737 m located in the centre of the orebody.

- Circular Concrete Shaft 4.25 m, 3500 tpd, October 1990.
  Total depth is 563 m, exploration from hanging wall drifts.

  The location of the Shaft, proposed in October 1990, is north of the orebody. Pilot holes indicated a major fault zone at elevation of 550 metres. For this reason the planned shaft depth had to be reduced. Underground exploration was to have been carried out from the hanging wall.


6901300 N
6901400 N
HOISTROOM
DUMP CHUTE
ELEV. 1100m
SCALE
0
50m
800m SHAFT STATION
ELEV. 800m
SHAFT PILOT HOLE
ELEV. 600m
580m LEVEL
ORE ZONE PROJECTION
LOST CORE
540m LEVEL (SHAFT BOTTOM)
FAULT ZONE
ELEV. 500m
MT. MYE FORMATION
SCALE: 1:1000
DESIGNED:
DRAWN: E.P.
CHECKED:
APPROVED:
DATE
CURRAGH RESOURCES INC.
DY DEPOSIT
SHAFT LONGITUDINAL
(LOOKING WEST)
CANADIAN MINE DEVELOPMENT
BRAMPTON, ONTARIO
CANADA
DWG. NO.
89-6-77
REV.

Surface Installations		$ 3,726,465
Shaft Sinking 1083-800	283 m á 11,703	
800-560	240 m á 11,703	
560-520	40 m á 11,703	
Depth	563 m	$ 6,588,789
Shaft Brattice Lining		$ 124,423
Station Excavation and Equipping at 800, 560 and 520 LP		$ 1,480,688
Other (air heating etc)		$ 728,770
Total Sinking Cost		$ 8,922,670
Total Plant and Sinking Cost		$ 12,649,135
Contractors G & A 11.5%		$ 1,454,649
Contractors Fee 8.5%		$ 1,075,176
Total Contractors Cost		$ 15,178,950
Contingency		$ 1,000,000
Total Contractors Construction Cost		$ 16,178,950
Owners Power Cost		$ 316,000
Total Estimate		$ 16,494,950

Say $ 16.5 Million. Cost equals 29,300 $/m

A cost estimate for a 4.75 m circular concrete shaft was carried out at the same time. The difference is the cost of shaft sinking (=excavating), which amounts to $ 14,027 per metre. Total shaft cost is estimated to be $ 18,329,561 (say $ 18.4 Million).

A cost estimate for a 6 m concrete lined shaft located in the centre of the deposit was carried out in December 1992. The shaft has a total depth of 777 m to bottom out at elevation 350 m. In this estimate, a cost of shaft sinking of 12,570.93 $/m is used. The total cost of the surface plant, sinking and lining 777 m of shaft and the excavation of four stations is $ 12,840,968. Copies of the original estimates are included in the following pages.

Contractor's general overhead, contractor's fees and contingency are added below to bring the final cost of this shaft to a level comparable to the cost estimates presented for the smaller shafts in October 1990.

Total plant and Sinking Cost		$ 12,825,348
Contractor's G & A	11.5%	$ 1,474,915
Contractor's Fee	8.5%	$ 1,090,155
Total Contractor's Cost		$ 15,390,418
Contingency		$ 1,000,000
Owner's Power Cost		$ 316,000
Total Estimate		$ 16,706,418

Cost Summary - Shaft (for general information only):

Diameter m	Depth m	Total Cost Mill.$	Sinking & Lining $/m	Total Project $/m
4.00	563	15.6	11,204	27,709
4.25	563	16.5	11,703	29,298
4.75	563	18.4	14,027	32,557
6.00	777	16.7	12,571	21,501

There is a discrepancy between the cost quoted for the smaller shafts and the 6 metre one. The 6 m shaft was part of a contract mining offer, which might be the reason for the substantial difference in unit price. There could also be a personal factor involved. The first estimates are signed by J.M. Proudfoot for Canadian Mine Development and addressed to M.H. Pelley of Curragh Resources Incorporated. The proposal for the 6 m shaft is signed by J.I. Tatak for CMD and addressed to C.K. Benner of CRI.

DY PROJECT
EXPLORATION SHAFT & DEVELOPMENT
PHASE 1 - COST ESTIMATE

SHAFT BOTTOM - 350 ELEV.
3 LEVEL EXPLORATION (480, 440, 400)

Item No.	Description			Unit	Unit Cost	Total
1.0	Shaft Pilot Hole			L.S.	$300,000	$300,000 *
2.0	Road Improvements			L.S.	$200,000	$200,000 *
3.0	Power Transmission Line			L.S.	$250,000	$250,000 *
4.0	Surface Plant Set-up					
	Surface Site Preparation			L.S.	$91,707	$91,707
	Shaft Collar			L.S.	$245,794	$245,794
	Headframe Civil Work			L.S.	$886,605	$886,605
	Hoist Civil & Mechanical Work			L.S.	$1,441,989	$1,441,989
	Site Utilities			L.S.	$628,423	$628,423
	Subtotal					$3,294,518
	SHAFT					
5.0	Contractor's Mobilization & Plant Set-up			L.S.	$628,157	$628,157
6.0	Shaft Sinking - 6.0 M Circular Concrete Shaft			777.0 m	$12,570.93 /m	$9,767,613
7.0	Station Excavation					
	760 Level			381 m^3	$247.29 /$m^3$	$94,217
	440 Level			812 m^3	$247.29 /$m^3$	$200,799
	400 Level			375 m^3	$247.29 /$m^3$	$92,734
	390 Level			140 m^3	$247.29 /$m^3$	$34,621
	Subtotal					$422,371
8.0	Station Equipping					
	760 Level	Brow Concrete		L.S.	$28,532	$28,532
		Electrical Power Distribution		L.S.	$16,723	$16,723
		Piping		L.S.	$6,914	$6,914
		Track		L.S.	N/A	N/A
		Pump Station		L.S.	$160,076	$160,076
						$212,245
	400 Level	Brow Concrete		L.S.	$28,532	$28,532
		Electrical Power Distribution		L.S.	$16,723	$16,723
		Piping		L.S.	$6,914	$6,914
		Track		L.S.	$14,064	$14,064
		Pump Station		L.S.	$160,076	$160,076
						$226,309
	360 Level	- Loading Pocket				
			Excavate	L.S.	$23,996	$23,996
			Equip.	L.S.	$308,731	$308,731
		- Waste Pocket				
			Excavate	L.S.	$20,352	$20,352
			Equip.	L.S.	$75,783	$75,783
						$428,862
	390 Level	Concrete Brow - Trackless		L.S.	$28,532	$28,532
9.0	Provisions For Ground Support			L.S.	$454,083	$454,083
10.0	Provisions For Grouting			L.S.	$517,510	$517,510
11.0	Shaft Power Cables			L.S.	$86,573	$86,573
12.0	Air Heating			L.S.	$68,713	$68,713 *
	TOTAL SHAFT COST					$12,840,968

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DY PROJECT
EXPLORATION SHAFT & DEVELOPMENT
PHASE 1 - COST ESTIMATE
SHAFT BOTTOM - 350 ELEV.
3 LEVEL EXPLORATION (480, 440, 400)

Item No.	Description	Unit	Unit Cost	Total
	LEVEL DEVELOPMENT & DIAMOND DRILLING			
13.0	Contractor's Mobilization & Plant Set-up	L.S.	$182,689	$182,689
14.0	Changeover	L.S.	$84,913	$84,913
15.0	Rockbreaker / Grizzly / Ore Pass Dump	2	$198,049	$396,098
16.0	Mine Development Exploration			
	400 Level X-Cut	125 m	$638.28 /m	$79,785
	Ramp - 3.0 x 4.0 m Trackless	710 m	$638.28 /m	$453,179
	400 Exploration	760 m	$670.11 /m	$509,284
	440 Exploration	795 m	$670.11 /m	$532,737
	480 Exploration	460 m	$670.11 /m	$308,251
	Raises	168 m	$678.39 /m	$113,970
	Track Installation	2,140 m	$167.18 /m	$357,765
	Indirect Support	180 days	$10,060.37 /day	$1,810,867
	Subtotal			$4,165,838
17.0	Diamond Drilling			
	"BX" Drilling, Assaying etc.	15,000 m	$60.00 /m	$900,000 **
	Drilling Support	50 days	$9,371.14 /day	$468,557
				$1,368,557
18.0	Provisions For Ground Support	L.S.	$231,000	$231,000
19.0	Mine Air Heating	L.S.	$150,000	$150,000 *
	TOTAL EXPLORATION DEVELOPMENT & DRILLING COST			$6,579,095
20.0	Christmas Shutdown	28 days	$1,469.92 /day	$41,158
	TOTAL COST			$23,505,739
	Contractor's G & A	11.5%		$2,591,758
	Contractor's Fee	5.0%		$1,081,851
	Total Contractor's Cost			$27,179,348
	Contingency			$1,000,000
	Power			$646,000
	Total Project Cost Estimate			$28,825,348
			Say	$28.8 Million

* - No Overhead, No Fee
** - No Fee

For general interest, two cost estimates are given for ramp drivage:

- October 1990 : 4.00 m x 5.50 m Access Decline to Elevation 480 metres, -20%, truck haulage. Total Length is 1,740 metres

Mobilization and Set Up			$ 441,616
Site Preparation			$ 272,748
Decline	0 - 300	300 m á 1,507.12	
	300 - 1000	700 m á 1,776,52	
	1000 - 1740	740 m á 1,952,58	
Total Ramp	1740 m		
	Misc. Excav.	169 m á 1,571.27	
Total Excavations			$ 3,391,303
Ground Support			$ 2,497,256
Water Control			$ 604,000
Air Heating			$ 165,000
Job Cost			**$ 7,371,923**
Contractor's Overhead	11.5%		$ 847,771
Contractor's Fee	8.5%		$ 626,613
Contractor's Cost			**$ 8,846,307**
Owner's Power Cost			$ 357,270
Roadbed Material			$ 120,000
Total			$ 9,323,577
Contingency			$ 750,000
Total Estimate			**$ 10,073,577**

Say $ 10.1 Million

CURRAGH RESOURCES - DY DEPOSIT
4.16 M X 6.20 M DECLINE
ACCESS TO 490 ELEVATION

RAMP
MINUS 20 PERCENT TRUCK HAULAGE
POWER INSTALLATION DELAYED TO 1ST QUARTER 1992

01-Feb-91
04:07 PM

DESCRIPTION	UNITS	UNIT RATE (CONTRACTOR'S COST)	TOTAL (CONTRACTOR'S COST)	UNIT RATE (FUEL COST)	TOTAL (FUEL COST)	UNIT RATE (OWNER'S COST)	TOTAL (OWNER'S COST)
MOBILIZATION		L.S.	$ 130,302		-		$ 800
SET UP CONTRACTOR'S FACILITIES		L.S.	546,889		-		429,912
SET UP TEMPORARY ELECTRICS		L.S.	11,890		-		207,200
PREPAYMENT (300 M DECLINE)		L.S.	192,915		-		-
			$ 881,996		$ 0		$ 637,912
EXCAVATION							
ACCESS TRENCH Overburden (By Others)	0 m³	-	$ 0	-	$ 0	-	$ 0
Rock	0 m³	16.54	0	0.32	0	0.81	0
COLLAR	10 m	1,608.59	16,086	33.02	330	220.88	2,209
GROUND SUPPORT							
ROCKBOLTS	400 bolts	27.56	11,024	0.99	396	9.37	3,748
MESH	250 m²	33.72	8,430	1.55	388	6.78	1,695
			$ 35,540		$ 1,114		$ 7,652
DECLINE 0 - 300 m	290 m	851.92	$ 247,057	27.94	$ 8,103	177.20	$ 51,388
PREPAYMENT CREDIT	300 m	(643.05)	(192,915)	0.00	0	-	0
DECLINE 300 - 1,000 m	700 m	896.70	627,690	49.99	34,993	177.20	124,040
DECLINE 1,000 - 1,500 m	500 m	949.56	474,780	73.31	36,655	177.20	88,600
DECLINE 1,500 - 2,000 m	260 m	985.82	256,313	90.18	23,447	177.20	46,072
MISC. EXCAVATIONS	354 m	912.03	322,857	59.78	21,164	-	0
			$ 1,735,782		$ 124,362		$ 310,100
GROUND SUPPORT							
ROCKBOLTS	8,094 bolts	27.56	$ 223,071	0.99	$ 8,013	9.37	$ 75,841
MESH 725 m	6,337 m²	33.72	213,684	1.55	9,822	6.78	42,965
SHOTCRETE (50 mm) 453 m	5,391 m²	157.91	851,293	3.40	18,329	2.50	13,478
			$ 1,288,048		$ 36,164		$ 132,284
WATER CONTROL							
GROUT COVER	50 shifts	1,593.28	$ 79,664	88.91	$ 4,446	100.00	$ 5,000
PROBE HOLES 150 @ 15 m	2,250 m	23.70	53,325	0.78	1,755	1.67	3,758
GROUT HOLES 225 @ 10 m	2,250 m	23.70	53,325	0.78	1,755	1.67	3,758
GROUT	2,500 bags	-	0	-	0	16.00	40,000
			$ 186,314		$ 7,956		$ 52,516
DIESEL GENERATED POWER							
Collar	7.0 days	485.04	$ 3,395	214.22	$ 1,500	-	$ 0
0 - 300 m	63.0 days	883.44	55,657	600.47	37,830	-	0
300 - 1,000 m	159.0 days	965.28	153,480	929.91	147,856	-	0
1,000 - 1,500 m	111.0 days	1,451.28	161,092	1,141.70	126,729	-	0
Christmas Shutdown	14.0 days	1,187.28	16,622	608.64	8,521	-	0
1,500 m - 2,000 m	7.0 days	1,491.84	10,443	1,306.42	9,145	-	0
ELECTRIC POWER 1,500 - 2,000 m	54.0 days	-	0	-	0	743.77	40,164
			$ 400,689		$ 331,581		$ 40,164
INDIRECTS DURING EXCAVATION							
Trench	0.0 days	2,192.33	$ 0	0.00	$ 0	150.00	$ 0
Collar	7.0 days	3,120.58	21,844	0.00	0	150.00	1,050
0 - 300 m	63.0 days	8,756.02	551,629	16.91	1,065	550.00	34,650
300 - 1,000 m	159.0 days	11,885.59	1,889,809	118.54	18,848	800.00	127,200
1,000 - 1,500 m	111.0 days	13,057.32	1,449,363	118.54	13,158	850.00	94,350
Christmas Shutdown	14.0 days	2,987.50	41,825	33.88	474	150.00	2,100
1,500 m - 2,000 m	61.0 days	15,468.71	943,591	152.42	9,298	1,050.00	64,050
			$ 4,898,061		$ 42,843		$ 323,400
AIR HEAT			-		-		114,521
JOB COST			$ 9,425,430		$ 544,020		$ 1,618,549
ROADBED MATERIAL							165,000
SUBTOTAL							$ 1,783,549
CONTINGENECY							0
SUBTOTAL			$ 9,426,430		$ 544,020		$ 1,783,549
TOTAL							$ 11,753,999

26% markup

or 112,258

4.0 M X 5.50 M DECLINE
ACCESS TO 480 ELEVATION

RAMP
MINUS 20 PERCENT TRUCK HAULAGE

DESCRIPTION MOBILIZATION/SET-UP	UNITS	UNIT RATE	TOTAL	
MOBILIZATION	-	L.S.	$ 92,533	
SET UP CONTRACTOR'S FACILITIES		L.S.	349,083	
				$ 441,616
EXCAVATION				
ACCESS TRENCH	6100 M^3	31.01	$ 189,161	
COLLAR	10 M	2,692.21	26,922	
GROUND SUPPORT				
ROCKBOLTS	400	90.65	36,260	
MESH	250 M^2	81.62	20,405	
				$ 272,748
DECLINE 0 - 300	290 M	1,507.12	$ 437,065	
DECLINE 300 - 1000	700 M	1,776.52	1,243,564	
DECLINE 1000 - 1740	740 M	1,952.58	1,445,131	
MISC. EXCAVATIONS	169 M	1,571.27	265,543	
				$ 3,391,303
GROUND SUPPORT				
ROCKBOLTS 1,740 M	6351	90.65	$ 575,718	
MESH 725 M	8723 M^2	81.62	711,971	
SHOTCRETE (75 MM) 335 M	4701 M^2	257.30	1,209,567	
				$ 2,497,256
WATER CONTROL				
GROUT COVER	40 SHIFTS	4,404	$ 176,160	
PROBE HOLES 120 @ 15M	2250	76.40	171,900	
GROUT HOLES 225 @ 10 M	2250	76.40	171,900	
GROUT	2500 BAGS	16.00	40,000	
				$ 604,000
AIR HEAT				165,000
JOB COST				$ 7,371,923
CONTRACTOR'S OVERHEAD 11.5%				847,771
CONTRACTOR'S FEE 8.5%				626,613
CONTRACTOR'S COST				$ 8,846,307
OWNER'S POWER COST INCL 4 MONTHS DIESEL GENERATED				357,270
ROADBED MATERIAL				120,000
TOTAL				$ 9,323,577
CONTINGENCY				750,000
TOTAL				$ 10,073,577
			SAY	10.1 MILLION

· February 1991: 4.16 m x 6.20 m Access Decline to Elevation 490 m, -20%, truck haulage.

