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CURRAGH RESOURCES INC.

REVIEW OF
MINERAL PROPERTIES OF
CURRAGH RESOURCES INC.
AND AFFILIATES

OCTOBER, 1989

Submitted By:

KILBORN LIMITED
2200 Lake Shore Blvd. West
Toronto, Ontario
M8V 1A4

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CURRAGH RESOURCES INC.

REVIEW OF
MINERAL PROPERTIES CONTROLLED BY
CURRAGH RESOURCES INC.

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1.0 INTRODUCTION, SUMMARY AND CONCLUSIONS

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1.1 INTRODUCTION

This review of the properties and operations owned or controlled by Curragh Resources Inc., (Curragh) has been carried out at the request of Mr. Marvin Pelley, Executive Vice President - Mining, of Curragh. The executive offices of Curragh are located at 95 Wellington Street West, Toronto, Ontario.

Kilborn personnel have provided assistance to Curragh since the Faro Division's activities began at the end of 1985. This included provision of staff on site until permanent employees could be hired. More recent study assignments have kept some of Kilborn's staff familiar with Curragh's mineral projects.

This report may be used in its entirety by Curragh for any legitimate purpose, but the report may not be condensed or excerpted without the written consent of Kilborn Limited (Kilborn).

1.1.1 Scope of Work

Kilborn addressed Curragh's 11-year plan for the Faro Division entitled "Mine Plan S89: Alpha II" including the in-place mineral reserves, mining recovery from in-place mineral reserves, mining operations, mill feed grade, mill recovery, and concentrate grade - for the existing operations at the Faro open pit. Also addressed were the projected development and operation of the Vangorda and Grum open pits, the Faro underground and Dy underground deposits, and the milling of ores from these new sources.

In the area of Faro Division support facilities, Kilborn investigated the arrangements for power supply to the operations, employee housing, employee relations, transportation of concentrate to the ocean port of Skagway, and loading of the concentrate at Skagway.

Kilborn reviewed the projected capital expenditures and operating costs for the Faro Division over the 11-year period from 1990 to 2000 inclusive.

There are a number of other mineral deposits, at various stages of exploration and development, which were included in the assets purchased from Cyprus Anvil and, more recently, from other sources. The more important of these deposits are addressed in this report, including the Swim deposit in the Faro Division and the separate Cirque, Mt. Hundere and Westray projects. Comments on these properties are based on earlier reports by Kilborn or by independent engineers.

1.1.2 Faro Division Properties

Curragh was formed in 1985. In November of that year, Curragh and Cyprus Anvil Mining Corporation (Cyprus Anvil) closed on their agreement whereby the assets of Cyprus Anvil were sold to Curragh. These assets included the Faro lead-zinc-silver mine and extensive mill facilities, plus the Town of Faro in the Yukon. Except for a small part of the Town of Faro, the foregoing assets were not in use and had been prepared for indefinite cold storage.

The acquisition also included the Vangorda, Grum, Dy and Swim deposits of the Faro Division in the Yukon, and control of the Cirque lead-zinc-silver deposit in British Columbia, plus extensive mining claims in western Canada and the associated geological information.

Starting in late 1985, Curragh reactivated the Faro mine and mill complex, and concentrate production began in mid-1986. During 1987, the first complete year, Curragh mined 29.3 million tonnes of ore and waste; the mill treated 4.5 million tonnes of ore to produce 185 thousand tonnes of lead concentrate (containing 4.2 million ounces of silver) and 348 thousand tonnes of zinc concentrate. The comparable figures for 1988 were: 23.4 million tonnes of ore and waste mined, 4.1 million tonnes of ore milled, 202 thousand tonnes of lead concentrate produced (containing 3.5 million ounces of silver), and 312 thousand tonnes of zinc concentrate produced. It is noted that production was affected adversely by a four-week strike in mid-1988.

Pit development work is under way at the Vangorda and Grum deposits; ore mined from these deposits will replace a large part of the ore now being mined from the Faro pit which will be almost completed by the end of 1991.

1.1.3 Other Properties

In April 1989, Curragh purchased an 80 percent joint venture interest in the Mt. Hundere lead-zinc-silver property located 45 kilometres north of Watson Lake, Yukon Territory, some 550 kilometres southwest of Faro. Site investigations are in progress and an ore reserve estimate is being prepared.

The Cirque property, acquired from Cyprus Anvil, is located in northeastern British Columbia, 280 kilometres north of MacKenzie. An underground exploration and development program is in progress, scheduled for completion during first quarter 1990. This will provide the necessary sampling and metallurgical testing, mine planning and design data required for a feasibility study.

Curragh has an agreement in principle to acquire, from an affiliated company, control of the Westray coal project located in Pictou County, Nova Scotia, near the town of New Glasgow. A 1987 feasibility study for Suncor Inc. (the previous owner) indicated a viable project. Development work began via twin declines but has been temporarily suspended pending resolution of financial arrangements.

1.2 SUMMARY AND CONCLUSIONS

This report relies upon geological and property holdings data provided by Curragh, and upon open pit designs for the Faro and Swim deposits prepared by Mr. Ion Vintila, an independent consultant retained by Curragh.

Kilborn prepared independent check calculations of diluted, mineable ore reserves, in the proven and probable classifications, which provide confirmation of the adequacy of Curragh's ore reserve estimate. Specifically, Kilborn's total tonnage estimate is within 4 percent of Curragh's total, and the average lead, zinc and silver grades and the contained metal totals are all within 10 percent of Curragh's corresponding figures.

Curragh's estimates of operating costs were reviewed for specific mining methods and locations, and in aggregate over the life of the 11-year Mine Plan. No corrections are considered necessary.

Curragh's design cut-off grades of 4 percent combined lead plus zinc for open pit mining and 9 percent for underground mining were reviewed. Based on Curragh's operating cost estimates, current metal prices, recent historical mill recoveries, and the assumption that revenue is 65 percent of metal value contained in concentrate, these cut-off grades were found to be between 15 percent and 30 percent above breakeven. These are suitable values for mine design. Kilborn endorses Curragh's proposal to stockpile material containing 3 to 4 percent lead plus zinc which must be removed from within the pit boundary.

Curragh's open pit mining plans, schedules and equipment requirements were reviewed and accepted with the provision that the maintenance department must be brought up to, and maintained at, the scheduled force.

Curragh's plans for underground mining of the Faro and Dy deposits are based on proposals from a contractor which have been reviewed by Kilborn and accepted with the reservation that additional waste development may be required at Faro depending upon the degree of irregularity of the ore horizon.

Curragh's estimates of capital expenditures over the 11-year period were reviewed. Kilborn has recommended an increase of $3.8 million in the mill capital in 1990 over the amount shown in the Mine Plan, and has expressed reservations regarding the development cost of the Faro Underground deposit.

In general, Curragh's estimates for the Faro Division are considered to be adequate.

The Mt. Hundere, Cirque and Westray Coal properties were reviewed on a project basis. Kilborn has not researched ownership nor royalties on any of the properties described in this report.

2.0 OVERVIEW OF FARO DIVISION GEOGRAPHY, ENVIRONMENT AND HISTORY

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2.1 GEOGRAPHY

2.1.1 Location

The position of Curragh's Faro operation, with relation to the other centres in the Yukon territory, is shown on Figure 2-1. Whitehorse, a city of approximately 20,000 people, is the largest community - the total population of the territory is near 30,000.

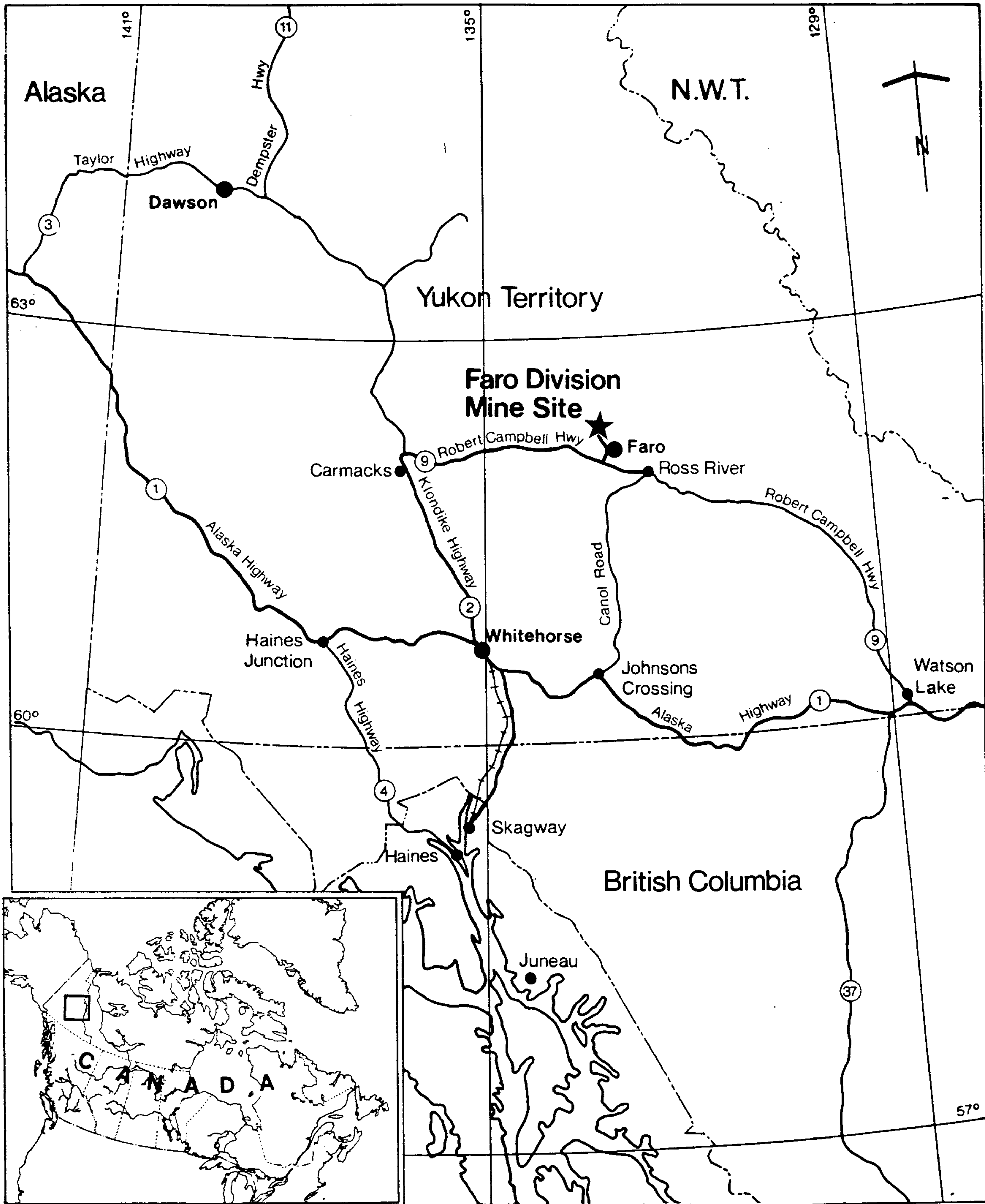
2.1.2 Services

Whitehorse is the distribution centre for the Yukon, and there are good support facilities for construction and for machinery repairs. Inland Canada can be reached via the Alaska Highway to Dawson Creek, British Columbia, and from there to Edmonton, Alberta, or to Vancouver, British Columbia. An ocean port, Skagway, can be reached by highway south from Whitehorse. Substantial distances of both routes are paved, open year round and are suitable for heavy highway transport vehicles.

2.1.3 Access

The town of Faro is connected to Whitehorse by paved highway, open year round, a distance of approximately 450 kilometres via Carmacks. From Faro, a gravel road - capable of accepting heavy highway transport vehicles - leads to Curragh's Faro mine and plant site.

There is an airstrip at Faro which is suitable for small aircraft. Many chartered aircraft use this airstrip and there are some scheduled flights to Whitehorse.



TITLE: LOCATION MAP	SECTION:	
KILBORN	AREA NO:	REV. NO:
CLIENT: CURRAGH RESOURCES — PROJECT NO: 3680-15	DRAWING NO: FIGURE 2-1	A
APPROVED: — DATE: MAY-89		

2.1.4 Town of Faro

Before the previous operators closed the mine in 1981, the Town of Faro was a thriving community of more than 2,000 people. The facilities in the town at that time were: grade school, high school, post office, social service office, liquor store, municipal service centre, fire hall, medical centre, bank, hotel, department store, supermarket, garages, convenience type retail outlets, curling rink, recreation centre, playgrounds, union hall, Legion hall and Royal Canadian Mounted Police detachment.

Before Curragh started to re-activate the mining operation in late 1985, the population of Faro had fallen to less than 100 people. The federal, territorial, and municipal services remained in a somewhat diminished fashion, and the hotel, an automotive station and a convenience store remained open.

After Curragh started its re-activation program, the various facilities in the town were re-opened, as the demand increased, until today there is available, in town, virtually every service that existed before the mine shut down in 1981. Curragh sold off all but about seven of the more than 400 company houses.

2.2 ENVIRONMENT

2.2.1 Climate

The climate is sub-arctic, with long cold winters and short warm summers. Total annual precipitation is about 400 millimeters per year, divided almost equally between snow in winter and rain in summer.

Whiteouts from snow in winter or fog in summer had not been significant problems until the unusual winter of 1988/89.

In late December and early January, there is less than four hours of daylight, and in late June and early July, there is almost no darkness.

2.2.2 Topography

In this part of the Yukon, a broad valley runs roughly north-south with mountain ranges on both sides. The Pelly River drains this valley to the north, into the Yukon River system.

The Town of Faro is located beside the Pelly River, at approximately 750 metres above sea level, and the mining sites are approximately 300 metres higher. The terrain, in the vicinity of the town and the mining sites, is hilly but not rugged.

2.2.3 Land Use

The cold climate limits agricultural use of the land to a few home gardens. There is some useful timber in the Yukon, though not in the immediate mine area, but the long time for regrowth indicates that it may be unwise to consider harvesting any trees.

Pack horses are left outside to graze all winter, but it is doubtful if a viable cattle ranching industry could be supported on the vegetation available.

Game is fairly plentiful in the area and sports fishing is available year round.

2.3 HISTORY

2.3.1 Ownership

Cyprus Mines Corporation (Cyprus Mines) of Los Angeles, California, joined with Dynasty Exploration Ltd. (Dynasty) in late 1965 to conduct exploration activities in the Vangorda Creek area of the Anvil Range. Upon the successful delineation of 60 million tonnes of lead-zinc-silver ore, known as the Faro Deposit, the Anvil Mining Corporation Ltd. was formed to complete the exploration and development of these reserves, and to direct the design and construction of mining and milling facilities. Cyprus Mines maintained a controlling interest of approximately 60 percent, with Dynasty holding the remaining shares in the Company. Mining operations were started in 1969 and milling commenced in 1970.

An amalgamation of Anvil Mining Corporation Ltd. and Cyprus Mines took place in 1975 whereby the Cyprus Anvil Mining Corporation was formed, 63 percent owned by Cyprus Mines.

In 1979, Standard Oil (Indiana) bought Cyprus Mines, thereby acquiring the latter's interest in Cyprus Anvil. Following unsuccessful negotiations between Amoco Minerals Co. (a subsidiary of Standard Oil) and the Canadian Foreign Investment Review Agency, Standard Oil sold its interest in Cyprus Anvil in 1981 to Hudson's Bay Oil and Gas, controlled by Continental Oil, based in Calgary, Alberta. Shortly thereafter, Dome Petroleum purchased Hudson's Bay Oil and Gas, thus acquiring the majority interest in Cyprus Anvil.

The operations were shut down at the end of 1981, although some pit waste was removed in 1983 and 1984. Dome Petroleum publicly stated its wish to divest its interest in Cyprus Anvil Mining Corporation.

In May 1985, C. H. Frame and associates entered into an agreement with Dome Petroleum to evaluate the assets for purchase, and for reopening of the mine and mill. In November 1985, Curragh Resources, a private partnership of which Mr. C. H. Frame was Chairman and Chief Executive Officer, purchased all the assets of Cyprus Mining.

2.3.2 Production

The initial (early 1970s) milling capacity was 5,000 tonnes per day, and 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 9100 tonnes of ore per day. In June 1986, Curragh commenced milling at a designed throughput rate of 11,400 tonnes per day. There was a concurrent upgrading in the mining equipment as the milling capacity was increased.

The principal production statistics since 1970 are shown below:

Year	Ore and Waste Mined BCY* (000)	Ore Milled DMT** (000)	Concentrate Produced DMT (000)
1970	6,344	1,779	242
1971	7,006	2,425	389
1972	5,168	2,636	402
1973	5,570	2,630	425
1974	6,456	2,654	388
1975	5,677	2,926	411
1976	3,693	1,520	183
1977	8,500	3,116	358
1978	10,483	3,280	414
1979	8,121	2,823	368
1980	9,344	2,825	326
1981	11,357	2,737	313
1982	000	000	000
1983	9,700	000	000
1984	4,500	000	000
1985	000	000	000
	MT***(000)		
1986	23,440	1,943	199
1987	29,336	4,539	532
1988	24,232	4,126	516

* BCY - Bank Cubic Yards

** DMT - Dry Metric Tonnes

*** MT - Metric Tonnes

3.0 PROPERTY HOLDINGS, FARO DIVISION

3.0 PROPERTY HOLDINGS, FARO DIVISION

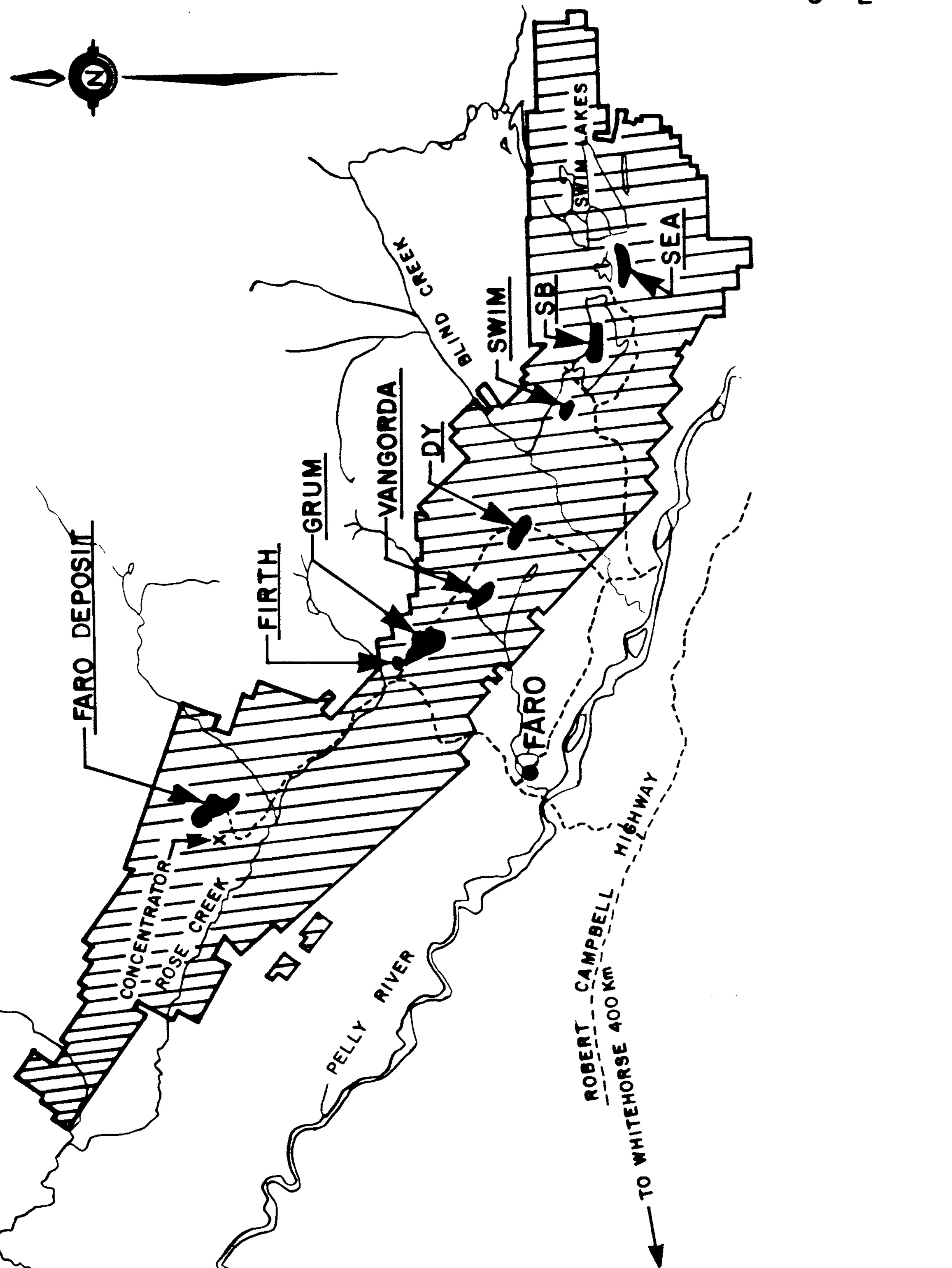
The information in this section was provided by Curragh. Kilborn did not verify ownership and expiry dates of the mining claims or the mineral leases, nor did Kilborn verify the precise locations of the mineral claims or deposits.

The boundary of the principal block of mining claims, held by Curragh in the Faro district, is shown on Figure 3-1. The claim block is approximately 60 kilometres long and, at most points, more than 5 kilometres wide.

Full claims usually are 1,500 feet by 1,500 feet, although these dimensions can vary. There are 2,159 full claims and 224 fractions of claims within the claim block. Curragh holds 1,539 full claims and 129 fractions outright, and the others are held with other parties or are included in joint ventures. Certain of the claims have been converted to mineral leases.

Curragh holds 12 mineral leases that cover the Faro deposit and the immediate vicinity. A portion of the potential Faro underground mine is not included in the present leases. This portion is held as a mining claim and can be converted to a mineral lease. The current status of the 12 mineral leases is shown here:

Mineral Lease Number	Claim Name	Claim Number	Grant Number	Expiry Date
3427	Faro	39	92225	16 Nov 2009
3428	Faro	41	92227	16 Nov 2009
3429	Faro	42	92228	16 Nov 2009
3430	Faro	43	92229	16 Nov 2009
3431	Faro	44	92230	16 Nov 2009
3432	Faro	45	92231	16 Nov 2009
3433	Faro	46	92232	16 Nov 2009
3434	Faro	53	92239	16 Nov 2009
3435	Faro	54	92240	16 Nov 2009
3436	Faro	55	92241	16 Nov 2009
3437	Faro	56	92242	16 Nov 2009
3438	Whi	8FR	94573	16 Nov 2009



TITLE FARO AREA -- CLAIM BLOCK		SECTION	
KILBORN		AREA NO:	REV. NO.
CLIENT CURRAGH RESOURCES	PROJECT NO: 3680-15	DRAWING NO: FIGURE 3-1	A
APPROVED	DATE: OCT. 89		

The Faro deposit now being mined is located on Mineral Leases 3428 and 3429.

The potential underground mine is located on Mining Claim Faro 66, expiry date March 1, 1997.

In the Grum/Vangorda area, Curragh holds 41 mineral leases which were carried over from the previous owner. These leases cover the Grum and Vangorda deposits, and their current status is shown below:

Mineral Lease Number	Claim Name	Claim Number	Grant Number	Expiry Date	Owner
3195	Bix	2	70440	28 Jan 2006	CR/CNR
3196	Bix	3	70441	28 Jan 2006	CR/CNR
3335	Champ	3	66702	25 Jan 2008	CR/CNR
3336	Champ	4	66703	25 Jan 2008	CR/CNR
3337	Champ	5	66704	25 Jan 2008	CR/CNR
3338	Champ	6	66705	25 Jan 2008	CR/CNR
3206	Chuck	1	66760	28 Jan 2006	CR/KA/CNR
3207	Chuck	2	66761	28 Jan 2006	CR/KA/CNR
3208	Chuck	5	66764	28 Jan 2006	CR/KA/CNR
3209	Chuck	6	66765	28 Jan 2006	CR/KA/CNR
3210	Chuck	7	66766	28 Jan 2006	CR/KA/CNR
3211	Chuck	8	66767	28 Jan 2006	CR/KA/CNR
3329	Elle May	1	66680	25 Jan 2008	CR
3330	Elle May	2	66681	25 Jan 2008	CR
3331	Elle May	3	66682	25 Jan 2008	CR/CNR
3204	Firth	6	66741	28 Jan 2006	CR/KA/CNR
3205	Firth	8	66743	28 Jan 2006	CR/KA/CNR
3200	Grum	1	66752	28 Jan 2006	CR/KA/CNR

Mineral Lease Number	Claim Name	Claim Number	Grant Number	Expiry Date	Owner
3201	Grum	2	66753	28 Jan 2006	CR/KA/CNR
3202	Grum	3	66754	28 Jan 2006	CR/KA/CNR
3203	Grum	5	66756	28 Jan 2006	CR/KA/CNR
2124	Hank	1FR	77898	21 Aug 1994	CR
2125	Hank	2FR	77899	21 Aug 1994	CR/CNR
2126	Hank	3FR	77900	21 Aug 1994	CR/CNR
2127	Hank	4FR	77901	21 Aug 1994	CR/KA/CNR
2128	Hank	5FR	77902	21 Aug 1994	CR/KA/CNR
2129	Hank	6FR	77903	21 Aug 1994	CR/KA/CNR
2130	Hank	7FR	77904	21 Aug 1994	CR/KA/CNR
2131	Hank	8FR	77905	21 Aug 1994	CR/KA/CNR
3197	Rocky	2	66673	28 Jan 2006	CR
3212	Rocky	3	66674	01 Jun 2006	CR/CNR
3213	Rocky	4	66675	01 Jun 2006	CR
3214	Rocky	5	66676	01 Jun 2006	CR/CNR
3327	Rocky	6	66677	01 Aug 2007	CR
3215	Rocky	7	66678	01 Jun 2006	CR/CNR
3328	Rocky	8	66679	01 Aug 2007	CR/CNR
3198	Wynne	1	66684	28 Jan 2006	CR
3332	Wynne	2	66685	25 Jan 2008	CR
3199	Wynne	3	66686	28 Jan 2006	CR
3333	Wynne	4	66687	25 Jan 2008	CR
3334	Wynne	5	66688	25 Jan 2008	CR

CR - Curragh Resources
CNR - Canadian Natural Resources Ltd.
KA - Kerr-Addison Mines Limited

The Vangorda deposit now being developed is located on Mineral Leases 3198, 3199, 3213 and 3330.

The Grum deposit now being developed is located on Mineral Leases 2128, 2129, 3200, 3201, 3202 and 3208.

The Dy deposit, which is scheduled for shaft sinking and underground exploration, is located on the following Mineral Claims:

Claim Name	Claim Number	Grant Number	Expiry Date	Owner
Dy	41	85922	01 March 1999	CR
Dy	43	85924	01 March 1999	CR
Dy	45	85926	01 March 1999	CR
Dy	144	Y4359	01 March 1999	CR
Dy	183	93116	01 March 1998	CR
Dy	184	93117	01 March 1998	CR
Dy	185	93118	01 March 1998	CR
Dy	186	93119	01 March 1998	CR
Dy	43AFR	YA24932	01 March 1992	CR
Que	37FR	Y10675	01 March 1997	CR
Que	32FR	Y10670	01 March 1997	CR
Gale	25	Y67343	01 March 1997	PRM
Gale	27	Y67345	01 March 1997	PRM
Gale	46	Y67364	01 March 1997	PRM

Approximately 5 percent of the Dy deposit lies on the Gale claims which are owned by Pelley River Mines; a company which is controlled by Curragh.

