CURRAGH RESOURCES

005920

LONG RANGE PLAN FOR
FARO, VANGORDA AND GRUM DEPOSITS
FARO, YUKON

APRIL, 1987

<u>VOLUME II</u>

<u>TECHNICAL REVIEW</u>

Prepared by

KILBORN

CURRAGH RESOURCES

LONG RANGE PLAN FOR

FARO, VANGORDA AND GRUM DEPOSITS

FARO, YUKON

VOLUME II

TECHNICAL REVIEW

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1.0 INTRODUCTION

1.0 INTRODUCTION

1.1 SCOPE OF WORK

Mr. Clifford H. Frame, Chairman and Chief Executive Officer, Curragh Resources (Curragh) requested Kilborn Engineering (B.C.) Ltd. (Kilborn) to prepare a Technical Review of the currently operating Faro Open-Pit Mining Operation and Long-Term Production Plans to mine the Faro, Vangorda and Grum deposits.

The preferred plan considers the mining of all three deposits. The plan is based on a start date of April 1, 1987. Kilborn was requested to evaluate preliminary production plans and, guided by 1987 budgeted productivities and operating costs, develop project costs to the end of production in 1987 first-quarter funds, in conjunction with Curragh personnel.

The study is confined to the Faro, Grum and Vangorda deposits. Other deposits in the area are not considered.

Volume I, Summary, provides an overview of the production plan and cash costs for the study.

Volume II, Technical Review, (this volume) describes the ore deposits and reviews the parameters used in this long range plan for mining and milling the ore and for transporting the lead and zinc concentrates produced.

Volume III, Capital and Operating Cost Estimates, provides details of the cash costs associated with this plan.

1.2 SETTING

Curragh owns a large open-pit lead-zinc-silver mine near the community of Faro, approximately 200 air kilometres northeast of Whitehorse, Yukon Territory (see Figure 1.2-1).

TITLE: FARO AREA DEPOSITS LOCATION PLAN

SECTION:

KILBORN ENGINEERING (B.C.) LTD.

AREA NO:

REV. NO: A

CLIENT: CURRAGH RESOURCES FARO, YUKON

PROJECT NO: 3509-19

DRAWING NO: FIGURE 1.2-1

APPROVED:

DATE: MAR 3/87

The operation is amongst the major world producers of lead and zinc concentrates. The plant was initially constructed with a capacity of 5,000 tonnes per day, and in 1980-81 was expanded to its present capacity of 13,500 tonnes per day.

The mining and milling operations are located at an elevation of 1,200 to 1,300 metres above sea level in a rough, semi-arid subarctic region of the Yukon.

1.3 HISTORY

The initial mineral discovery in the Anvil Range was what is now known as the Vangorda deposit. The deposit was drill tested by Prospector Airways, a predecessor to Kerr Addison Mines.

The Faro deposit was discovered in 1964 while drill testing airborne anomalies. This deposit was developed and brought into production by the predecessor of Cyprus Anvil Mining Corporation (Cyprus Anvil) in 1969. Subsequently, additional deposits were discovered: Grum, 1973, by Kerr Addison Mines, and DY by Cyprus Anvil.

Following the commencement of operations in 1969, the Anvil operations assumed a position of prominence amongst the major world producers of lead and zinc concentrates. The initial plant capacity of 5,000 tonnes per day was increased in 1974 to 9,300 tonnes per day. Further major mill modifications were predicated upon maintaining the 9,300 tonnes per day milling rate at a finer grind. Considerable flexibility was incorporated in the design to facilitate anticipated increases in production levels in the future.

Cyprus Anvil curtailed production in June 1982 due to the depressed state of metal prices.

Open-pit waste stripping operations commenced in June 1983 in accordance with the terms of an agreement between Cyprus and the Federal Government which provided financial assistance for this program.

Approximately 9.7 million bank cubic yards of waste were removed during the period June 1983 to October 1984.

Unsuccessful labour negotiations resulted in a lock-out of all unionized employees, and all the facilities were mothballed by the late summer of 1985.

At this time Dome Petroleum publicly stated its wish to divest its interest in Cyprus Anvil.

Curragh acquired the assets of Cyprus Anvil in November 1985. The operation was rehabilitated and stripping operations recommenced in January 1986. The mill was reactivated and started ore processing in June 1986.