The ramp has a total length of 1,730 metres. The method of cost estimate varies from the previous cases, since some cost items are carried by the owner. Total cost is $ 11,753,999.

Decline	0 - 300	300 m á 851.92	255,576
	300 - 1000	700 m á 896.70	627,690
	1000 - 1500	500 m á 949.56	474,780
	1500 - 2000	230 m á 985.82	226,739
	Misc. Exc.	354 m á 912.03	322,859
Total Excavations			$ 1,907,644

Cost Summary - Ramp (-20%):

Size m	Length m	Total Cost Mill. $	Cost for Excav. $/m	Cost for Project $/m
4.00 X 5.50	1,740	10.1	1,950	5,790
4.16 X 6.20	1,730	11.8	1,100	6,795

As with the shafts, there is a price difference that is hard to understand. The figures presented can not be used in a cost estimate, but serve as general guidelines. It should be noted that the unit price per metre for the total project is 4 to 6 times higher than the cost of excavation.

Various access possibilities are mentioned in the reports. No study is available that analyses the pros and cons of the different routes that could be considered for underground access.

9.7 Consideration of the specific Circumstances at Dy.

9.7.1 Background.

Twenty years have passed since the existence of mineral resources at Dy became known. Undoubtedly, many reasons exist that would explain why this deposit has not yet been made into a mine.

Surface exploration shows a mineral deposit that has the potential for exploitation. However, the wide spacing (150 metres) of surface drill holes does not permit a good enough interpretation of the orebody.

Still, the information available is sufficient to warrant the decision to go underground. Further exploration must be carried out from there. Only then will it be possible to define the shape of the orebody and develop mining methods and proper mine plans.

At first glance, one is tempted to select an exploration shaft for getting underground. However, close scrutiny of the Dy setting puts the choice of a shaft in doubt. Local geology and the shape of the deposit make it very difficult to find a proper location for a shaft. None of the locations suggested in the past are, in the opinion of this writer, suitable.

A shaft situated in bedrock north of the deposit would be at the upper end of the orebody and demand underground exploration from the hanging wall. This, in itself, is possible, but does not permit the use of exploration drifts for production. Mine development would make internal ramps necessary. Pilot holes have been drilled at a selected shaft location to the north. They indicate a fault, and difficult ground conditions in the area intended for the loading pockets and the shaft bottom.

The shaft location in the barren zone within the orebody is not acceptable either. The barren zone might contain ore. Phyllite in the hanging wall would probably require a larger shaft safety pillar than planned for. This, in turn, would tie up ore reserves.

The proposed Blind Creek ramp, on the other hand, would make a suitable access for underground exploration. For the transportation of ore to surface it does, however, pose environmental problems, since the danger of spilling acid material into the Blind Creek, a salmon spawning river, would exist.

Whether shaft or ramp is chosen, it will become the entry to the mine for years to come. It is, therefore, important to take all factors of influence into consideration when deciding on type of underground access, location and purpose.

Over the past 15 years, Cyprus Anvil Mining and Curragh Resources Incorporated have developed various concepts for reaching the Dy orebody. Shaft access and ramp entry have both been considered, alone and in combination.

A shaft is generally preferred for deep orebodies and high tonnage. It requires more capital up-front, is expensive and usually takes longer to construct than a ramp. Shaft operations are not flexible, provide, however, a cheap and proven way of hoisting ore and waste.

Underground access by ramp is longer than by vertical shaft. However, one metre of ramp costs about ten times less than one metre of shaft. Various ways of driving a ramp are available; a ramp operation also allows for a more flexible operation than a shaft does. Advanced technology in conveyor haulage, and the use of electric trucks, make these systems competitive with shaft hoisting of ore and waste.

In the reports reviewed, shaft access has been the preferred way of entering underground at times, the ramp option was favoured at others.

With the information of previous plans in mind, an attempt is here made to define an underground access that reaches the deposit at reasonable cost and time and serves future production requirements as well.

Factors that influence the choice of underground access include:

- Depth of the deposit
- Shape of the deposit
- Location of shaft collar
- Location of ramp portal
- Depth of shaft
- Length of ramp
- Gradient of ramp
- Location of shaft entry into the orebody
- Location of ramp entry into the orebody
- Environmental considerations
- Project cost
- Project time
- Production requirements
    - ore and waste hoisting
    - ore and waste haulage
    - ventilation requirements
    - transport of people
    - transport of materials
    - transport of equipment

The terrain covering the deposit starts at an elevation of around 1,195 metres and falls towards the southeast to about 1,025 metres. It is at elevation 1,125 metres at the southern edge.

The deposit starts at an elevation of 600 metres in the north west corner and dips to an elevation of about 400 metres on the east side and about 200 metres in the south. Thus the overburden/rock mantle varies from 595 metres in the northwest to 825 metres in the east and some 925 metres in the south.

9.7.2 Shaft Access.

Because of the dip and the shape of the orebody, it is difficult to find a suitable location for a shaft. CMD focused on two locations, one to the north of the deposit, the other in the barren zone in the centre of the orebody.

There are three reasons for rejecting the location north of the orebody (see sketch on next page):

- Six test holes were drilled in 1989/90 around the proposed shaft site to the north of the orebody. Core loss occurred between elevations 555 and 540 metres, a fault zone became apparent below elevation 540 metres. Poor ground conditions would make shaft construction difficult and expensive.

- Underground exploration to be carried out from the hanging wall. Exploration drifts can not be utilized for mine development.

- Internal ramps and long drifts are required for mine development.

A shaft in the centre of the orebody would serve development and production needs, is still unacceptable, however, when one takes into consideration the question of the barren zone, and the shaft safety pillar.

9.7.2.1 Barren Zone.

The existence, the size and the exact location of the barren zone must be established with a degree of certainty before it is decided to place a shaft within the orebody.

Drill logs of holes within the designated barren zone, as shown below, put existence and extent of a barren zone in doubt. The possibility of finding mineable ore in this zone can not be excluded at this point.

The shaft site selected is located at the southwest border of polygon 79X09. Mineralized intersections in this polygon are:

·· Drill hole 79X09:

From	To	Interval in m	% Pb+Zn
636.8	640.9	4.1	9.28
640.9	642.9	2.0	7.58
642.9	644.9	2.0	4.23
644.9	646.9	2.0	8.95
Total thickness		10.1 m	

– The proposed shaft location at the ore horizon is about halfway between holes 79X09 and 79X03. The distance between some of the holes in the area in question are:

· between holes

79X09	and	79X03	=	190 m
79X09	and	79X01	=	150 m
79X09	and	80X11	=	130 m

The distance between drill holes is too great to prove that a barren zone large enough to provide an area of sufficient size for a shaft safety pillar exists.

Some critical distances at the ore horizon between the proposed shaft location and some drillholes are:

· between shaft and hole

79X09	=	80 m
79X03	=	110 m
79X01	=	100 m
78X01	=	135 m
78X09	=	185 m
80X11	=	100 m

A shaft safety pillar with a cone of 10 degrees, which might not be big enough, requires an area with a radius of 117 m. Thus, the shaft safety pillar would cover most of polygon 79X09, which is fully included as mineable ore in the ore reserves calculations.

Mineralized intersections of other polygons within the barren zone in the vicinity of the proposed shaft location are:

·· Drill hole 78X01:

From	**To**	**Interval (m)**	**% Pb+Zn**
616.4	618.4	2.0	10.98
618.4	620.0	1.6	5.94
631.7	633.7	2.0	4.93
633.7	635.7	2.0	10.49
635.7	637.7	2.0	8.93
637.7	639.7	2.0	8.61
639.7	640.7	1.0	3.09
640.7	642.5	1.8	7.99
642.5	645.5	3.0	1.80
645.8	647.8	2.0	11.21
647.8	649.5	1.7	9.55
Total thickness		21.1 m	

·· Drill hole 79X01:

554.5	556.5	2.0	5.39
556.5	558.5	2.0	4.62
558.5	560.5	2.0	5.83
560.5	561.6	1.1	7.43
650.9	652.9	2.0	7.40
652.9	654.9	2.0	6.38
654.9	656.9	2.0	5.30
Total thickness		13.1 m	

·· Drill hole 79X03

864.5	866.5	2.0	9.03
866.5	868.0	1.5	9.88
868.0	868.7	0.7	0.22
868.7	869.7	1.0	3.93
869.7	870.4	0.7	0.22
870.4	871.2	0.8	0.44
871.2	872.1	0.9	10.85
Total thickness		7.6 m	

·· Drill hole 80X11:

612.9	613.4	0.5	6.65
613.4	615.5	2.1	10.58
Total thickness		2.6 m	

·· Drill hole 78X02:

674.3	676.3	2.0	10.16
676.3	678.3	2.0	10.42
678.3	680.3	2.0	8.51
680.3	682.3	2.0	5.52
682.3	684.3	2.0	3.32
684.3	686.3	2.0	9.07
686.3	687.6	1.3	13.29
687.6	688.6	1.0	4.85
688.6	690.5	1.9	6.48
690.5	692.5	2.0	12.61
692.5	694.5	2.0	9.88
694.5	696.5	2.0	4.17
696.5	697.5	1.0	4.50
697.5	698.9	1.4	6.50
698.9	700.9	2.0	10.06
700.9	702.2	1.3	9.17
702.2	703.9	1.7	0.00
703.9	705.4	1.5	10.28
Total thickness		32.2 m	

·· Drill hole 79X14:

788.8	789.1	0.3	14.97
789.1	790.3	1.2	1.30
790.3	791.6	1.3	8.47
791.6	792.1	0.5	13.61
792.1	792.7	0.6	9.07
792.7	794.2	1.5	10.60
794.2	794.5	0.3	7.82
794.5	795.7	1.2	6.53
795.7	796.1	0.4	14.21
796.1	798.1	2.0	13.02
798.1	800.1	2.0	10.90
800.1	802.1	2.0	4.24
802.1	804.1	2.0	5.20
804.1	804.6	0.5	17.00
804.6	805.1	0.5	11.37
805.1	807.1	2.1	5.52
807.1	808.7	1.6	1.07
808.7	809.1	0.4	17.58
809.1	811.5	2.4	0.13
811.5	812.4	0.9	18.80
812.4	814.3	1.9	4.20
614.3	815.6	1.3	1.13
815.6	817.6	2.0	0.66
817.6	819.3	1.7	2.80
819.3	821.8	2.5	6.47
821.8	822.5	0.7	17.11
822.5	824.8	2.3	7.21
Total thickness		36.1 m	

·· Drill hole 77X09:

625.5	627.5	2.0	11.69
627.5	629.5	2.0	7.37
629.5	631.5	2.0	7.54
631.5	633.5	2.0	8.28
633.5	639.5	6.0	6.19
Total thickness		14.0 m	

·· Drill hole 91DY05

584.9	608.1	23.2	12.57 (average)

·· Drill hole 78X09

556.3	562.1	5.8	9.35
575.2	580.2	5.0	10.83
Total thickness		10.8 m	

The proposed shaft site is in polygon 79X09. Immediately adjacent are polygons 80X11, 79X03, 79X01, 78X01, and 78X02, some of which are being considered as "barren", as well as polygons 78X09 and 91DY05, which are deemed mineable. Grade values and thickness of the drill holes shown above clearly indicate the existence of mineralized zones in the so-called barren zone within the massive sulphides.

Since this is not clear from looking at reserve drawings and existing mine plans, assays of sections of holes in the barren zone are shown here to demonstrate the occurrence of mineralized zones that put the existence, or at least the extent, of a barren zone in doubt.

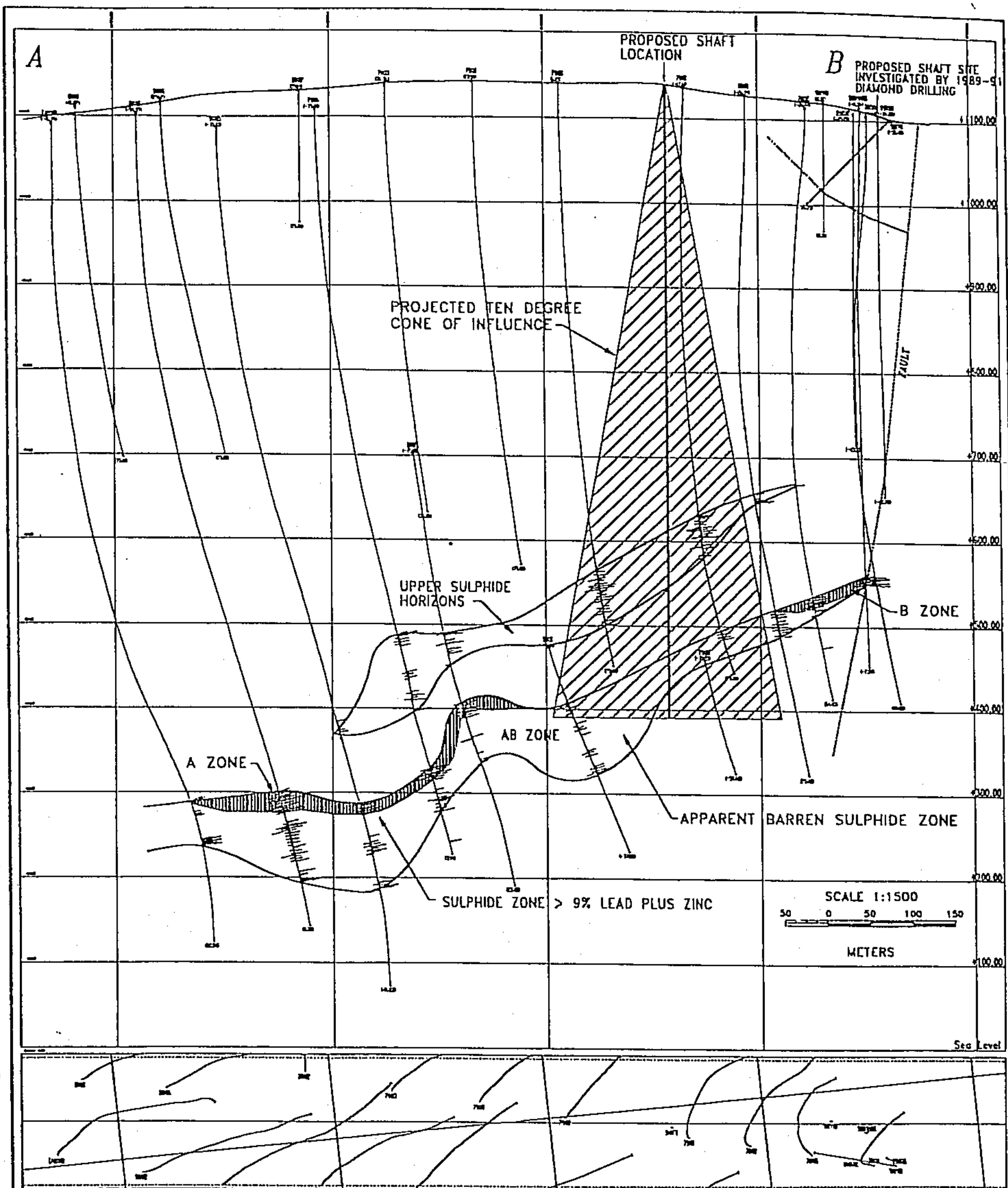
9.7.2.2 Shaft Safety Pillar.