4.0 MINERAL DEPOSITS AND STATE OF DEVELOPMENT

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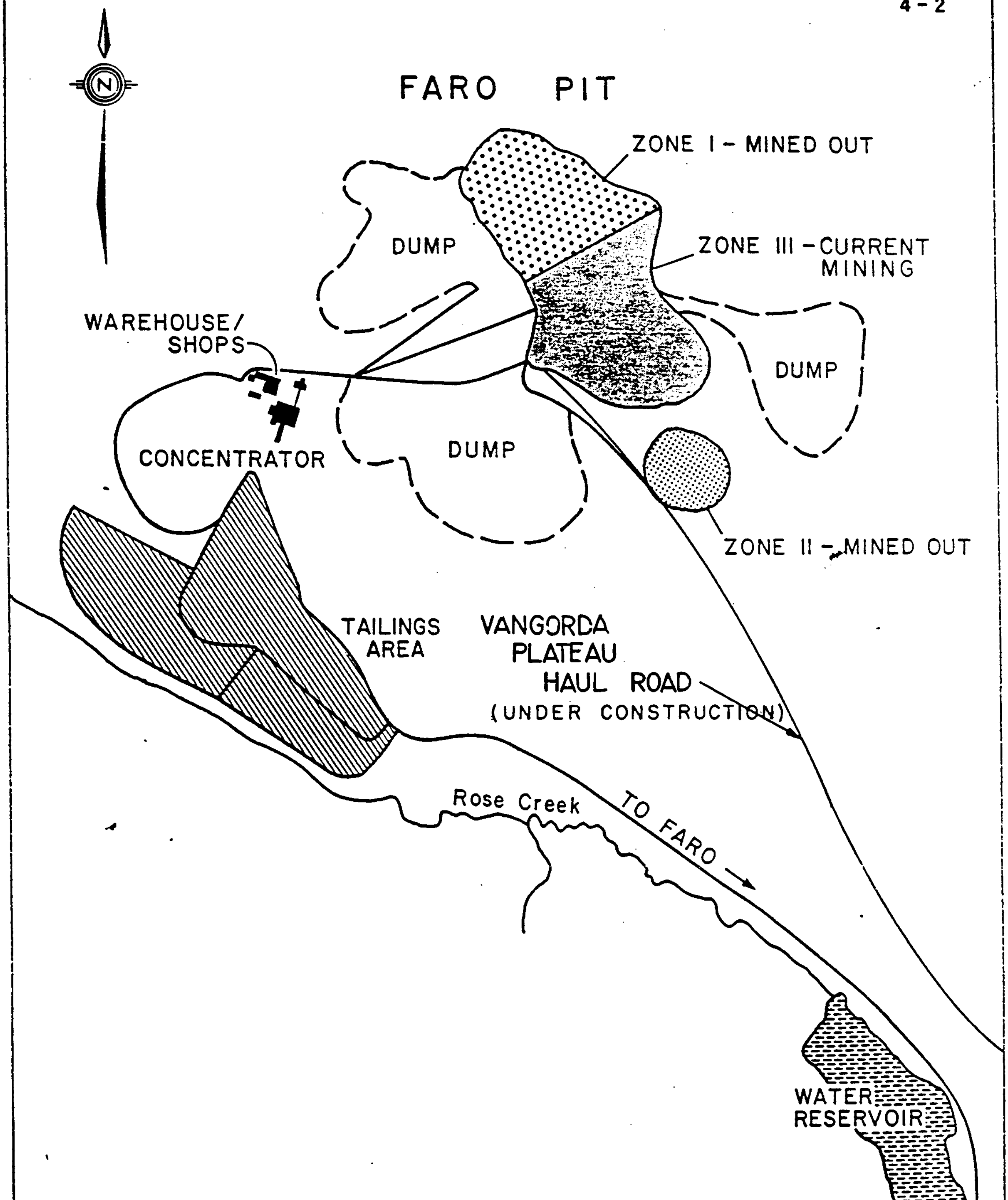
There are numerous known mineral deposits on Curragh's properties in the Anvil District and at Mt. Hundere in the Yukon, and in British Columbia at the Cirque property. In several cases, these occurrences have been subdivided into open pit and underground deposits in terms of the most likely mining method. In addition, Curragh controls the Westray coal project in Nova Scotia.

4.1 DEPOSITS SCHEDULED FOR MINING

The deposits being mined, or scheduled to be mined, under the Mine Plan S89: Alpha II, are all in the Faro Division as follows:

- Faro: Open pit
- Faro: Underground
- Grum: Open pit
- Vangorda: Open pit
- Dy: Underground

Current operations are centered around the Faro open pit (see Figure 4-1). The facilities consist of a large open pit mine, a 12,500 tonnes per day concentrator, a fully equipped maintenance facility, heating plant, offices, water supply reservoir, as well as waste dumps and a tailings disposal area which, together with the Faro pit when mined out, can serve for the projected life of the scheduled deposits. Electric power is supplied by the Yukon Electric Company by way of a high voltage transmission line, connecting to the Yukon power grid near Whitehorse. Curragh has standby diesel-electric generators on site for emergency power.



TITLE: FARO MINE - SITE PLAN		SECTION:	
KILBORN		AREA NO:	REV. NO:
CLIENT: CURRAGH RESOURCES	PROJECT NO: 3680-15	DRAWING NO: FIGURE 4-1	A
APPROVED:	DATE: MAY-89		

Since the Faro open pit was started in 1969, more than 275,000,000 tonnes of material have been excavated. Rates of excavation by Curragh since 1986 have exceeded the rates achieved by the previous owners. The ultimate pit design will be reached in approximately two years without the necessity of continuing at the recent high rates of excavation. It is unlikely that the life of the Faro pit will be extended past the present ultimate pit design.

As the Faro pit becomes exhausted, it is planned to supply the Faro concentrator from other deposits. Run-of-mine ore will be trucked to the crusher at the concentrator. Road connections from the other deposits will be constructed as required.

The Faro underground mine is scheduled to produce from a mineralized zone that extends under a final wall of the Faro open pit. A relatively short development program will be carried out starting from an adit already collared inside the open pit near one of the haulage ramps; ore delivered to the adit collar from underground will be transported to the crusher using existing pit equipment. The Faro underground mine is scheduled to produce 1,178,000 tonnes of ore over three years.

The Vangorda and Grum deposits are located close together, approximately 15 kilometres east of the Faro concentrator. Stripping of overburden and other pit preparation work is in progress for both deposits. A special ore haulage road has been constructed from both directions, using waste from the Faro pit and the Grum overburden stripping; a short extension to the Vangorda pit remains to be completed. This haulage road will accommodate the large pit equipment, and there will be no intermingling with public traffic.

Vangorda, the smaller of the two deposits, is scheduled first for production. Preproduction work involves a modest amount of initial stripping and the diversion of a small creek.

The larger Grum deposit was explored, in part, from an adit driven down through the mineralized zone by a previous owner. Minimal facilities were on site at the time of the underground exploration work, but nothing of significant value was left, and the workings were allowed to flood. Preproduction at Grum involves removal of a large quantity of overburden (this is now under way), followed by a major quantity of waste rock.

The Dy deposit is located approximately 20 kilometres east of the Faro concentrator, and can be serviced by an extension to the Grum/Vangorda haul road. The mineralized zones lie between 500 metres and 900 metres below surface, too deep for open pit mining. The mineral inventory is based on 54 diamond drill holes drilled from surface over the period 1976 to 1981.

Curragh plans to start developing an underground mine on the Dy deposit in 1990, and to start producing ore in mid-1992. The 11-year operating forecast for Curragh's Faro district operations shows 8,500,000 tonnes of ore to be extracted from the Dy underground mine by the end of year 2000.

The ores scheduled from the five foregoing deposits will provide feed at near capacity for the concentrator until the end of the year 2000.

4.2 FARO DIVISION DEPOSITS NOT YET SCHEDULED FOR MINING

Curragh has three known deposits at its Faro Division for which exploration to date has established a mineral inventory but which are not yet scheduled for production. Two of these are extensions of the Grum deposit.

4.2.1 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 portion of these resources, above a cut-off grade of about 9 percent 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 by 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.

4.2.2 Champ Zone

The Champ Zone is an eastward extension of the Grum Main Zone. It is reported to contain a probable geological inventory of 1.7 million tonnes at 3.5 percent lead, 4.3 percent zinc and 46 g/t silver, but this potential open pit material has not yet been evaluated.

4.2.3 Swim Deposit

The Swim deposit is located approximately 25 kilometres east of the Faro concentrator. It is a near surface deposit potentially amenable to open pit mining. From the limited drilling data, it is believed that the Swim deposit resembles the Grum deposit in physical irregularity, but is much smaller and of slightly lower average grade.

A recent preliminary manual pit design by Mr. Ion Vintila, an independent consultant, indicates 3.74 million diluted recoverable tonnes from a mineral inventory of 5.1 million possible tonnes above a 4 percent lead plus zinc cut-off. This must be confirmed by further exploration and detailed evaluation.

4.3 OTHER PROPERTIES NOT YET SCHEDULED FOR MINING

Curragh owns or controls five other properties in addition to those of the operating Faro Division. These are Mt. Hundere in the Yukon; Cirque, Fluke and Elf in British Columbia; and the Westray coal project in Nova Scotia. The Fluke and Elf properties both reached the diamond drilling stage but have no mineral inventories and remain low priority targets.

4.3.1 Mt. Hundere Property

The Mt. Hundere property is located 45 kilometres north of Watson Lake, Yukon, some 550 kilometres southeast of Faro. There are three defined lead-zinc-silver deposits, Jewelbox Hill, Gribbler Ridge and North Hill, and two other known targets.

The 1989 sitework and diamond drilling program is nearing completion and ore reserve calculations are in progress. Portions of the Jewelbox and North Hill (Burnick Zone) deposits appear to be amenable to open pit mining. The balance of the deposits, if feasible, would be mined by underground methods, probably room and pillar.

4.3.2 Cirque Deposit

The Cirque deposit is located in British Columbia, 475 kilometres northeast of Prince Rupert, the nearest ocean port, and 280 kilometres north of MacKenzie, the nearest railhead. Cirque is a massive sulphide/barite tabular deposit, drilled between 1978 and 1982 with 74 holes totalling 23,400 metres.

Preliminary feasibility studies indicate that 86 percent of the 22.2 million tonne mining reserves, above an 8 percent lead plus zinc cut-off grade, could be recovered by room and pillar mining and could support a 3,500 tonne per day concentrator. To further evaluate the Cirque deposit, an underground exploration program is in progress at an estimated cost of $13 million.

4.3.3 Westray Coal Project

The Westray coal project is located in Pictou County, Nova Scotia. In a July 1987 Feasibility Study for Suncor Inc. (the previous owner), Associated Mining Consultants Limited estimated "demonstrated" reserves available for mining of 45 million tonnes of low (0.8%) sulphur coal of which 40 percent could be recovered by room and pillar mining. A wash plant is proposed to process 3,200 tonnes of raw coal per day and produce 1.0 million tonnes of clean coal per year.

Development work began earlier this year via twin declines but has been temporarily suspended pending resolution of financial arrangements.

5.0 GEOLOGY OF THE FARO DIVISION

5.0 GEOLOGY OF THE FARO DIVISION

The geological descriptions and illustrations in this report are reproduced from data and reports supplied by Curragh which Kilborn has relied upon.

5.1 REGIONAL GEOLOGY

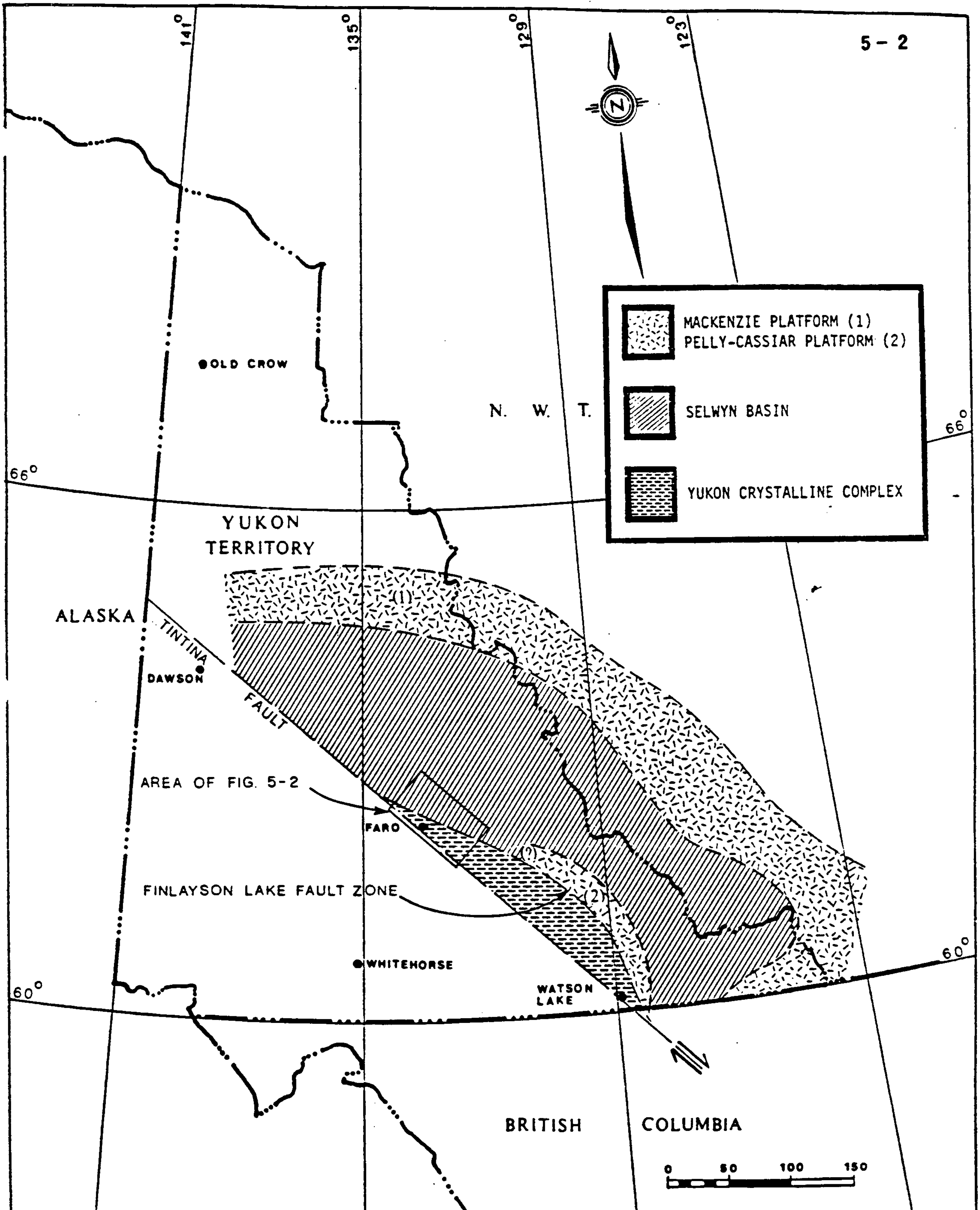
The Selwyn Basin is a large area of the central Yukon where deep water shales accumulated along the ancient North American continental margin during the Paleozoic. Shales of the Selwyn Basin host most of Canada's large stratiform lead-zinc deposits, making it a metallogenic province of worldwide significance.

Mineral deposits near Faro often are referred to as being in the Anvil Range Lead-Zinc-Silver District; the Anvil District is part of the Selwyn Basin. Figure 5-1 shows the regional geology - the Selwyn Basin and vicinity and the location of the Anvil District within the basin.

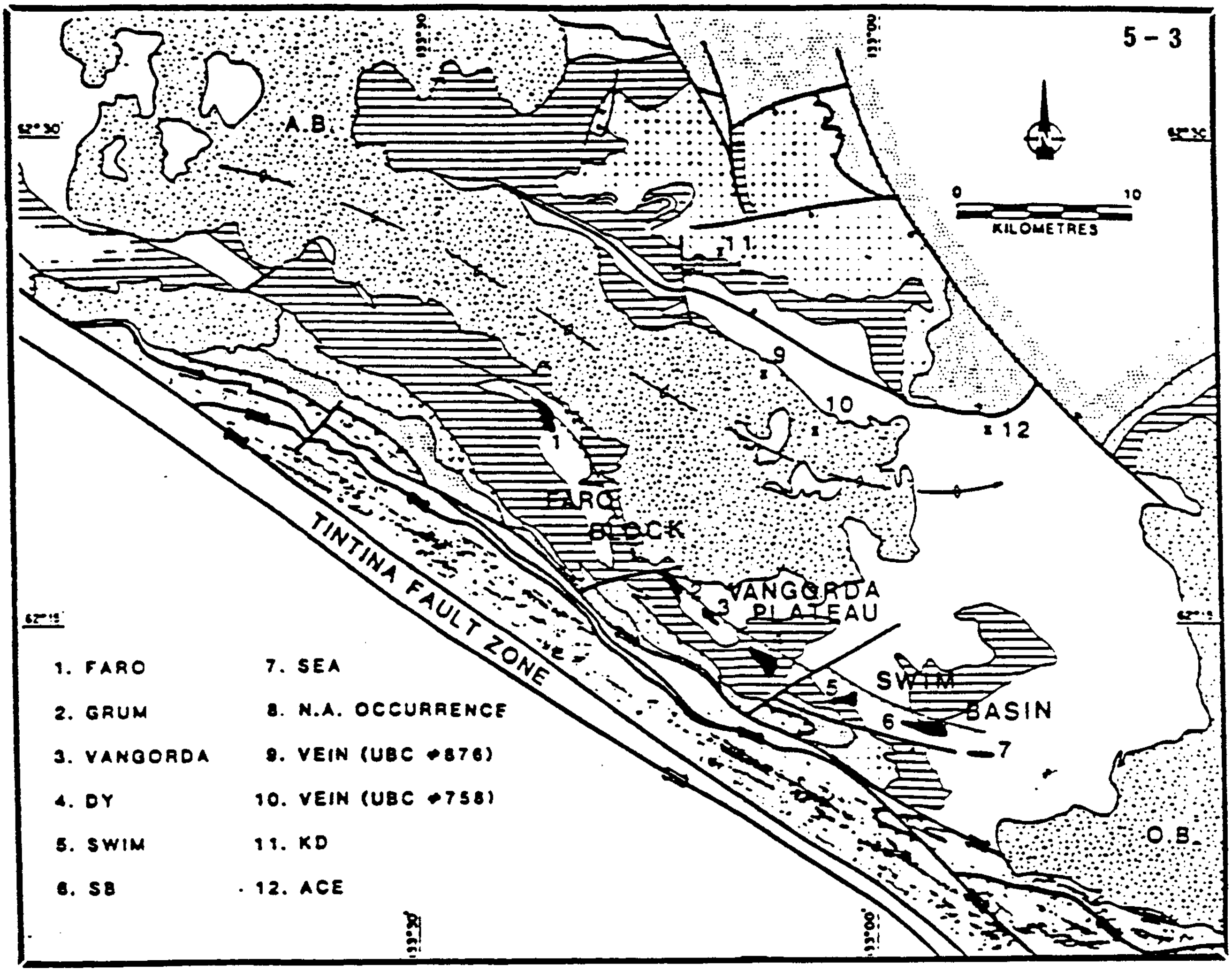
5.2 ANVIL DISTRICT GEOLOGY

5.2.1 General

Unlike the remainder of the Selwyn Basin, the rocks and ores of the Anvil District are metamorphosed and the shales are converted to phyllites and schists. The central part of the district is underlain by a large granitic body that cores an elongate dome exposing the metamorphic sequence (Figure 5-2). The district contains several stratiform, lead-zinc-silver bearing, pyritic, massive sulphide deposits hosted by Cambrian metasediments on the southwest flank of the dome. The Tintina Fault, one of the major right lateral Cordilleran strike slip faults, passes just south of the district (Figures 5-1 and 5-2), but is not directly related to the ores.



TITLE: FARO AREA DEPOSITS REGIONAL GEOLOGY	SECTION:
KILBORN	AREA NO: / REV. NO:
CLIENT: CURRAGH RESOURCES / PROJECT NO: 3680 - 15	DRAWING NO: FIG.— 5 - 1 / A
APPROVED: / DATE: MAY - 89	



contact

normal fault

axis of Anvil Arch

strike slip fault

thrust fault

northeast edge of subbasin

granodiorite and quartz monzonite
AB = Anvil Batholith OB = Orcnay Batholith

VANGORDA FORMATION calcareous phyllite and equivalent calcsilicates, metabasite

EARN GROUP block shale, chert, chert pebble conglomerate, limestone, quartzite (includes undifferentiated Askin Group, Silurian and Devonian dolomite and quartzite locally)

MOUNT MYE FORMATION non calcareous phyllite and schist

MENZIE CREEK FORMATION metabasalt flows breccias and tuffs, graphitic phyllite (includes undifferentiated Road River Group black shales locally)

undifferentiated rocks southwest of Finlayson Lake fault zone includes rocks of Yukon Cataclastic complex, Triassic sedimentary rocks, ultramafic and mafic plutonic rocks and basalt and varicolored chert of Permian or Pennsylvanian Anvil Range Group

TITLE: FARO AREA DEPOSITS ANVIL RANGE GEOLOGY		SECTION:	
KILBORN		AREA NO:	REV. NO:
CLIENT: CURRAGH RESOURCES	PROJECT NO: 3680-15	DRAWING NO: FIG. — 5 - 2	A
APPROVED:	DATE: MAY-89		

5.2.2 Stratigraphy

The stratigraphic sequence of Anvil District ranges in age from latest Precambrian to Permian. Two major divisions or assemblages of strata are present. They are separated by a poorly exposed interval of black shale of uncertain affinity which contains late middle Devonian limestone lenses (Tempelman-Kluit, 1972).

The lower division ranges in age from late Precambrian to perhaps early Silurian. It is approximately 5 kilometres thick and divisible into three major mappable units (Figure 5-3). From the base, these are non-calcareous metapelite of Mount Mye formation, calcareous metapelite of Vangorda formation and basalt and black phyllite of Menzie Creek formation. Established formal stratigraphic nomenclature does not apply directly to this area, but the rocks are very similar to those of Kechika Group, south of the district in the Pelly Mountains. The lead-zinc deposits occur within a restricted portion of the lower division.

The upper division includes rocks ranging in age from Devonian to Permian. In contrast to the lower division, the upper division is characteristically cherty and conspicuously coarsely clastic. All or part of the upper division may be allochthonous with respect to the lower. The upper division is host to stratiform barite deposits and to a number of interesting geologic problems beyond the scope of this summary.

(a) Mount Mye Formation

The Mount Mye formation varies from non-calcareous, biotite-muscovite schist to non-calcareous, weakly carbonaceous, light to medium gray muscovite-chlorite phyllites with lesser, interlayered, black graphitic phyllite, marble, calc-silicate phyllite or schist, metabasite and psammitic schist. At Faro, the formation is dominated by schistose variants of these rock types. The formation is at least 2 kilometres thick, its base is not exposed in the district.

Slate and Quartzite

Menzie Creek Volcanic Unit

Pillowed and Massive Flows,
Volcanic Breccia Tuff,
Interlayered Black Slate

1000 m ±

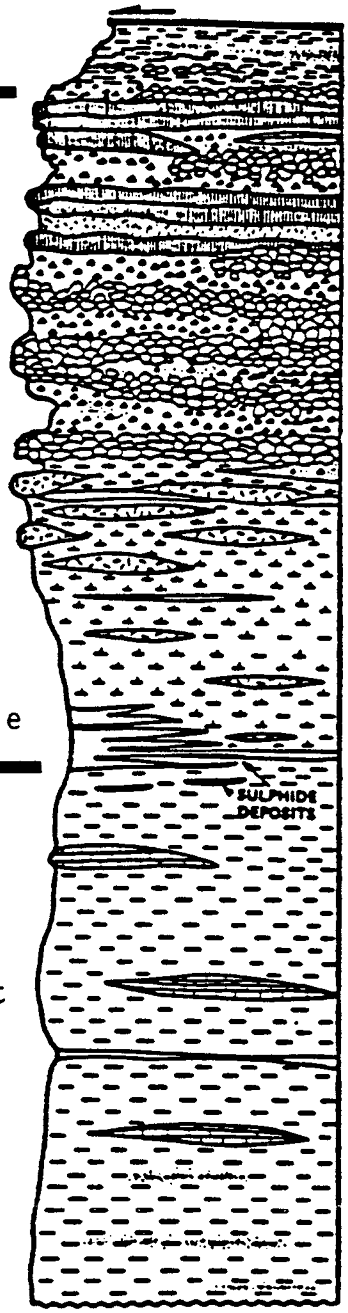
Vangorda Formation

Calcareous Phyllite and
Calc-Silicate, Metabasite,
Carbonaceous Phyllite,
Chloritic Phyllite Minor Marble
Approx. 1000 m

Mount Mye Formation

Non-Calcareous Phyllite and
Schist, Marble and Calc-
Silicate Lenses, Carbonaceous
Schist, Minor Pssamitic Schist
and Metabasite

2000 m +



CORRELATIVE UNITS

Early Ord. Graptolites

Road River Group

Kechika Group

Rabbitkettle Formation

I€p & €p (Gordey '78)
8a (Blusson '66)

?

U. Grit Unit?

Diagramatic stratigraphic section of the lower Palezoic of Anvil Range showing the ore deposits in relation to stratigraphy. Note that the bulk of the metavolcanics or metabasites are younger than the ore deposits but that the deposits are approximately coincident with the first appearance of substantial mafic igneous material in the section. Note also the anomalous thickness of carbonaceous rocks near the ore deposit trend.

TITLE: MAPPABLE SUBDIVISIONS OF THE LOWER DIVISION OF ANVIL RANGE	SECTION:	
KILBORN	AREA NO:	REV. NO:
CLIENT: CURRAGH RESOURCES PROJECT NO:	DRAWING NO: FIG. 5-3	
APPROVED: DATE: MAY-89		

KE(BC)-13

The upper portion of the formation is very similar to the buff weathering mudstone and blue-gray mudstone units described by Gordey (1978) to the east near Howards Pass and to Unit 8A of Blusson (1966) near Cantung. Correlation with these units would imply the top of the formation is lower Cambrian or possibly middle Cambrian. Parts of the Mount Mye also resemble rocks underlying those presumed correlative units locally, implying the Mount Mye probably includes rocks as old as Hadrynian.

(b) Vangorda Formation

The Vangorda formation is characterized by light to medium-gray, calcareous, phyllitic rocks made up of very thin (0.1 to 2.0 centimetres) interlayers of: (i) medium gray, non-calcareous, weakly carbonaceous, muscovite-chlorite pelite, and (ii) light gray, generally calcareous quartz plus or minus calcite plus or minus dolomite siltstone. In areas of more intense metamorphism, such as near the Faro deposit, the calcareous phyllite is altered to a harder, banded, green, purplish-brown and cream coloured calc-silicate. Other rock types interbedded with the calcareous phyllite include metabasite and meta-tuffs, graphitic phyllite and phyllitic limestone.

Most metabasite bodies are medium-grained and equigranular, thus they may have been sills; however, locally amygdaloidal margins and a common association with thin bedded, tuffaceous rocks suggest at least some were flows. Whole rock compositional data shows that the metabasites are all of basaltic composition. The bodies range from one to 100 metres in thickness and are up to several kilometres in length.

The Vangorda formation varies between 0.5 and 2 kilometres in apparent thickness with basic igneous rocks comprising approximately 15 percent of the section. The formation becomes more calcareous up section, paralleling an increase in metabasaltic units. A major carbonaceous member occurs at the base of the formation.

The Vangorda formation is lithologically similar to, though more argillaceous than, the Rabbitkettle formation seen to the east. Based on this correlation, the Vangorda formation may range in age from middle or upper Cambrian through lower Ordovician.

(c) Menzie Creek Formation

The Menzie Creek formation is a unit of basaltic metavolcanic rocks consisting of pillowed and massive flows with comparable amounts of massive, coarse, monolithic breccias and lesser, thin-bedded tuff and/or volcanic sandstone and siltstone. Carbonaceous phyllite and brown siltstone interbeds northeast of the Anvil Batholith contain graptolites of middle Ordovician or lower Silurian age, suggesting correlation with the widespread Road River formation black shale and chert to the northeast. The Menzie Creek formation varies from zero to about 1.5 kilometres in thickness in and near the district. It has been traced for 100 kilometres along strike and 30 kilometres across strike, showing that it is one of the largest of several basaltic units of its age in and around the Selwyn Basin.

(d) Relation of Stratigraphy to Ore Deposits

The ore deposits of the Anvil District are stratiform and strata-bound to an approximately 150 metre thick interval straddling the contact of the Mount Mye and Vangorda formations. The deposits consist of one to five horizons of sulphide mineralization stacked one above the other within this interval. They appear to be related to facies changes involving the basal carbonaceous member of the Vangorda formation.

5.2.3 Deformation Metamorphism

The structural and metamorphic history of the Anvil Range is complex and of considerable significance to the form and nature of the ore deposits. During mid-Mesozoic, the district suffered two periods of intense fold deformation and concurrent metamorphism during which the gross structure of the mineral deposits was determined.

The first deformation produced a regional metamorphic foliation axial planar to tight to isoclinal mesoscopic folds in bedding. Mesoscopic early folds are rarely preserved in the district. Northeasterly inclined to upright, northeasterly verging megascopic folds with shallow northwesterly or southwesterly plunging axes appear to have formed at that time.

During the second event, the initial foliation was strongly crenulated and ubiquitous close to tight mesoscopic folds were produced. Some of the largest megascopic folds known to have been formed during this event are those at the Grum deposit. Parallel to the axial planes of these folds is a crenulation cleavage which imparts a well developed lithon structure and pronounced fissility to most rocks of the district.

Three later, less intense periods of folding and associated faulting followed. These later events generally produced open folds and weak crenulations. An important exception to this general rule is found in the vicinity of the Faro deposit where the fourth event is quite intense.

During the later stages of the fold deformation history, a large granitic body (Anvil Batholith) was intruded into the metamorphic sequence. Anvil Batholith ranges in composition from granodiorite to quartz monzonite and textures include equigranular massive, megacrystic massive and various strongly to weakly foliated variants. Several potassium/argon ages on the granitic rocks yield ages of 85-100 million years. Intrusion of the Anvil Batholith further deformed the metamorphic sequence so that the overall structure of the district is an elongate dome cored by the Batholith (Figure 5-2).

In the later stages of batholith emplacement, large extensional fault displacements occurred along its margins. These faults determine the present day limits of several of the deposits.

Metamorphism was concurrent with deformation and was most intense during the early deformations. Metamorphic facies developed range from middle amphibolite facies to lower greenschist facies in a low pressure Buchan facies series. Metamorphic isograds are roughly concentric about the Anvil Batholith. Faro, close to the batholith, (Figure 5-2) is strongly metamorphosed, while deposits such as Vangorda are less intensely metamorphosed. This difference in metamorphism is reflected in decreased grain size, increased degree of mineral intergrowth, and lesser iron content of sphalerite in the less metamorphosed deposits. This has a significant impact on metallurgical performance of Anvil District ores.

5.2.4 Ore Deposits

The lead-zinc-silver deposits of the Anvil Range are of the sediment hosted, stratiform, massive pyritic sulphide type. They occur as a single thick sulphide lens with little or no interbanded metasedimentary rocks (e.g., Faro) or as multi-layered deposits with several thinner lenses stacked approximately one above the other with substantial metasedimentary or metavolcanic interlayers (e.g., Grum and Dy). An individual mineralized layer was deposited parallel to the bedding of the host sediments. It consisted of an upper, often centrally positioned, lead-zinc rich, massive sulphide facies and a lower and peripheral, lower grade, quartzose, disseminated sulphide facies.

These sulphide sheets, or horizons, have since been deformed into complex fold structures. The deposits are thus elongate parallel to the fold axes and associated lineations in the host metasediments. The Faro deposit, which appears to be an exception to this generalization, actually shows great internal complexity in the geometry of high grade and waste layers.

Present day deposit lengths are generally two to three times widths; unfolded deposit dimensions range up to 4,000 metres across their ameboid shapes. Individual sulphide horizons commonly are 10 to 40 metres in thickness. The upper and lower contacts of sulphide horizons are invariably sharp, while laterally the sulphides grade into the enclosing host rocks.

All deposits are composed of a small number of different sulphide rock types. As noted above, the sulphide rock types are broadly divisible into massive sulphides and quartzose, disseminated sulphides. There are pyritic, baritic, pyrrhotitic and carbonate-bearing variants of massive sulphide types and carbonaceous and non-carbonaceous variants of the quartzose sulphide rock types.

All deposits show a variably developed, white mica-dominant, alteration overprint in the wallrocks.

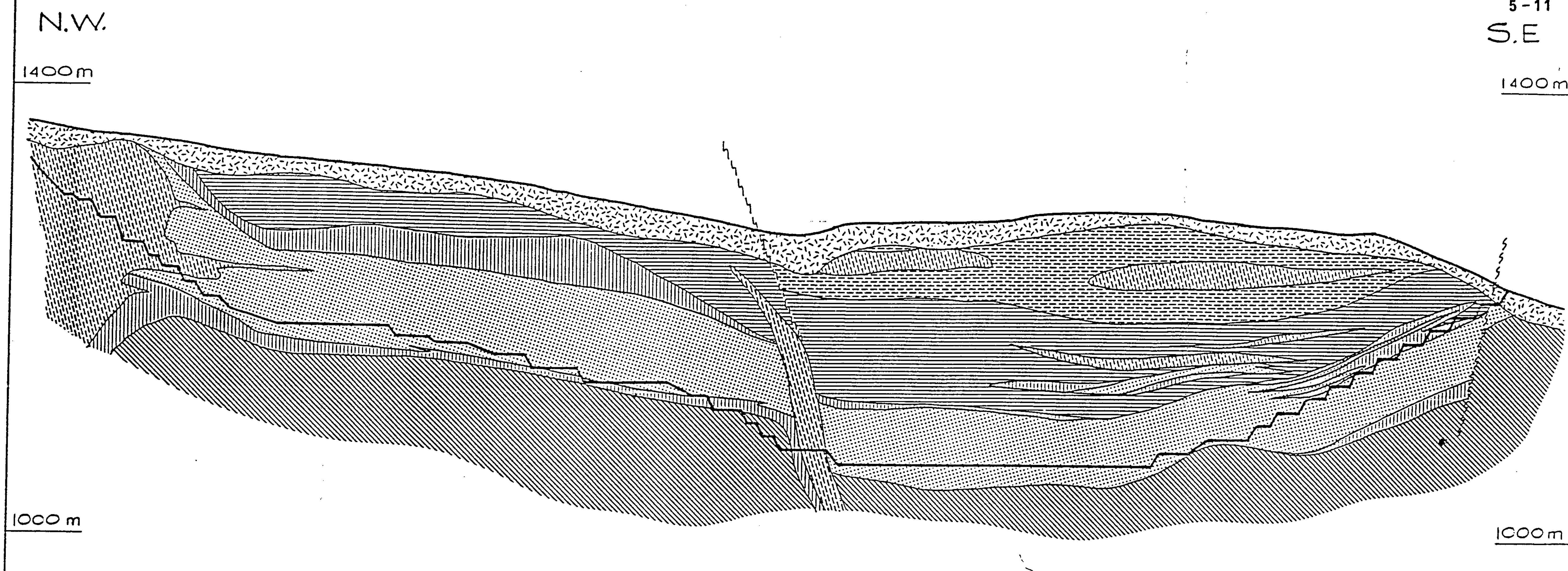
There are presently five known lead-zinc bearing mineral deposits along a prominent curvilinear trend on the south flank of Anvil Arch (Figure 5-2). From northwest to southeast, they include Faro, Grum, Vangorda, Dy and Swim. Additionally, two lead-zinc deficient sulphide occurrences, the SB and Sea, are known.

5.3 FARO GEOLOGY

5.3.1 General

Before mining, the Faro deposit was 2,000 metres along strike, 800 metres across strike, and from a few metres to 90 metres thick. The deposit is a flat lying, elongate, asymmetric lens with a thick northeast side and a thin tapering southwest side.

Figures 5-4 and 5-5 are schematic longitudinal and cross-section views respectively, of the Faro deposit.



LEGEND

OVERBURDEN

INTRUSIVE ROCKS

VANGORDA FORMATION

CALCAREOUS PHYLLITE; CALC-SILICATES

GRAPHITIC PHYLLITE/SCHIST.

SULPHIDE HORIZON(S)

ALTERATION OVERPRINT

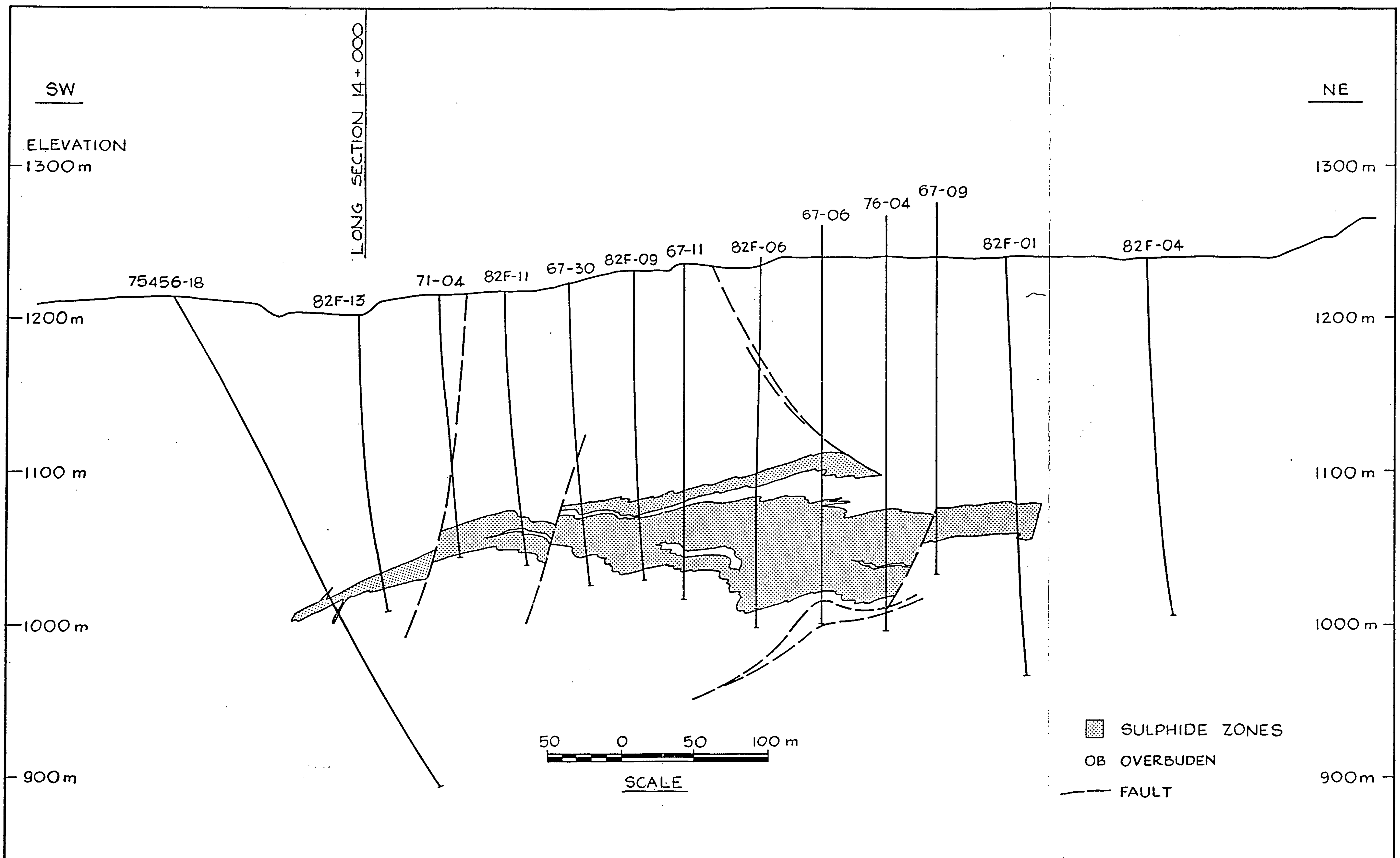
MT. MYE FORMATION

NON-CALCAREOUS PHYLLITE/SCHIST

0 100 200 300 m

SCALE - METRES

REVISIONS	SCALE	DATE	CLIENT CURRAGH RESOURCES	FARO AREA DEPOSITS FARO DEPOSIT LONGITUDINAL SECTION	
	DESIGNED		LOCATION		
	DRAWN PAH	MAR 3/87			
	CHECKED				
NO / DATE / BY	APPROVED	MAR 3/87	KILBORN	PROJ. NO 3680·15 / DWG. NO. FIG. 5 - 4	REV. A



				SCALE AS SHOWN	DATE	CLIENT CURRAGH RESOURCES	FARO AREA DEPOSITS
				DESIGNED J.B.F.	MAR87		FARO DEPOSIT
				DRAWN Z.A.	MAR87	LOCATION FARO, YUKON	X-SECTION (124 + 22)
ISSUED FOR TECHNICAL REVIEW	A	MAR87	ZA	CHECKED		KILBORN	PROJ. NO 3680-15
REVISIONS	NO	DATE	BY	APPROVED			DWG. NO. FIG. 5 - 5 REV. A

K.E.B.C.9

5.3.2 Stratigraphy and Lithology

The Faro deposit occurs approximately 100 metres beneath the Mount Mye/Vangorda formation boundary. Stratigraphically, this may equate to the position of the lowest horizons in the Vangorda Plateau deposits.

The immediate host rock of the orebody is biotite-muscovite-andalusite schist that grades downwards into a coarse, gneissic biotite-muscovite schist. A discontinuous graphitic phyllite unit about 6 metres thick is interlayered with the schists about 25 metres above the ore deposit. There are also several thin interbands of strongly foliated chlorite-actinolite schist, or bleached and carbonated equivalents of this mafic schist, above the orebody.

The Vangorda formation at Faro is represented by hard, dense, banded calc-silicates rather than the calcareous phyllite that characterizes the Vangorda Plateau. This fact is of considerable importance in blasthole drilling at Faro because of the rock's hardness. Amphibolite up to 10 metres thick is interbanded with the calc-silicates and there are several thin graphitic phyllite layers. The basal unit of the Vangorda formation in the Faro deposit area consists of graphitic phyllite, amphibolite and calc-silicates mixed in subequal amounts.

Post metamorphic igneous intrusive rocks are more widely developed at Faro than elsewhere in the district. There are two clans of importance: (i) an equigranular to subporphyritic hornblende diorite to quartz diorite clan, and (ii) a quartz-feldspar porphyry clan. The former occurs as a large dyke truncating the deposit at its northwest end, a smaller dyke along the fault between Zones 1 and 3, an inferred sill beneath the breccia cap, and several smaller dykes. The latter forms highly irregular and unpredictable intrusive bodies in the north part of Zone 3.

Associated with these dykes or irregular intrusive bodies, and the intersection of two important faults, is a large mass of heavily silicified post metamorphic breccia at the northeast edge of the deposit in Zone 3. This 'breccia cap' complicates the problems of blasthole drilling because of its extreme hardness.

5.3.3 Structure

Faro is deeper in the structural sequence than other parts of the Anvil District. Consequently, the structural picture is rather different. The second deformation effect is very strong at Faro. Virtually all signs of the first deformation have been completely over-printed.

The planar schistosity is strongly developed and is the plane of greatest fissility in all metamorphic rocks of the Faro area. It dips 10 to 20 degrees towards the southwest or west. Second phase folds are generally isoclinal with shallowly northwest or southeast plunging axes. The ore deposit is a tabular body parallel to compositional layering.

Generations of folds deform compositional layering and the ore deposit into close to tight northeasterly verging folds with axial planes dipping 45 to 60 degrees towards the southwest and generally west or northwest plunging axes. The late folds commonly have amplitudes of approximately one metre and folds of several tens of metres are inferred in the base of the deposit. The size of these folds and the extent to which the deposit margin and internal banding geometry is defined by late folds as opposed to faults is one of the major uncertainties in ore reserve estimation at Faro.

Faults postdating the fold deformation (and concurrent metamorphism) are widely developed at Faro. Two sets are particularly important: (i) a N20°W striking and steeply west dipping set, and (ii) an east or N60°E striking and generally steeply to moderately south dipping set. These two sets define a graben structure. Zone 3, which contains the remaining reserves, is the central down-thrown block, and Zones 1 and 2, now mined out, are the up-thrown blocks. Many other fault sets are more locally developed.

The shallow to moderate southwest dip means that the northeast wall of the pit is relatively unstable. Failures to date appear to be surficial and involve platy rock fragments sliding slowly towards the pit. The possibility of larger scale failures, involving slip backed by some of the larger faults dipping towards the pit, cannot be dismissed. The relatively more massive and stronger rock mass of the northeast edge of the sulphide deposit is expected to buttress the northeast pit wall as the pit deepens.

5.3.4 Deposit Geology

There is essentially one thick horizon at Faro, although this horizon contains numerous cycles and several thin phyllite waste bands are included. Locally, a thin upper horizon is differentiated from the main mass of the deposit, but this is generally too thin to be mineable. Low grade sulphide interbanding with the high grade ore is widespread especially in the northeast part of the deposit. The low grade or waste sulphides pose a major dilution problem, unlike phyllitic waste they cannot be visually differentiated, thus are much more difficult to control. Only blasthole assays can define sulphide waste. The thickness of high grade and sulphide waste or low grade interbanding is commonly less than the 6 metre (20 foot) bench height, particularly in the northeast part of the deposit; this places basic limits on dilution control.

Ore type zoning is particularly strong at Faro. It follows the scheme outlined above with a massive variably baritic upper portion and a quartzose variably carbonaceous lower part. In addition, there is a prominent, very low-grade semimassive zone along the northeast edge of Zone 3 and unusually abundant (compared to other Anvil District deposits), but erratically distributed, pyrrhotitic mineralization in the southwest part of the deposit. Grade zoning follows ore type zoning so that the base and northeast edge of the deposit contains the lower grade mineralization, whereas the upper and southwest portion contains the higher grade mineralization. Zoning was also obvious in plan view at Faro. Zone 1 was rich in baritic ores, thus high grade; Zone 2 at the other end of the deposit was rich in carbonaceous quartzose ore types, thus low grade and metallurgically undesirable. Zone 3 has intermediate characteristics.

The greatest continuity in the deposit is along the deposit elongation. Across this trend the horizontal continuity is relatively poor with gradual changes in rock type and grade in the northeast half of the deposit, and less abrupt grade variation in the southwest half. The vertical continuity is poor since there are rapid changes in rock type and grade across the subhorizontal layering. The ore deposit thus has a feather edge assay boundary along its northeast edge, a better defined lower assay boundary, and a relatively sharply defined upper boundary. The southwest limit of the open-pit ore is defined by the gradual thinning of the deposit and the gentle southwest dip.

It is easy to make generalizations such as those above in order to convey an impression of the deposit; however, the Faro deposit shows very complex internal variation. Between drill holes, variability is so great that commonly the rock type and assay distribution in adjacent drill holes seem to bear no relation to one another. This great variability places some basic limits on the reliability of local reserve estimates.

5.4 GRUM GEOLOGY

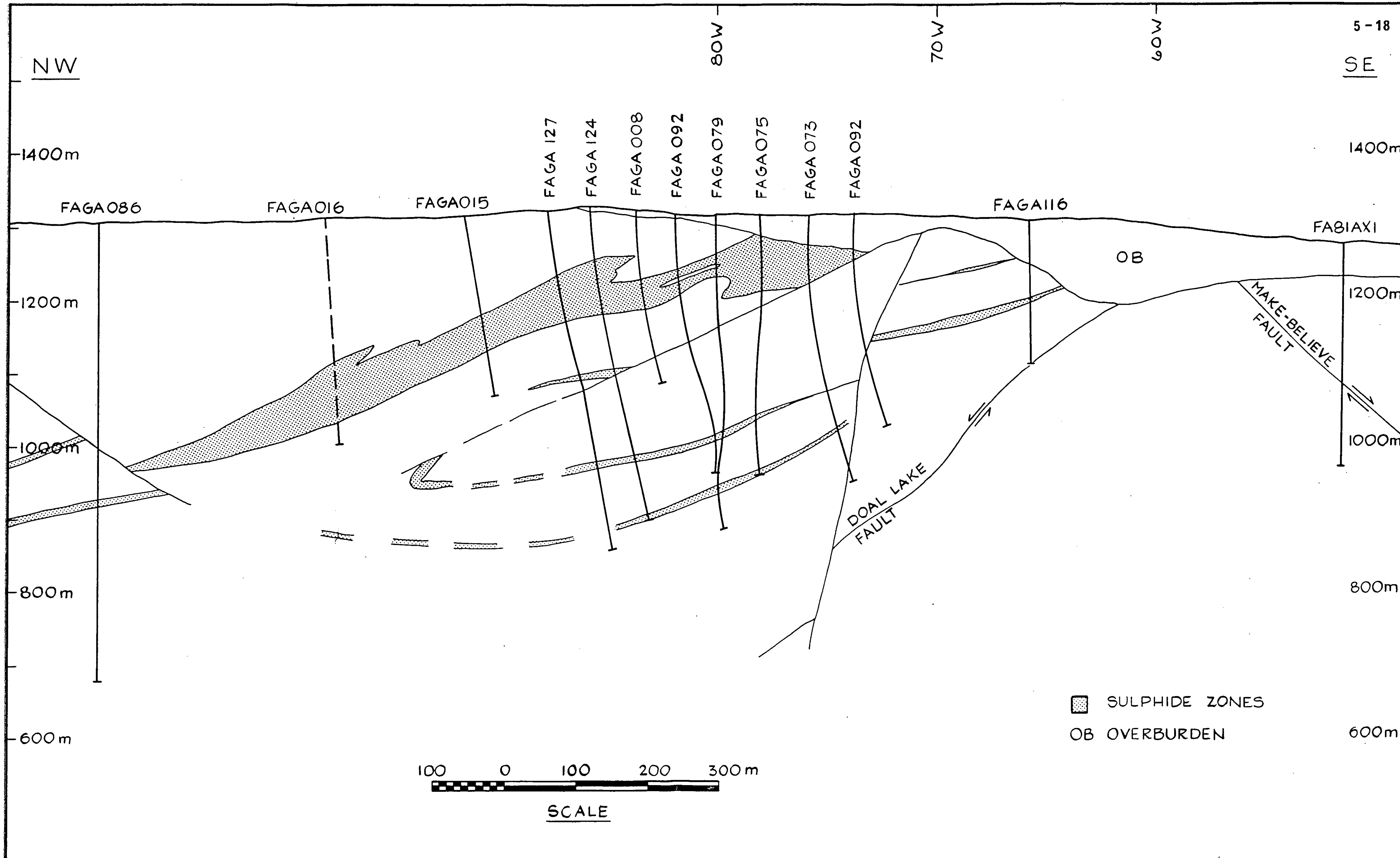
5.4.1 General

Figures 5-6 and 5-7 are, respectively, longitudinal and cross-sectional views of the Grum deposit.

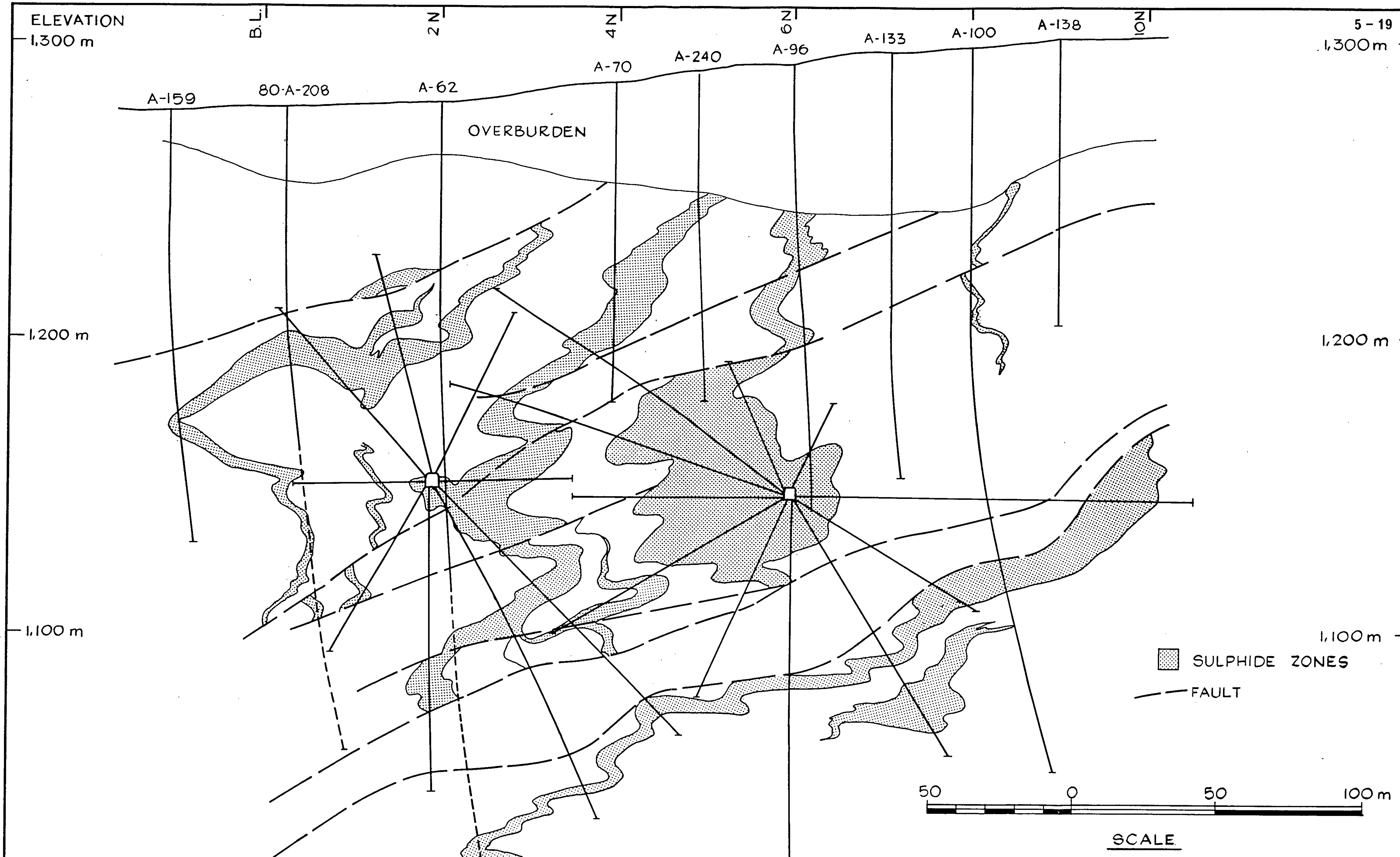
Grum, like Vangorda and Dy, has several characteristics that distinguish it from Faro. In large part, this is due to the lower metamorphic grade the deposit has reached. The most outstanding difference between Grum and all the other Vangorda Plateau deposits, as opposed to Faro, is the form of the deposit. The Vangorda Plateau deposits consist of several distinct, highly contorted horizons separated by barren phyllite waste. Faro, on the other hand, is essentially one thick horizon in overall outline with lesser phyllitic waste but substantial barren sulphide waste banding. This implies that dilution by phyllite will be higher at Grum than at Faro. Faro, however, contains considerable internal sulphide waste, thus its mining dilution is higher than might appear at first glance. Nonetheless, Grum has a higher potential for mining dilution and will have more complex mining problems than Faro. However, the dilutant at Grum will be predominantly the more easily identifiable phyllite rather than low grade sulphides as at Faro. Experience at Faro shows that phyllite dilution is much easier to control than low grade sulphides.

5.4.2 Stratigraphy and Lithology

The Grum deposit consists of three to five highly contorted layers of massive and disseminated sulphide mineralization within a 150 metre section of barren phyllite. The most important mineralized horizon occurs just beneath the basal carbonaceous member of the Vangorda formation. There are thin low-grade horizons within the Vangorda formation and more important horizons in the upper part of the Mount Mye formation.



				SCALE 1:5000	DATE	CLIENT CURRAGH RESOURCES	FARO AREA DEPOSITS
				DESIGNED J.B.F.	MAR.87		GRUM DEPOSIT
				DRAWN Z.A.	MAR.87	LOCATION	LONGITUDINAL SECTION
				CHECKED		KILBORN	
REVISIONS	Nº	DATE	BY	APPROVED			PROJ. Nº 3680-15 DWG. Nº FIG. – 5 – 6 REV.