A cone of 10 degrees has been suggested for the shaft safety pillar. Phyllite in the hanging wall, combined with the geometry of the orebody, might require a safety pillar of more than 10 degrees.

The collar of a shaft in polygon 79X09 would be located at elevation 1,140 metres. The hanging wall of the orebody at this location is at elevation 476 metres. Hence, overburden is 664 metres. A safety pillar with an angle of 10 degrees would require a cone with a radius of 117 m at the orebody. A 20 degree angle requires 242 m, and a 30 degree angle 383 m.

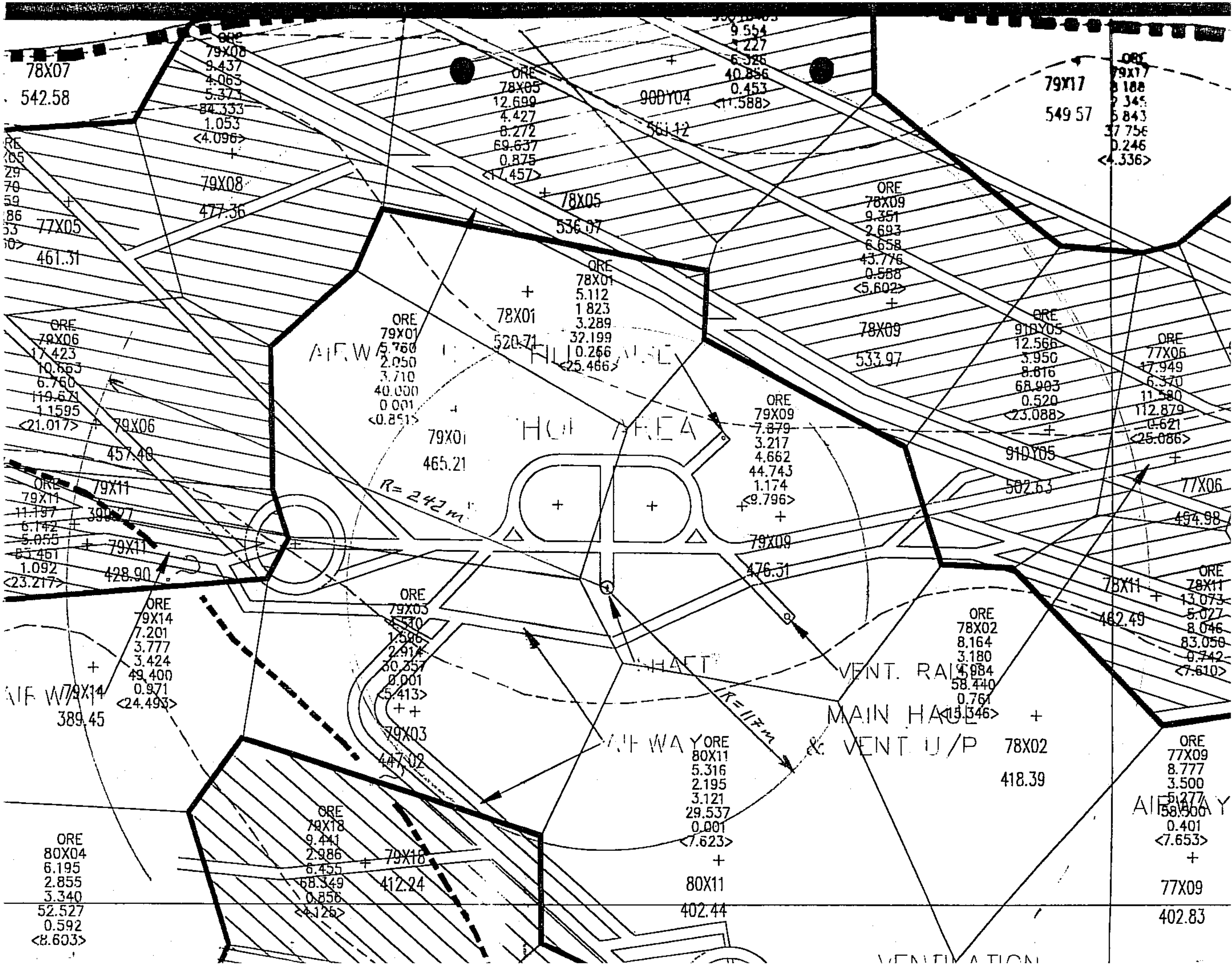
Even a 10 degree cone would cover a substantial portion of good ore in polygon 79x09.

The application of backfill should not be used as an argument for a reduction in the size of safety pillar, since it is not possible, at this time, to predict whether the orebody can economically support the cost of backfill in all areas of the mine.



Curragh Inc.	Dy Deposit	VERTICAL SECTION SHAFT LOCATION STUDY Oct 10, 1992

Figure 5


78X07
542.58
ORE
79X08
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4.063
5.373
84.333
1.053
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79X08
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ORE
78X05
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8.272
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78X05
536.07
9.554
6.326
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90DY04
79X17
549.57
ORE
79X17
37.756
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77X05
461.31
ORE
78X09
9.351
2.693
6.658
43.776
0.588
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78X09
533.97
ORE
78X01
5.112
1.823
3.289
32.199
0.266
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78X01
520.71
ORE
79X06
17.423
10.663
6.760
119.671
1.1595
<21.017>
79X06
457.40
ORE
79X01
5.760
2.050
3.710
40.000
0.001
<0.851>
79X01
465.21
HOIST AREA
R= 242 m
ORE
79X09
7.879
3.217
4.662
44.743
1.174
<9.796>
79X09
476.31
ORE
91DY05
12.566
3.950
8.616
68.903
0.520
<23.088>
91DY05
502.63
ORE
77X06
17.949
6.370
11.580
112.879
0.621
<25.086>
77X06
494.98
ORE
79X11
11.197
6.142
5.055
65.461
1.092
<23.217>
79X11
428.90
ORE
79X14
7.201
3.777
3.424
49.400
0.971
<24.493>
79X14
389.45
ORE
79X03
4.530
1.586
2.914
30.357
0.001
<5.413>
79X03
447.02
SHAFT
VENT. RAISE
MAIN HAUL
& VENT. U/P
R=117 m
ORE
78X02
8.164
3.180
4.984
58.440
0.761
<15.346>
78X02
418.39
78X11
402.49
ORE
78X11
13.073
5.027
8.046
83.050
0.742
<7.610>
AIRWAY
ORE
80X11
5.316
2.195
3.121
29.537
0.001
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80X11
402.44
ORE
79X18
9.441
2.986
6.455
68.349
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79X18
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ORE
80X04
6.195
2.855
3.340
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0.592
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ORE
77X09
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5.277
58.500
0.401
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77X09
402.83
VENTILATION

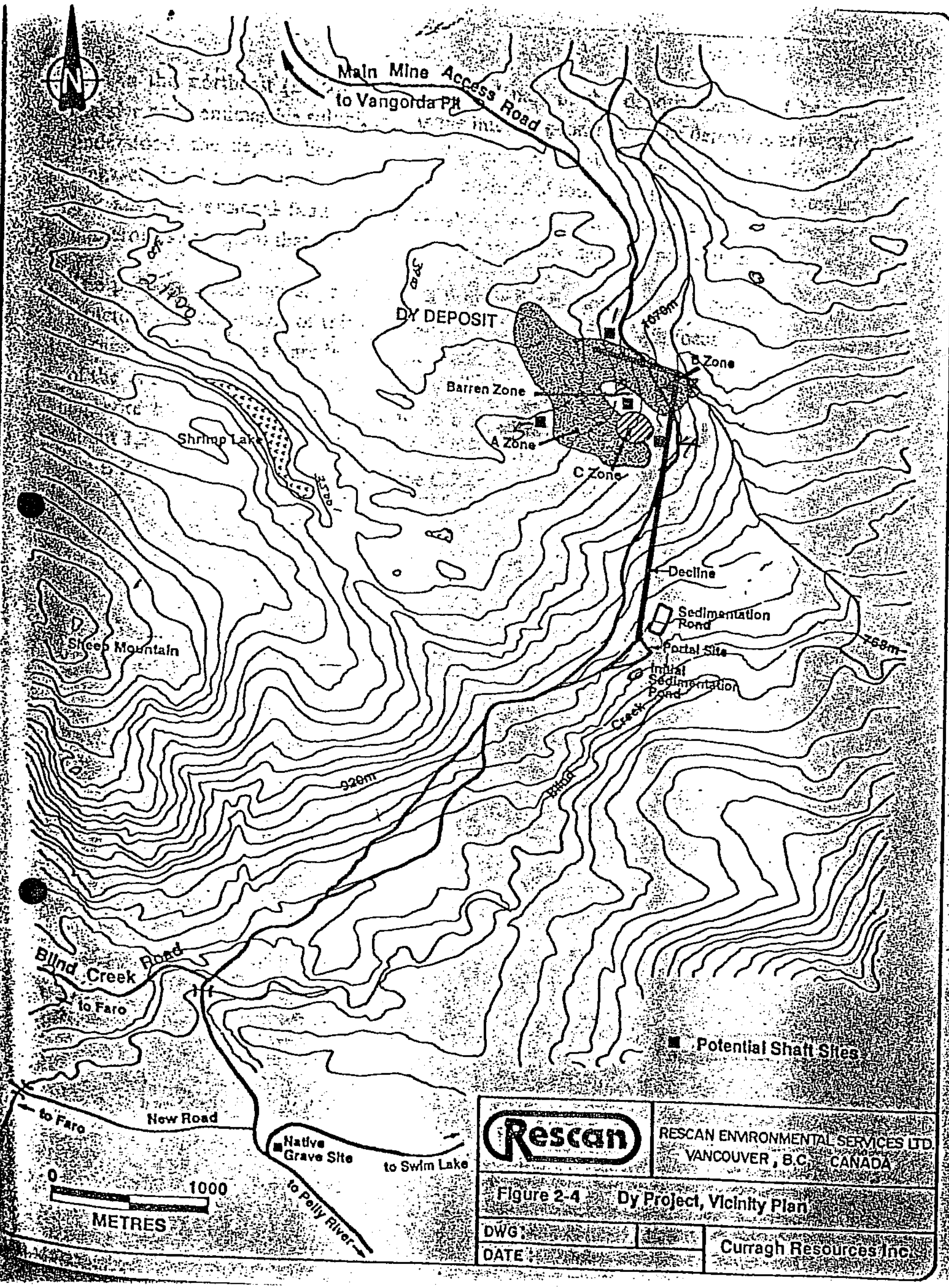


Figure 2-4 Dy Project, Vicinity Plan

The Rescan report shows five potential shaft sites. See fig. 2-4.

- Potential shaft site Nr. 1 : Located north of the orebody is unacceptable for the three reasons explained above.

- Potential shaft site Nr. 2 : Located in the so called barren zone in the centre of the orebody; is unacceptable for the two reasons explained above.

- Potential shaft site Nr. 3 : Located on the east side of the orebody; is unacceptable because it is within the orebody. The same conditions and reasons as for Nr. 2 apply.

- Potential shaft site Nr. 4 : Located southeast of the orebody is unacceptable because the orebody is open in this direction. (Similar to Nr.2).

- Potential shaft site Nr. 5 : Located southwest of the orebody is unacceptable because the orebody is open in this direction. (Similar to Nr.2).

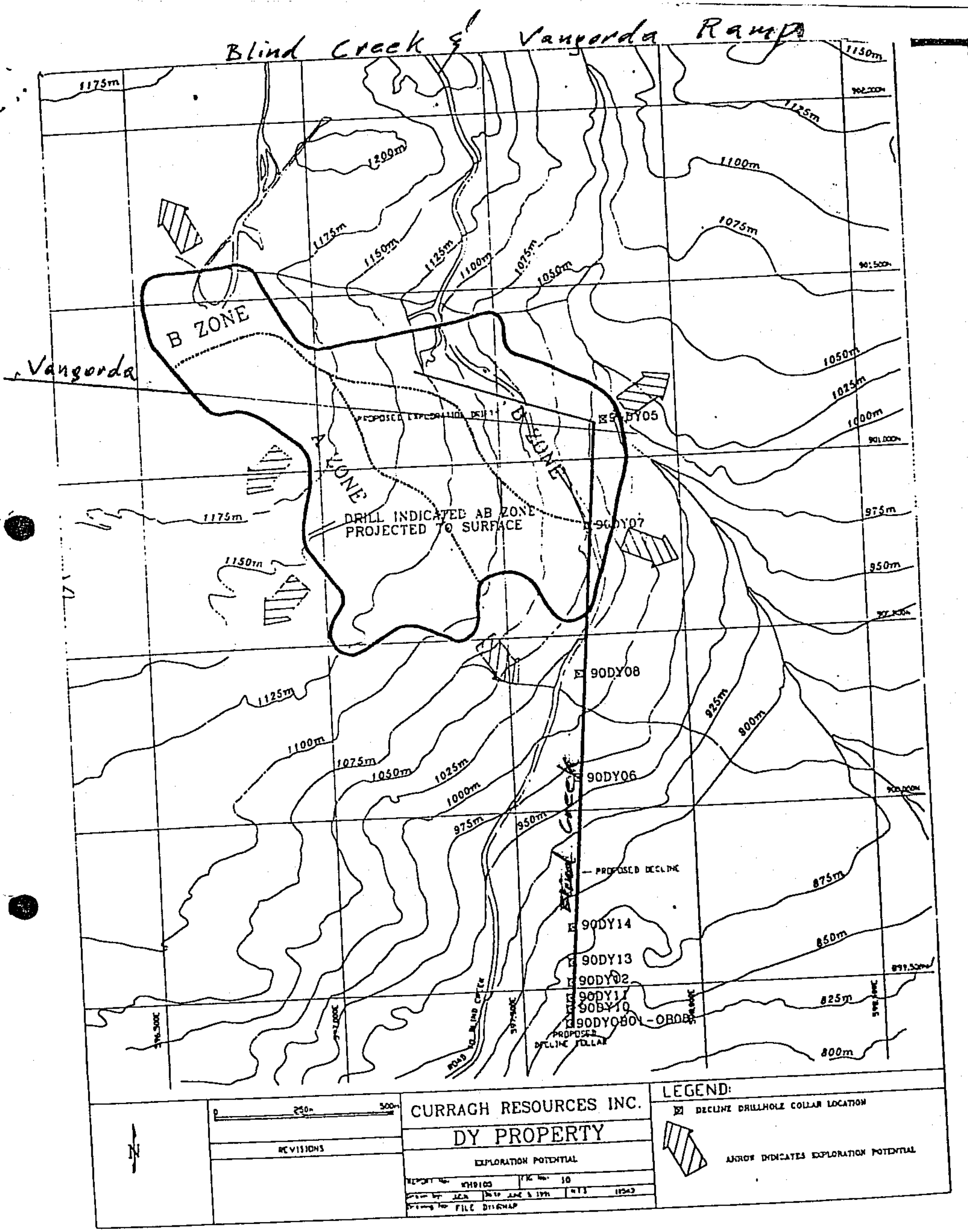
Additional information from surface drilling would be required in order to define a suitable shaft location. Hence, a shaft is not being considered for underground access at this time.