				SCALE AS SHOWN	DATE	CLIENT CURRAGH RESOURCES	FARO AREA DEPOSITS GRUM DEPOSIT X-SECTION (70W)
				DESIGNED JBF	MAR.87		
				DRAWN ZA	MAR.87	LOCATION	
				CHECKED		KILBORN	
REVISIONS	Nº	DATE	BY	APPROVED			PROJ. Nº 3680-15 DWG. Nº FIG. — 5 – 7 REV.

KE.B.C.9

At Grum, the Vangorda formation consists of soft, highly fissile, calcareous phyllites. Metabasites in the Grum area are minor and tend to be highly foliated chlorite phyllite rather than the blocky, massive greenstones that typify the Vangorda formation elsewhere. The basal carbonaceous member of the formation thickens across the deposit from about 10 metres in the northeast to as much as 80 or 100 metres in the southwest of the deposit. The sulphide horizons appear to be associated with the northeast pinchout of this unit. Immediately above the main ore horizon the carbonaceous rocks are soft, highly sheared and gouged, but elsewhere they are moderately hard, highly fractured, black siliceous phyllites.

The Mount Mye formation also consists of soft phyllites which are distinguished from those of the Vangorda formation by being non-calcareous and less distinctly banded.

There are no significant post metamorphic dykes at Grum. The Anvil Batholith crops out 1.5 kilometres northeast of the deposit, but is separated from it by major faults. The batholith is unrelated to the deposit and does not appear to have affected it significantly.

5.4.3 Structure

The ore layers at Grum are contorted into a complex, shallowly northwest plunging, polyphase fold structure. The prominent "S" shaped folds are second phase structures. They are superimposed on a larger "Z" shaped first phase fold. The dominant plane of fissility in the phyllites at Grum is axial planar to the second phase folds and dips shallowly (10 to 30 degrees) generally to the southwest. This fissility is a major factor in assessing slope stability for the Grum pit. The overall deposit elongation parallels the axial direction of the second phase folds (315 degrees trend, 11 degrees plunge).

There are several important faults at Grum. The largest displacements occur on moderately (35-45 degrees) dipping structures that truncate the deposit at both its northwest and southeast ends.

Neither of these structures would crop out in an open pit, but smaller subparallel faults will be found in the pit. A steeply northwest dipping fault trending about 060 degrees, passes between sections 70W and 72W and downdrops the deposit about 60 metres to the northwest. A myriad of smaller faults were mapped underground by Kerr-Addison Mines, trending on the average 080 degrees and dipping steeply. Joints mapped underground and on surface tend to strike 060 degrees and dip subvertically.

5.4.4 Deposit Geology

As with other deposits in the Anvil Range, a given ore horizon at Grum tends to have a massive sulphide upper and central portion and a quartzose, disseminated sulphide lower and peripheral portion. The horizons can be up to 30 metres thick, but are mostly 15 metres or less thick. Grade is strongly partitioned into massive, particularly baritic, sulphides thus the tops of horizons tend to be high grade and the bottoms low grade (except, of course, where the horizons are overturned). The sulphide horizons are separated by significant thicknesses of barren phyllite. Interfaces between ore and waste tend to be sharp at the stratigraphic hangingwall contact against barren phyllite, and gradational both at the footwall and laterally against sulphide waste.

Grum ores have a finer grain size and more complex material intergrowth than Faro ores. This may require finer grinding for satisfactory recovery of metal in concentrate.

At a given lead-zinc cut-off grade, ores at Grum are higher grade than those remaining at Faro, and are higher in precious metals relative to base metals. The average gold content of Grum is several times higher than Faro. Similarly, other elements that tend to be geochemical associates of gold, i.e., mercury and arsenic, tend to be higher at Grum. The sphalerite at Grum, and likely other Vangorda Plateau deposits, is richer in zinc and lower in iron content due to lower metamorphic grade.

A feature unique to Grum among the Vangorda Plateau deposits is the relative abundance of quartzose ore types, particularly carbonaceous pyritic quartzites which comprise about 35 percent of the reserves above 4 percent lead plus zinc.

5.4.5 Surficial Geology

The subcrop of the ore deposit is covered by up to 100 metres of morainal material (tills) and better sorted glaciofluvial silts, sands and gravels. These unconsolidated sediments are water saturated and may contain pockets of permafrost. The northeast wall of any pit designs at Grum must contend with thick sections of these sediments.

5.5 VANGORDA GEOLOGY

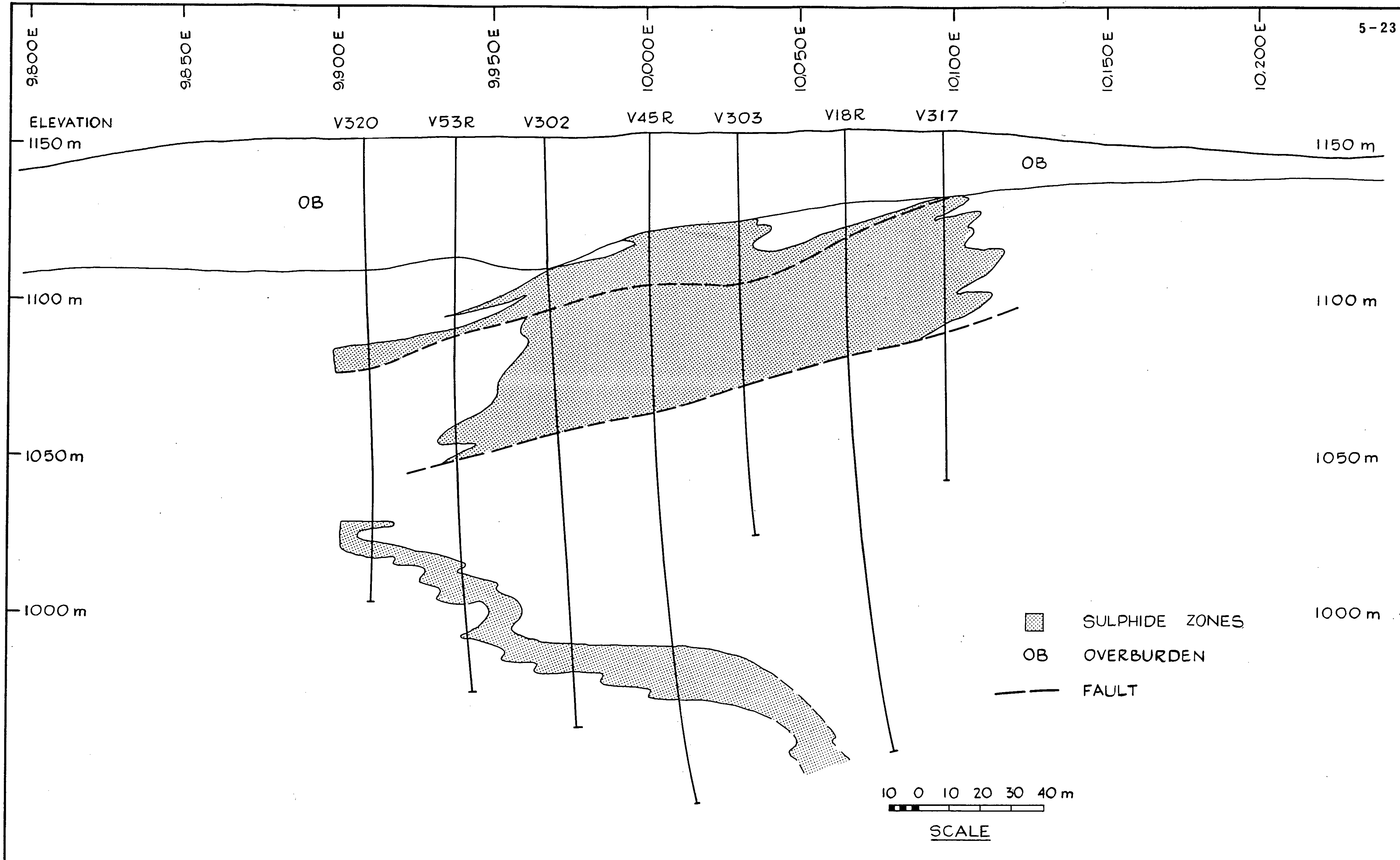
5.5.1 General

A cross-sectional view of the Vangorda deposit is shown on Figure 5-8.

The deposit is quite shallow; in most places, subcropping beneath glacial till. The till blanket is up to about 30 metres thick in the northwest part of the deposit, but thin in the southeast. Northwest of Vangorda Creek, till cover is also quite thin. Locally, the basal overburden and uppermost broken bedrock are cemented by iron oxides into a tough breccia.

5.5.2 Stratigraphy and Lithology

The Vangorda deposit consists of one major sulphide horizon about 50 to 120 metres beneath the basal carbonaceous member of the Vangorda formation. The host rocks for the deposit are dominantly non-calcareous phyllites, probably part of the Mount Mye formation; however, formational assignments near this deposit are ambiguous.



				SCALE AS SHOWN	DATE
				DESIGNED J.B.F.	MAR.87
				DRAWN Z.A.	MAR.87
				CHECKED	
REVISIONS	NO	DATE	BY	APPROVED	

CLIENT CURRAGH RESOURCES

LOCATION

KILBORN

FARO AREA DEPOSITS
VANGORDA DEPOSIT
X-SECTION (8E)

PROJ. NO 3680-15 | DWG. NO. FIG. – 5 – 8 | REV. A

K.E.B.C.9

The reason for the ambiguity is largely due to the strong wall rock alteration developed around the deposit. Most phyllites, especially in the deposit footwall, are bleached, locally silicified and/or chloritic- and sulphide-bearing.

A number of thin sulphide horizons occur above the main horizon; one, at the base of the carbonaceous phyllites southwest of (stratigraphically above) the deposit, may equate to the main horizon at Grum. In general, these horizons are too thin or too low grade to be mineable.

5.5.3 Structure

The Vangorda deposit occurs in the hinge of a large second phase fold. Overall, the deposit has the shape of a reclining "M" or a "3" in cross-section; however, there is some uncertainty in the details of fold morphology. The deposit is elongate in the northwest-southeast direction parallel to fold axes. It has been traced over a 1,300 metre by 200 metre area.

The northwest half of the deposit plunges about 10 degrees towards the northwest, while the southeast half has a subhorizontal plunge. The foliation dips shallowly toward the southwest as at Grum, but is locally quite variable.

The deposit is truncated by a steep normal fault at its northwest end. Many gouge zones were observed in drill core, but the orientation of the structures responsible for them is not well known. A number of faults parallel to foliation are predicted. These are "required" to make the structure and stratigraphy fit. These low angle structures are best thought of as sheared-out fold limbs; they are not generally gouge zones and will pose no more serious a problem for slope stability than the foliation itself and the myriad small gouge zones that parallel it.

5.5.4 Deposit Geology

The deposit consists of the same sulphide rock types as the Faro and Grum deposits, but two rock types are particularly prominent. In the footwall (also the interpreted stratigraphic footwall) of the deposit is a sulphide-rich quartzite which grades downwards into siliceous phyllite and ultimately altered phyllite. Parallel to this downward decrease in silica is a downward decrease in the abundance of sulphides from quartz-rich semimassive sulphide at the top to weakly pyritic altered phyllite at the base. Most of the sulphides in the quartzite are pyrite; however, pyrrhotite is generally present and locally abundant or dominant. Magnetite is unusually well developed in the quartzite. The quartzite contains only minor lead and zinc, but is relatively rich in copper and unusually high in gold. The quartzite is similar to the semimassive zone along the northeast edge of Zone 3 at Faro, and one of the lower ore panels at Grum.

From a reserve estimation point of view, the barren pyritic quartzite is significant in that it is located beneath, and sharply delineated from, the high grade massive sulphides. For this reason, it was important to restrict the selection of assay composites during grade interpolation to equivalent geology, thus preventing excessive averaging of grades into the deposit footwall and lowering of the overall deposit grade.

The massive sulphides that overlie the pyritic quartzite are commonly baritic and rich in lead and zinc. Of the mineralization exceeding 6 percent lead and zinc, 90 percent is barite-bearing massive sulphides. Most pyritic quartzite is sulphide waste on the basis of lead-zinc content.

The shallow depth of burial of the deposit may create metallurgical difficulties because of oxidation. Early metallurgical work seemed to show this; however, later work carried out by Cyprus Anvil on fresh core achieved better results. The limited core was not visibly oxidized and oxidation is not extensively described in most drill logs below the first few metres of bedrock. Diamond drill core recoveries in massive sulphides were generally good except locally near Vangorda Creek and in the southeast end of the deposit (at Faro, oxidized massive sulphides yield poor core recoveries). In much of the southeast end of the deposit, information on core recoveries from recent drilling is not available, but it is known that the older holes did not core well. It seems prudent to assume that the portion of the deposit southeast of 12E, where till cover is thin, will be oxidized. This could affect up to 1,900,000 tonnes of baritic sulphides or 37 percent of the baritic sulphides in the geological reserves.

5.6 DY GEOLOGY

The Dy deposit is similar in general to the other deposits of the district but is less well known because of the sparse drill density and more completely preserved because of its great depth. (Approximately 500 m to top of ore zones).

The deposit consists of five sulphide horizons. The uppermost and lowermost horizons are of relatively little significance in the total mineral inventory. The exact correlation and geometry of the ore zones is not well known due to wide spaced drilling. The overall deposit geometry may be more similar to that of Grum than is currently appreciated once more drilling data is available.

The ore zones are generally subparallel to the second phase foliation which dips approximately 25 degrees toward the southwest as is typical for the Vangorda Plateau. The ore horizons are spread through a stratigraphic thickness of approximately 200 m but only comprise approximately 25 percent of that interval. Upper and lower contacts with barren interbanded phyllite are sharp, laterally the ore horizons thin gradually or fade out into the host phyllites. Several steeply dipping faults are known to offset the sulphides but fault systematics are not well known. A major low angle extensional fault occurs below the deposit. This fault may have detached Dy from the up plunge extensions of Vangorda.

In plan view there are two major lobes of the deposit separated by a nearly barren pyritic sulphide zone. The north lobe parallels the strike of the deposit and runs along the north edge of the known deposit. The north of "B2" zone is zinc rich relative to lead and very high grade. The B2 zone ore types are dominantly quartzose disseminated material about half of which is carbonaceous. The south or "A2" zone is lead rich relative to zinc and is dominated by baritic massive ore types. The A2 zone rakes across the deposit dip at the south limit of the deposit. The overall plunge of the A2 zone is thus south at about 20 degrees. Sulphides continue between the A2 and B2 zones but are lead-zinc poor.

The deposit is open locally to the south and deep extensions to the southwest are possible.

6.0 ORE RESERVES AND MINERAL RESOURCES OF THE FARO DIVISION (WTHIN CURRENT PLANS)

6.0 ORE RESERVES AND MINERAL RESOURCES OF THE FARO DIVISION (WITHIN CURRENT PLANS)

6.1 SUMMARY

Kilborn has checked, in various ways, the mineable ore reserves used in Curragh's 11-year Mine Plan, S89: Alpha II. For the three open pit deposits which were modeled on computer, Kilborn did complete manual reserve calculations using interpreted geological section drawings supplied by Curragh. Most of this work was started in October 1987. For the Faro underground deposit, Kilborn independently duplicated the polygonal calculations using Curragh's drill hole data and interpretations. For the Dy underground deposit, Kilborn retained an independent geologist to audit and reclassify the old polygonal reserves which were calculated by a previous owner.

Comparative results are summarized below and explained in Sections 6.2 to 6.6. Reserves in the currently active Faro pit have been adjusted to mid-year 1989. Cut-off grades are 4% combined lead plus zinc for open pit reserves, and 9% for underground reserves.

6.1.1 Faro Open Pit Mineable Reserves - Proven

	Tonnes	Lead Percent	Zinc Percent	Silver g/Tonne
Kilborn Estimate				
Diluted, Jan.1/89	13,302,000	3.10	5.45	36.02
Less, Mined in First Half, 1989	2,783,000	2.77	4.41	32.0
Diluted, Jul.1/89	10,519,000	3.19	5.73	37.1
Curragh Projection				
Diluted, Jul.1/89	12,009,000	2.75	4.68	31.5

6.1.2 Faro Underground Mineable Reserves - Probable

	Tonnes	Lead Percent	Zinc Percent	Silver g/Tonne
Kilborn Estimate				
Undiluted	1,820,000	5.03	7.78	67.75
Diluted	2,000,000	4.57	7.07	61.59
Curragh Projection				
Diluted	1,178,000	4.11	6.27	60.00

6.1.3 Grum Deposit Mineable Reserves - Proven

	Tonnes	Lead Percent	Zinc Percent	Silver g/Tonne	Gold g/Tonne
Kilborn Estimate					
Undiluted	24,110,000	3.42	5.72	58.42	N.A.
Diluted	24,110,000	3.08	5.15	52.58	N.A.
Curragh Projection					
Diluted	25,161,000	2.96	5.01	50.00	0.81

6.1.4 Vangorda Deposit Mineable Reserves - Probable

	Tonnes	Lead Percent	Zinc Percent	Silver g/Tonne	Gold g/Tonne
Kilborn Estimate					
Undiluted	6,327,000	3.99	4.94	52.83	0.63
Diluted	6,327,000	3.59	4.45	47.55	0.57
Curragh Projection					
Diluted	6,935,000	3.49	4.51	48.00	0.65

6.1.5 Dy Deposit Mineable Reserves - Probable

	Tonnes	Lead Percent	Zinc Percent	Silver g/Tonne	Gold g/Tonne
Kilborn Estimate					
	11,510,000	4.99	6.46	78.6	0.87
Curragh Projection					
Diluted Reserve	11,300,000	5.82	6.84	83.0	0.94
*(Scheduled for mining)	(8,500,000)	(5.80)	(7.40)	(89.0)	(0.94)

* Curragh's schedule starts in a 'higher than average grade' part of the deposit.

6.1.6 Faro Stockpile - Proven

Curragh maintains a stockpile near the concentrator. Its primary purpose is to store marginal grade material (currently between 3 and 4 percent combined lead plus zinc), but it also serves as a surge pile to store ore when the concentrator is down or to supply feed if there are problems in the pit.

As of July 1, 1989, Curragh reported that the stockpile contained 1,215,000 tonnes with average grades of 2.04 percent lead, 3.26 percent zinc and 27 g/t silver.

Kilborn has not checked the stockpile quantities or grades.

6.2 FARO DEPOSIT (OPEN PIT)

Kilborn's check on the Faro open pit mineable ore reserves was based on the following information which was supplied by Curragh:

(a) Faro drill hole data;
(b) Geological cross-sections through the deposit;
(c) Plan of existing open pit as of 1988 year-end;
(d) Plan of ultimate open pit.

The portion of the Faro deposit which now is being mined by open pit methods has been identified by drilling and ongoing bench mapping during operations. The deposit has been computer modeled numerous times. These models appear to be consistent with actual production but tended to be up to 5 percent higher in metal content than what is experienced. A new model was produced recently from assay data composited over bench height, rather than over rock type, with the objective of improving the definition of high grade areas.

The geological interpretation which was reviewed by Kilborn was done by Cyprus Anvil and Curragh and was found to be reasonably consistent with what had been experienced. A new interpretation was completed by Curragh late in 1988 and was used to prepare the new computer model.

Kilborn did not verify the drill logs, assay methods nor drill hole surveys.

Kilborn did random checks of the drill hole plots on the sections, plotted the year-end 1988 and ultimate pits on the sections, and determined the areas of influence of the drill holes. The mineral reserve was calculated by the cross-sectional method using the following criteria:

(a) Minimum in-situ grade of 4 percent combined lead plus zinc;
(b) Minimum interval of low grade in an ore section of 3 metres before rejection from the reserve;
(c) Distance of influence, halfway to the next drill hole.

The results are as follows:

IN-SITU RESERVE ESTIMATE

Ore Type	Quantity Tonnes	Lead Percent	Zinc Percent	Silver g/Tonne
A	886,399	1.92	4.49	27.61
H	732,400	3.91	6.34	51.00
Other	11,683,170	3.54	6.15	40.27
TOTAL Within Pit	13,301,969	3.45	6.05	40.02

To determine the minable reserves, the quantity of material was left constant and the grades were reduced by ten percent to compensate for the mixing of ore and waste on the contacts of the deposit. This assumes that ten percent of the ore in-place is hauled to the waste dump and an equal quantity of waste is mixed with the ore and hauled to the mill.

RECOVERABLE ORE RESERVE ESTIMATE

Ore Type	Quantity Tonnes	Lead Percent	Zinc Percent	Silver g/Tonne
All Types, Jan. 1/89	13,302,000	3.10	5.45	36.02
Less, Mined 1st half '89	2,783,000	2.77	4.41	32.0
After mid-1989	10,519,000	3.19	5.73	37.1

The ore reserve quality is considered as proven.

In Curragh's 1989 forecast plus the 11-year projection, the Faro open pit is scheduled to produce:

	Quantity Tonnes	Lead Percent	Zinc Percent	Silver g/Tonne
Second Half 1989	2,269,000	3.07	4.58	40.0
1990 and 1991	9,740,000	2.68	4.70	29.5
After mid-1989	12,009,000	2.75	4.68	31.5

Curragh's mining reserve calculations are stated to include dilution within the new computer model, and a reduction of 5 percent for mining losses (i.e. 95% recovery).

Curragh's tonnage is 14% higher than Kilborn's estimate, but the grades are correspondingly lower. Thus, the contained metal is in close agreement while Curragh's estimate appears to include considerably more dilution. Some part of the difference may also be due to the reinterpretation of geology. In either case the result is a more conservative projection of costs per pound of metal. Therefore, it is considered that Curragh's mine plan for the Faro open pit can be accepted by Kilborn.

6.3 FARO DEPOSIT (UNDERGROUND)

Kilborn's check calculation of mineable ore reserves was based on the following information supplied by Curragh:

(a) Faro drill hole data base;
(b) Geological interpretation of ore type.

That part of the Faro deposit which is more suitable for underground mining, has been identified by various core drill holes over a long period of time. Approximately one-half of the holes were drilled in the early 1970s, and a sizeable extension to the mineralized zone was identified by drilling in 1982 and 1983.

The geological interpretations were done by Cyprus Anvil and by Curragh.

Kilborn did not verify the drill logs, assaying methods nor the drill hole surveys.

Kilborn did plot the drill holes on plan, and marked out the polygons to be used in calculating the mineral resource. The mineral reserve was calculated by the polygon method with the following restrictions:

(a)	Minimum in-situ grade	:	9 percent (lead plus zinc)
(b)	Minimum mining height	:	2.1 metres
(c)	Maximum radius of influence	:	46 metres
(d)	Minimum pillar between underground mineral reserve and the final open pit wall	:	15 metres

The results are listed below:

IN-SITU RESERVE ESTIMATE

Ore Type	Quantity Tonnes	Lead Percent	Zinc Percent	Silver g/Tonne
2A	58,000	1.98	3.88	13.86
2BG	2,288,000	5.09	7.69	67.51
2H	264,000	5.21	9.26	82.17
TOTAL In-Situ	2,610,000	5.03	7.76	67.80

The mineable ore reserve was calculated from the in-situ reserve using these parameters:

(a) Dilution: 10 percent at zero grade;
(b) Mining Recovery: 75 percent of in-situ resource.

RECOVERABLE ORE RESERVE ESTIMATE

	Quantity Tonnes	Lead Percent	Zinc Percent	Silver g/Tonne
Diluted	2,153,000	4.58	7.05	61.73
	=========	=====	=====	======

These reserves are classed as probable.

Closely similar tonnes and grades were scheduled by Curragh, in the Case 010 forecast of March 1989, as shown below. However, the mining plan has been revised. It is now planned to leave the graphitic bands of ore grade material in place since they would aversely affect mill recovery. The new planned extraction is also shown below:

	Quantity Tonnes	Lead Percent	Zinc Percent	Silver g/Tonne
Curragh Case 010	2,000,000	4.56	6.97	62.0
Curragh S89 Alpha II	1,178,000	4.11	6.27	60.0

As stated previously, the overall reserves are in close agreement. The planned production tonnages and grades are evidently possible but Kilborn has not had an opportunity to assess the relative economics of processing the graphitic material or leaving it in the ground.

6.4 GRUM DEPOSIT

The ore reserve and mineral inventory check calculations by Kilborn were based on the following information supplied by Curragh:

(a) Grum drill hole data base;
(b) Sections through the deposit with geologically interpreted ore zones and ultimate pit outline;
(c) Grum deposit, Cyprus Anvil "Simpson Adamson" Reserve Calculation - December 1982.

The Grum deposit has been core drilled from surface by Prospectors Airways and, to a lesser extent, by Kerr-Addison. Following this drilling, Kerr-Addison undertook an underground exploration program to bulk sample the deposit and did additional core drilling from underground. Cyprus Anvil did additional drilling after acquisition of the property from Kerr-Addison. No additional drilling has been done since 1981. Sections through the deposit at 60.5 metres, which were provided, showed the ultimate pit outline based on Curragh pit planning. The geological interpretation was that used by Cyprus Anvil in 1982. The pit outlines shown on the sections were the work of Curragh in 1986.

Kilborn did not verify the drill logs, assay methods, drill hole surveys nor the plotting of the drill holes on the sections.

Kilborn did check that the ore intersections, as listed, were plotted correctly on the sections.

Kilborn used the ore zones as outlined in the Grum cross-sections. The areas of mineralization were outlined and measured by planimeter, and projected halfway to the adjacent drilled sections.

Tonnages were calculated from the planimetered area, the distance between sections and the specific gravity assigned to each ore intersection.

Although the ore deposit is quite well defined, Kilborn has classed the ore reserve as probable, because of the post mineralization deformation and the drill hole spacing in some areas.

The mineral resource has been calculated at a cut-off grade of 4 percent combined lead plus zinc. The in-place resource was divided into in-pit and out-of-pit material and further divided into A type ore and other ore types. Out-of-pit material is listed in section 7.0.

The in-pit reserves at four percent cut-off are as follows:

IN-SITU RESERVE ESTIMATE

Ore Type	Quantity Tonnes	Lead Percent	Zinc Percent	Silver g/Tonne
A	11,702,000	2.77	5.03	48.73
Other	12,408,000	4.19	6.68	70.44
TOTAL Within Pit	24,110,000	3.50	5.88	59.91

To determine the mineable reserves, the quantity of material was left constant and the grades were reduced by 10 percent to compensate for the mixing of ore and waste on the contacts of the deposit. This assumes that 10 percent of the ore in-place will be hauled to the waste dump and this quantity is replaced by an equal amount of waste which is mixed with the ore and hauled to the mill.

RECOVERABLE ORE RESERVE ESTIMATE

Ore Type	Quantity Tonnes	Lead Percent	Zinc Percent	Silver g/Tonne
A	11,702,000	2.49	4.53	43.86
Other	12,408,000	3.77	6.01	63.40
TOTAL Diluted	24,110,000	3.15	5.29	53.92

In the Curragh 11-year projection, the Grum open pit is scheduled to produce only 23,982,000 tonnes but it is believed that actual production will be close to the Curragh in-pit mineable reserve of:

Quantity Tonnes	Lead Percent	Zinc Percent	Silver g/Tonne	Gold g/Tonne
25,161,000	2.96	5.01	50.00	0.81

Curragh's reserve calculations are stated to include a 5 percent mining loss and 15 percent dilution at zero grade.

The diluted tonnage listed by Curragh is approximately 5 percent greater than the diluted tonnage calculated by Kilborn. The lead contents and zinc contents of these diluted tonnages are within 2 percent of each other. The Kilborn calculation shows an 8 percent greater silver content, and Kilborn did not calculate the gold content. The percentage differences between the two calculations are small, therefore, Kilborn considers that Curragh's calculations for the Grum open pit are acceptable.

6.5 VANGORDA DEPOSIT

The check calculation of ore reserves by Kilborn was based on the following information supplied by Curragh:

(a) Vangorda drill hole data base, dated July 31, 1987;
(b) Section through deposit, with drill holes plotted, geological interpretations and pit outlines;
(c) Sections through deposit, with computer plotted grade contours;
(d) Internal report entitled 'District Geology, Vangorda/Grum, September 1987'.

The Vangorda deposit had been drilled by Prospectors Airways and, to a lesser extent, by Kerr-Addison. Cyprus Anvil re-drilled the deposit in 1979, with a small amount of fill-in drilling in 1981. The Vangorda drill hole data base contains only the Cyprus Anvil drilling results; Curragh drilled three additional holes during 1988, but these were not included.

For the Cyprus Anvil drilling, the deposit was divided into sections at 100 feet apart, and a fence of core holes was drilled on every second section. The holes are almost all vertical, and they are roughly 100 feet apart along the section line. Geological interpretations and the open pit outline are shown on the sections; the interpretations and the pit design were done by Curragh in 1986.

Kilborn did not verify the drill logs, assaying methods, drill hole surveys nor the plotting of the drill holes on the sections.

Kilborn did check that the ore intersections, as listed in the data base printout, were plotted correctly on the sections. A few missing intersections were added, and a few adjustments were made to the lengths of intersections used in the calculations, but in general the plotting related accurately to the printout.

Kilborn projected the mineralization from hole to hole along the sections; the geological interpretations governed as to type of ore and, in some cases, as to which ore intersections were joined in adjacent holes. The areas of mineralization were outlined and were measured by planimeter, and projected half way to the adjacent drilled sections. However, before calculations were made, the adjacent sections were examined to determine if there were related structures on the other sections; in a few cases the mineralized outlines had no adjacent related structures and these mineralized outlines were not included in the calculations.

Tonnages were calculated from the planimetered area, the average distance between drilled sections (60.96 metres) and the average specific gravity of the ore type.

Although they are quite well defined, Kilborn has classed the ore reserves as 'probable', because of the distance between drilled sections and because of the qualitative nature of the geological interpretations.

The ore reserves and mineral resources are listed at a cut-off of 4 percent combined lead and zinc. Curragh had planned to provide plus 5 percent combined mill feed during active mining, and to stockpile the plus 4 percent minus 5 percent fraction to be blended later with higher grade ore from Grum. In the tabulation below for Vangorda, the mineralization is divided into ore reserves at 5 percent combined lead plus zinc cut-off and ore reserves at plus 4 percent minus 5 percent, for material within the pit limits. All grades shown are undiluted.

IN-SITU RESERVE ESTIMATE

Type	Tonnes	Pb %	Zn %	Ag g/t	Au g/t
>5% Pb+Zn	5,968,209	4.10	5.10	54.24	0.63
>4% <5% Pb+Zn	359,456	2.17	2.34	29.26	0.69
TOTAL Within Pit	6,327,000	3.99	4.94	52.83	0.63

To determine the mineable reserves, the same assumptions of 90% recovery and 10% dilution at zero grade were applied to the Vangorda in-situ reserves.

RECOVERABLE ORE RESERVE ESTIMATES

	Tonnes	Pb %	Zn %	Ag g/t	Au g/t
Diluted	6,327,000	3.59	4.45	47.6	0.57

In Curragh's 11-year projection, the Vangorda open pit is scheduled to produce:

Quantity Tonnes	Lead Percent	Zinc Percent	Silver g/Tonne	Gold g/Tonne
6,935,000	3.49	4.51	48.00	0.65

Curragh's reserve calculations are stated to include a 5 percent mining loss and 15 percent dilution at zero grade.

The diluted tonnage listed by Curragh is approximately 10 percent greater than the diluted tonnage calculated by Kilborn. The metal contents of these diluted tonnages are greater in the Curragh calculations by approximately 7 percent for lead, 11 percent for zinc, 13 percent for silver, and 25 percent for gold. The two estimates are not directly comparable because the Curragh projections are taken from a later pit design by the independent consultant, Ion Vintila. Kilborn has not yet had the opportunity of applying the new pit outlines to the old, sectional reserve blocks to determine the extent of the changes. Also, Kilborn has been informed that the specific gravity figures used at the time had been reduced by 5 to 10 percent from the measured pulp densities to allow for voids in the solid rock. Experience at Faro had shown that the allowance was unnecessary but the Vangorda density data had not been restored at the time of Kilborn's estimate. This explanation should also be confirmed. However, in the context of the total reserves for this project, the variations are not significant.

6.6 DY DEPOSIT

The ore reserves and mineral inventory are based on the following information supplied by Curragh:

(a) Cyprus Anvil "B.V. Hall" 1981 geological reserve calculations, with accompanying plans and sections;
(b) Cyprus Anvil "Rollings" 1982 ore calculations.

The Dy deposit has been explored by 54 core holes, diamond drilled from surface over the period 1976 to 1981, totalling 46,000 metres and averaging 852 metres. These deep holes were surveyed, at regular intervals down the hole, so that the various sample points could be plotted with reasonable accuracy.

B. V. Hall estimated the mineral inventory as follows:

Classification	Quantity Tonnes	Lead Percent	Zinc Percent	Silver g/Tonne
Drill Indicated	17,388,056	5.82	6.84	83.1
Drill Inferred	3,946,071	5.03	7.45	75.3
TOTAL DEPOSIT	21,334,127	5.68	6.95	81.6

This was based on a minimum mining width of 3.5 metres, a cut-off grade of 9 percent combined lead plus zinc, a polygonal area of influence for each intercept, measured by planimeter, and the measured specific gravity for that intercept. Tonnages were collated into four stratigraphic horizons designated 2, 3, 4 and 5.

Rollings used the same methodology, but reduced the number of stratigraphic horizons to three, designated A2, B2 and 3A. This estimate was broken down by rock type (for milling characteristics) and included gold and copper values, but was not classified. Results are summarized as follows:

Horizon	Quantity Tonnes	Lead Percent	Zinc Percent	Silver g/Tonne	Gold g/Tonne
A2	13,611,629	5.87	6.14	85.8	1.06
3A	960,601	5.03	5.76	63.3	0.59
B2	6,487,750	4.91	8.14	82.6	0.76
TOTAL	21,059,980	5.54	6.74	83.8	0.95

The Rollings total of 21.06 million tonnes was carried as 'possible' ore in Curragh's mineral inventory for several years because there were no plans to develop the deposit. When Curragh decided to include the Dy in the Case 010 12-year plan, Kilborn retained an independent consulting geologist, P. C. Coltas, to review the previous estimates and to reclassify the inventory. His report is appended (see Appendix C).

For comparison with Rollings, the total P.C. Coltas inventory is summarized as follows:

Horizon	Quantity Tonnes	Lead Percent	Zinc Percent	Silver g/Tonne	Gold g/Tonne
A2	12,927,675	5.84	6.04	86.1	1.01
B2	7,187,150	4.83	8.08	81.5	0.75
INVENTORY	20,114,825	5.47	6.77	84.5	0.91

Basically, the 3A horizon (based on two totally isolated intercepts) has been eliminated, and some material has been assessed more stringently in the A2 and more generously in the B2 horizons.

The classified P. C. Coltas estimate is summarized as follows:

Classification	Quantity Tonnes	Lead Percent	Zinc Percent	Silver g/Tonne	Gold g/Tonne
A2 Probable	9,358,975	5.74	6.19	85.8	1.03
B2 Probable	5,561,550	5.03	8.40	85.5	0.77
PROBABLE RESERVE	14,920,525	5.45	7.02	85.7	0.93
A2 Possible	3,568,700	6.23	5.64	86.9	0.96
B2 Possible	1,625,600	4.14	7.00	68.1	0.66
POSSIBLE INVENTORY	5,194,300	5.57	6.07	81.0	0.87

Kilborn assessed the P. C. Coltas probable reserves in terms of mineability. The proposed mining method is room and pillar—the Dy deposit shares some physical characteristics with the Elliot Lake uranium camp. Mining recovery was assumed to be 70 percent in thicknesses of up to 12 metres, and 75 percent where greater than 12 metres. Multiple intersections were assessed on the thickness of the intervening waste:

- Less than 6 metres - one lens eliminated;
- From 6 to 15 metres - recovery from lower lens reduced;
- Over 15 metres - 70 percent recovery from both lenses.

Dilution was assumed to be 10 percent at zero grade.

The diluted, mineable reserves are summarized as follows:

Source	Quantity Tonnes	Lead Percent	Zinc Percent	Silver g/Tonne	Gold g/Tonne
A2 Probable	7,515,000	5.19	5.62	78.0	0.96
B2 Probable	3,995,000	4.62	8.02	79.7	0.69
MINEABLE RESERVE	11,510,000	4.99	6.46	78.6	0.87

The Curragh reserves are based on B.V. Hall's Drill Indicated (Probable) inventory and assume that 65 percent of this material can be mined with zero dilution by room-and-pillar methods leaving a thin layer of ore on the hanging-wall and footwall of the openings. Mine production over the 11-year projection is described in section 9.6 and summarized below:

Source	Quantity Tonnes	Lead Percent	Zinc Percent	Silver g/Tonne	Gold g/Tonne
Curragh Reserves	11,300,000	5.82	6.84	83	0.94
Alpha II Production	8,500,000	5.80	7.40	89	0.94

7.0 MINERAL INVENTORY (NOT IN CURRENT PLANS)

7.0 MINERAL INVENTORY (NOT IN CURRENT PLANS)

7.1 SUMMARY

Curragh has various sources of mineral inventory which are not included in the current Long Range Plan, S89: Alpha II. These include stockpiles of lower grade material near the Faro mill; unmined material left in the ground at the Grum, Dy and Swim deposits; the Cirque deposits in British Columbia; the Mt. Hundere deposits in the Yukon Territory; and the Westray coal project in Nova Scotia.

7.2 FARO DIVISION

7.2.1 Stockpiles

Curragh expects to have the following material stockpiled at the end of the scheduled production in 2000:

Source	Tonnes (thousands)	Lead percent	Zinc percent	Gold g/t	Silver g/t
Faro >4%	2,059	1.70	2.82	0.10	22
Grum >4%	1,746	2.42	4.21	0.81	37
Mixed >3% <4%	4,373	1.27	1.99	N.A.	20

Stockpiles form proven resources whose values are dependent on metal prices and operating costs at those times when processing capacity becomes available.

7.2.2 Unmined Inventory

Curragh expects to have the following unmined inventory at the end of the scheduled production period:

Property	Tonnes (thousands)	Pb %	Zn %	Ag g/t	Cut-off Grade % (Pb+Zn)
Grum - Champ Zone:	1,700	3.5	4.3	46	4
- Main Zone:	6,539	3.03	4.57	46.3	4
Dy	8,888	5.84	6.30	77	9
Swim	5,130	3.5	4.4	47	4

Note: Kilborn did not calculate the above mineral inventories, other than Grum Main Zone.

Grum Main Zone

The main Grum deposit covers a 792-metre strike length (2,600 feet from 6100W to 8700W). The geological reserve below and beyond the designed pit perimeter, at a 4 percent combined lead plus zinc cut-off grade, is derived by subtraction as follows:

	Tonnes	Pb%	Zn%	Ag g/t
Main Zone (61W to 87W)	30,649,000	3.4	5.6	57
In-Pit Reserves	24,110,000	3.50	5.88	59.9
Out-of-pit Reserves	6,539,000	3.03	4.57	46.3

The 4 percent cut-off used for open pit reserves is too low to cover the much higher unit costs of underground mining. Kilborn assessed the plotted cross-sections (62W to 86W) for mineable blocks above a cut-off grade of 9 percent lead plus zinc. Continuity between sections was addressed and some small, isolated blocks were ignored. The in-situ inventory and the diluted mineable inventory, assuming 95 percent recovery and 10 percent dilution at zero grade, were estimated as follows:

	Tonnes	Pb%	Zn%	Ag g/t
In-situ Inventory >9%	3,064,700	4.83	8.20	77
Recoverable Inventory	3,200,000	4.39	7.45	70

Curragh has classified the Main Zone geological reserve of 30.65 million tonnes as Proven in the context of open pit mining. However, Kilborn would classify the 3.2 million tonne underground recoverable inventory as probable pending fill in drilling and a more rigourous evaluation of the 9 percent cut-off grade which was assumed to cover several mining methods.

Grum - Other Zones

Two adjacent mineralized zones are known to abut the Main Zone: the Champ Zone and the North West Extension. The Champ Zone, east of Main Zone (4900W to 6100W), is reported to contain probable geological reserves of 1.7 million tonnes at 3.5 percent lead, 4.3 percent zinc, and 46 g/t silver above a cut-off grade of 4 percent lead plus zinc: this is assumed to be open pit material but has not yet been evaluated.

The North West Extension (8700W to 10100W) has the potential to contain reserves of several million tonnes above the 4 percent cut-off. Ten holes were drilled on section 88W, but further west the information becomes very sparse. There are four holes on 92W and single holes on sections 96W, 98W and 100W. They all encountered multiple intercepts above a 4 percent cut-off, but there are very few above the 9 percent lead plus zinc cut-off grade used to establish underground mineable reserves. Additional drilling is required in this area.

Dy Deposit

Curragh projects an unmined inventory of 8.89 million tonnes at the end of the year 2000, including pillars and remnants. This is calculated by subtracting the projected production from the B.V. Hall undiluted in-situ reserve of 17.39 million tonnes.

To obtain a realistic estimate of the remaining mineable reserve, the production of 8.5 million tonnes must be subtracted from the estimated diluted mineable reserve which, using Curragh's figures, leaves a remainder of 2.80 million tonnes at 5.88% Pb, 5.14 % Zn, 65 g/t Ag and 0.94 g/t Au. Kilborn believes that, with the exception of the lead grade, these figures are reasonably conservative.

Swim Deposit

The Swim deposit is located a further 5 kilometres east of Dy, a haul distance of 28 kilometres from the Faro concentrator. The geological reserves are reported as 4.3 million possible tonnes at 3.8 percent lead, 4.7 percent zinc, and 51 g/t silver above a cut-off of 6 percent lead plus zinc (and 5.1 million tonnes at 3.5% Pb, 4.4% Zn and 47 g/t Ag above a 4% cut-off). This is a near surface deposit suitable for open pit mining.

A recent preliminary manual pit design by Mr. Ion Vintila, an independent consultant, indicates the following undiluted quantities within the pit boundary which includes a haulage ramp:

	Volume (m^3)	S.G.	Tonnes	Pb%	Zn%	Ag
High Grade (>5%)	771,000	3.85	2,970,000	4.0	4.9	51
Low Grade (>4% <5%)	171,000	3.60	610,000	2.0	2.4	32
Overburden	2,000,000	2.20	4,400,000			
Waste Rock	8,658,000	2.70	23,370,000			
TOTAL IN-SITU	11,600,000		31,350,000			

From the limited drilling data, it is believed that the Swim deposit resembles the Grum in physical irregularity. Curragh has assumed 95 percent recovery (5 percent lost within the waste rock blasts) and 15 percent dilution at zero grade. This appears to be a conservative assumption: the dilution quantity could be higher but would contain low grade mineralization. The mineable inventory and waste tonnages within the pit boundary become:

	Tonnes	Pb%	Zn%	Ag	Strip Ratio
Overburden	4,400,000				
Waste Rock	23,040,000				
High Grade (>5%)	3,245,000	3.48	4.23	44	8.66:1
Low Grade (>4% <5%)	665,000	1.95	2.35	32	
TOTAL ROCK	31,350,000				
TOTAL INVENTORY	3,910,000	3.22	3.91	42	7.02:1
	==========	====	====	==	======

In addition, Curragh has assumed a gold head grade of 0.8 g/t, the average of the other deposits in the area. Gold assays were not available for many of the Swim samples.

7.3 MT. HUNDERE PROJECT

Curragh owns an 80 percent joint venture interest in the Mt. Hundere lead-zinc-silver property located 45 kilometers north of Watson Lake, Yukon Territory. Development of the project would involve mining and processing at the site and haulage of concentrates via the Alaska Highway to Skagway - in B-Train units as presently used from Faro. The existing Whitehorse administration office, some 400 kilometers east of Watson Lake, could serve the Mt. Hundere operations as well as continuing to serve the Faro Division.

The original surface showings were discovered in 1962. Successive owners had reportedly carried out geochemical and geophysical surveys, trenching and 23,484 metres of diamond drilling in 190 holes before Curragh purchased the property in April 1989.

The skarn-type lead-zinc-silver mineralization occurs in shallow dipping beds (usually less than 25 degrees) cut by a number of steeply dipping faults with vertical displacements which can exceed 100 metres.

In February 1989, an independent geological consulting company prepared a mineral inventory for Canamax (the previous owner) which identified 5,267,000 tonnes of material with average grades of 5.3% lead, 13.4% zinc and 61 g/t silver, in three separate deposits: Jewelbox Hill, Gribbler Ridge and North Hill (where the Attila and Burnick Zones have been shown to join together). Two other promising targets were noted.

Metallurgical testwork by Bacon, Donaldson and Associates Ltd. indicated that high grade lead and zinc concentrates could be produced from Mt. Hundere sulphide and oxidized samples.

A portion of the Jewelbox Hill deposit appears to be amenable to open pit mining. The balance of the deposits, if feasible, would be mined by underground methods, probably room and pillar.

To further evaluate the deposits, an exploration program is in progress comprising 29,000 metres of diamond drilling in 142 holes, site soils and environmental investigations, and upgrading of the access road. While the program is not complete, results to date appear to confirm the mineral inventory, and reserve calculations will be available in a few weeks.

7.4 CIRQUE PROJECT

The Cirque project is located in northeastern British Columbia, 925 kilometres north of Vancouver. It is 475 kilometres northeast of Prince Rupert, the nearest deep water port, and 280 kilometres north of Mackenzie, the nearest railhead. This deposit would be developed separately from the Faro Division operations.

The deposit was discovered in 1977. Diamond drilling was carried out between 1978 and 1982. A total of $21,000,000 was spent on the project by the former owner. Included in the total is $11,000,000 drilling at Cirque and South Cirque (74 holes totalling 23,400 metres and 28 holes totalling 21,250 metres respectively); $3,250,000 on other claim groups and on regional exploration and $5,300,000 on road and airstrip construction. It is understood that Curragh recently sold a 15 percent equity interest in the property to Asturiana de Zinc (ADZ) for $10,000,000.

Cirque is a stratiform, sediment hosted, massive sulphide/barite deposit. The deposit is a tabular body 1000 metres long, 300 metres wide and 2 to 70 metres thick. It dips 30 to 40 degrees towards the southwest, opposite to the topographic slope, and plunges south.

Cirque geological reserves are estimated at 32.2 million tonnes averaging 2.15 percent lead, 7.88 percent zinc, and 47.7 g/t silver. A high grade zone contains a mining inventory of 22.2 million tonnes averaging 2.74 percent lead, 9.25 percent zinc, and 57 g/t silver above an 8 percent lead plus zinc cut-off. Eighty-six percent of this in-situ inventory is judged to be extractable by room and pillar underground mining.

Metallurgical test work done to date indicates that Cirque ores produce a high quality zinc concentrate and an average quality lead concentrate using conventional flotation. Silver recovery in base metal concentrates is low, however preliminary work shows high silver recovery by autoclaves leaching a pyrite concentrate.

The Cirque Project Development Plan of May 1988 indicated that the deposit could support a 3,500 tonnes per day underground mine and surface concentrating plant under certain conditions. Annual output from such a facility would be 170,000 tonnes of zinc concentrate and 54,000 tonnes of lead concentrate, containing 1,000,000 ounces of silver.

To further evaluate the deposit, an underground exploration program with a total estimated cost of $13,000,000 is in progress. This program will include 1,600 metres of underground development and 123,000 metres of underground diamond drilling, necessary bulk sampling and metallurgical testing, mine planning and design work.

Preliminary drilling one kilometre south of the Cirque deposit has indicated the presence of an additional deposit, South Cirque, with potential to contain 20,000,000 tonnes of ore of tenor similar to Cirque. The South Cirque deposit is located below the proposed production adit for Cirque.

7.5 WESTRAY COAL PROJECT

Curragh has an agreement in principle to acquire, from an affiliated company, control of the Westray coal project in Pictou County, Nova Scotia near the town of New Glasgow.

The Pictou County coalfield was first discovered in 1798 and has already produced more than 57 million tonnes of coal. The Westray property lies immediately east of the old Allan Mine workings which were primarily on the Foord Seam. In a 1987 feasibility study for Suncor Inc. (the previous owner), Associated Mining Consultants Limited estimated in-situ coal resources of 54 million tonnes in the Foord Seam. After delineating logical mining blocks and a 100 m boundary pillar, and excluding high (>24%) ash content material at the top and bottom of the seam, the reserves available for mining were estimated as 45.0 million tonnes of "demonstrated" (proven plus probable) low sulphur coal with an average ash content of 23.8 percent, and a further 2.0 million tonnes of "inferred" (possible) coal with 21.0 percent ash. There is potential for additional reserves in extensions to the Foord Seam and in the McLeod and Cage Seams.

The proposed mining method is room and pillar in order to provide the flexibility to cope with faulting, coal extraction heights of up to 7 metres, and surface subsidence restrictions. Total planned production from first pass mining and second pass pillar recovery is 18.7 million tonnes, equivalent to 40 percent extraction. Planned access is through twin declines on which development work has started but is temporarily suspended pending resolution of financial arrangements.

A wash plant is planned which will process approximately 3,200 tonnes of raw coal per day to produce 1.0 million tonnes per year of clean coal in one of two grades:

- Regular, 20.0% Ash ($\pm$ 1.5%), 0.8% sulphur, 10,900 Btu/lb
- Premium, 13.5% Ash ($\pm$ 1.0%), 0.8% sulphur, 12,000 Btu/lb

The two products will be dried to 8% total moisture and separately stored for rail or truck haulage to Nova Scotia Power Corporation (contracted for 700,000 tonnes per year) and other customers.

8.0 MINING PROPERTIES WITHOUT MINERAL INVENTORIES

8.0 MINING PROPERTIES WITHOUT MINERAL INVENTORIES

8.1 SWIM BASIN, FARO DIVISION

The Swim Basin (southeast of Vangorda Plateau) is a structural basin that represents the southeast third of the Anvil District. Curragh controls 830 claims in the basin encompassing about 12,000 hectares.

The Swim deposit along with two non lead-zinc bearing massive sulphide occurrences are in the Basin. The Swim Basin is an area of very poor outcrop exposure thus its mineral potential is difficult to evaluate. There has been extensive electromagnetic, magnetic and gravity surveying carried out in the basin between 1964 and 1981. An overburden drilling program was carried out in the early 1970s. About 50 diamond drill holes are scattered through the Basin. Despite this amount of work, there are exploration targets remaining. Three holes (total 250 metres) were drilled in 1988.

8.2 AKIE DISTRICT, BRITISH COLUMBIA (NOT INCLUDING CIRQUE)

8.2.1 Elf

The Elf claims cover 3,200 hectares, 35 kilometres southeast of Cirque. There are 26 drill holes totalling 10,500 metres on the property. Approximately $2,200,000 has been spent.

Mineralization on the Elf claims is stratiform and, like Cirque, is hosted by Devonian sedimentary rocks. The steeply dipping mineralized zone has been traced by drilling for 800 metres along strike and 600 metres down dip. The best intersection, 300 metres down dip from the discovery showing, averages 13.8 percent lead plus zinc and 27 g/t silver over 11 metres.

8.2.2 Fluke

The Fluke claims are located 16 kilometres southeast of the Cirque deposit. The property is 4,000 hectares in area. From 1980 to 1982, the former owner carried out drilling and other work on the property totalling $890,000. There are seven holes totalling 3,295 metres on the claims.

The style and setting of Fluke mineralization is similar to the South Cirque deposit. Limited drilling to date has intersected mineralization 100 metres down dip from the discovery showing. Although intersections to date are low grade, the type of mineralization warrants more extensive investigation by further drilling.

9.0 MINE PRODUCTION PLANS FOR FARO DIVISION

9.0 MINE PRODUCTION PLANS FOR FARO DIVISION

9.1 SUMMARY

Curragh has prepared an 11-year Faro Division Mine Plan (S89:ALPHA II) in which ores are produced from various sources, as illustrated below:

	Year 1990	91	92	93	94	95	96	97	98	99	2000
Faro (Open pit)	9,740,000 tonnes										
Faro (Underground)	1,178,000 tonnes										
Vangorda (Open pit)		6,935,000 tonnes									
Grum (Open pit)			23,981,000 tonnes								
Dy (Underground)			8,500,000 tonnes								

The quantities of ores mined at the different sources may vary from year to year, and stockpiles are used to avoid large changes in the milling rate. The milling rate averages 4,445,000 tonnes per year over this 12-year forecast; the high is 4,730,000 tonnes in 1990 and the low is 4,300,000 tonnes in the year 1999.

The milling schedule by sources of ore, shown on Figure 9-1, differs from the mining forecast shown previously, because of stockpiling. The stockpiles will make it possible, if required, to process a single ore type for extended periods - say one month - before changing to another ore type.

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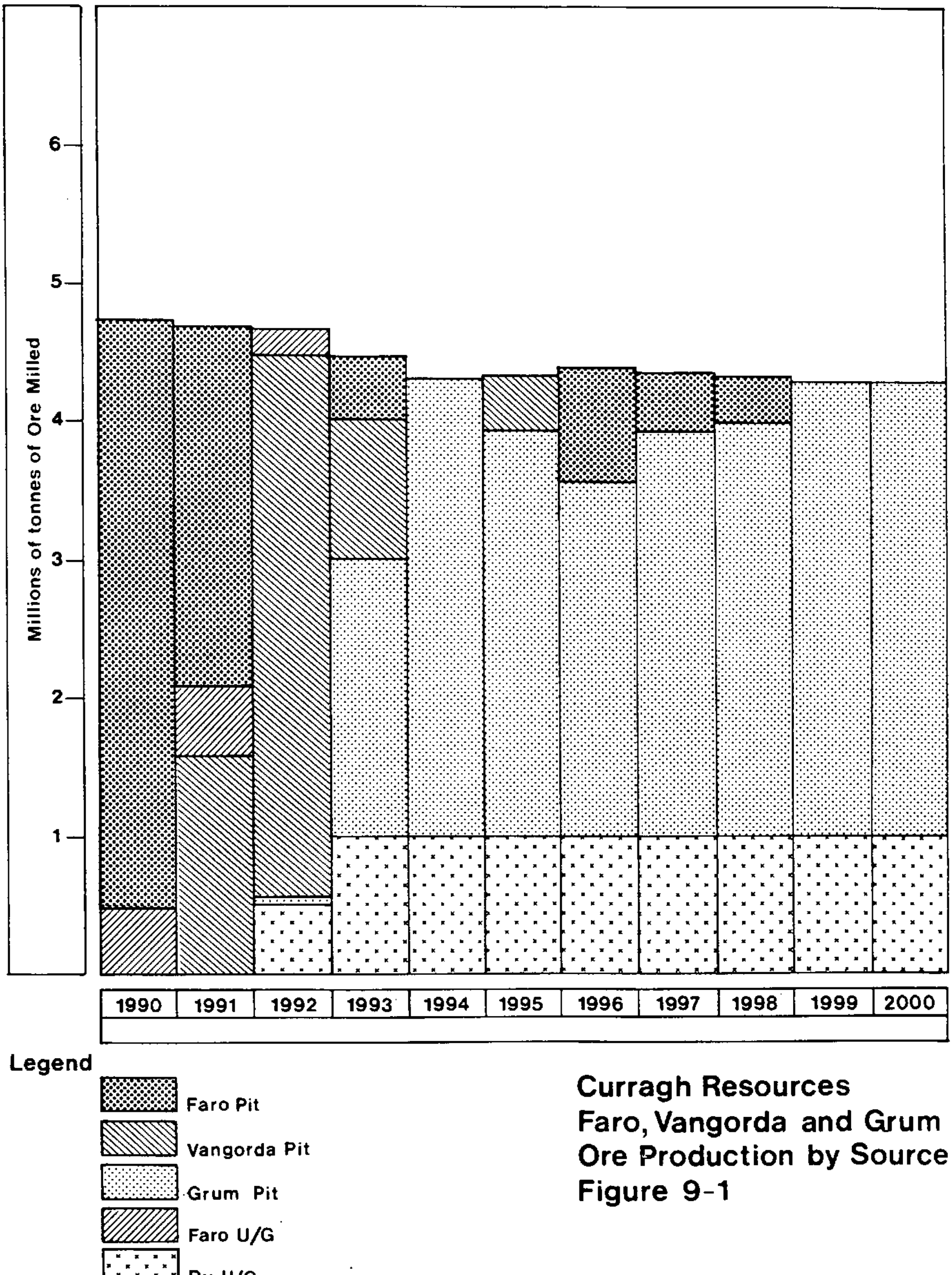


Curragh Resources
Faro, Vangorda and Grum
Ore Production by Source
Figure 9-1

At the end of this 11-year schedule, there will be mineral resources remaining in the Dy underground mine and outside the pit limits at Grum. These resources may be mined after 2000, particularly if new open pit deposits are developed to maintain mill throughput. The other deposits (Faro and Vangorda) will be mined out.