9.7.3 Ramp Access.

9.7.3.1 Ramp Portal.

Ramp planning should not only contemplate the necessity of underground exploration, but take into consideration future production needs as well. Consequently, the right choice of ramp size, gradient, and type (straight, switchback, or spiral ramp), is important. Surface location, and designated ramp bottom, will dictate the vertical distance to be overcome. This, in combination with the gradient selected, will determine the length of the ramp. Geological conditions will greatly affect ramp size, advance, support methods and final cost.

The two locations for ramp access considered in the past are north of the orebody, and in the valley of Blind Creek. The site north of the orebody has interest only in connection with a shaft at this same location.


Blind Creek & Vangorda Ramp
B ZONE
A ZONE
D ZONE
Vangorda
PROPOSED EXPLORATION DECLINE
DRILL INDICATED AB ZONE PROJECTED TO SURFACE
90DY05
90DY07
90DY08
90DY06
90DY14
90DY13
90DY02
90DY11
90DY10
90DY08B-08BB
PROPOSED DECLINE
PROPOSED DECLINE COLLAR
ROAD TO BLIND CREEK
Blind Creek
1175m
1200m
1150m
1125m
1100m
1075m
1050m
1025m
1000m
975m
950m
925m
900m
875m
850m
825m
800m
CURRAGH RESOURCES INC.
DY PROPERTY
EXPLORATION POTENTIAL
REVISIONS
LEGEND:
DECLINE DRILLHOLE COLLAR LOCATION
ARROW INDICATES EXPLORATION POTENTIAL
FIG. No. 10
FILE DYSMAP

At an elevation of 835 metres, the valley of Blind Creek offers a starting point at a low elevation. Nevertheless, the vicinity of Blind Creek does impose environmental problems. Switch-backs are required if one is to enter the orebody at an elevation of around 400 metres.

The Blind Creek ramp offers the shortest distance. It could serve underground exploration, but should not be used for ore haulage, because of the environmental hazards this would pose for Blind Creek.

The proposed settling ponds and water treatment would suffice for ramp drivage and underground exploration. Environmental problems will hardly occur during Phase I of the Dy development, since only small amounts of ore will be handled and much attention will be paid to avoiding environmental predicaments. It is during production, when all attention focuses on the daily operating problems, that difficulties have a tendency to crop up.

CMD proposed a gradient of 20% for the Blind Creek ramp. This is too steep if the ramp is to be used for anything but access for underground exploration, ventilation, emergency exit and conveyor haulage. But even conveyor haulage at this angle might cause problems, with chunks of ore rolling back on the belt to create safety hazards and obstacles to a smooth operation.

The Blind Creek ramp comes from the south and curves into the main zone of the orebody from the east. 14 test holes have been drilled for ramp design. Two of these holes have intersected mineralized zones.

The proposed mine plan by N. Rose, August 1992, depicts the Blind Creek ramp as entering the orebody between elevations 500 and 550 meters. It has, therefore, little value as a starting point for underground exploration, unless a drift in the hanging wall is utilized for this purpose.

The Rose plan shows a shaft more or less in the centre of the orebody. With this arrangement, the ramp would serve as ventilation and emergency exit.

An alternative to the Blind Creek ramp option would be that of a ramp from the Vangorda pit to the Dy deposit, a horizontal distance of roughly 3,050 metres. A portal at Vangorda is away from the Blind Creek water shed and is in line with the flow of ore to the concentrator. A massive sulphide showing at the southeast edge of Vangorda Pit at an elevation of 1094 metres provides a possible starting point for a ramp.

Further investigation might reveal a suitable portal location at a lower elevation. The example below shows the difference in total length between a starting point at Blind Creek and one at Vangorda, and with the bottom of both ramps at elevation 400 m and an alternative at elevation 500 m with different grades.

		Blind Creek	Vangorda Pit
Portal elevation		**835 m**	**1,094 m**
Ramp bottom		**400 m**	**400 m**
Vertical distance		435 m	694 m
Length of ramp	**at 15 %**	**2,945 m**	**4,695 m**
	at 18 %	2,465 m	3,930 m too steep
	at 20 %	2,215 m	3,530 m too steep
	at 22 %	2,010 m	3,205 m too steep
Alternative with **Ramp bottom**		**500 m**	**500 m**
Vertical distance		335 m	594 m
Length of ramp	**at 15 %**	**2,265 m**	**4,020 m**
	at 18 %	1,900 m	3.365 m too steep
	at 20 %	1,705 m	3,025 m too steep
	at 22 %	1,550 m	2,745 m too steep

Elevation 400 m would provide a better starting point for underground exploration, but the 500 metre level is possible, if certain exploration limitations are accepted.

9.7.3.2 Ramp Bottom.

Once the ramp has reached bottom, horizontal drifts below or above the orebody are established to allow for underground exploration.

The Blind Creek ramp bottoms out east of the orebody, the Vangorda ramp comes from the northwest. The horizontal distance between Vangorda and the west edge of Dy as it is known today, is about 3050 m. To overcome the vertical distance of 694 m with a 15 % ramp to reach the western edge of Dy at the 400 m elevation requires a switchback. However, it would be possible to cross the orebody with a straight ramp from the west, through the low grade centre zone.

The last portion of the ramp could be steeper as it would be used for conveyor haulage from a crusher station. This important detail should be kept in mind as part of ramp planning.
A ramp approach from the west offers, in the opinion of this writer, more flexibility for exploration and mining.

The deposit has been subdivided into B-Zone, A-Zone, and Central Zone. For the purpose of this analysis, the northwest corner of the B-Zone could be regarded as a separate zone. Thus, the following four zones are contemplated, to see which approach would offer better exploration opportunities: (See sketch).

- B-Zone (Main Zone)
  along the north edge of the orebody, between polygons 79X12, 80X01 and 79X14 to the west and 79X04, 79X05 and 78X06 to the east, extending more or less 1,000 metres from west to east and some 300 metres from north to south. The Main Zone lies roughly between the 500 and 400 metre elevations.

- Northwest Zone
  in the northwest corner of the orebody, including, and north of, polygons 77X04 and 77X03. The northwest Zone extends about 350 metres from west to east and 300 metres from north to south. It lies roughly between elevations 675 and 450 metres.

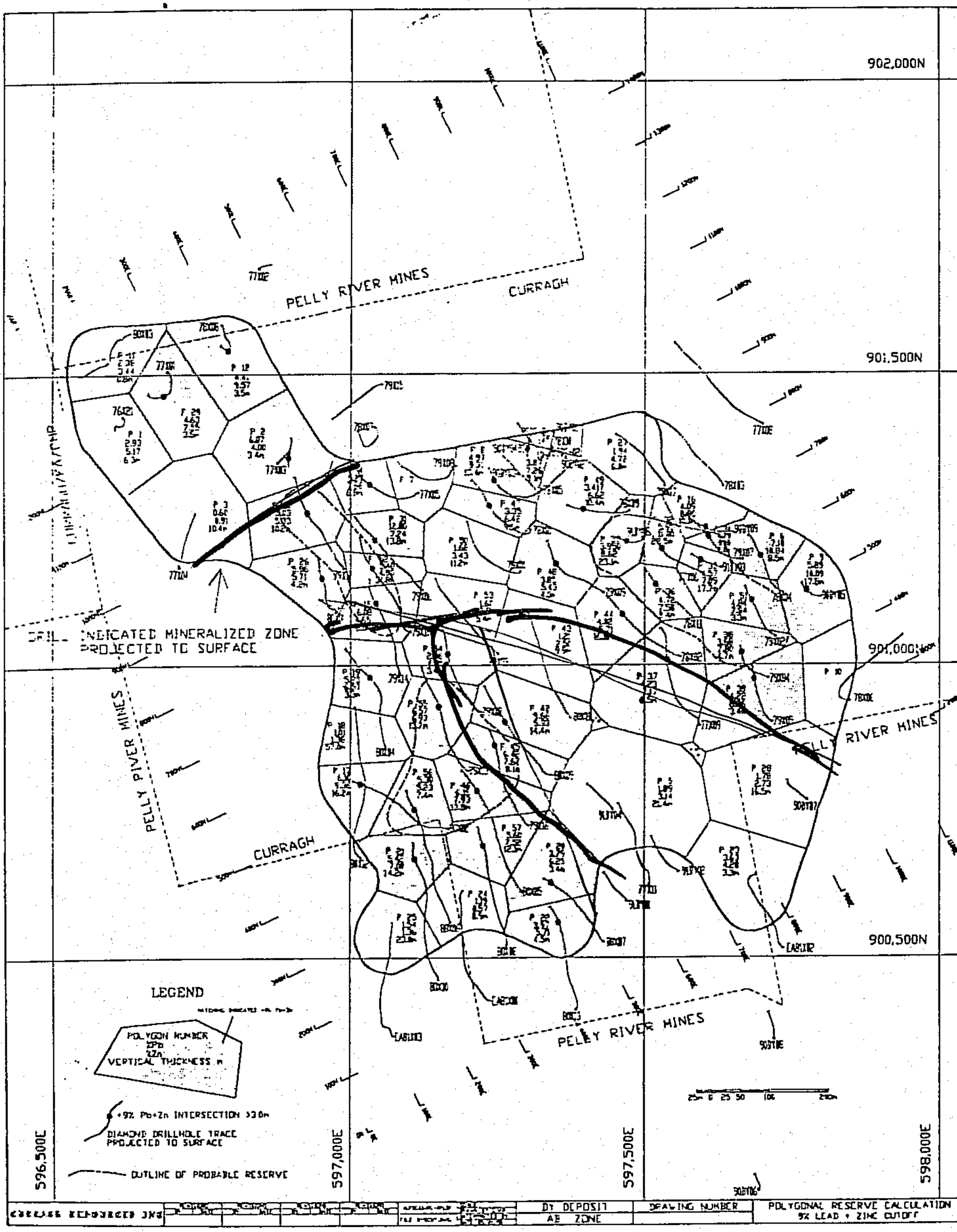
- A-Zone
  along the west edge of the deposit as it is depicted now, including, and south of, polygons 80X04 and 79X18, to the south edge of the deposit. It extends some 400 metres from west to east and roughly 550 metres from north to south between elevations 400 and 200 metres.

- Central Zone
  includes polygons 79X03, 80X11, 78X02, 77X09 and continues southeast to the southern border of the deposit. It is about 250 metres wide and 500 metres long, between elevations 450 and 400 metres.

The Vangorda entrance can be utilized for the exploration of the B-Zone from the west, the Northwest Zone, the A-Zone and the Central Zone. The Blind Creek approach would only serve the B-Zone from the east and the southeastern part of the Central Zone.

The perception of the Dy deposit will change as underground exploration is carried out and underground experience is gained. The deposit is open in several directions, its extent and shape will certainly change as time passes. The present shape of the deposit is largely determined by the lack of drill information.

Future underground work will show faults, differences in dip, thickness, elevation, grade, etc. that will provide parameters for the delineation of ore blocks for mining. Nevertheless, the four zones defined above will help in assessing the best access to the orebody by ramp, since it points out the benefit to exploration of a west entry.



902,000N
901,500N
901,000N
900,500N
596,500E
597,000E
597,500E
598,000E
PELLY RIVER MINES
CURRAGH
DRILL INDICATED MINERALIZED ZONE
PROJECTED TO SURFACE
LEGEND
POLYGON NUMBER
%Pb
%Zn
VERTICAL THICKNESS m
+9% Pb+Zn INTERSECTION >3.0m
DIAMOND DRILLHOLE TRACE
PROJECTED TO SURFACE
OUTLINE OF PROBABLE RESERVE
DY DEPOSIT
AB ZONE
DRAWING NUMBER
POLYGONAL RESERVE CALCULATION
9% LEAD + ZINC CUTOFF

Figure 9

9.7.3.3 Environmental Considerations.

The proximity of a salmon spawning river makes a portal in the valley of Blind Creek an environmental hazard. Any water that were to leave the site without mitigation could influence the water quality of Blind Creek. A mine entrance at this location would be an on-going operating concern.

A portal at Vangorda, on the other hand, presents quite a different situation: The Vangorda Creek has been diverted to pass through a culvert around the open pit. The existing closure plan calls for filling the pit with water from the creek, once operations have ceased. The creek will then pass through the resulting lake. It is stipulated that the water quality of the creek leaving Vangorda lake must be the same as the quality of the water entering the lake.

An access to the Dy deposit from the Vangorda pit will keep Vangorda active as long as Dy is in operation. The money required to comply with the Vangorda closure plan, which includes the treatment cost of the water of Vangorda Creek, would not have to be spent now.

Any mine water from the Vangorda ramp would go into the pit, from where it would go through the existing water treatment system before being discharged into the natural drainage system.

The Vangorda Creek diversion would be maintained for the life of the Dy operation. Any acid water from Dy would go through the Vangorda pit to the existing water treatment system. Waste and ore dumped or spilled from Dy into the Vangorda pit would not change the composition of the material there now. Therefore, the existing Vangorda closure plan should be equally valid for the Dy closure.

9.7.3.4 Production Requirements.

Besides providing the first access underground for exploration, the ramp must support production through the following functions:

- Ventilation
- Ore haulage
- Waste haulage
- Transportation of
    - people
    - materials
    - equipment

For a production of over one million tonnes per year, truck haulage is insufficient, but conveyor haulage is well suited. Options for conveyor haulage include:

- Several flights (for switch-back ramp, or if smaller drives are used)
- Large drive for long conveyor and steel cord belting
- Cable belt conveyor

Primary crushing underground is required for moving ore and waste by conveyor. Separate bins and chutes are necessary for ore and waste. Transportation of ore and waste happens on the same conveyor at different times. The conveyor is roof mounted to provide good access for cleanup and maintenance, and a roadway for the movement of vehicles.

The cross section of the ramp must be designed to provide room for conveyor installation and roadway as well as ventilation.

9.8 Concluding Remarks on Shaft versus Ramp.

Only a few years ago, one would not have considered accessing a mine with the depth of the Dy deposit through anything but a shaft. Times change, however, and it is not uncommon any more to use ramp access for deposits similar in depth to the Dy. The reasons can be found in better methods of ramp drivage, the acceptance of conveyor haulage in hardrock mining, and the fact that modern raise bore methods make the construction of a shaft of any size much cheaper once muck removal is possible from underground through a ramp.

Four reasons are given in this report to demonstrate why shaft access for underground exploration is not the answer at Dy:

- A shaft at the proposed north end of the orebody should be rejected due to bad ground conditions and poor utilization of a shaft located at the upper end of the orebody.

- The uncertainty as to the existence of a barren zone and, if it does exist, the question of its size.

- The size of the proposed shaft safety pillar with a cone of 10 degrees is probably not sufficient. With phyllite in the hanging wall and considering the characteristics of Dy, a larger safety pillar is suggested. Since backfill might not be used, at least not in all parts of the mine, support from backfill can not be assumed in the calculations for the design of a shaft safety pillar at this time.

- The difficulty of finding a suitable shaft location outside the ore boundary as it is defined now, since the orebody is open in many directions.

Several factors will influence the decision as to whether a ramp should begin in the Blind Creek valley or in the Vangorda Pit, or at another suitable location, yet to be found.

The cost comparison is obviously influenced by the vertical distance between the portal and underground, which, together with the selected gradient, will determine the length of the ramp. The actual cost today is still to be established. For the purpose of comparison, however, the figures presented by CMD some five years ago will be used here (see section 9.7 of chapter 9 of this report).

The CMD figures vary greatly. Average prices for excavation only and for the total project are shown below.

· Excavation only	:	1,465 $/m
· Total project	:	6,215 $/m

The price of the complete project includes contractors' fees and extras. To obtain a price for comparison, the cost of excavation, ground support and water control presented by CMD for advancing a ramp of 1,740 m only is used.

· Excavation	:	3,391,303 $
· Ground support	:	2,497,256 $
· Water Control	:	604,000 $
Total	:	6,492,559 $: 1,740 m = 3,731 $/m
· Advance only	:	3,730 $/m (for comparison only)

	Blind Creek	Vangorda
· Portal elevation	835 m	1,094 m
· Entry underground	400 m	400 m
· Vertical depth	435 m	694 m
· Gradient	15 %	15 %
· **Length of ramp**	**2,945 m**	**4,695 m**
· Advance only	3,730 $/m	3,730 $/m
· Total cost (advance) (for comparison only)	10,985,000 $	17,512,000 $

Using CMD cost figures
for the total ramp project:

· Cost per metre	6,215 $/m	6,215 $/m
· Total cost	18,023,500 $	28,756,805 $

Alternative entry underground at elevation 500 m:

· Portal	835 m	1,094 m
· Bottom	500 m	500 m
· Vertical depth	335 m	594 m
· Gradient	15 %	15 %
· **Length of ramp**	**2,265 m**	**4,020 m**
· Cost á 3,730 $/m	8,448,450 $	14,994,600 $
· Cost á 6,215 $/m	14,076,975 $	24,984,300 $

Theoretical costs for comparison of **alternatives** for **Blind Creek only** (=access for underground exploration, and ventilation during production):

	Length	Total Cost á 3,730 $/m	Total Cost á 6,215 $/m
· Bottom at 400 m:			
Gradient 18 %	2,465 m	9,194,450 $	15,319,975 $
20 %	2,215 m	8,261,950 $	13,766,225 $
22 %	2,010 m	7,497,300 $	12,492,150 $
· Bottom at 500 m:			
Gradient 18 %	1,900 m	7,087,000 $	11,808,500 $
20 %	1,705 m	6,359,650 $	10,596,575 $
22 %	1,550 m	5,781,500 $	9,633,250 $

Figures above are based on cost information presented in the CMD proposals and prorated to the length of the various ramp alternatives. These are not actual figures, but show the order of magnitude and the need to obtain competitive prices from more than one contractor.

At the time of writing, certain details concerning contours and geography of the Vangorda Pit were not available. It makes common sense to reduce the vertical distance between portal and ramp bottom by moving the portal further down into the pit if this is possible. A site visit will resolve this question.

There are a number of significant factors that are important in selecting the location of the ramp. An economic evaluation will eventually govern the final decision, but the reader should be aware of the many details that need to be taken into consideration.

The example shown below of rating is opinionated, but will assist in the process of selecting the right location of ramp access to the orebody.

1 = lowest 10 = highest

	Blind Creek	Vangorda
· Cost	10	1
· Construction time	10	1
· Environmental concern	3	8
· Knowledge of geology	8	3
· Possibility of finding mineable ore from ramp	2	4
· Additional exploration area	3	7
· Use for production	3	10
· Surf. access-haul road	1	10
· Geological Info avail.	7	3
· Anticipated roof cond.	3	4
· Underground exploration	3	8
· Mine infrastructure	3	8
· Ore handling undergr.	5	7
· Conveyor application	3	10
· Ore haulage to surface	2	10
· Ore handling on surface	4	8
· Entry for mining	4	7
· Dry facilities	1	10
· Travel time to portal	2	7
· Travel time to workplace	4	6
· Surface Maintenance shop	1	10
· Mine office	1	10

The chart reflects the opinion of one individual. It includes most of the factors of influence. A final decision must be made in Faro taking into consideration these and other factors. This writer sees many advantages to the Vangorda ramp over the Blind Creek ramp concerning environmental issues, ore handling and the transportation of people, materials and equipment.

9.9 Ramp Advance.

Geological conditions at Dy will be different from those at Faro and Grum. Nevertheless, experience from the Faro and Grum ramps will help in assessing ground conditions for the ramp to the Dy deposit.

Fourteen test holes for the Blind Creek ramp were drilled in preparation for ramp advance.

A few holes only have been drilled on the Vangorda plateau between Vangorda and Dy. Hole 76X22 intersected the Dixon Creek Fault at about 375 metres. This fault dips at 20 to 30 degrees to the southeast and continues underneath the Dy deposit. It is considered the shut off fault of Dy mineralization. At the fault intersection the core is intact. It is brecciated. Movement of groundwater can be expected in all faults. Water will drain, recharging is not expected.

S^2 foliation dips at about 20 degrees to the southeast. This will influence ground conditions and have a bearing on the size of the ramp and other underground openings.

The Vangorda ramp is oriented in the same direction as the S^2 foliation. Wedges will form that must be bolted. Roofbolting can be anticipated throughout the full length of the ramp.

The Blind Creek ramp intersects the S^2 foliation at an angle. Loose will form in chunks that will have to be secured continuously by bolting.

Besides the Dixon Creek Fault at the Vangorda ramp, other vertical and angle faults are to be expected in both ramps.

Good people, experienced with difficult ground conditions and the knowledge and capability to manage all types of ground support as well as water inflows, will be the key to the successful completion of the ramp.

Methods of ramp advance apply for both ramps, except for dumping of muck and discharge of mine water. Poor ground conditions make the selection of the right cross section of the ramp as well as the most efficient advance method important.

An appraisal of mechanized cutting is significant. If applicable, it has many advantages. Less damage to the roof and side walls, better ventilation, faster advance, lower costs are significant factors.

The application of tunnel bore machines (TBM) is not advisable, mainly because of the presence of water bearing faults.

However, mechanized cutting with roadheaders should be considered. Small quartz veins at the face can be cut with an AM-75 type, or a heavier machine. For thick quartz veins and dykes, blasting would be necessary. The economy of mechanized advance will depend on bit usage, which will depend on the silica content of the material.

Methods of muck removal are mentioned in Section 9.5 of this chapter. Scooptram and truck(s) would be the preferred way of muck removal with conventional drilling and blasting, while conveyor haulage can be adapted quite well to a system of advance by roadheader. In this case, discharge from the conveyor on surface could be done with the help of a radial stacker, as shown on the sketch on the next page.

The following two advance methods are possibilities for Dy, and are briefly discussed here.

9.9.1 Conventional Drilling and Blasting.

This method is well known and does not need further description. It is fairly flexible and can be adapted more easily to changing geological conditions than mechanized cutting. The advance of 4.5 m/day used by CMD sounds reasonable, considering the poor ground conditions and the need for continuous ground support.

9.9.2 Mechanized Advance with Roadheader.

Advance by roadheader requires good geological data at the planning stage. Advancing through faults and water bearing ground will probably be done with conventional drilling and blasting.

Various types of roadheaders are available from several manufacturers. Examples are Dosco in England, Voest-Alpine in Austria, Paurat and IBS (Industrie und Bergbau Services) in Germany, and others.

Good performance with roadheaders can be achieved with rock strengths of up to 80 Mpa. Heavier machines of 90 to 100 tonnes are capable of cutting rock of up to 120 to 140 MPa. Yet, not only compressive strength, but other factors as well, need to be considered to make mechanized cutting a success.

Technical data below will acquaint the reader with some of the machine capabilities. Theoretical performance is data supplied by manufacturers, actual performance is based on practical experience in German coal mines and the construction industry in Europe.

· 1 m^3 equals 1 m^3 *in situ*

	Rock Type	Compr.Str. MPa	Performance m^3/h Theoret.	Actual
Paurat				
E 134	Mudstone	60	55	25
		80	30	17
		100	20	12
E 200	Mudstone	60	75	35
		80	45	25
		100	30	17
	Sandstone	120	20	10
		140	14	7
E 250	Mudstone	60	80	45
		80	45	30
		100	35	20
	Sandstone	120	25	15
		140	14	8
Voest Alpine				
AM 75	Mudstone	60	60	30
		80	40	20
		100	20	10
AM 85	Mudstone	60	70	35
		80	56	28
		100	30	15
	Sandstone	120	10	6
AM 105	Mudstone	60	80	40
		80	70	35
		100	30	18
	Sandstone	120	15	8
		140	8	5

The ramp section will be determined by a number of factors. A section of 5.50 m x 5.25 m used here is an example only, to obtain the amount of muck from 1 m of advance in order to relate the data given above in cubic metres to advance figures in metres.

· 5.50 m x 5.25 m x 1 m = 28,88 m^3

· Cutting performance

m^3/h	Advance m/h	Advance per 8 hour shift, 3 hrs cutting per shift m/shift	m/day
40	1.4	4.2	12.6
30	1.0	3.0	9.0
20	0.7	2.1	6.6

Even with cutting performance kept low intentionally, ramp advance is such that it makes mechanized cutting interesting and worthwhile to investigate further. Better roof conditions obtained from cutting, and a possible reduction in support cost, might be the real advantage of mechanized cutting that will have roof support incorporated as part of the system of advance.

The application of a roadheader should not be considered if the floor gets soft when in contact with water as it would not support the weight of the machine.

9.10 Access Options.

A vertical shaft is rejected as initial access for reasons explained in Section 9.7.2 of this chapter. Using raise boring technology, shaft construction can be done at favourable costs once ramp access allows muck removal from underground.

An appropriate zone for a shaft, if desirable, can be found after underground exploration has revealed a better picture of the orebody.

The Blind Creek ramp is not suitable for production, mainly for environmental reasons, but could serve as initial access for underground exploration.

The Vangorda ramp forms a good access for underground exploration and is well suited for production, but costs more than the Blind Creek ramp. For lack of sufficiently detailed information, only elevation 1094 metres has so far been considered as a starting point.

Based on information available today, the following options are open:

· Vangorda ramp first, then
  ·· Shaft for ventilation and emergency exit
  or
  ·· Blind Creek ramp for ventilation and emergency exit

· Blind Creek ramp first, then
  ·· Vangorda ramp for production and main access

or

  ·· Shaft for production and main access

Options that have been contemplated in the past are numerous. Some are summarized here for general interest.

· Vangorda Plateau, 1,100 m: Shaft 800 m deep, ramp at 20 % is 5,400 m long.

· North of orebody at elevation 925 m: Shaft, ramp at 20 % is 2,200 m long.

· Blind Creek valley for ramp at elevation 835 m: Ramp at 20 % is 1,735 m long. Bottom at 400 m.
  A shaft is located west of the orebody.
  Total project will take 443 weeks.
  Underground exploration to take 25 weeks.

· Shaft and ramp located at about 1,100 m north of orebody.
  Bottom at 440 m. Switchback ramp from elevation 935 m at 20 % is 2,200 m long and will assist in shaft sinking. Shaft, 660 m deep, will be for production.

· Shaft, 668 m deep, located north of the orebody.
  For exploration and production. Bottom at 440 m. Bored raise for ventilation and second exit.

Shaft to cost		20,626,140 $ and take 30 months.
Loading pockets	=	7,815,466 $
Total	=	28,441,606 $

· Ramp (4.0 m x 5.5 m, at 20 %) from Blind Creek valley, 14.5 months at 4.5 m/day, exploration access.
  Shaft (options of various sizes) located north of the orebody for production.
  Cost of total project for 3,500 tpd/d:

·· Ramp	=	10,675,000 $
·· Undergr. expl.	=	3,869,000 $
·· Shaft 4.25 m	=	18,724,000 $
·· Crusher	=	4,657,000 $
·· Mine developm.	=	10,665,000 $
Subtotal	=	48,590,000 $

·· Mining = 48,309,000 $
·· Conveyor ramp = 7,656,000 $
Total = 104,555,000 $
Conveyor ramp is parallel to the exploration ramp.
· Shaft at 21.0 million $ with bored raise at 3.2 million $.

The information shown should give the reader a feel for the variety of data that has been accumulated over the years. In spite of all the information available, there are three important areas that need clarification before a proper economic analysis can be made and a final decision taken.

- Management of Faro must weigh the importance of the environmental questions concerning Blind Creek.

- The possibility of finding a portal for the Vangorda ramp at a lower elevation.

- The possibility of finding a better location than Blind Creek and Vangorda.

It is expected that these questions will be resolved during the next visit by this writer to Faro in April.

9.11 Time Schedules and Cost Estimates.

It is reasonable to delay completion of this section until the key issues mentioned above have been taken care of, and realistic unit costs are available.

Costs presented by Canadian Mine Development (CMD) are high compared to known unit costs for shaft sinking and ramp advance.

For comparison, see costs obtained from the Canadian Mining Journal's Mining Source Book from 1988 and 1996, shown in Appendices I and II in this section of the report.

CMD's unit price for ramp advance has been given as $5,790 to $6,795 per metre, which is much higher than the prices shown in Appendices I and II.

One explanation for the large difference would be the fact, that some of the jobs listed in the Mining Source Book had been done in house by the mining company with no capital depreciation included. CMD, as contractor, would include an equipment write off plus profit margins.