Kilborn reviewed Curragh's designs for the three open pits to verify the quantities of excavation within these designs. It was found that the quantities listed by Curragh were accurate. It is Kilborn's opinion that the pit designs and schedules are realistic, and can be achieved by Curragh.

Mining plans for the Faro underground mine had been prepared by Kilborn in April 1987. Since then, Curragh has received a proposal from a contractor for mining this deposit based on a different mining concept; Curragh projections now are based on this proposal. Kilborn cannot agree or disagree with the proposal without an engineering study.

The Dy underground mine projections also are based on a proposal by a contractor. Again, it is difficult to agree or disagree with this proposal without detailed analysis.

9.2 FARO OPEN PIT

When the Cyprus Anvil operations were halted in 1981, there was almost no ore available for mining in the Faro open pit. The government-funded waste stripping program of 1983-84, plus the waste that Curragh removed in the first half of 1986, released enough ore for the reopening of the concentrator in June, 1986.

Curragh's waste stripping operations were under way by February, 1986, at a rate of excavation that exceeded the rates achieved in the past in this pit. The high rates of total excavation were necessary in 1986 (23,400,000 tonnes) and in 1987 (29,300,000 tonnes) in order for Curragh to achieve their schedules. However, in 1988, the configuration of the pit became such that less waste stripping was required in order to release the scheduled ore; total excavation was cut back to 23,400,000 tonnes in 1988, and 9,093,000 in the first half of 1989.

The excavations (in tonnes) scheduled for the remainder of the life of the Faro pit are shown below:

Year	Ore	Waste	Total Excavation
1989 (2nd half)	2,270,000	7,902,000	10,172,000
1990	6,146,000	7,231,000	13,377,000
1991	3,595,000	2,240,000	5,835,000

Curragh has a good record for planning and scheduling this pit over the last three years, and there is no reason to believe that the current schedule will not be achieved.

Kilborn visited the pit in December, 1988 and briefly reviewed the mining plans. Since then, an independent mining consultant, Mr. Ion Vintila, has refined the pit design, reducing the quantity of waste excavation and eliminating the need to remove an existing section of ramp built from waste rock. It is noted that there are three widely separated working areas; therefore, the effect of a confined working area should not be a factor until the last year - at the time when production demands are the lowest.

There is surplus equipment capacity at present. It is planned to transfer the mining equipment, as required, to the Grum/Vangorda operations. Curragh's schedule for assignment of major units (shovels, drills, trucks) does not appear to conflict with either operation.

9.3 FARO UNDERGROUND MINE

The deposit scheduled for underground mining is an extension to the southwest of the deposit which is now being mined by open-pit methods. A lens of high grade mineralization will outcrop in the final pit wall; the lens was not included in the designed pit because of its high stripping ratio. It is planned to recover this high grade mineralization by underground mining methods.

Mining practices at the Faro underground mine will be influenced by two major factors:

(a) The mineralized tabular zones dip at 15 degrees to 25 degrees to the horizontal.

(b) The rock comprising the hangingwall and the footwall of the underground mine openings is mineralized. This will reduce the negative effects of dilution when mining.

In the April, 1987 report, Kilborn proposed a conventional open stope and pillar mining method. The open stopes are mined straight up the dip. The broken ore is scraped down to a loading point since the dip is too flat for the ore to slide down by gravity, and too steep to use mobile equipment. Preproduction development is needed, some of which is in ore, but the main access is in waste beneath the ore zone.

Curragh since has received a proposal from a contractor for mining the Faro underground using a cross-pitch room and pillar mining method; mobile equipment is used for most tasks. In cross-pitch mining, the entry drifts are driven at an angle across the dip of the deposit, the angle selected allows the entry drift floor to be less steep than the dip of the deposit, and high productivity mobile equipment can be used at all mining faces. In addition, cross-pitch mining permits the primary extraction sequence (rooms) to proceed down dip in ore; in the case of Faro underground, full production would be reached with a minimum of advance development, and most development headings in waste would be eliminated.

The cross-pitch room and pillar method is common in coal mines, but it does not accommodate itself easily to changes in the dip or undulations in the hangingwall or footwall as are likely to occur in this deposit.

Curragh's 11-year forecast is based on the proposal for contract mining using the cross-pitch method; the times (and monies) listed are applicable to this system only. However, it is Kilborn's opinion that there is insufficient drilling information to base this projection exclusively on the cross-pitch method - it would be prudent to prepare a contingency plan to defer the ore scheduled in the first years from the Faro underground.

The schedule, as it is now, is shown below:

Year	Tonnes	% Pb	% Zn	Ag g/t	Au g/t
1990	478,000	4.1	6.2	57	0.35
1991	500,000	4.1	6.3	62	0.29
1992	700,000	4.1	6.3	62	0.29

9.4 GRUM OPEN PIT

A report entitled, "Curragh Resources, Long Range Plan For Faro, Vangorda & Grum Deposits", was issued by Kilborn in April, 1987. Curragh provided the design for the Grum open pit; Kilborn verified the quantities of excavation within the design. The verification calculations were based on planimetered areas of the bench plans, and it was found that the quantities listed by Curragh for this design were accurate.

Since April, 1987, Curragh has redesigned the Grum pit to exclude certain quantities of mineralized material that had very high waste stripping ratios, thus reducing the total excavation projection.

For the Grum pit, Curragh provided the revised plans and sections and Kilborn carried out manual checks on the differences between this design and the previous pit design. Kilborn found the revised quantities to be accurate (as listed in Curragh's production forecast, Case 8802, 88-03-03). In March 1989, an updated design, Case A010, 89-03-17, indicated a 5 percent increase in the quantity of ore over the 1988 case, for a very small increase in the total excavation.

The current 11-year forecast is based on Case S89: ALPHA II.

A comparison is given of excavations in the four cases referred to above:

Case	Overburden (Tonnes x 1000)	Waste (Tonnes x 1000)	Ore (Tonnes x 1000)	Total (Tonnes x 1000)
Curragh and Kilborn (1987)	40,645	190,677	24,993	256,315
Curragh 88-03-03	(Combined)	175,832	23,967	199,799
Curragh 89-03-17	41,221	133,859	25,161	200,241
Curragh ALPHA II	29,788	147,817	23,982	201,587

As stated previously, Kilborn checked Case 88-03-03 and found it to be accurate, but Kilborn did not check the small differences between Curragh's 1988 and 1989 cases.

At year-end 1988, Curragh had started removing the unconsolidated overburden from the Grum deposit - some rented equipment was being used. The unconsolidated overburden consists of morainal material (tills) and glaciofluvial silts, sand and gravels; these sediments are said to be water saturated. There was no problem in handling overburden in the cold weather, but some provisions may be required for handling wet material in the warm periods. If the wet material does not pose a serious problem, then there are no anticipated physical constraints on the Grum preproduction development - except the large volume to be moved.

Deliveries of ore from the Grum deposit follow the Vangorda deliveries by more than a year but, because of the major quantity of overburden, the stripping efforts must be concentrated on the Grum deposit in 1989 and 1990. Equipment released from the Faro open pit will be used.

9.5 VANGORDA OPEN PIT

The report, "Curragh Resources, Long Range Plan For Faro, Vangorda & Grum Deposits", was issued by Kilborn in April, 1987. Curragh provided the design for the Vangorda open pit; Kilborn verified the quantities of excavation within the design. The verification calculations were based on planimetered areas of the bench plans, and it was noted that the quantities listed by Curragh for this design were accurate.

Since April, 1987, Curragh has redesigned the Vangorda pit to exclude certain quantities of mineralized material that had very high waste stripping ratios, thus reducing the total excavation projection.

For the Vangorda pit, Curragh provided the revised plans and sections and Kilborn carried out manual checks on the differences between this design and the previous pit design. Kilborn found the revised quantities to be accurate (as listed in Curragh's production forecast, Case 8802, 88-03-03).

Curragh's 11-year forecast is based on Case A010, 89-03-17 which showed an increase in the quantities of both ore and total excavation. Reasons for the increase in the ore were discussed in Section 6.5 of this report.

A comparison is given of excavations in the three cases referred to above:

Case	Overburden & Waste (Tonnes x 1000)	Ore (Tonnes x 1000)	Total (Tonnes x 1000)
Curragh and Kilborn (1987)	21,487	6,459	27,946
Curragh 88-03-03	16,089	6,005	22,095
Curragh 89-03-17 and Alpha II	15,434	6,935	22,369

As stated previously, Kilborn checked Case 88-03-03 and found it to be accurate, but Kilborn did not check the small differences between Curragh's 1988 and 1989 cases. A limited drilling program in 1985 (three holes) provided information for a partial redesign of the Vangorda pit during December, 1988. Since the changes are not great in the context of the total 12-year forecast, Kilborn has accepted these changes without confirmation.

Stripping of the Vangorda pit will start in 1989. Till cover over the ore is thin at the southeast part of the deposit; operations will start in this area to obtain a bulk sample of ore for metallurgical testing.

Capital has been included for the diversion of the Vangorda Creek, which flows over the deposit.

9.6 DY UNDERGROUND

Curragh plans to explore and develop the Dy deposit starting early in 1990. It is proposed to use a contractor to sink a shaft to a depth of 668 metres, carry out a two-phase exploration and development program, and, if successful, put the mine into production at a rate of 1,000,000 tonnes per year. The shaft location has not been finalized because two pilot holes have encountered bad ground conditions.

Phase I, Exploration, includes 1,500 metres of cross-cuts and drill drifts, and 6,700 metres of diamond drilling to confirm the ore reserves in the shallowest portion of the deposit and to assess the extent of folding and faulting and other physical parameters which will affect the mining plant.

Phase II, Exploration and Development, includes 1,380 metres of drifts and ramps (460 metres in ore for a bulk sample), 660 metres of ventilation/escapeway raise, and provision of the first conveyorway to permit start of production in mid-1992. Production is scheduled at 500,000 tonnes during that first year and 1,000,000 tonnes per year thereafter.

Curragh's scheduled production tonnages and grades are set out by years in Table 9.6-1, together with comparative figures from a contractor's proposal. Curragh has advanced the project one year since receiving the proposal which means that shaft work must begin early in 1990. Given the present degree of knowledge and planning of the Dy orebody, either of these schedules is possible. There is a large quantity of higher than average grade ore near the proposed shaft location. However, it is Kilborn's opinion that the millhead grades are likely to be somewhat lower than those shown in the contractor's schedule over the years 1995 to 2000 which assumed that a 'skin' of ore would be left on the hangingwall and footwall of each lens and that dilution would be zero. Such a system is technically possible, but may be impractical.

TABLE 9.6-1

DY ANNUAL PRODUCTION SCHEDULES

Curragh S89: ALPHA II 11-Year Plan

Production Year	Production Tonnes	Average Grades % Pb	% Zn	Ag g/t	Au g/t
1992	500,000	6.2	9.2	98.0	0.9
1993	1,000,000	5.8	8.3	92.0	0.8
1994	1,000,000	5.8	7.5	90.0	0.9
1995	1,000,000	5.8	7.5	90.0	0.9
1996	1,000,000	5.7	7.1	87.0	1.0
1997	1,000,000	5.7	7.1	87.0	1.0
1998	1,000,000	5.7	7.1	87.0	1.0
1999	1,000,000	5.8	6.8	87.0	1.0
2000	1,000,000	5.8	6.8	87.0	1.0
TOTAL	8,500,000	5.80	7.40	89.0	0.94

Contractor's Proposal

Production Year	Production Tonnes	Average Grades % Pb	% Zn	Ag g/t	Au g/t
1993	500,000	5.54	9.29	91.60	0.85
1994	1,000,000	5.54	9.29	91.60	0.85
1995	1,000,000	6.22	8.09	95.23	0.90
1996	1,000,000	6.89	6.80	98.15	0.99
1997	1,000,000	7.06	6.61	96.86	1.12
1998	1,000,000	7.06	6.61	96.86	1.12
1999	1,000,000	6.97	6.97	95.59	1.15
2000	1,000,000	5.19	6.43	78.59	0.98
TOTAL	7,500,000	6.36	7.39	93.16	1.00

The contractor will provide the necessary surface plant (headframe, hoist, compressors, offices, dry, etc.) and all underground equipment required to mine ore and hoist it to a surface bin. Curragh will haul ore from the Dy bin to the mill, or stockpile, and will provide electric power and engineering, geological and assaying services. The mine will produce approximately 3,000 tonnes per day, seven days per week, 336 days per year.

Since efficient processing of the Dy ore requires that the Faro mill capacity of 4,800,000 tonnes per year be filled from other ore sources, and that pit reserves are exhausted in the year 2000, it appears that the Dy production rate of 1,000,000 tonnes per year is somewhat less than optimum.

9.7 SURFACE MINING EQUIPMENT - PRODUCTIVITY AND ALLOCATIONS

Detailed equipment schedules have been prepared by Curragh to accompany the 11-year projection. Productivity forecasting is based on past equipment performances and the projection is achievable provided that the equipment is mechanically capable of sustained operation.

A high rate of excavation is scheduled from open pits for the next six years. The rates are: 27,600,000 tonnes in 1990, rising to 29,000,000 tonnes per year in 1993 and 1995. Curragh excavated over 29,000,000 tonnes during 1987 with the present equipment, from a deep pit; the equipment has the capacity, but must be in good condition to keep up sustained production. The major items of mine production equipment are discussed below.

Shovels

Shovels often are used for 20 years or more. One of the four existing shovels will be replaced by a good, used machine in 1990. By 1996, when requirements decline, the other three will have approximately 15 years of service. The shovels should be capable of meeting the scheduled productivity and production if mechanical and electrical maintenance is kept up.

Drills

Productivities assigned to drills exceed the scheduled requirements by about 25 percent. Drills of this type often are kept in service for 20 years or more, and the two drills in service should be capable of achieving the 11-year projection.

Trucks

Truck productivities were calculated by Curragh to account for effective utilization, and to reflect the variations of haul distances as the pits are deepened and the waste dump extended. The number of trucks and the truck operating hours were not recalculated; however, spot checks were made using a fixed number of truck trips per shift, and no great differences were noted.

It is recognized that the older (105-tonne capacity) trucks are not as effective as the larger trucks now available. A fleet of used 154-tonne capacity trucks is forecast to be purchased over the period 1990 to 1992. These trucks plus the existing fleet of fairly new 154-tonne trucks will be able to achieve the 11-year forecast.

Equipment Allocations

The equipment schedules indicate to which operation each unit is assigned, on a quarterly basis. There is no apparent conflict with the allocations for shovels, drills and trucks.

10.0 ORE PROCESSING, FARO DIVISION

10.0 ORE PROCESSING, FARO DIVISION

10.1 FLOWSHEET AND PRODUCTION RATE

The Faro lead/zinc concentrator commenced production in January, 1970 at a capacity of 5000 tonnes of ore per day. In 1974, capacity was increased to 6000 tonnes per day.

Major mill modifications were carried out in 1980-1981 in order to increase throughput to 9100 tonnes of ore per day. Shortly after completion of the 1980-1981 modifications, the concentrator was shut down by Cyprus Anvil due to depressed metal prices.

Curragh commenced rehabilitation of the concentrator late in 1985 - some four years after it had ceased operations. Milling was started again on June 2, 1986 at a designed throughput rate of 11,400 tonnes per day. Throughput has been increased since startup to 12,500 tonnes per day.

Ore from the Faro open pit is crushed in three stages in order to produce fine ore for the grinding circuits.

Wet grinding is also carried out in three stages in order to produce feed for the lead rougher flotation circuit.

Lead rougher concentrate is reground and cleaned in three stages in order to produce lead concentrate assaying 58-60 percent lead (Pb) and 400-550 g/t silver (Ag).

Tailings from the lead rougher flotation circuit are pumped to the zinc rougher flotation cells. Zinc rougher concentrate is also reground and cleaned in three stages to produce zinc concentrate assaying 49-50 percent zinc (Zn).

Tailings from the zinc rougher and zinc first cleaner flotation cells flow by gravity to the tailings pond.

Lead and zinc concentrates are thickened, filtered and dried in parallel dewatering circuits. Dried concentrates from the rotary dryers are stored in a storage shed, reclaimed by front-end loader and trucked to Skagway, Alaska.

A composite flowsheet, Figure 10-1, is shown on page 10-3.

Concentrator throughput for 1987, 1988 and 1989 (to September 30) is shown below.

Year	Concentrator Throughput	
	Tonnes	Average Tonnes/Day
1987	4,539,000	12,437
1988*	4,126,000	11,273
1989 (Jan.1 - Sep.30)	3,289,000	12,048

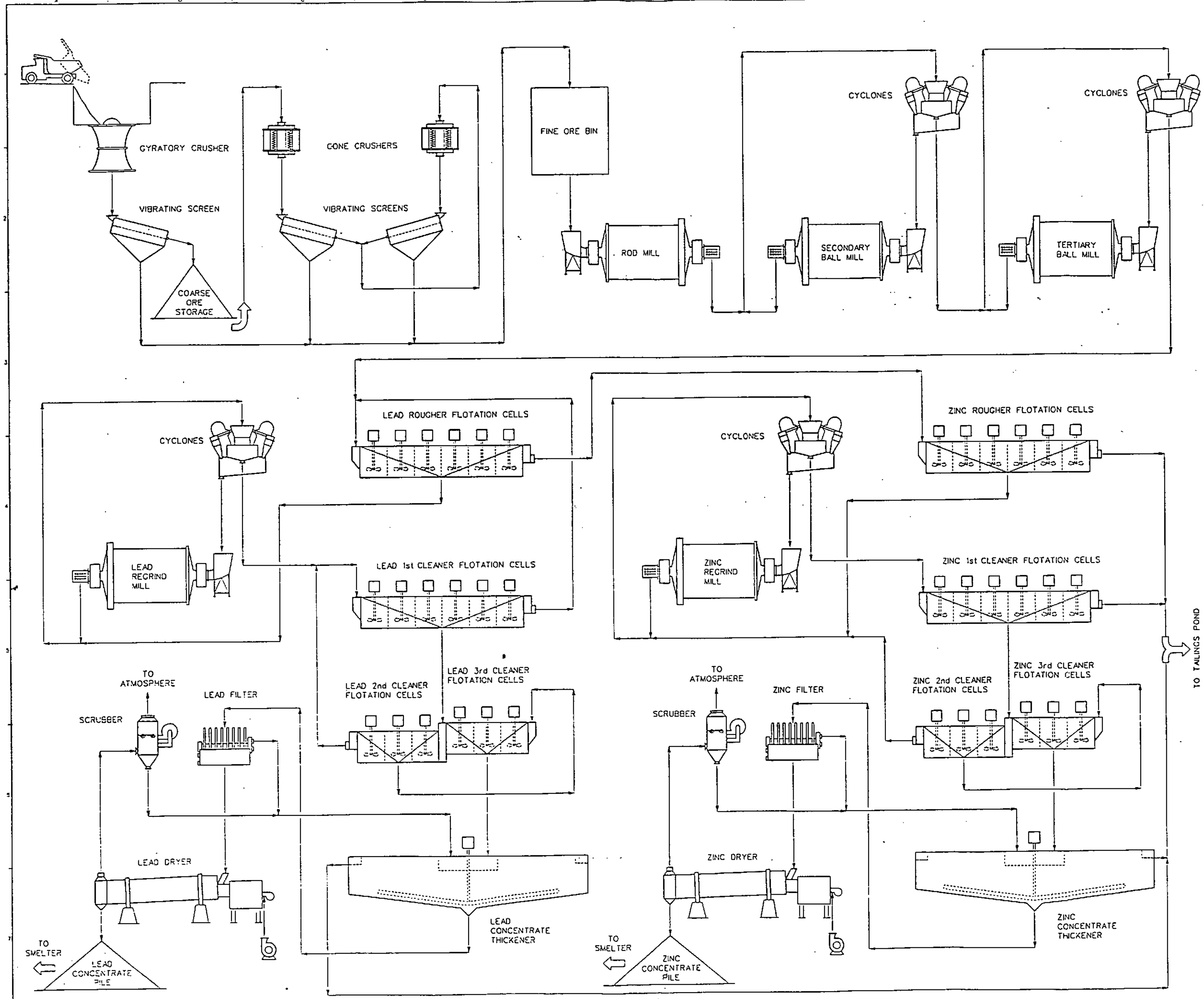
*Concentrator operated at reduced capacity by staff June 10 - July 7 during a strike.

The best twelve consecutive month throughput attained during the period April 1, 1987 to March 31, 1988 is shown below.

Concentrator Throughput	
Tonnes	Average Tonnes/Day
4,599,046	12,566

From August 1, 1989 to September 30, 1989 the concentrator throughput has averaged 12,547 tonnes per day (615 tonnes per hour at 85% operating time).

The grinding mills at Faro have the following nameplate and connected power.



A	RELEASED FOR STATUS REPORT		

REVISIONS

REFERENCE DRAWINGS

CLIENT: CURRAGH RESOURCES INC.
LOCATION: FARO YUKON

GENERAL LAYOUT

KILBORN

TITLE

FARO CONCENTRATOR
GENERAL
COMPOSITE FLOWSHEET

FIG. 10-1 A

	Nameplate HP	Service Factor	Connected HP
Rod Mill #1	450	1.15	517
Rod Mill #2	450	1.15	517
Rod Mill #3	450	1.15	517
Rod Mill #4	1,500	1.15	1,725
Ball Mill #1	450	1.15	518
Ball Mill #2	450	1.15	518
Ball Mill #3	450	1.15	518
Ball Mill #4	2,500	1.15	2,875
Ball Mill #5	2,500	1.15	2,875
Ball Mill #6	2,500	1.15	2,875
Total	11,700		13,455

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 below:

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

Concentrate production for 1987, 1988 and 1989 (to September 30) is shown below.

Year	Lead Concentrate Tonnes	Zinc Concentrate Tonnes	Total Concentrate Tonnes
1987	185,000	348,000	533,000
1988*	202,000	312,000	514,000
1989 (Jan.1 - Sep.30)	125,000	241,000	366,000

*Concentrator operated at reduced capacity by staff June 10 - July 7 during a strike.

The best twelve consecutive month concentrate production attained during the period April 1, 1987 to March 31, 1988 is shown below.

Lead Concentrate Tonnes	Zinc Concentrate Tonnes	Total Concentrate Tonnes
207,811	351,613	559,424

Concentrator throughput forecast by Curragh for 1990-2000 by orebody is shown below.

	Faro (Tonnes)	Vangorda (Tonnes)	Grum (Tonnes)	Dy (Tonnes)	Total (Tonnes)
1990	4,730,000	--	--	--	4,730,000
1991	3,097,000	1,602,000	--	--	4,699,000
1992	200,000	3,932,000	50,000	500,000	4,682,000
1993	450,000	1,000,000	2,010,000	1,000,000	4,460,000
1994	--	--	3,307,000	1,000,000	4,307,000
1995	--	400,000	2,950,000	1,000,000	4,350,000
1996	825,000		2,566,000	1,000,000	4,391,000
1997	418,000		2,930,000	1,000,000	4,348,000
1998	330,000		3,000,000	1,000,000	4,330,000
1999	--		3,300,000	1,000,000	4,300,000
2000	--		3,302,000	1,000,000	4,302,000

Concentrate production forecast by Curragh for 1990-2000 is shown below.

	Lead Concentrate (Tonnes)	Zinc Concentrate (Tonnes)	Total (Tonnes)
1990	200,835	366,485	567,320
1991	201,002	387,062	588,064
1992	244,778	349,324	594,102
1993	219,058	374,966	594,102
1994	219,388	375,799	595,187
1995	206,620	354,426	561,046
1996	213,639	369,154	582,793
1997	217,222	378,728	595,950
1998	223,820	373,958	597,779
1999	208,011	363,221	571,231
2000	178,042	299,319	477,361

The 11-year production forecast by Curragh shows increases in concentrator throughput and concentrate production over those which have been achieved during the best 12 consecutive months (April 1, 1987 to March 31, 1988).

In order to attain a concentrator throughput of 4,730,000 tonnes in 1990 grinding circuit operating time will have to be increased from the present 85% to 90%. In a letter dated September 19, 1989 from Kilborn to Curragh the following items were deemed essential in order for the concentrator to attain an annual throughput of 4,800,000 tonnes:

		CAPITAL COST
1.	- Bring mill maintenance manpower up to budget of 61 ASAP - With only 44 men on site at present, proper maintenance cannot be achieved	--
2.	- Speed up belt conveyor #13 to 580 fpm (1200 t/h) - Install 250 HP motor - Replace discharge chute	$260,000

3.	- Speed up belt conveyor #12 to 575 fpm (1200 t/h) - Install 30 HP motor - Replace discharge chute - Refurbish belt scale	$70,000
4.	- Speed up belt conveyor #11 to 606 fpm (1200 t/h) - Replace discharge chute - Install slow start on drive	$60,000
5.	- Speed up belt conveyors #10A and 10B to 220 fpm (440-880 t/h ea) - Replace discharge chutes	$30,000
6.	- Speed up belt conveyor #7 to 275 fpm (1000 t/h) - Install 20 HP motor - Replace discharge chute	$60,000
7.	- Speed up belt conveyor #6 to 452 fpm (1000 t/h) - Install 125 HP motor - Replace discharge chute - Refurbish belt scale	$130,000
8.	- Speed up belt conveyors #5A and 5B to 378 fpm (1000 t/h ea) - Install 25 HP motors - Replace discharge chutes	$50,000
9.	- Add remote control for vibrating feeders located under coarse ore storage shed	$50,000
10.	- Speed up belt conveyor #4 to 350 fpm (1600 t/h) - Install 40 HP motor	$30,000
11.	- Speed up belt conveyor #3 to 535 fpm (1600 t/h) - Install 250 HP motor - Replace discharge chute	$200,000

12.	- Speed up belt conveyor #2 to 350 fpm (1600 t/h)	$50,000
	- Install 30 HP motor	
	- Replace feed chutes	
	- Replace discharge chute	
	- Install heavy duty impact idlers; minimum spacing	
13.	- Decrease slope of primary screens; revise chutes (increase capacity to 800 t/h ea)	$115,000
	- Install dust covers	
14.	- Speed up belt conveyor #1 to 520 fpm (1600 t/h)	$220,000
	- Install 250 HP motor	
	- Replace feed chute	
	- Replace discharge chute	
	- Install skirting at headpulley	
	- Refurbish belt scale	
15.	- Speed up apron feeder (1600 t/h)	$20,000
16.	- Improve dust collection	$1,700,000
	- Coarse Ore storage	
	- Secondary Crushing	
	- Top of Fine Ore Bins	
	- Primary Crushing	Complete by Dec.'89
	- #1/6 Conveyor Gallery	Complete by Dec.'89
	- Transfer House	Complete by Dec.'89
17.	- Remove build-up of oxidized ore from 5 fine ore bins in order to maximize live load capacity	$50,000
18.	- Speed up rod mill #3	$50,000
19.	- Slowdown dryer feed belt conveyors to match discharge rate from filters	$20,000

20.	- Speed up zinc dryer discharge belt conveyor	$5,000
21.	- Overhaul vacuum pumps 1-5	$50,000
22.	- Overhaul disc filters 1-5 - Replace variable speed drives	$60,000
23.	- Overhaul rotary dryers 1-5 - Install better seals at feed and discharge ends - Install one piece feed chutes to minimize tramp air intake	$200,000
24.	- Install sumps and sump pumps at each pumpbox in the grinding circuit	$350,000
25.	- Modify bull gear guards on all Allis-Chalmers mills and collect grease in drums	$20,000
26.	- Install retaining walls, sumps and sump pumps in the flotation area	$160,000
27.	- Pump scrubber effluent from secondary crushing plant into grinding circuit	$50,000
28.	- Install retaining walls around tailings trench in thickener area to prevent concentrate from entering - Fill in trench from grinding circuit	$150,000
29.	- Install retaining walls around thickeners and clarifiers; install sumps and pumps	$280,000
TOTAL CAPITAL COST		$4,490,000

Curragh's revised capital estimate for 1990, a copy of which was received by Kilborn on October 19, 1989, lists the following projects and expenditures.