Appendix I

Ramp Development in Canada, 1988

1989 Mining Source Book

Published by Canadian Mining Journal

Company, Mine	Size (m)	Grade	Avg. Daily Advance (m)	Unit Cost ($/m)
Algoma Steel, MacLeod	3.0X4.0	16%		
Cominco Ltd., Polaris, Conveyor	5.0X5.0	20%	8.0-12.0	558.00
Dickenson, AW White	3.0X3.0			784.00
Falconbridge, Lockerby, service	4.3X3.0	15%	2.0	
Falconbridge, Lockerby service, haulage	4.5X4.0	10%	2.0	
Falconbridge, Strathcona	4.0X4.3	15%	7.6	
Giant Yellowknife, No. 1, service	4.6X3.0	17%	3.0	656.00
Hemlo Gold, Golden Giant	4.5X4.5	16%	6.4	914.40
Hudson Bay, Chisel Lake	4.0X3.4	13%		
Hudson Bay, Stall Lake	4.0X3.0	20%		

Company, Mine	Size (m)	Grade	Avg. Daily Advance (m)	Unit Cost ($/m)
Hudson Bay, Trout Lake	2.4X2.4	13%	1.8	
Inco Ltd., Copper Cliff South	4.0X3.7	20%		
Inco Ltd., Creighton No. 3	4.0X3.7	15% 18%		
Inco Ltd., Levack	4.0X3.7	20%		
Inco Ltd., Little Stobie	4.0X3.7			
Inco Ltd., Stobie	5.0X3.7	20%		
Kiena Gold Mines	5.0X3.5	15%	3.7	1,110.00
LAC Minerals, Doyon Main	4.3X3.4	15%	6.0	900.00
Lynn Gold, MacLellan	3.4X4.6	18%	3.0	1,200.00
Minnova, Ansil	4.9X3.7	8%		1,214.00
Minnova, Laura Lake Project	3.0X4.6	17%	3.0	1,840.00
Noranda, Norita	3.7X5.0	12%		
Renabie Gold Mines	3.0X4.0		1.5	
SMDC, Star Lake	3.0X4.0	15%	4.6	1,260.00

Company, Mine	Size (m)	Grade	Avg. Daily Advance (m)	Unit Cost ($/m)
Societé Miniére Louvenu	3.4X2.0	16%	4.0	
Teck-Corona, David Bell	3.7X5.0	15%	6.0	
Total Erickson Resources	3.0X3.0	14%	8.2	985.00

Appendix II
Ramp Development in Canada, 1995

1996 Mining Source Book

Published by Canadian Mining Journal

Company, Mine	**Size (m)**	**Grade**	**Avg. Daily Advance (m)**	**Unit Cost ($/m)**
Algoma, MacLeod	2.7X4.3	5%	6.7	558.00
Audrey, Bouchard Hebert	4.5X3.7	21%	7.0	803.00
Barrick, Bousquet No. 2	5.0X4.2		5.7	1,400.00
Claude, Seabee	3.4X4.6	17%	6.6	
Echo Bay, Lupin	5.0X3.4	15%		1,640.00
Goldcorp, Red Lake	4.0X2.7	17%	3.2	
Heath Steel, B-Zone	4.9X3.0	15%	5.6	
Heath Steel, C-Zone	4.8X3.9	15%	5.1	
Hemlo Gold, Golden Giant	4.5X5.0	15%		1,150.00
Homestake, Eskay Creek	4.3X3.1 3.8X3.1	15% 15%	2.4 2.4	3,000.00 2,800.00
Hudson Bay, Ruttan	4.7X4.7	16%	19.0	1,255.00

Company, Mine	Size (m)	Grade	Avg. Daily Advance (m)	Unit Cost ($/m)
Hudson Bay, Trout Lake	5.5X4.2	14%	8.0	
Inco, Birchtree	5.2X4.0	+17%	4.9	
Inco, Copper Cliff	4.9X4.9	20%		9,900.00
Iuco, Frood	4.3X4.3	10%	7.6	
Inco, Little Stobie	4.6X4.9	15%	2.7	
Inco, Stobie	4.9X3.7	15%	6.0	
Inco, T-1	4.6X3.2 3.7X3.2	15% 15%	4.3 3.7	
Inco, T-3	4.4X4.3	17%	5.8	
Inmet, Winston Lake	3.9X4.5	15%	3.0	1,237.00
Mines d'OrKiena	4.5X5.0	15%	3.1	1,300.00
Mines Silidor	4.5X3.8	15%		714.00
Noranda, Norita	4.2X4.2	15%	6.7	1,312.00
Placer Dome, Campbell	4.0X4.0	15%	4.6	
Royal Oak, Giant	2.7X3.0	15%		722.00

Company, Mine	**Size (m)**	**Grade**	**Avg. Daily Advance (m)**	**Unit Cost ($/m)**
Royal Oak, Pamour No. 1	3.2X4.9	17%	3.4	
Teck-Corona, David Bell	4.0X4.4	15%	9.6	1,483.00
TVX Gold New Britania	4.6X3.4	15%	6.7	1,050.00

10. UNDERGROUND EXPLORATION.

Underground exploration would clearly prove the existence of an orebody and is needed to define mining areas for production. Geological and geotechnical information would be obtained from the ore zone as well as from the hanging wall and foot wall in order to lay out a test stope, allow selection of suitable mining method and develop proper mine plans.

Exploration drifts in the footwall or hanging wall, some distance above or below the ore zone, will permit fan drilling of some 7 to 10 holes every 30 m.

Time and money can be saved by exploring the B-Zone from an exploration drift in the hanging wall. It would also be possible to investigate the mineralized zone of the upper level from a hanging wall drift.

The Grum underground exploration program, referred to in Chapter 13.4.3. page 80, gives an indication of a similar effort made during the early stages of underground exploration for the Grum pit.

The details of the Dy underground exploration program will be designed at Faro while underground access advance is in progress.

The need and practicality of supplying 100,000 to 200,000 tonnes of ore for a pilot test in the Faro concentrator requires further scrutiny.

11. MINING METHODS.

It is, in the opinion of the author, premature to expect anything but a general discussion of mining concepts at this time, since the information needed for the development of specific methods for the Dy does not yet exist.

A simple method of mining the ore efficiently and economically at the highest extraction rate (without backfill if possible), should be the objective.

Some form of panel and pillar mining with pillar robbing in retreat, leaving post pillars as temporary support, is a cheap mining method that could be contemplated.

Mining with backfill could be considered in certain high grade and thick areas of the orebody, where this method might be the best technical and economical solution and could offer opportunities for bulk mining as well.

Little is known about the geological structure of the hanging wall. It is not possible to identify vertical joints through surface drill holes. It is, therefore, suggested by this author to plan on the use of roof support instead of relying on protecting mine openings with a skin of ore one metre thick, as was done at Faro underground and suggested in some reports. Ore left in the roof would amount to a loss in recovery of 8 %.

N.D. Rose, Fox Geological Consultants Ltd. (Chapter 13.4.7, page 86), has proposed several concepts of mining, and has also presented an underground mine plan. The suggested mining method is concrete pillar mining.

Using a compressive strength of 14 to 35 MPa perpendicular to foliation for hanging wall material in his calculation of ore recovery, Rose suggests an extraction rate of around 50 % or less. He suggests the use of cemented backfill to increase ore recovery to about 85 % or more. Calculations seem to be based on the "Tributary Area Theory". This theory is, as has been pointed out by D. Coates, valid only for deposits where the width of the orebody is greater than twice the depth, which is not the case at Dy.

Should mining with backfill be required, the following considerations would require a technical and economical evaluation:

- Backfill methods possible for Dy
  - Paste fill
  - Hydraulic fill
  - Dry fill

- Source of backfill material
  - Underground waste
  - Mill tailings
  - Open pit waste
  - Surface material to be excavated, if possible, close to the mine

- Transportation to the mine.

- Transportation underground.

- Placement in the stope.

- Availability and cost of cement.

12. ENVIRONMENTAL CONSIDERATIONS.

Chapter 9.7.3.3, page 9-25, deals with the environmental impact of the location of the ramp, and deems a ramp portal for production at Vangorda to be much better placed than a portal in the Blind Creek valley.

Management at Faro is familiar with permitting requirements and will look after these aspects of the new project.

13. COMMENTARY ON AVAILABLE INFORMATION.

13.1 Summary.

Chapter thirteen consists of the following four sections:

- 13.1 Summary
- 13.2 Reports and Information concerning Ore Reserves
- 13.3 Reports and Proposals by Canadian Mine Development (CMD)
- 13.4 Other Information

The reports mentioned in section 13.2 form the backbone of all information available concerning the Dy deposit.

The CMD proposals in section 13.3 contain valuable background information. It could be questioned whether all this information should have been amassed in the first place, when one considers the slenderness of the drilling information on which it was based. However, it constitutes a useful source of reference.

Again, the reports listed in Section 13.4 offer valuable background information. No further comments are needed here, as the statements made in the paragraph above apply in this case too.

13.2 Reports and Information concerning Ore Reserves.

13.2.1 Summary.

The knowledge gained from the reports included in this section and listed below is the basis for understanding the Dy deposit as we see it today and has been used in all presentations, and throughout the present report.

The reports are factual and need no further comment. The following seven reports were available for review:

- 13.2.2 Assay Plots
  1991
- 13.2.3 Estimate of Geological Reserves
  June 1991
- 13.2.4 Mineral Inventory
  December 1991
- 13.2.5 Mineral Inventory
  May 1993
- 13.2.6 Grum Deposit
  March 1977
- 13.2.7 Grum Deposit
  May, 1993

13.2.2 Assay Plots 1991

Vertical cross sections of all diamond drill holes at a scale of 1 : 1,000 exist. The cross sections are collected in three volumes that make up one of the most important sources of assistance in the interpretation and perception of the Dy deposit.

13.2.3 CRI - DY DEPOSIT
Estimate of Geological Reserves
June 1991
N0.WH 9103

Report by P.J. Chornoby and C.V. Reed. The report consists of one volume of text with the following appendices included in the same volume:

Appendix I CRI 1991 Polygon Reserve Calculation
Calculation tables

Appendix II CRI 1991 Reserves within Pelly River Mines Claim
Boundaries
Calculation tables

Appendix III CAMC Dy Reserve Calculation
Hall 1981
Bound separately, was not available.

Appendix IV CAMC Dy Reserve Calculation
Rollings 1982
Bound separately, not available

Appendix V Kilborn Ltd.
Dy Reserve Calculation
Coltas 1989

Appendix VI CRI Dy Exploration Potential at
6% cutoff above AB Zone
6% cutoff below AB Zone
8% cutoff above AB Zone
8% cutoff below AB Zone

Appendix VII Dy Deposit
Summary Drill Logs Core Assays
1976 to 1991

Appendix VIII Dy Deposit
Reserve Polygons
Plan View
Scale = 1:2000

13.2.4 CRI - DY DEPOSIT
Mineral Inventory
December 1991
No. WH9103

Appendix I CRI 1991 Polygon Mineral Inventory Calculation
Calculation tables

Appendix II CRI 1991 Mineral Inventory Within Pelly River Mines
Claim Boundaries
Calculation Tables

Appendix III CAMC Dy Reserve Calculation,
Hall 1981
Summary only, the full report is bound separately

Appendix IV CAMC Dy Reserve Calculations,
Rollings 1982
Summary only, the full report is bound separately

Appendix V Kilborn Ltd.
Dy Reserve Calculation
Coltas 1989

Appendix VI Dy - Additional Potential Mineralization above and below the AB Zone

Appendix VII Dy Deposit - Additional Drill Core Assays
1976 to 1991

Appendix VIII Curragh Resources Inc.
1991 Mineral Inventory Calculation Composites

Appendix IX Dy Deposit - Mineral Inventory Polygons
Plan View
Scale = 1 : 2000
6 % Lead + Zinc Cutoff
8 % Lead + Zinc Cutoff
9 % Lead + Zinc Cutoff

Appendix X Dy Deposit - AB Zone Composites
Mineral Inventory
Vertical Long Sections
Scale = 1 : 1,250
Volume II, No. WH9103

Appendix XI Dy Deposit
Mineral Inventory
Vertical Long Sections
Scale = 1 : 1,250
Volume III, No. WH9103

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

13.2.6 Grum Deposit,
March 1977
Mineral Inventory

13.2.7 Curragh Inc.
May 1993
Geology, Mineral Inventory and Reserves of the Grum Deposit

13.3 Reports and Proposals by Canadian Mine Development.

13.3.1 Summary and Comments.

The following seven proposals from Canadian Mine Development (CMD) have been reviewed:

- 13.3.2 June 1988
- 13.3.3 March 1989
- 13.3.4 April 1989
- 13.3.5 September 1990
- 13.3.6 October 1990
- 13.3.7 January/February 1991
- 13.3.8 December 1992

Each CMD proposal generally consists of an outline of the offer, cost estimates, including backup information, and drawings with technical details.

Alternatives for reaching the Dy Deposit are presented as well as ways of exploring, developing and mining the orebody.

Various combinations of shaft and ramp access are considered and costed out.

The data on hand constitutes good background information. With the exception of Blind Creek, none of the proposals fits the requirements of today, or would be of use now. Costs are generally too high.

The development of the property was proposed carried out in two phases. Phase I includes development of access to the deposit and underground exploration (drifting and drilling) leading to the decision point for the "go ahead".

Phase II includes development work for production and mining. Most offers include contract mining.

A number of alternative proposals for getting underground were considered and costed out. At one time, a decline from a low point in the Blind Creek valley was the favoured access route for underground exploration. Shaft access was supported at other times.

The last proposal, dated December 1992, suggests a production shaft with a diameter of 6 metres located in the centre of the orebody to be used to accomodate underground exploration. A bored emergency-ventilation raise was proposed for the second entry to the mine.

CMD is a mining and shaft sinking contractor. Their own interest and expertise show in the presentation of the proposals. They lean towards accessing the Dy deposit by means of a vertical shaft and contract mining for production.

The wishes of the owners are strongly reflected in the different proposals. The key factor for selection of one method over another at any particular time seems to have been strongly influenced by the scheduling required for getting underground and into production.

A brief summary of the seven proposals follows:

· **Proposal of June 1988:**

Ramp from Blind Creek, portal elevation 835 m, 20 %, 1,735 m long, enters orebody at 400 m(?), 43 weeks.
(Note: The ramp should be 2,115 m long, in order to reach a depth of 400 m)
Production 2.3 million tonnes.
Underground exploration 25 weeks.
Shaft located within the orebody.
Alternate shaft location west of the orebody.
Two ventilation raises, one at the northeast edge of Block 2, another at the southern border of the orebody.
Room and Pillar Mining, 12 m rooms and 8 m pillars

· **Proposal of March 1989:**

Rectangular, timbered shaft, north of orebody, 660 m deep, bottom at 440 m for underground exploration and production.
Bored raise located close to the shaft for ventilation and emergency exit.

Alternative:
Switchback ramp to start at elevation 925 m to intersect future shaft at midpoint to accommodate shaft construction. Length of ramp is 2,200 m at 20 %.
Ramp for underground exploration, shaft for production.

· **Proposal of April 1989:**

Rectangular shaft for underground exploration and future production. Shaft is north of the orebody. Depth is 668 m, shaft bottom is at 440 m with the loading pocket at 480 m.
Cost = $20,626,140, construction time = 30 months
Exploration drilling, underground development and loading pockets to cost an additional $7,815,466.

Bored raise for ventilation and emergency exit.
Production = 1 million tonnes per year.
= 3,000 tonnes per day.

Summary of mining methods and production cost:

· Room and pillar pilot	52.4 %	19.73 $/t of ore
· Room and pillar bench	27.6 %	15.53 $/t of ore
· Longhole mining	10.0 %	16.99 $/t of ore
· Slusher stoping	10.0 %	32.65 $/t of ore
Average mining cost		19.50 $/t of ore

- **Proposal of September 1990:**

Various studies are presented, including a 2.7 million tonne per year production rate with ramp to be driven first for underground exploration.

For production, the following variations are suggested:

- 6 m diameter production shaft for two skips and a service cage with a waste skip as counterweight
- Ramp plus a 5.0 m diameter production shaft for two skips and an emergency cage with counterweight
- Ramp plus a 4.25 m production shaft for two skips and a manway
- Exploration ramp plus a second, parallel ramp for ore transport

The shaft location selected is north of the orebody.
Attempts were made to drill shaft pilot holes.

The proposal also contains details about mining.

· **Proposal of October 1990:**

Shaft pilot holes to the north, outside the orebody, have intersected a major fault at the designated depth of the loading pockets. A future shaft should now bottom out at elevation 520 m. Underground exploration drilling is now planned to be carried out from a drift in the hanging wall.

Access is now again favoured by a ramp from Blind Creek to take 15 months at a cost of $10.1 million.

Options under consideration (shafts to be concrete lined, inside diameter):

· 4.00 m diameter shaft = $15.6 million

· 4.25 m diameter shaft = $16.5 million

· 4.75 m diameter shaft = $18.4 million

· Shaft first - underground exploration - ramp second

· Ramp first - underground exploration - shaft second

· Ramp first - underground exploration - second ramp with or without conveyor installation

· Truck haulage in the ramp can obtain 1,340 - 2,500 TPD

· **Proposal of January/February 1991:**

Ramp access
Cost estimates for contract mining
Cost estimates for mining by the owner

· **Proposal of December 1992:**

Phase I includes the sinking of a shaft of 737 m to reach the deposit, and underground exploration. The shaft is designed for production. Ventilation is initially provided by dividing the shaft through the installation of a curtain wall.

Phase I - Details:

· Shaft to be sunk to elevation 390 m

· Underground exploration on levels 400, 440 and 480 m

· Shaft to be extended to elevation 350 m

· Completion of infrastructure for Phase I (see details of item 13.3.8)

· Total cost of Phase I = $28.8 million

· Total time = 32 months

Phase II consists of deepening the shaft to the final depth, boring a vertical raise for ventilation and emergency exit, and development for production.

Phase II - Details:

· Shaft to be deepened to elevation 190 m

· Construction of bored ventilation-escape raise

· Completion of infrastructure for Phase II (see details of item 13.3.8)

· Cost of shaft deepening = $5.8 million

· Cost of vent. escape raise = $4.6 million

Mining Operation:

· Operating cost estimate in $/t for 3,500 tpd:

	Contractor (rented equipment)	Owner (rented equipment)	Owner (capitalized equipment)
Room & Pillar Jumbo, ST-8	25.48	22.10	20.31
Jackleg, ST-3.5	26.60	23.04	21.74
Jackleg and Slusher+ST-3.5	30.28	26.19	25.19

· Capital cost estimate in $:

Jumbo, ST-8	6,325,000
Jackleg, ST-3.5	4,035,000
Jackleg, slusher, ST-3.5	3,425,000

The seven offers present alternatives for reaching the orebody. The last offer proposes the option of the production shaft as first access, with a vertical bored raise as second exit.

Of the various possibilities proposed by CMD, only the Blind Creek ramp is still under consideration. However, for environmental reasons this ramp is unacceptable for ore haulage. Furthermore, it does hardly create opportunities for finding additional ore reserves that could be mined at an early stage. It could serve as exploration access and for ventilation.

The CMD offers contain many good ideas. Much work has gone into the preparation of the various proposals. The concepts presented concerning mining are limited by the low level of information available. The deposit was then, as now, insufficiently understood for the development of proper mine plans.

It is a pity that only one company was involved in bidding for the work at Dy.

13.3.2 Exploration, Mine Development and Production June 1988

One document, drawings only, text was not available for review.
Drawing 88-24-001 shows an exploration ramp with the portal at elevation 835 m, a shaft location within the ore body, an alternate shaft location on the west side of the ore body as well as mining blocks. Production rate is 2,300,000 tonnes per Year.

Site preparation plus the excavation of a 1,735 m long ramp at 20% is scheduled to take 43 weeks. For underground drifting, plus exploration drilling to define Block B2, an additional 25 weeks are allowed.

Drawing 88-24-004 shows a typical mine plan for block B2 with the last section of the ramp entering the orebody at 18 %, one ventilation raise at the northeast edge of Block B2 and a second one on the southern border of Block B2a, as well as a shaft located in the centre of the ore body. This mine plan calls for room and pillar mining with 12 m rooms and 8 m pillars. Various cross sections form part of this report.

1,735 m exploration ramp with portal near Blind Creek at elevation 835 m to take 43 weeks. 1,410 m of exploration drifts, cross cuts and drill stations to take another 20 weeks. This is to be followed by exploration drilling. The decision point of phase II of the project would be reached 66 weeks after start up.

13.3.3 Exploration Access Alternatives March 1989

This document consists of one volume made up of a short text and a number of drawings. It contains a comparison of the time and cost of access by shaft versus access by ramp.

The shaft is located in bedrock north of the orebody, and provides reasonable access to the B Zone. A vertical ventilation raise is bored close to the shaft and provides for mine ventilation and serves as second mine access during production.

The shaft has a depth of some 660 m, with the collar at about 1,100 m and the bottom at 440 m. The timbered shaft is rectangular, 25' 6" x 9'0" (7.75 m x 2.75 m) with a skip/cage compartment, a skip compartment and a service compartment with manway.

As an alternative version of shaft/ventilation raise, the proposal considers a 4.27 m x 4.57 m (14' x 15') ramp at -20 %, which would have the portal at elevation 925 m and a total length of 2,200 m. The ramp would intersect the location of a future shaft at mid-shaft to allow for shaft excavation once the production decision has been made.

13.3.4 Exploration Shaft Access. April 1989

Under this heading two different proposals, consisting of four documents, have been reviewed. The first proposal, of April 1989 is dated April 6, 1989. Changes to this proposal were initiated at a meeting on April 7. These changes are reflected in the proposal dated April 10, 1989.

In the second proposal, various items in the first version have been altered, mainly to simplify the project and speed up its execution.

Access to the ore horizon for underground exploration and future production is planned with a rectangular shaft, 23'10" x 9'0" (7.25 m x 2.25 m), and two 10 ton skips.

- Production rate = 1,000,000 tonnes per year
- Hoisting rate = 3,000 tonnes per day
- Shaft bottom = 440 m

- Loading pocket = 480 m
- Pump station = 800 m
- Total depth = 668 m
- Shaft location = north of the ore body.
- Cost of Phase I = $ 20,626,140 (includes shaft, headframe, hoist and underground exploration)

Production development, loading pockets and additional exploration drilling is estimated to cost $ 7,815,466.

The shaft will be divided by a curtain wall to provide initial ventilation. At a later stage a ventilation raise will be bored. Total construction time is 30 months.

Initial production will be from the high grade B2 block. Minimum mining height is 3.5 m. It is anticipated that 60 % of production will be from room and pillar pilot mining and 40 % from benching. Other alternatives include:

Ore production:
- Room and pillar mining with benching in the thicker areas.
- Longholing in areas of drag folds.
- Slusher stopes in low mining areas. Minimum mining height = 3.5 m.

Based on available drill data, the orebody is divided into eight blocks as follows:

	Tonnes	% Pb+Zn
Block 2	3,979,837	14.88
B2 Horizon	2,507,877	10.14
A2 West Block 1	2,151,170	14.48
A2 West Block 2	971,652	10.62
A2 East Block	2,954,192	13.77
A 2.4 Block	1,346,638	13.11
A2 Horizon	6,187,977	10.29
3A Horizon	960,601	12.28
Total	21,059,960	12.28

Production Schedule (1,000 tonnes):

Year:	1	2	3	4	5	6	7	8
Block								
B2	500	1,000	700	300				
A2.4			300	300				
A2West				400	600	600	350	
A2East					400	400	650	350
Other								650
Total	500	1,000	1,000	1,000	1,000	1,000	1,000	1,000

This proposal calls for a total of 7.5 Mio tonnes of ore in the first 8 years. It is based on contract mining, where the contractor would supply and maintain all equipment.

Basic data:

3,000 tonnes per day

room and pillar pilot	52.4 %	at 19.73 $ per tonne
room and pillar bench	27.6 %	at 15.53 $ per tonne
slusher stoping	10.0 %	at 32,65 $ per tonne
longhole mining	10.0 %	at 16.99 $ per tonne

Average cost is $ 19.50 per tonne of ore.

13.3.5 Exploration and Mining Cost Estimate. September 1990

This document consists of two volumes of text and drawings dated September 10, 1990 and September 21, 1990, and one volume containing backup calculations for the proposal.

Various possibilities of access by ramp and shaft have been analyzed. A ramp driven with drill jumbo, scoop tram and truck provides the earliest access for underground exploration and shows the lowest costs among the proposals discussed in this section.

The following variations were considered:

- Ramp plus 5.0 m production shaft (ore skips and emergency cage)
- 6 m production shaft (two ore skips, plus service cage with waste skip as counterweight)
- Second conveyor ramp

- Ramp plus 4.25 m production shaft, ore skips and manway

Mining costs at a production rate of 2.7 million tonnes per year were also developed in this study.

An exploration ramp with a cross section of 4.0 m x 5.5 m at -20 %, using trucks, allows for the best costs and the shortest time to reach the exploration phase. The selection of the best system for mine access and production ore haulage will be made once underground exploration is completed. The ramp will be driven in 14.5 months at 4.5 m/day.

Mining costs at 7,500 tpd are given as $21.45 with conveyor haulage in a ramp. The cost calculations include incremental costs for ore haulage from the portal to a stock pile.

For the option "shaft access", a site north of the ore body was selected as shaft location. Options include:

- 6.0 m circular shaft fully concrete lined, with one double drum hoist for the cage and a double drum hoist for skips. Installation of a crusher-loadout system. The second exit is provided by raise boring and equipping a 4 m borehole at the east end of the orebody.
- 22'00" x 14'6" (6.70 m x 4.40 m) Rectangular Shaft.
- 4.0 m Exploration Shaft.
- Ramp and 5.0 m Circular Shaft.
- 5 m Circular Shaft.

13.3.6 Exploration and Mining Cost Estimate. October 1990

The document consists of two volumes dated October 22, 1990 and October 29, 1990.

Access by ramp is favoured, but shaft investigations continue.

The shaft pilot hole intersected a fault zone below the ore zone. Consequently, the shaft bottom was raised from elevation 480 m to elevation 520 m.

Exploration drilling will now be accomplished from a drift in the hanging wall.

For the completion of the ramp, 15 months and $10.1 million are estimated. For truck haulage in a ramp, rates are given as 1,800 tonnes per day, and with larger trucks as 2,500 TPD.

The production rate would be higher with a conveyor. However, for truck haulage prior to the completion of the second ramp and a conveyor installation, the capacity is given as 1,340 tpd.

Cost of shafts:

4.00 m diameter	=	\$15.6 million
4.25 m diameter	=	\$16.5 million
4.75 m diameter	=	\$18.4 million

Volume 2 of this proposal, entitled "Initial Exploration and Development Program", dated October 29, 1990, suggests the following options:

- Shaft first - then underground exploration - ramp second
- Ramp first - then underground exploration - shaft second
- Ramp first - then underground exploration - second ramp complete with conveyor

13.3.7 Cost Update. January/February 1991

This document consists of two volumes, the first dated January 30, 1991 and the second February 1, 1991. They contain backup information for the detailed cost estimates of the execution of actual work as well as the cash flow required.

The volume containing the scope of work with the necessary drawings were not available to this author.

The cost figures in the study dated January 30, 1991 show owner's involvement. The figures in the February document are based on all work to be done by contractor. Costing is for an access ramp 4.16 m x 6.20 m at 20 % and a total length of about 1,730 m to allow access to the orebody at an elevation of 490 m.

13.3.8 Exploration Shaft and Development. December 1992

This proposal consists of two volumes, both dated December 14, 1992. The first volume contains the text of the proposal and drawings, while the second volume is made up of the cost backup.

This study proposes access to the mine through a shaft with a bored vertical raise for emergency exit and ventilation. The shaft is located in the zone of barren massive sulphides in the centre of the orebody, while the raise would be at its eastern edge. The shaft collar is at elevation 1,127 m. The shaft has an inside diameter of 6 m and is concrete lined. A brattice wall is installed in the shaft to facilitate ventilation until completion of the bored raise at the eastern edge of the orebody.
Phase I calls for the shaft to be sunk to elevation 390 m to allow for exploration on the 400, 440 and 480 m levels (drawing 92-DY-32). Transportation on the levels is by rail to reduce ventilation needs. The shaft will be extended to 350 m while exploration on the three levels is in progress. The 440 level and the 480 level will be reached via a +20 % ramp in the ore.

Summary of proposal:

· Shaft	=	737 m
· 440 Level X-Cut	=	125 m
· Ramp from 390 to 480, 3.0m x 4.m	=	440 m
· 400 Level Exploration Drift	=	760 m
· 440 Level Exploration Drift	=	795 m
· 480 Level Exploration Drift	=	460 m
· Raises	=	56 m
· Diamond Drilling "BX"	=	15,000 m
· Track Installation	=	2,140 m
· Cost Estimate for Phase I	=	$28.8 Million
· Time for the execution of Phase I	=	32 months

Additional infrastructure for Phase I includes:

- Pump station at 760 m level
- Shaft station plus pump station at 400 m level
- Rockbreaker-grizzly plus ore pass at 480 m level
- Rockbreaker-grizzly plus ore pass at 440 m level
- Conveyor on the 400 level to the coarse ore bin
- Coarse ore bin at 57^0 and a capacity of 800 tonnes
- Waste bin for 150 tonnes
- Loading pockets at the 360 m level
- Ramp to the shaft bottom

Phase II provides for the shaft bottom to be extended to final depth at elevation 190 m. The cost estimate for deepening the shaft to this level is $5,820,973; for the ventilation raise-escapeway an additional $4,556,747 is required.

Additional infrastructure for Phase II includes:

- Haulage levels at 360, 320, 280 and 240 m levels
- Haulage drift and ore passes to the grizzly at the 240 m level
- Rockbreaker and grizzly on the 240 m level
- Coarse ore bin at 57^0 and a capacity of 800 tonnes
- Waste bin for 150 tonnes
- Loading pockets at the 220 m level
- Pump station near shaft bottom
- Ramp to shaft bottom

There are no headframe bins provided on surface. Ore (and waste) is dumped on the ground and must be rehandled with a front-end loader.

Operating Cost for 3,500 tpd and the following options:

- Contractor Operated
- Owner Operated - Rental Equipment
- Owner Operated - Capitalized Equipment

Mining Cost includes mining, ore handling underground and hoisting.
Mining will be done in the B Zone in the Northeast part of the orebody (see drawing 92-Dy-18).

Polygon	Tonnes	% Pb+Zn	Remarks % Pb+Zn
32	813,000	15.16	> 15
4	820,000	15.08	> 15
6	1,248,000	15.98	> 15
5	642,000	12.70	> 12
50	133,000	12.68	> 12
26	404,000	12.99	> 12
34	924,000	12.59	> 12
49	137,000	12.03	> 12
47	587,000	10.04	> 10
51	254,000	11.13	> 10
33	566,000	10.32	> 10
45	497,000	8.94	> 08
15	234,000	8.19	> 08

Total tonnage considered for the first stage of mining:

Tonnes	Average Grade % Pb+Zn	Cumulative Tonnage	%Pb+Zn
2,881,000	**15.49**		
2,240,000	12.66	**5,121,000**	**14.26**
1,407,000	10.35	**6,528,000**	**13.41**
731,000	8.70	**7,259,000**	**12.06**

Summary of mining cost in $ per tonne:

	Contractor using Rental Equ.	Owner using Rental Equ.	Owner using Cap.Equ.
Room & Pillar Jumbo, ST-8	22.10	21.80	20.31
Room & Pillar Jackleg, ST 3.5	26.60	23.04	21.74
Room & Pillar Jackleg, Slusher ST 3.5	30.28	26.19	25.19

Information reflecting ideas about mining are repeated here to provide general knowledge.