		Capital Cost
1.	Primary crusher dump hopper and feeder	$2,200,000
2.	Secondary crushing dust collection	$1,000,000
3.	#13 conveyor upgrade	$170,000
4.	#12 conveyor upgrade	$70,000
5.	#11 conveyor upgrade	$100,000
6.	Primary crusher foundation	$250,000
7.	Replace two coarse ore bin feeders and install remote controls	$150,000
8.	Replace slot feeders #1 - 6 and #11	$460,000
9.	Maximize fine ore bin #5 capacity	$50,000
10.	Upgrade heating plant pumps	$40,000
11.	Upgrade dewatering circuit	$500,000
12.	Flotation spillage handling	$160,000
13.	Motor control center cooling	$100,000
14.	Upgrade switch gear	$100,000
15.	Spare motor for #4 rod mill	$250,000
16.	Process control	$1,000,000
17.	Upgrade sampling	$300,000
18.	Install lead circuit flotation column	$2,000,000
19.	Replace OK flotation cell froth launders	$150,000
20.	Upgrade mill lighting	$60,000
21.	Purchase inching device for grinding mills	$90,000
22.	Upgrade assay lab and equipment	$100,000
23.	Upgrade bucking room and metallurgical lab	$70,000
24.	Replace 950 loader	$225,000
25.	Purchase new forklift	$55,000
26.	Purchase new Bobcat	$45,000
27.	Carry out recycle water study and design	$100,000
28.	Revamp crushing plant elevator	$85,000
29.	Upgrade gland water system	$50,000
30.	Upgrade flotation circuit pumps	$100,000
TOTAL CAPITAL COST		$10,030,000

On completion of the items listed above, the concentrator capacity should be very close to the planned production levels. However, Kilborn recommends that the 1990 capital budget be adjusted to include the $270,000 estimated cost of upgrading belt conveyors 5A, 5B, 6, 7, 10A and 10B (items 5 to 8 on page 10-7) in order to ensure maximum throughput of the secondary crushing plant. This will improve grinding circuit operating time to 90 percent and concentrator throughput to 4,800,000 tonnes of ore per year.

In order to attain lead concentrate production of 244,778 tonnes in 1992 and 387,062 tonnes of zinc concentrate in 1991 the installation of a pressure filter will be essential. Filtration tests on the finer concentrates expected from the Vangorda and Grum ores should be carried out as soon as possible in order to finalize sizing of the pressure filter. Capital cost for a new pressure filter has been estimated to be $2,600,000.

Mr. R. L. Coleman, an independent mineral processing consultant, was retained by Curragh to evaluate the Faro concentrator. His report, "An Assessment of Curragh Resources Inc. Concentrator Operations for Long Term Planning", is included in Appendix B of this report.

Mr. Coleman states in his report that the Faro concentrator can be modified in order to attain an annual throughput rate of 5,000,000 tonnes of ore.

10.2 METALLURGY

Actual mill feed grade and metallurgical results for 1987 to September 30, 1989 and those forecast by Curragh for 1990-2000, are listed below:

Year	Mill Feed % Pb	Mill Feed % Zn	Lead Concentrate % Pb	Zinc Concentrate % Zn	Recoveries-% Pb	Recoveries-% Zn
1987	3.31	4.93	60.32	49.44	74.00	76.84
1988	3.62	4.87	58.22	48.98	78.83	76.0
1989	2.91	4.69	58.96	49.77	71.11	78.05
1990	3.2	5.1	60.5	52.1	79.2	78.7
1991	3.2	5.3	59.5	52.3	80.4	80.6
1992	3.8	5.0	59.7	54.5	82.7	80.9
1993	3.6	5.5	59.7	53.7	80.7	81.7
1994	3.8	5.7	59.6	54.4	80.3	82.7
1995	3.5	5.4	59.4	54.3	79.9	82.2
1996	3.6	5.5	59.7	53.8	79.8	81.9
1997	3.7	5.8	59.6	54.3	80.1	82.1
1998	3.8	5.7	59.6	54.3	80.6	82.4
1999	3.6	5.6	59.6	54.5	80.1	82.8
2000	3.1	4.6	59.7	54.1	78.8	81.3

From testwork carried out to date on Vangorda and Grum ores, Mr. Coleman states in his assessment report that the following concentrate grades and recoveries can be expected:

Orebody	Concentrate Grades: Lead Concentrate % Pb	Concentrate Grades: Zinc Concentrate % Zn	Recoveries - % Pb	Recoveries - % Zn
Vangorda	62.0	54	84	82
Grum	61.0	52	82	82

In order to treat the Vangorda/Grum ores in the Faro concentrator the following changes will have to be made:

- Reroute lead rougher concentrate to the regrind mill.
- Reroute lead scavenger concentrate to the regrind mill.
- Reroute lead regrind cyclone overflow to the 1st cleaners.
- Reroute lead first cleaner tails to the #1 zinc conditioner.
- Install a high intensity conditioner in the zinc cleaner circuit.

- Use two regrind mills for lead and one regrind mill for zinc.
- Install a 317/3418 collector system.
- Install a thiourea depressant system.
- Install a sodium silicate system.

Future testwork will have to be undertaken in order to project metallurgical results when milling Faro/Vangorda, Faro/Vangorda/Grum/Dy, Grum/Dy, Vangorda/Grum/Dy, Faro/Grum/Dy and Grum/Dy ores.

Mixing of wet/warm underground ores with cold open pit ores during the winter months is a practice not recommended by Kilborn. These ores should be processed separately during the cold months in order to avoid freezing problems.

Stockpiling of the massive sulphide ores for extended periods of time will result in oxidation. Concentrate grades and recoveries will be adversely affected by oxidized ores. All efforts should be made to minimize stockpiling of high grade ore.

10.3 TAILINGS DISPOSAL

Concentrator tailings are presently being disposed of in an area developed by Cyprus Anvil. Capital money is budgeted in each of the years 1989, 1990 and 1991 for raising the dam. Curragh has estimated that the practical limit of disposal in this area will be reached sometime between July, 1993 and October, 1994.

Use of the mined-out Faro pit for tailings disposal is planned. Capital money is budgeted in 1993 for 'In-pit Tailings Disposal'; this is a full year after the open pit is scheduled to be mined out, and is the last year the underground mine is working from an entrance in the pit wall. Some coordination may be required before 1993, to ensure that the mining and tailings disposal operations do not interfere with each other.

The total capacity for tailings disposal at the two locations mentioned, is estimated by Curragh to be sufficient for 15 to 19 years' production.

Capital money is budgeted in 1994 to prepare the present tailings area for abandonment. There is no mention of abandonment plans for the tailings deposited in the Faro pit.

11.0 CONCENTRATE HANDLING, FARO DIVISION

11.0 CONCENTRATE HANDLING, FARO DIVISION

11.1 SUMMARY

Curragh has developed an integrated shipping system for concentrates leaving the mill site for world markets and for the backhaul of supplies necessary for the operation.

Overland transport is by B-train trucking to tidewater at Skagway, Alaska. Concentrates are loaded through a bulk terminal at Skagway and shipped to smelters in the Orient and western Europe in 30,000 to 40,000 tonne bulk freighters.

11.2 HIGHWAY TRANSPORTATION

Through agreements with the Yukon and Alaska governments, concentrates are trucked under contract to the port of Skagway, a distance of 550 kilometres by all-weather road. The concentrate trucking system was custom designed for the Faro-Skagway haul. Each B-train comprises a specially geared tractor plus two semi-trailers. Each semi-trailer carries two 12.5 tonne capacity concentrate pots for a total payload of 50 tonnes. A fleet of 40 B-train units is operated by a contractor 24 hours a day, seven days a week. The contractor owns the tractors. Curragh owns the trailers through a lease arrangement.

The truck route is from the mine site through the community of Carmacks to Whitehorse where drivers are changed. From Whitehorse, the route followed is the Klondike Highway through the community of Carcross to Skagway, Alaska. The round trip takes 24 hours.

Supplies are backhauled from Skagway to the mine site on the flatbed trailers of the B-trains. To permit the use of standard freight-shipping containers, the empty concentrate containers are repositioned on the trailers.

The B-train fleet has the capacity to transport in excess of 600,000 dry tonnes of concentrates per year, and has up to 10 times the required backhaul freight capacity. Backhauled commodities include lime, soda ash and grinding media, and could include fuel oil.

11.3 PORT FACILITIES

The concentrate storage and ship loading facilities at Skagway are leased by Curragh from White Pass and Yukon Route. These facilities have storage capacity for 100,000 tonnes and can handle well in excess of 600,000 tonnes of concentrates annually. The ship loading system operates at 750 tonnes per hour.

Concentrates are shipped from Skagway to the Orient or Europe in 30,000 to 40,000 tonne capacity bulk cargo ships.

12.0 ADMINISTRATION AND SERVICES, FARO DIVISION

12.0 ADMINISTRATION AND SERVICES, FARO DIVISION

12.1 SUMMARY

The present administration and service facilities are adequate for the Faro operations. Some semi-portable facilities are in place at the Grum/Vangorda site to service the current work; these facilities are being upgraded, but it is not intended to duplicate any service which can be provided from the existing Faro facilities. Underground mining of the Dy deposit will require construction of plant for hoisting, dry, warehouse, and office.

12.2 ADMINISTRATION

Curragh functions out of three locations. There is an Executive Office in Toronto, Ontario. An office, located in Whitehorse, is responsible for concentrate shipping, government liaison, environmental planning, and regional exploration. Operations at Faro are managed from offices located at the Faro plant site.

Operations at the Faro site are under the General Manager. Reporting to the General Manager are the Mining Manager, Processing Manager, Director of Accounting Services, Director of Materials, and Director of Human Resources. The Mining Manager and Processing Manager are responsible for all operation, maintenance and associated technical support for their particular areas.

12.3 HOUSING

The established town of Faro provides adequate housing and amenities for present and future planned open pit mining operations. Curragh does not provide housing for its employees. More than 400 houses were acquired with the assets of the previous mine owner, but these were sold to the Yukon territorial government who, in turn, made an arrangement with an independent real estate company. The real estate company offers the houses to Faro residents under a variety of rental and purchase options.

When the Dy deposit is developed for underground mining, some additional housing will be required. Curragh plans to use a mining contractor, therefore, the need may be for single-type accommodation which is more easily available than family-type accommodation.

12.4 ELECTRICAL POWER

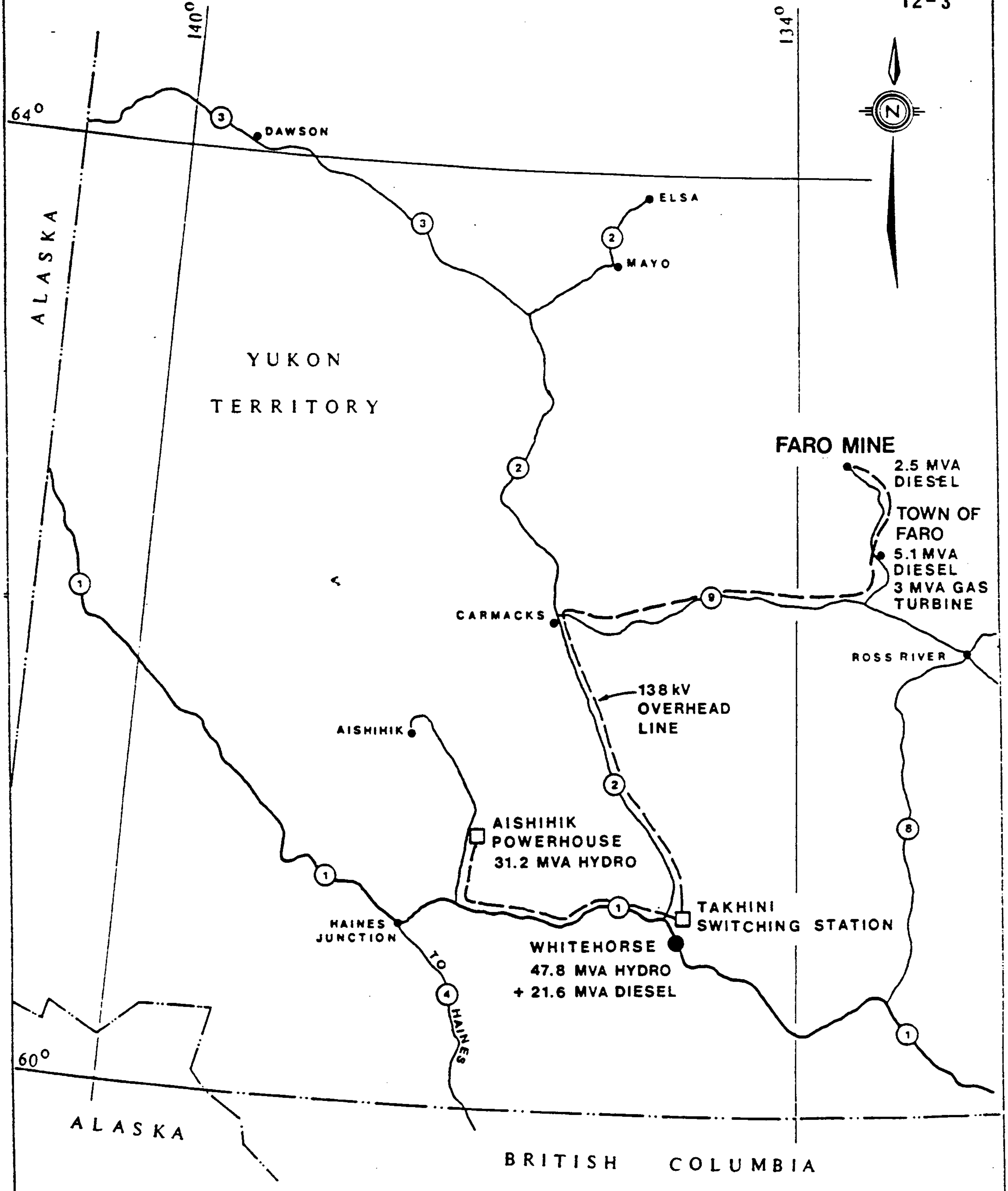
The Yukon Electric Company (YEC) electrical power generating stations and distribution grid provide power to the Faro minesite. See Figure 12-1.

The YEC generates hydro electric power at Aishihik and Whitehorse. Diesel generation capacity is located at Whitehorse and Faro. YEC hydro generation capacity is 84 MVA and diesel generation capacity at Faro is 5 MVA.

In addition to YEC capacity, a 2.5 MVA diesel emergency generator is owned by Curragh and operated as required.

The general decline in the economic growth of heavy power users in the Yukon has resulted in an excess of hydro electric power being available. The Faro operations energy requirements are serviced by hydro electric generators.

The current average total mine site energy consumption is 14,000,000 kilowatt hours per month; the power costs are based on a peak of 22.1 MVA. It is anticipated that no increase in the average consumption rate will result from the mining of Vangorda Plateau ores.



TITLE: YEC YUKON POWER SYSTEM	SECTION:	
KILBORN	AREA NO:	REV. NO:
CLIENT: CURRAGH RESOURCES — PROJECT NO: 3680-15	DRAWING NO:	
APPROVED: — DATE: MAY-89	FIGURE - 12-1	A

12.5 WATER SUPPLY

Since start-up in mid-1986, the Faro mine has drawn all of its fresh water supply for the mill from the water storage reservoir located on the south fork of Rose Creek approximately 3 kilometres east of the concentrator and tailings area. Studies were carried out, prior to and after start-up, on the adequacy of the water supply reservoir for the increased concentrator throughput, and hence greater water demand. There is a surplus of surface runoff water during the summer months; however there was a perceived shortfall for an assured water supply in the late winter to early spring period (March-April) following a dry summer season. Several alternatives were examined to provide a means for supplementing the water supply namely:

(a) Raising the present reservoir by some 6.5 metres to increase the winter storage capacity; and
(b) Installing ground water wells in the sand and gravel aquifer near the junction of the North and South Forks of Rose Creek. The wells would only be used in the late winter period prior to spring runoff.

The option selected for economic and technical reasons, following extensive ground water hydrological investigations, was to install four large diameter production wells. In January 1987, the wells were commissioned and provided approximately 2400 U.S. gallons per minute on a trial basis. The wells were operated from December 1987 until May 1988, and were started again in mid-December 1988.

There were some geotechnical concerns about the present dam when water is at the highest levels. During 1989, a berm was placed along the downstream toe of the dam to alleviate these concerns.

Water supply for the town of Faro is from wells located adjacent to the Pelly River. There is more than ample guaranteed water for present and forecast demand.

12.6 MAINTENANCE FACILITIES

The equipment maintenance facilities at the Faro site comprise: equipment repair bays, shop facilities, parts warehouse, and offices in one heated building covering approximately 5 acres. This facility is well equipped to service and maintain the pit production and service equipment fleets. Light vehicles are maintained on site, under contract, by a maintenance service organization. Major rebuilds of heavy equipment engines and components are carried out offsite by the suppliers or their agents. A tire supply company provides tires and tire servicing at a cost based on usage.

Curragh plans certain support facilities for efficiency at the Grum/Vangorda operations. An equipment lubrication area, a day-to-day supplies storage building, a changehouse and an office have been built. As production at the Faro pit drops off, Curragh will be able to move some of the equipment servicing arrangements to Grum/Vangorda - for instance, the contracted tire servicing and the field support for shovel repair.

12.7 EMPLOYEE RELATIONS

The hourly-paid employees are represented by the United Steel Workers of America, Local 1051; the Collective Agreement, effective until October 31, 1991, covers production and maintenance workers only.

The union was re-certified on December 8, 1987 and contract negotiations began in January 1988. There was a short strike (June 10, 1988 to July 7, 1988) and Curragh's first collective agreement came into effect after the strike.

Major items in the contract are as follows:

Hours of Work

For most workers	- 12 hours per day Four days on, four days off Average 42-hour week Two hours paid as overtime
For a few workers	- Eight hours per day Five days on, two days off

Wage Rates

Approximately the same as Afton Mines in southern British Columbia, $1.00 per hour less than Quintette Coal in central British Columbia, and $2.50 to $3.00 per hour less than Cassiar Asbestos in northern British Columbia.

Other Benefits

- Group Life Insurance;
- Dental Care and Vision Care;
- Statutory Holidays;
- Vacation pay is 6 percent of annual earnings after one year;
- Shift differentials;
- Tool and clothing allowances ($100 - $150 per year).

Union Dues

0.928 percent of gross earnings; collected by the company from all eligible employees whether union members or not (Rand formula).

Grievances

From July to December 1988, there were 10 grievances, two of which were taken to arbitration. During 1989 to date, there were 72 grievances. One of these has been taken to arbitration and approximately 8 more have been referred.

Contracting Out

Curragh may contract out work of the type done by union members, by agreement with the union, on payment of $3,000 to the union for this privilege.

12.8 ENVIRONMENTAL

A Vice President - Exploration and Environment, located in Whitehorse, handles the liaison with the appropriate authorities. An Environmental Engineer, with assistants as required, monitors the areas of environmental concern at the Faro operation.

It is planned to use the Faro pit for tailings disposal after the present tailings dam reaches its capacity in 1993 or 1994. In preparation for the abandonment of the present tailings disposal area, the testing of covers is being carried out - this testing is a four-year project. The covers being tested are: till, shallow water, bog, and a composite cover.

Waste from the Vangorda pit is indicated to be acid-generating. Environmental concerns have been addressed in the planning stage, and a water treatment facility is being constructed.

The permitting process for mining operations to acquire a water use licence from the Yukon Territory Water Board has been initiated for Vangorda and Grum. Timely receipt of the licence is anticipated by Curragh.

The environmental clean-up group is available for, and would take action in potential emergency situations such as chemical spills.

13.0 OPERATING COSTS, FARO DIVISION

13.0 OPERATING COSTS, FARO DIVISION

13.1 SUMMARY

Curragh's 11-year operating cost forecast for the Faro Division has annual costs ranging from $122.2 million in 1990 to $154.0 million in 1994, in constant mid-1990 dollars, and covering the total cash costs of producing and delivering concentrates on board a freighter at the Skagway ocean terminal. In terms of production, the total operating cost per tonne of ore milled ranges from $25.84 in 1990 to $35.75 in 1994, while the total operating cost per tonne of concentrate produced ranges from $210.74 in 1991 to $273.01 in 1995. The variations arise from changing sources of ore, mixing of mining methods, and varying metal grades.

The operating cost forecast is largely based on Curragh's operating experience over the last three years, with the exception of underground mining costs. The costs fall into two broad classes: production costs, built mainly from unit costs (per tonne or per hour), and general and administration (G & A) costs which are generally fixed for a given staff requirement.

Curragh uses a standard list of 24 expense types to collect (and to forecast) costs for the various departments.

13.2 MINING

13.2.1 Surface Mining Costs

Curragh's forecast of surface mining costs covers ore production of 9,740,000 tonnes from the Faro pit, 6,935,000 tonnes from the Vangorda pit, 23,982,000 tonnes from the Grum pit, the non-capital waste stripping, and all other associated mine costs.

The surface mining budget is based on recent historical costs. The 11-year forecast shows the costs of excavation, before deductions for capitalized waste stripping, as follows:

	1990	1991	1992	1993	1994	1995	1996	1997	1998	1999
$	1.44	1.50	1.44	1.43	1.34	1.32	1.77	1.67	1.53	1.43

There is a small reduction due to economy of scale in the high production years of 1990 to 1995; then the unit costs increase as production drops off, tempered in the later years by in-pit disposal of waste at Grum. The "spike" in 1991 is caused by the inefficiencies of shutting down Faro production and opening up Vangorda.

The unit costs per tonne of the principle items are shown for the first 5 years:

	1990	91	92	93	94
Salaries	$0.12	.12	.11	.10	.11
Wages	$0.48	.51	.52	.50	.45
Explosives	$0.10	.08	.12	.15	.15
Fuels and Lubes	$0.16	.17	.14	.14	.12
Tires	$0.07	.07	.06	.06	.05
Maintenance Parts	$0.27	.30	.26	.25	.23
Power	$0.07	.07	.07	.07	.07

It is indicated that forecasting was done on the basis of unit costs, and there are no anomalies that require explanations. The unit costs used are taken from past performance at the Faro pit and are acceptable for the remaining excavations in the Faro pit and the excavation in the Vangorda and Grum pits.

13.2.2 Underground Mining Costs

Curragh's forecast of underground mine costs covers the Faro mine production of 1.18 million tonnes over the years 1990 to 1992, and the Dy mine production of 8.50 million tonnes from 1992 to 2000.

Faro Underground

The forecast Faro underground costs total $28.30 per tonne of ore mined, comprised of:

- $21.66 per tonne of ore for contract mining, including supply of equipment and diamond drilling;
- $127,000 per year for consultants;
- $478,800 per year for Curragh salary costs;
- $6,000 per year for office supplies and expenses;
- $0.418 per tonne for power;
- $5.037 per tonne for operating supplies; and
- $0.110 per tonne for maintenance parts.

This cost was re-estimated recently by Curragh on the basis of fully mechanized cross-pitch room and pillar mining with equipment, labour and supervision supplied by a contractor, and with operating supplies and engineering services provided by Curragh. As described in Section 6.3 of this report, Kilborn has reservations about the effectiveness of this method. However, it should be noted that, two years ago, Kilborn estimated the Faro underground cash operating cost at $23.50 per tonne for producing 500,000 tonnes per year by a more flexible form of room and pillar mining. Adding escalation would bring this figure to about $26.50 in mid-1990 dollars, without including capital (or depreciation) costs for mining equipment.

Dy Underground

The forecast Dy underground costs total $31.23 per tonne of ore mined, comprised of:

- $25.00 per tonne of ore for contract mining including supply of equipment and diamond drilling;
- $127,000 per year for consultants;
- $532,000 per year for Curragh salary costs;
- $6,000 per year for office supplies and expenses;

- $0.418 per tonne for power;
- $5.037 per tonne for operating supplies; and
- $0.110 per tonne for maintenance parts.

This cost is based on a contractor's proposal for mining 1.0 million tonnes of ore per year by a mixture of methods including highly mechanized room and pillar (pilot drift, pillar slash and bench), slusher stoping in steeper dipping areas, and longhole mining in the thickest sections, followed by conveying to the shaft and hoisting to the ore bin. An additional amount, averaging $1.10 per tonne, is estimated (under Faro Division, Surface Mining) for hauling the ore from Dy to the Faro mill.

These costs appear to be reasonable. The cost of pilot drifting (50 percent of total tonnage) is based on an extremely high rate of productivity, but contractor's crews generally deliver superior performance. Also, contingencies totalling approximately $3.00 per tonne have been built into the estimate. Curragh claims that $2.00 per tonne is assigned, over the eight full production years, to ongoing development and construction work to open up new mining blocks.

13.3 MILLING

Operating costs for the Faro concentrator over the 11-year period (1990-2000) have been forecasted by Curragh to range from $7.38 to $7.73 per tonne of ore milled, in constant 1990 dollars. A breakdown of the estimated 11-year average of $7.60 per tonne is presented below:

Expense Type	Cost $/Tonne of Ore Milled	% of Total
Salaries and Wages	2.00	26.3
Fuels and Lubes	0.33	4.4
Reagents	1.77	23.3
Grinding Media	1.17	15.4
Liners	0.27	3.6
Contractors	0.11	1.5
Consultants	0.03	0.5
Maintenance Parts	0.70	9.2
Power	1.03	13.4
Operating Supplies	0.19	2.4
TOTAL	7.60	100.0

All operating costs appear to be acceptable for the Faro concentrator.

13.4 CONCENTRATE HANDLING

The major items in Curragh's 11-year forecast of costs for concentrate handling are:

(a) Concentrate Shipment

This item includes the contracted costs for haulage of concentrate from Faro to Skagway and contracted cost to store and load concentrate at the Port of Skagway, but does not include the marine terminal lease.

(b) Lease and Rentals

This item includes leasing of office space in Whitehorse and the marine terminal at the Port of Skagway.

(c) Taxes and Insurance

This item includes part of a $1.00 per wet metric tonne royalty to the Yukon government, insurance and related costs for road transport and certain property taxes.

Items (a), (b) and (c) represent more than 95 percent of the budgeted costs for concentrate handling of $56.19 per tonne. These three items are governed by term agreements, and are fairly predictable. Item (a) was renegotiated in June, 1989, and the contract increases are included in the forecast of costs. Item (a) represents, by itself, over 80 percent of the cost of concentrate handling.

13.5 FREIGHT BACKHAUL

The costs of the freight backhaul system covers the movement of reagents, grinding media, and, potentially, petroleum products from Seattle, Washington to Faro. The freight, which is carried in 20-foot containers, is loaded in Seattle, barged to Skagway, and carried to Faro on empty concentrate haulage trucks.

Curragh has been using this backhaul system for more than three years. The quantities of materials are known and the freight rate is by agreement, therefore, the budgeted estimate can be accepted. The 11-year forecast varies only slightly from year to year. The average annual expenditure is $1,564,000.

13.6 GENERAL AND ADMINISTRATION - ALL

This classification of costs includes:

- Materials Management;
- Human Resources Management;
- Accounting Services.

(The three foregoing groups are sometimes taken together and called Faro G & A.)

- G & A - Whitehorse;
- G & A - Toronto.

The 11-year forecast for each of these classifications is described below:

Materials Management

The major items in the costs, of approximately $3,084,000 per year until 1996, are freight ($2,011,000) and salaries and wages ($927,300). After 1996, the cost for salaries and wages is reduced gradually, but the cost for freight remains unchanged. (The freight cost under Materials Management is exclusive of freight cost shown under 'Freight Backhaul'.)