13.4 Other Information.

13.4.1 Summary.

Information available and not incorporated in sections 13.2 and 13.3 has been reviewed and is specified in this chapter. Some documents included have only been scanned. Documents are listed without much attention to chronological order or relative importance.

Information presented in documents of this section has also been used in other chapters of the present report.

The following reports have been reviewed:

- 13.4.1 Summary
- 13.4.2 Faro Underground, Daily Records
  1990
- 13.4.3 Cyprus Anvil, D.J. Hanson
  Early 1980's
- 13.4.4 Kilborn Limited
  April 1987
- 13.4.5 Kilborn Limited
  October 1989
- 13.4.6 Rescan
  March 1991
- 13.4.7 Fox Geological Consultants
  October 1992
- 13.4.8 Hydrogeological Consultants Ltd.
  July 1989
- 13.4.9 Curragh Inc.
  May 1993
- 13.4.10 Curragh Resources
  1991
- 13.4.11 Curragh Resources
  1986
- 13.4.12 Curragh Inc.
  Geology, Mineral Inventory, and Reserves of the Grum Deposit
  May 1993
- 13.4.13 PSA- Proposal
  August 1995
- 13.4.14 Lakefield Research
  Recovery of Lead, Zinc, and Silver from Dy Samples
  September 1992

13.4.2 Faro Underground Daily Records

There are six boxes containing the daily records of production and development advance in the storage room at the mine office.

This author had a brief look through some of the boxes, but did not take the time to go through the information collected.

An example is shown here for September 2, 1990:

· Daily advance summary:

·· Development 16'x 16' (4.90 m x 4.90 m):
12' 10' 8' 7.5' 9' 10' 8' 10'

Total development = 74.5'

·· Production 16'x 12.5' (4.90 m x 3.80 m) at +20%

·· Total excavation = 24,227 ft^3

·· Total production : = 2,670 ton (24,227 ft^3 x 0.11)

·· Manpower = 59

·· Underground map sheet - irregular geology

The Dy deposit is expected to be quite different from Faro underground. In spite of this, one can probably find details in the Faro daily reports that will be of interest at Dy.

The information available should be examined by the people who will be involved in the Dy Project whenever time permits.

13.4.3 Cyprus Anvil, D.J. Hanson
The Development of the Vangorda Plateau Ore Deposits

This report has no date, but was probably written in the early 1980's. It contains valuable background information.

Example: Initial surface exploration at the Grum deposit consisted of 17,000 metres from holes drilled 60 metres apart on section lines that were 120 metres apart.

Surface and underground Exploration: *Upon completion of the initial drilling phase, an extensive underground development program was started. The deposit was explored by means of an 800 metre decline on a 16 percent grade. Two ramps, 120 metres apart, were driven along the plunge of the mineralization with connecting crosscuts at 120 metre intervals. Ring drilling in the plane of the geologic section was conducted from the ramps at 60 metre intervals. The program included 2,900 metres of openings and 15,000 metres of diamond drilling.*

Concurrent with the underground development, a further 24,000 metres of surface diamond drilling was carried out to complete the grid at 60 metre spacing on lines 60 metres apart from section 60W to 88W. Fill-in holes at 30 metre spacing were also drilled on section 64W and 68W.

The strike length of the Grum deposit examined from underground is 700 metres.

Ore Haulage: *Various ore haulage systems were examined to determine the most dependable and cost efficient means of transporting 1.7 million tonnes of ore per year from the Vangorda Plateau to the existing mill site; a distance of approximately 14 kilometres. In consideration of possible mill expansions, and the requirement to produce the entire mill ore supply from the Vangorda Plateau in the future, each of the systems investigated would be capable of providing increased haulage rates to meet project requirements.*

A preliminary engineering study was commenced in the fall of 1979 to evaluate a number of potential feasible ore haulage systems which included:

a) Trucking utilizing off-highway equipment

b) Rail transportation

c) Conveyor by means of conventional belt system

d) Trucking using smaller highway trucks

e) Conveying ore on a cable belt

A feasibility study of the haulage system was conducted by Swan Wooster Engineering Co. Ltd. in early 1980. The final choice was between off-highway trucking and cable belt conveyor.

13.4.4 Kilborn Limited, Toronto, Ontario
April 1987

The title of the report is "CURRAGH RESOURCES, LONG RANGE PLAN FOR FARO, VANGORDA AND GRUM DEPOSITS". The report consists of three volumes:

Volume I	Summary
Volume II	Technical Review
Volume III	Capital and Cost Estimates

Only volume II was reviewed. Although the study is confined to the Faro, Grum and Vangorda deposits, it does contain information of value for Dy.

Planning Faro underground is included. Development will consist of a main ramp, mining levels, and a ventilation raise.

The ramp is 4.3 metres high and 5.2 metres wide, and at a grade of 15 % will be 1,220 metres long. There will be 579 metres of drift on the 3400 ft level and 550 metres on the 3200 ft level. The bored ventilation raise will have a diameter of 2.4 metres, a total length of 245 metres and is equipped with a manway for emergency escape. Fresh air will enter the mine through the ventilation raise at a rate of 300,000 cfm. Mine air heating will be provided.

Personnel for the underground operation is estimated to consist of a total of 100 people:

· Supervision and Technical	14
· Operating Labour	72
· Maintenance Labour	14
Total	100

For the Grum Deposit, the overall density of drilling is 15 metres by 30 metres, with local areas being even closer. Geological blocks for mine planning are 8.0 metres across deposit trend, 4.5 metres high and 15.0 metres along deposit trend.

Due to the wide spacing of drill holes in most Anvil District deposits, geostatistical analyses have been difficult. At Grum the drill pattern is dense enough to produce meaningful analysis once the basic geological data is organized into thirty metre spaced interpretation.

For the Vangorda deposit, total exploration drilling of 98 holes is reported, producing a 60 metre by 30 metre drill pattern.

13.4.5 Kilborn Limited, Toronto, Ontario
Review of Mineral Properties of Curragh Resources Inc. and Affiliates
October, 1989

Kilborn personnel have provided assistance to Curragh since the Faro Division's activities started at the end of 1985. The report presents the 11 year mine plan for the Faro Division from 1989 to 2000.

It includes the in-place mineral reserves, mining recovery from in-place reserves, mining operations, mill feed grade, mill recovery, and concentrate grade for the existing operations at the Faro open pit.

Also addressed were the Vangorda and Grum open pits, the Faro underground and the Dy underground deposits, and the milling of ore from these new sources.

Initial mill capacity in the early 1970's was 5,000 tonnes per day. This was increased in 1974 to 6,000 tonnes per day. Additional major mill modifications were carried out in 1980 and 1981 in order to increase throughput to 9,100 tonnes per day. Curragh started milling operations in 1986 at a designed throughput rate of 11,400 tonnes per day.

The Faro underground mine is scheduled to produce 1,178,000 tonnes of ore over three years.

In the 11 year mine plan, the Dy deposit was supposed to be developed in the early 1990's to supply ore to the mill by 1992. By the year 2000, ore delivery from Dy to the concentrator was supposed to have been a total of 8,500,000 tonnes.

Grum Underground: *When the currently planned Grum open pit has been mined, ore reserves will remain below and around the pit floor and walls and mineral inventory will remain in the contiguous North West Extension. It is likely that some of these resources, above a cut-off grade of about 9 % lead plus zinc, could be mined by underground methods. Access could be by ramp(s) driven from the pit floor or by shaft or possibly a combination of service ramp and small, ore hoisting shaft. The Grum underground is a low priority project, since the open pit is scheduled to produce through 1999.*

In the late 80's, concentrator throughput was over 12,500 tonnes per day. This is equivalent to 615 tonnes per hour at 85 percent operating time.

Based on the work indexes and grind requirements of the various orebodies, power required (kWh/t) and maximum tonnage throughput (t/h) based on 90% of full power draw at the pinions are presented here:

	Faro	Grum	Vangorda	Dy
Work Index kWh/t	13.6	13.4	10.0	6.0-12.0
Grind K80 Microns	74	65	45	40
Power required at Pinion kWh/t	14.82	15.7	14.19	9.05-18.10
Maximum Tonnage Throughput t/h	610	576	637	999-499

Mixing of wet/warm ore from underground with cold open pit ore is not recommended. Stockpiling of massive sulphide ores for extended periods of time will result in oxidation, which will adversely affect mill recoveries.

Estimates of operating costs and capital expenditure for all areas of activity are included in the report.

Appendix A consists of the "ESTIMATE OF GEOLOGICAL ORE RESERVES FOR THE DY DEPOSIT", prepared by P.C. Coltas.

13.4.6 Rescan, Project Description March 1991

The report is titled "ADVANCED EXPLORATION AND DEVELOPMENT OF THE DY UNDERGROUND MINE" and was prepared by Rescan Environmental Services Ltd, Vancouver, British Columbia.

It deals mainly with environmental issues and mitigative measures for all facets of the Dy Project, and describes permitting procedures for the various activities. Each operation includes a section on the impacts of closure and decommissioning.

The Rescan report states as an accepted fact that the mine will be accessed by the Blind Creek ramp. The report has been issued in draft form. Some statements by Rescan are quoted here for general information.

Exploration Cost to Date	:	*Approximately $12 million*
Development Cost	:	*Approximately $35 million*
Estimated Total Capital Cost	:	*Approximately $75 million*
Mining Method	:	*Combination of Room and Pillar* *Longhole Stoping* *Cut and Fill*
Production Rate	:	*7,500 tpd maximum* *1,500 tpd initially*
Proposed Mine Life	:	*15 years*
Reserves Geological	:	*21 million tonnes at 5.8 % Pb, 6.8 % Zn, 83 gr/t Ag, and 0.94 g/t Au*
Mineable (prel.)	:	*11.3 million tonnes*
Potential for Additional Reserves	:	*Significant (10 mio. tonnes)*
Construction Workforce	:	*40*
Operation Workforce	:	*120*

Development of the Dy Project has been structured in two phases, advanced exploration, and production. The two phases will be separated by a production decision once the feasibility of mining is established. During the advanced exploration phase, access to the deposit will be gained by excavation of a 1,700 m long decline.

Development of the decline portal site will cause a limited amount of surface disturbance from the portal collaring and the initial access construction.

Development of the site will also subject the area to the possibility of erosion in the event of spring freshet or summer rainfalls. The runoff, if allowed to leave the site without mitigation, could have implications on the water quality of Blind Creek and the valuable fisheries habitat which it supports.

The technicalities of settling ponds for the various stages of ramp advance and underground drifting are mentioned. Reference is made to the requirement of having a qualified professional geotechnical engineer present for inspections during construction of the settling ponds to ensure stability.

A description of the ramp, technical details of the equipment and the execution of ramp construction follow. Work is to be done in three 8 hour shifts with a total crew of about 26.

Development to a point 1,700 m from the portal will be done in an effort to encounter the main orebody known as the B Zone.

Once the B Zone is reached, a drift will be driven to the northwest into the B Zone to permit drilling, test mining and bulk sampling in this area.

The bulk sampling program for piloting purposes will involve 100,000 tonnes from the B Zone.

The initial operation will begin at 1,500 tonnes per day and will gradually increase to 3,500 tonnes per day. It is likely that much of the initial mining will be room and pillar in areas of shallow dip. Long hole stoping will probably be developed in fold noses. Cut and fill will be used in narrow, high grade areas.

Ventilation raises will also be installed during this component of the exploration program; one will be installed on the decline adjacent to the C Zone, should drifting into that zone occur, and another will be installed where the drift to the B Zone takes off to the northwest.

A conveyor will be installed in the decline during the development to facilitate the delivery of ore and waste from the excavation and bulk sampling operation to the surface. A floor mounted conveyor system is planned which will be capable of delivering 3,500 tonnes per day to the surface. A final decision on the timing of the conveyor installation has not yet been made.

One of the objectives for the advance exploration and initial mining phases will be to evaluate the deposit for potential shaft locations close to the deposit centre of gravity. Once a suitable site is established and a sufficient reserve base is established, a shaft will be raised from the underground working to the surface and a headframe/hoist system established with sufficient capacity to hoist up to 7,800 tonnes per day.

Access to the portal of the decline and the shaft would consist of a standard haul road with a 30 metre running surface flanked by 1 metre safety berms.

Mitigative measures for the protection of the Blind Creek water shed are mentioned frequently in the Rescan report.

13.4.7 **Fox Geological Consultants, Vancouver, B.C.**
October 1992
DY DEPOSIT, MINEABLE RESERVE ESTIMATE AND UNDERGROUND MINE PLAN, by N.D. Rose

As indicated by its title, the report gives a description of the Dy Deposit, shows calculations of mineable ore reserves and deals with underground mine planning.

Based on available information, the report covers these three areas very well. The need to obtain more information is frequently referred to. The study was done in the summer of 1992. At that time the Faro underground operation was still fresh in the minds of the people on site. The authors had the benefit of using the Faro underground experience for Dy.

The study shows the Blind Creek Ramp, and a shaft in the massive sulphides in the centre of the orebody. It indicates a cone of 10 degrees for a shaft safety pillar. It is suggested to give particular emphasis to polygons 78X01 and 79X09, which contain above 9% lead + zinc cut-off grade quartzites in the apparently massive sulphide zone local to the shaft.

Some relevant paragraphs are quoted here:

Adjustments in the grade composites for polygons 80X04 and 80X13 would have lead to their inclusion into the mineable inventory at a 9% lead + zinc cut-off grade.

The Dy deposit appears to be a complex, but poorly defined orebody. Grade variation and zonation of massive sulphides is evident and may pose problems in mining. A more detailed investigation of the geology of the deposit is warranted.

Much more work is necessary to adequately define Dy reserves and the 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.

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.

The Dy underground mine plan includes comments on the geotechnical background, selection of mining method, underground mine design and mining sequence, and the development-production schedule for eleven years.

13.4.8 Hydrogeological Consultants Ltd.
July 1989
Curragh Resources Ltd., Grum Pit
1989 Dewatering Assessment

The study was carried out for the surface conditions of the Grum Pit. It is difficult to relate the findings to an underground operation. The information did, however, contribute to the present report.

13.4.9 Curragh Inc.
Curragh Mining Properties
Claim/Lease Holdings
May 30, 1993
Source of additional information for this report.

13.4.10 Licence IN 89-002, Curragh Resources
Vangorda Plateau Mines
1991 Annual Report
to Yukon Territory Water Board
Whitehorse, Yukon

Source of additional information for the report.

13.4.11 Curragh Resources
Introduction to Curragh Resources
The Faro Ore Body, Mining and Milling Operations
Probably 1986

Provided background information for this report.

13.4.12 Curragh Inc.
Geology, Mineral Inventory and Reserves of the Grum Deposit
May 1993

13.4.13 PSA Proposal for Ore Slurry Transport System
August 1995

Prepared by: PSA - International Joint Venture Group, comprised by *Salzgitter Anlagenbau* in association with *Proton International Engineering Corporation*

This proposal, if carried out, will strengthen Dy Property economics.

13.4.14 Lakefield Research
Recovery of lead, zinc and silver from Dy samples
September 1992