The 11-year forecast for Materials Management costs is based on known quantities and it is considered that the budgeted estimate can be accepted. The reduction in salaries and wages in the later years is consistent with the shift of the operations to contracted underground mining.

Human Resources Management

Costs for Human Resources Management are listed at approximately $3,900,000 per year for the entire 11-year forecast. The largest single item (approximately $1,262,000 per year) is for administrative expenses. Other major items are salaries, contractors and employee-related expenses, each over $500,000 per year.

Employee-related costs include recruiting, training, donations, medicals, Town of Faro costs, and legal fees. These costs are repetitive and fairly predictable. The other costs in the Human Resources Management account are based on past records, therefore, the budgeted estimate can be accepted.

Accounting Services

This classification contains the costs for on-site accounting and management information systems. Eighty-five percent of the approximately $1,300,000 annual budget is to be spent on salaries and taxes and insurance.

The accounting staff increases from 13 people in 1990 to 14 for the period 1991 to 1996 then declines gradually to 10 in 2000. Since both major items are fairly predictable, the 11-year forecast for expenditures is considered to be acceptable.

General and Administration (G & A) - Whitehorse

The 11-year forecast for this classification contains the costs for administration of concentrate handling, regional geology, and environmental and governmental relations. Annual expenses predicted are the same for each year, approximately $1,288,000.

Four major items—salaries, consultants, leases and rentals, and employee-related expenses—make up almost 80 percent of the total costs. The costs are based on current expenses and the forecast is considered to be acceptable.

General and Administration (G & A) - Toronto

Curragh's long-range forecast for G & A - Toronto includes the costs for executive management, management at Faro, corporate accounting, product sales, insurance in Yukon, and general Toronto office expenses. The total annual cost of approximately $7,460,000 is predicted to be the same over the 11-year forecast.

Major items are salaries ($2,380,000) and consultants ($2,340,000). The other items are as expected in the operation of the head office of a mining company.

14.0 CAPITAL COSTS, FARO DIVISION

14.0 CAPITAL COSTS, FARO DIVISION

14.1 SUMMARY

Curragh's planned Faro Division capital expenditures over the 11-year period total $177,472,000. Table 14.1-1 summarizes the expenditures by years and reflects the heavy investments in developing new orebodies during the early years, 1990 to 1993, followed by lower expenditures for continuing equipment replacement and regional exploration.

The capital estimate appears to be of the right order of magnitude, but there are some omissions and discrepancies, described in the following subsections of this report, which will tend to increase the total.

14.2 MINE CAPITAL

This classification lists the sums of money requested to replace or rebuild mining equipment, to improve the equipment repair facilities, and to upgrade the mine engineering services. The largest part of the capital expenditures is allocated to mining equipment.

Surface mining requirements, for major production equipment, were reviewed in Section 9.7 of this report, and the scheduled capital costs are compatible with this review. Other large expenditures, not discussed in Section 9.7, are for bulldozers, rubber-tired bulldozers, and a recovery unit. The service life of a bulldozer (track type) usually is much less than 10 years, and the schedule of expenditures appears to replace the current complement of bulldozers. A new equipment recovery unit, capable of carrying 200 tonnes, remains to be purchased in 1989 - there will be a need for this item because of the distance between Grum/Vangorda and the present mine equipment repair facilities.

TABLE 14.1-1

CURRAGH RESOURCES INC.

11-YEAR CAPITAL COST SUMMARY

($ X 1,000)

Item	1990	1991	1992	1993	1994	1995	1996	1997	1998	1999	2000	Totals
Mine:												
Mobile Equip.	8,396	4,515	4,230	1,900	1,230	600	3,500	200	600	200	200	25,571
Environmental Studies	422	100	100	100	100	100	100	100	100	100	0	1,322
Geotechnical Eng.	113	0	0	0	0	0	0	0	0	0	0	113
Mill:												
Improvements	6,433	4,650	1,700	1,500	3,050	845	1,280	620	870	520	20	21,488
Tailings and Diversion Dam	0	0	1,725	500	2,000	0	0	0	0	0	0	4,225
Concentrate Handling:												
Road Transportation	1,220	100	100	100	100	100	100	100	100	100	0	2,120
Ship Loading	0	50	50	50	50	50	50	50	50	50	50	500
G & A	404	450	450	450	450	450	450	450	450	450	450	4,904
Grum - Exploration & Development	15,071	16,511	20,288	6,512	0	0	0	0	0	0	0	58,382
Vangorda - Exploration & Development	2,741	1,950	0	0	0	0	0	0	0	0	0	4,691
Vangorda Plateau:												
Services	2,450	400	0	0	0	0	0	0	0	0	0	2,850
Dy:												
Road & Sitework	0	630	165	0	0	0	0	0	0	0	0	795
Metallurgical Testwork	0	0	86	90	0	0	0	0	0	0	0	176
Surface Powerline	0	500	0	0	0	0	0	0	0	0	0	500
Shaft, Stations & Development	4,000	9,400	4,800	5,000	0	0	0	0	0	0	0	23,200
Exploration Drilling	900	0	400	400	0	0	0	0	0	0	0	1,700
Environmental	0	150	500	500	0	0	0	0	0	0	0	1,150
Faro Underground:												
Exploration	0	500	0	0	0	0	0	0	0	0	0	500
Deveopment	985	0	0	0	0	0	0	0	0	0	0	985
G & A:												
Environmental	0	100	100	100	100	100	100	100	100	100	0	900
Exploration, Anvil Dist.	1,000	800	800	800	800	800	1,000	700	700	1,500	1,500	10,400
Exploration, Elsewhere	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	11,000
TOTAL CAPITAL	45,135	41,806	36,494	19,002	8,880	4,045	7,580	3,320	3,970	4,020	3,220	177,472

In total, the capital expenditures are considered to be adequate for the tasks in the 11-year forecast. However, it is essential that the shovel and drill overhauls proceed as scheduled (or earlier) and that replacement trucks be purchased.

14.3 CONCENTRATOR CAPITAL

The Curragh 11-year capital cost forecast, as set out in Mine Plan S89: Alpha II, indicates expected expenditures of $25,713,000. Yearly capital costs are shown below:

Year	Capital Cost Expenditures $ Total/Year	Cost $/Tonne of Ore Milled
1990	6,433,000	1.36
1991	4,650,000	0.99
1992	3,425,000	0.73
1993	2,000,000	0.45
1994	5,050,000	1.17
1995	845,000	0.19
1996	1,280,000	0.29
1997	620,000	0.14
1998	870,000	0.20
1999	520,000	0.02
2000	20,000	0.01
TOTAL	25,713,000	0.53

Of the total capital expenditures, $4,225,000 is estimated for tailings diposal and abandonment projects over the period 1992 to 1994. Within the concentrator, the 1990 estimate has been revised to $10,030,000, an increase of $3,597,000. As set out in Section 10.1, Kilborn suggests that this figure be adjusted to include $270,000 for upgrading of certain belt conveyors. This would increase the 1990 budget to $10,300,000 and the 11-year total to $29,580,000., an increase of $3,867,000.

14.4 CONCENTRATE HANDLING CAPITAL

The concentrate handling system will be upgraded in 1990 ($1,220,000) including repairs and improvements to the loadout system, and improvements to the haulage equipment, and will be maintained by expenditures of $100,000 per year. For additions or replacements to the fleet of 40 B-Trains, Curragh intends to continue to lease trailers, pots and accessories, and to continue to contract supply and operation of the tractors.

14.5 GENERAL AND ADMINISTRATION CAPITAL (OFFICES)

An annual sum of money is allocated for upgrading and replacement of computers and office equipment. In 1990, the capital ($404,000) is for specific items. In the following years, the capital ($450,000 per year) is not identified specifically.

14.6 GRUM AND VANGORDA CAPITAL

14.6.1 Grum - Exploration and Development

Table 14.1-1, Capital Cost Summary, shows an estimated expenditure of $15,071,000 in this classification during 1990. The largest part ($13,981,000) is for Grum stripping. Quantity of stripping forecast is 12,363,000 tonnes, which equates to a cost of $1.13 per tonne. This stripping unit cost is compatible with the surface mining costs shown in Section 13.2.1 of this report. The other cost in 1990 is $1,090,000 for deep wells and pit dewatering equipment.

In the three subsequent years shown on Table 14.1-1, the forecast Grum capital expenditures are almost entirely for stripping. The unit costs of stripping are: $1.15 for 1991, $1.30 for 1992, and $1.39 for 1993. The increasing unit costs in these years can be attributed to increasing depths and surface haul distances.

It is considered that the capital cost forecast for 'Grum - Exploration and Development' is acceptable, provided that the quantities of excavation listed are acceptable. The total quantities of excavation were discussed in Section 9.4 of this report and are considered to be within acceptable limits. However, the ore available for initial production was not checked.

14.6.2 Vangorda - Exploration and Development

Table 14.1-1, Capital Cost Summary, shows an estimated expenditure of $2,741,000 in this classification during 1990. The sum of $2,569,000 is allocated for Vangorda stripping; for a quantity of 1,658,000 tonnes, this is equivalent to $1.55 per tonne. Other costs are $172,000 for a metallurgical test in 1990 and $1,950,000 for stripping 1,439,000 tonnes in 1991, a unit cost of $1.36 per tonne with crews being transferred from Faro pit.

It is considered that the capital forecast is acceptable, provided that the quantities of excavation listed are acceptable. This matter was discussed in Section 9.5 of this report and it was stated that the total quantities of excavation were within acceptable limits.

14.6.3 Vangorda Plateau

In preparation for mining operations on the Vangorda Plateau (site of the Grum, Vangorda and Dy deposits), services must be installed and environmental programs carried out. The capital expenditures in this classification are for items that are common to a large part of the deposits scheduled for mining. Many of these services will be completed under the 1989 capital program including the haul road, powerline, mine dry building, lube station, and telephone system.

The amount of $2,450,000 estimated for 1990 includes $1,400,000 for a water treatment plant, $650,000 for equipping and furnishing of the new mine dry building and warehouse, and an allowance of $400,000 for switchgear and miscellaneous equipment. With a further allowance of $400,000 in 1991, these monies appear adequate.

14.7 DY CAPITAL

The Curragh capital forecast of $27.52 million for exploration and development of the Dy deposit is largely based on an early estimate by a contractor, and which has since been revised upward by approximately $4.0 million.

Having reviewed the current plan and discussed it briefly with the contractor, Kilborn believes that the estimated capital is adequate to put the 'B2 Block' into production. Efficient production from the rest of the deposit will require development and construction of some 1800 metres of conveyorways, with associated dumps, passes and feeders, and deepening of the shaft by approximately 80 metres plus a new station, ore bin and loading pocket. Kilborn estimated that such a program would add about $16 million to post-production capital. However, Curragh has included this work in the operating cost forecast at the rate of $2.00 per tonne over the 8-year period of full production of 1.0 million tonnes per year.

14.8 FARO UNDERGROUND CAPITAL

The 1990 capital expenditure estimate of $985,000 is the complete sum for underground development of this mine which is scheduled to produce 1,178,000 tonnes over three years starting in 1990. This relatively small sum of money, and short development time, presumes that a contractor supplies all facilities and that a mining method is used which requires almost no advance development.

Kilborn has expressed its reservations about the concept for developing and mining this deposit (see Section 9.3 of this report), and the same reservations apply to the sum of money allotted to start this mine.

A total of $500,000 is budgeted for underground exploration. This sum is considered to be adequate.

14.9 GENERAL AND ADMINISTRATION - WHITEHORSE

Curragh's geological staff in Whitehorse carries out exploration not associated directly with the currently scheduled mining operations. Estimated capital expenditures are $1,000,000 in 1990 for the Anvil District, followed by amounts ranging from $800,000 to $1,500,000 over the years 1991 to 2000. Most of this will be spent exploring known targets. An annual budget of $1,000,000 is provided throughout the 11-year forecast for exploration outside the Anvil District.

The proposed capital to be spent on exploration does not contribute to Curragh's 11-year forecast, but is aimed at extending the operations beyond the year 2000.

An annual sum of $100,000 is shown in this classification for general environmental studies from 1991 to 1999.

APPENDIX "A"

ESTIMATE OF GEOLOGICAL ORE RESERVES
FOR DY DEPOSIT OF
CURRAGH RESOURCES INC.
FARO, YUKON

FEBRUARY, 1989

Prepared by:

P. C. COLTAS, P.Eng.
Consulting Geologist

[3680_15.RPT/2]

SUMMARY

An ore reserve estimate was carried out on the Dy deposit of Curragh Resources Inc. (Curragh). This deposit is one of a series of strataform, stratabound lead, zinc deposits located in the Anvil District, Faro, Yukon.

Ore Reserves

Ore reserve estimates are as follows:

	Tonnes	% Pb	% Zn	Ag g/t	Au g/t
Probable Reserves	14,920,525	5.44	7.02	85.7	0.93
Possible Reserves	5,194,300	5.57	6.07	81.0	0.87

Premises and Methods

Ore reserves in the Dy deposit have been calculated using the following premises and methods:

1. A minimum 3.5 metre mining width has been used, all intersections less than 3.5 metres were rejected.

2. Qualifying intersections are identified with a 9 percent combined lead plus zinc cut-off grade.

3. No assays less than this cut-off are used in defining an ore section unless they are enclosed by assays greater than these cut-offs with the average of the entire section greater than the cut-off.

4. Anomalous silver values (high) are taken into account such that intersections with less than cut-off grades combined lead-zinc but greater than cut-off grades in silver (i.e., 9 gms/MT per one percent combined lead plus zinc) are included in an ore intersection (one case).

*5. "Tonnages are calculated using a polygon 'area of influence' method wherein qualifying intersections are plotted in plan on all drill holes in a stratigraphic horizon. Straight lines are drawn through

adjacent intersections and perpendiculars dropped from each intersection to the bounding connecting lines. Intersections of these perpendiculars and connecting lines form unique polygons about each intersection"..."The area of all polygons is measured by planimeter. Final tonnages are calculated by multiplying this area by the drilled, approximate true thickness by the measured, average specific gravity for the entire intersection."

6. New sections were drawn to show all ore intersections and also holes with interesting low grade material. These sections were used to classify ore intersections into probable or possible reserves:

   (a) Probable Reserves: Where two or more holes on one section join two or more holes on an adjacent section or sections on the same stratigraphic horizon.

   (b) Possible Reserves: Where two or more holes are in a stratigraphic horizon, however, do not join a similar horizon on adjacent section or sections.

   (c) A number of ore grade intersections were rejected, because they were completely isolated from all other ore intersections. The holes in question are as follows: 77-1 (9+00 E), 77-5 (upper intersection) (12+00 E), 77-11 (18+00 E), 79-3 (13+50 E), and 80-5 (lower intersection) (15+00 E).

7. Calculation sheets accompany these reserves (in appendices). The grades-tonnes, etc., are all assigned to sections, **"A-2 Horizon and B-2 Horizon". Two plans accompany this report:

   (a) Probable reserves;
   (b) Possible reserves. All ore intersections (including low grade) are shown on these plans, and are coded for their classification.

8. The sections accompanying this report, and used as previously mentioned to classify ore intersections, show the location of all ore

intersections including low grade, and are coded to indicate their classification.

9. All available information, cross and long sections and assay logs for all holes were critically reviewed, including checking the sections for the exact location of the ore intersection (longitude, latitude and elevation). All assay logs were reviewed, and all ore intersections recalculated. Some previously included ore intersections were rejected (below cut-off grade), a number of other previously used intersections were shortened (material on fringe area below cut-off grade), and a number of previously used ore intersections were combined, still meeting the cut-off grade.

* Direct quote from B. V. Hall's (1981) ore reserve report.

** Taken from B. V. Hall's (1981) ore reserve report.

DY DEPOSIT RESERVE SUMMARY - 1989

(3.5 m Minimum Mining Width - 9% Pb+Zn Cut-off)

A-2 HORIZON

Section	Probable Tonnes	% Pb	% Zn	Ag g/T	Au g/T	Possible Tonnes	% Pb	% Zn	Ag g/T	Au g/T
10 + 50	603,275	5.02	5.33	71.6	0.67	-	-	-	-	-
12 + 00	2,676,425	6.54	5.04	87.8	1.04	919,300	5.88	6.08	94.9	1.11
13 + 50	954,650	3.82	6.63	68.3	1.03	-	-	-	-	-
15 + 00	3,898,100	5.80	6.71	92.4	1.09	1,564,500	7.87	5.61	101.6	0.89
16 + 50	1,226,525	5.38	7.16	81.2	0.96	1,084,900	4.14	5.32	59.0	0.94
TOTAL	9,358,975	5.70	6.19	85.8	1.03	3,568,700	6.23	5.64	86.9	0.96

B-2 HORIZON

Section	Probable Tonnes	% Pb	% Zn	Ag g/T	Au g/T	Possible Tonnes	% Pb	% Zn	Ag g/T	Au g/T
9 + 00	-	-	-	-	-	386,500	4.41	9.57	81.5	0.90
10 + 50	-	-	-	-	-	368,375	6.07	4.99	81.0	0.62
12 + 00	-	-	-	-	-	168,625	4.35	5.64	92.6	1.09
13 + 50	607,000	4.32	8.16	68.6	0.93	-	-	-	-	-
15 + 00	511,200	4.07	6.02	63.0	1.05	-	-	-	-	-
16 + 50	2,249,650	5.40	9.01	92.6	.71	512,750	2.75	6.70	44.6	0.47
18 + 00	1,539,925	4.54	7.84	82.7	0.74	-	-	-	-	-
19 + 50	653,775	6.30	9.66	100.5	0.70	189,350	3.38	7.66	57.8	0.39
TOTAL	5,561,550	5.03	8.40	85.5	0.77	1,625,600	4.14	7.00	68.1	0.66
GRAND TOTAL	14,920,525	5.45	7.02	85.7	0.93	5,194,300	5.57	6.07	81.0	0.87

I, P. C. Coltas, have critically reviewed all available information on the Dy Deposit of Curragh Resources Inc., Faro, Yukon. This information included the following:

(a) Ore Reserves, B. V. Hall, 1981;
(b) Ore Reserves, Rolling, 1982;
(c) Cross and long sections;
(d) Plans - ore reserve, diamond drill hole locations and topographic;
(e) All diamond drill assay logs.

The ore reserves reported are an accurate estimate of the mineral inventory of the Dy deposit.

P. C. Coltas, P.Eng.
Consulting Geologist

NOTE: Original signed by 'P.C. Coltas', on file at Kilborn's office.

APPENDIX "B"

AN ASSESSMENT
OF
CURRAGH RESOURCES INC.
CONCENTRATOR OPERATIONS
FOR
LONG TERM PLANNING

JANUARY 31, 1989

Prepared by:

R. L. COLEMAN, P.Eng.
Mineral Processing Consultant

[3680_15.RPT/21]

AN ASSESSMENT OF CURRAGH RESOURCES INC. CONCENTRATOR OPERATIONS FOR LONG TERM PLANNING

INTRODUCTION

R. L. Coleman has been asked to study and comment on the proposed FARO concentrator operating plan of Curragh Resources Inc. for the treatment of ores from the Faro, Vangorda and Grum deposits.

The process plant has been mineral processing the ores from the Faro open pit and thus has built up a tonnage and metallurgical history on which to base this assessment. A bulk scale test of Vangorda material is scheduled for late 1989 but the significant tonnages are to be handled starting in 1991. Some basic research was conducted on this deposit by he previous owners but the significant major research was performed by Lakefield Research in 1988. There have been no recent investigations on the Grum ores so that my recommendations will be based on the report of pilot plant testing in 1977 by the Noranda Milling Committee. Treatment of the Grum ores is scheduled to start in 1992 but significant tonnages will not be processed until 1994.

The remarks I now make will be based on the above comments and site visits and consulting at the site for the past 1-1/2 years. My background experience has been accumulated during the past 50 years of research, design, construction, operation, maintenance and supervision of precious metal, base metal and nonmetallic processing schemes. I will contribute no advice on mining plans.

CURRENT OPERATIONS

The Faro concentrator has been treating a complex lead-zinc ore since January 1970 with a short shutdown period due to low metal prices. The original designed capacity was 5,500 short tons per day with two expansions to 6,600 SDTPD and 10,000 SDTPD. It is currently operating at a nominal 650 metric tonnes per hour but averaging about 550 tonnes per hour on a monthly basis (exclusive of a labour strife period) in 1988. Peak monthly average was 12,657 MTPD or 422 MTPH and peak daily tonnage was 15,916 MTPD or 633 MTPH in 1988. (Limited number of figures noted.)

Curragh Resources inherited a concentrator which, in my opinion, had a number of poorly designed features and the previous owners ceased operations without due regard to appropriate shutdown procedures. Thus on restart there were a number of problems and considerable maintenance to attend to before an efficient operation could be claimed. All of the built-in or accumulated disadvantages have not yet been rectified.

LONG TERM PLAN

In the long term planning, Curragh Resources has established a milling rate of 5,000,000 metric tonnes per year (and probably a metals in concentrate output, the details of which I am not aware). The crushing plants should be able to supply this tonnage with better operating practice and maintenance.

In order to attain and maintain the targeted primary tonnage, the grinding circuits must average 600 MTPH which I expect is a reasonable figure with improved equipment, operating procedures, planned maintenance programs and process control. At the same time, due to the variable hardness in the deposits and because the Vangorda and Grum ores require finer grinds for efficient metallurgy, the tonnage target may have to be sacrificed on occasion in favour of the concentrate grade or metal output objectives. The grinding mills may have to be speeded up.

The flotation roughing capacity is adequate for the expected head analyses. However, the flowsheet in the cleaning and regrind circuits must be altered for the Vangorda and Grum ores. The research on these materials would suggest that the revised flowsheets recommended would improve the metallurgy of the Faro ores – particularly the concentrate grades. Studies are necessary to determine how much additional grind, classification, pumping and flotation capacity is required. Column flotation cells might well fit in here.

The Lakefield research on the Vangorda ores also shows that the suite of reagents currently used on the Faro ores is unsuitable for the new deposits. Thus reagent types quantities and points of addition will have to be changed. It is probable that the new reagents will be very beneficial in treating the Faro ores.

This Curragh operation does not recycle internal waters nor tailing dam effluents. It is my opinion that the use of internal water would reduce density problems and operating costs while the use of the tailing dam effluent will be required by law in the time span planned in your proposals. This may prove to be not entirely detrimental because experience at other properties has shown reduced reagent costs.

In order to produce higher grade zinc concentrates, particularly from Vangorda ores, conditioning for the circuit feed should be tested and high speed conditioning will be necessary in the cleaning part of the circuit.

Blending of ores from the three deposits has not been researched and may not be practical. However, if it is required for mining or other reasons, the research should be initiated immediately.

The recommended finer primary and secondary regrinds will produce finer concentrates and thus will increase the thickening, filtering and drying requirements. Since these areas are under capacity now, you will have to proceed with more or different filters and/or dryers.

To optimize the metallurgy and the efficiency of the entire milling complex, you should proceed with more instrumentation and process control hardware.

All the above mentioned changes will incur appreciable capital. I have been unable to find any reference to the amount of capital required for new milling equipment during the planning periods except for the 1992 one million dollars referred to by HMC in their exhibit 22A for tailings pumps. I hesitate to estimate the cost of the suggestions made at this time because I do not know the viewpoints of you and your staff on these matters, and I do not know what specific improvements have been made in the past 8-9 months.

METALLURGICAL ASSESSMENT

In order to comment on the specifics of your long range plan, I felt that it would be helpful to draw a brief table of the data available or referred to. The crushing and grinding capacities have already been discussed.

METALLURGICAL COMPARISONS

	FARO OPERATIONS			VANGORDA				GRUM			
	Record	Proposed		Curragh		Proposed		Curragh		Propose	Research
	1988	1989	1999	1992	6 YR	HMC	RLC	1992	6 YR	HMC	Noranda
ORE GRADE											
Pb%	3.62	3.2	3.0	3.4	3.0	-	-	5.8	3.0	-	4.0
Zn%	4.83	4.9	4.9	4.5	5.12	-	-	8.3	5.12	-	8.0
Au g/t	-	-	-	0.7	0.76	-	-	0.8	0.76	-	0.77
Ag g/t	52.0	35.0	35.0	48.0	49.0	-	-	92.0	49.0	-	60.0
LEAD CONC.											
Pb%	58.36	60.9	61.8	54.5	58.0	60.0	62.0	60.7	58.0	58.0	62.0
Zn%	7.7	-	-	-	-	-	8.0	-	-	-	8.0
Au g/t	-	-	-	4.5	6.0	3.9	8.0	2.6	6.0	5.0	4.7
Ag g/t	550.0	418.0	444.0	369.0	744.0	650.0	700.0	625.0	744.0	744.0	870.0
ZINC CONC.											
Pb%	2.46	-	-	-	-	-	1.0	-	-	-	2.0
Zn%	49.05	49.6	50.6	56.0	55.0	52.0	54.0	50.0	55.0	52.0	56.0
RECOV.											
Pb%	79.16	79.2	79.6	83.9	86.4	80.0	84.0	84.0	86.0	80.0	80.0
Zn%	75.48	78.0	78.7	81.1	93.0	78.0	92.0	82.0	93.0	80.0	84.0
Au%	-	-	-	-	35.0	50.0	62.0	-	35.0	32.0	33.0
Ag%	51.5	-	-	-	67.0	60.0	70.0	-	67.0	66.0	72.0

Based on the 1988 metallurgical results and the improvements that have been made or are planned for the concentrator, the concentrate grades and recoveries for the Faro deposit shown in the adjusted mill feed schedule for 1989, 1990 and 1991 are attainable. The concentrate grades will be improved from the 1988 results if the changes in lead circuit collector, flowsheet lead regrind and zinc circuit conditioning are implemented.

When processing Vangorda ores, the lead concentrate grades should be better than shown in your proposal if the collector, regrind, flowsheet and flotation capacity are modified. Lead recovery should be 84% while gold and silver recoveries should be about 50% and 70% respectively. With the addition of high speed conditioning in the zinc cleaners, zinc concentrate grades should improve to 54% and zinc recovery to 82%. These comments assume that the porous carbonaceous (perhaps oxidized) material will not be treated as ore until a scheme of treatment is developed. The non porous graphitic ore should yield acceptable metallurgy although not as good as the majority of the deposit.

Since there is no recent laboratory or pilot plant research to use for judging the proposed response of the Grum ores to treatment in the Curragh concentrator, it is difficult to predict success or lack of it. Thus, I suggest you use the results of the pilot plant test as reported on by the Noranda Milling Committee in December 1977 with some modifications. With the new technology, reagents and flowsheet as suggested for Vangorda, the lead grade from Grum ores should reach 61%, and the zinc grades 52%. Both lead and zinc recoveries should reach 82% while the precious metal recoveries should remain as shown in your proposal. (I would have some hope that the gold recovery would be better.)

The above remarks do not cover the effect of stockpiling ore for later treatment in the concentrator. If stockpiling is to be practiced as implied in your proposals, I would be seriously concerned about the effects of oxidation. Concentrate grades and metal recoveries will not be as good. Nor do these remarks apply should blending of the ores from the three pits be considered.

The concentrator operating costs will increase due to the labour costs, the finer grinds, the new types of reagents and their consumption and the probable recycle of tailing dam effluents. On the other hand, the returns will be enhanced by better concentrate grades, lower dewatering costs and more efficiency with process control.

SUMMARY

To summarize this assessment, I suggest that if:

(a) a new collector is used in the lead circuit;
(b) the flowsheet in the lead regrind circuit is changed;
(c) high speed conditioning is employed in the zinc circuit;
(d) the dewatering equipment is modified or changed;

the objectives, as outlined in your basic 6 year plan, can be attained and you might expect:

1. An annual milling rate of 5,000,000 metric tonnes.

2. Lead concentrate grades of 62%.

3. Lead recovery of 80 to 84% depending on deposit.

4. Gold recoveries from 40 to 60%.
5. Silver recoveries up to 70%.
6. Zinc concentrate grades of 52% plus.
7. Zinc recoveries up to 82%.

* * * * * *

NOTE: Original signed by 'Dick Coleman', on file at Curragh's office.