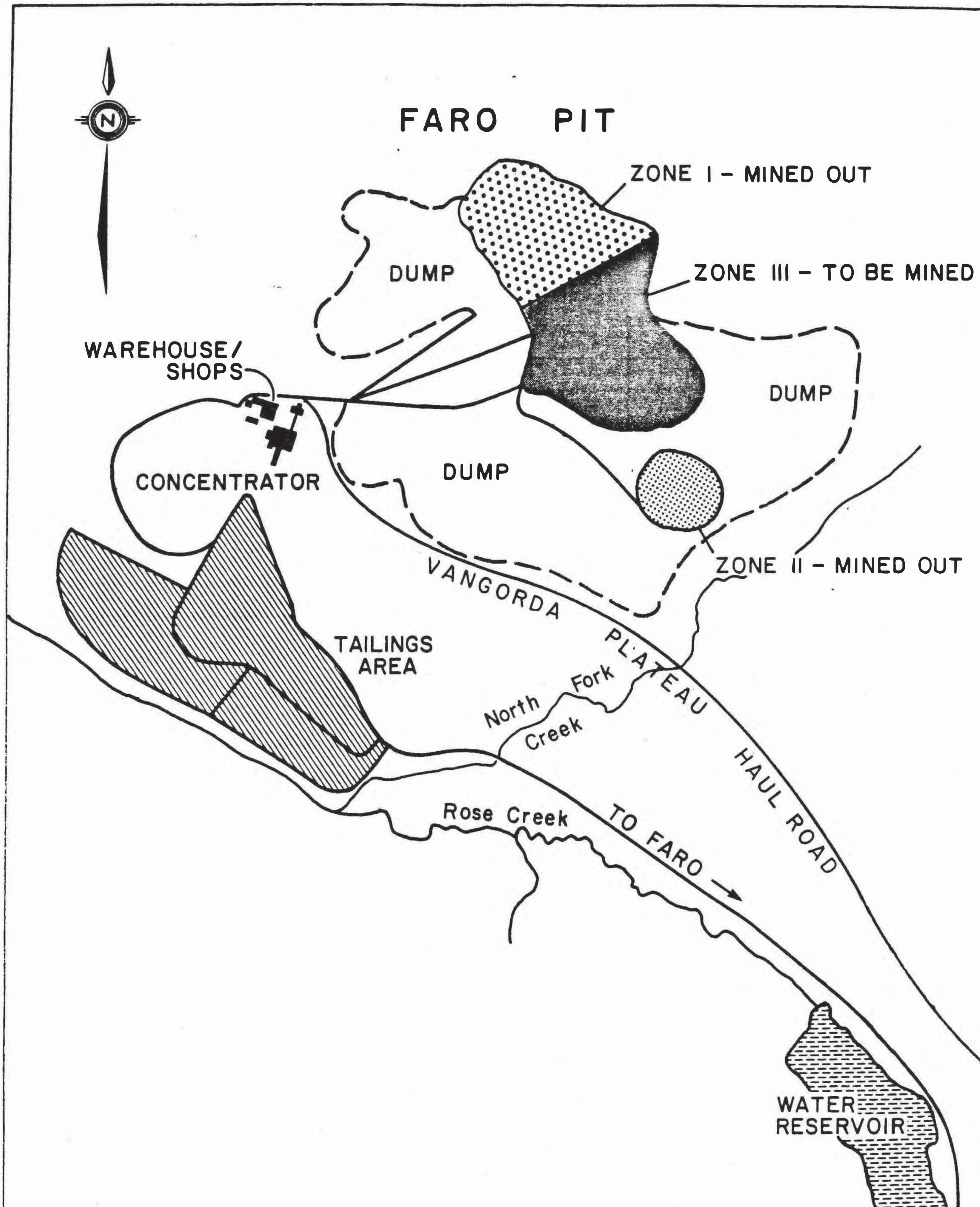
The mill is currently operating at in excess of design capacity.

1.4 PROPERTY

The Curragh mineral holdings in the Anvil District are shown on the District Claim Map Figure 3.1.2-2.

The Faro Mine Site Layout shown in Figure 1.4-1 indicates the relative location of the Faro open pit, concentrator, service buildings and the tailings impoundment ponds.

The locations of the Vangorda and Grum pits in relation to the Faro pit and concentrator are shown in Figure 1.4-2.



TITLE: FARO AREA DEPOSITS FARO MINE - SITE PLAN	SECTION:
KILBORN ENGINEERING (B.C.) LTD.	AREA NO: / REV. NO:
CLIENT: CURRAGH RESOURCES FARO, YUKON — PROJECT NO: 3509-19	DRAWING NO: FIGURE 1.4-1 — A
APPROVED: — DATE: MARCH 87	

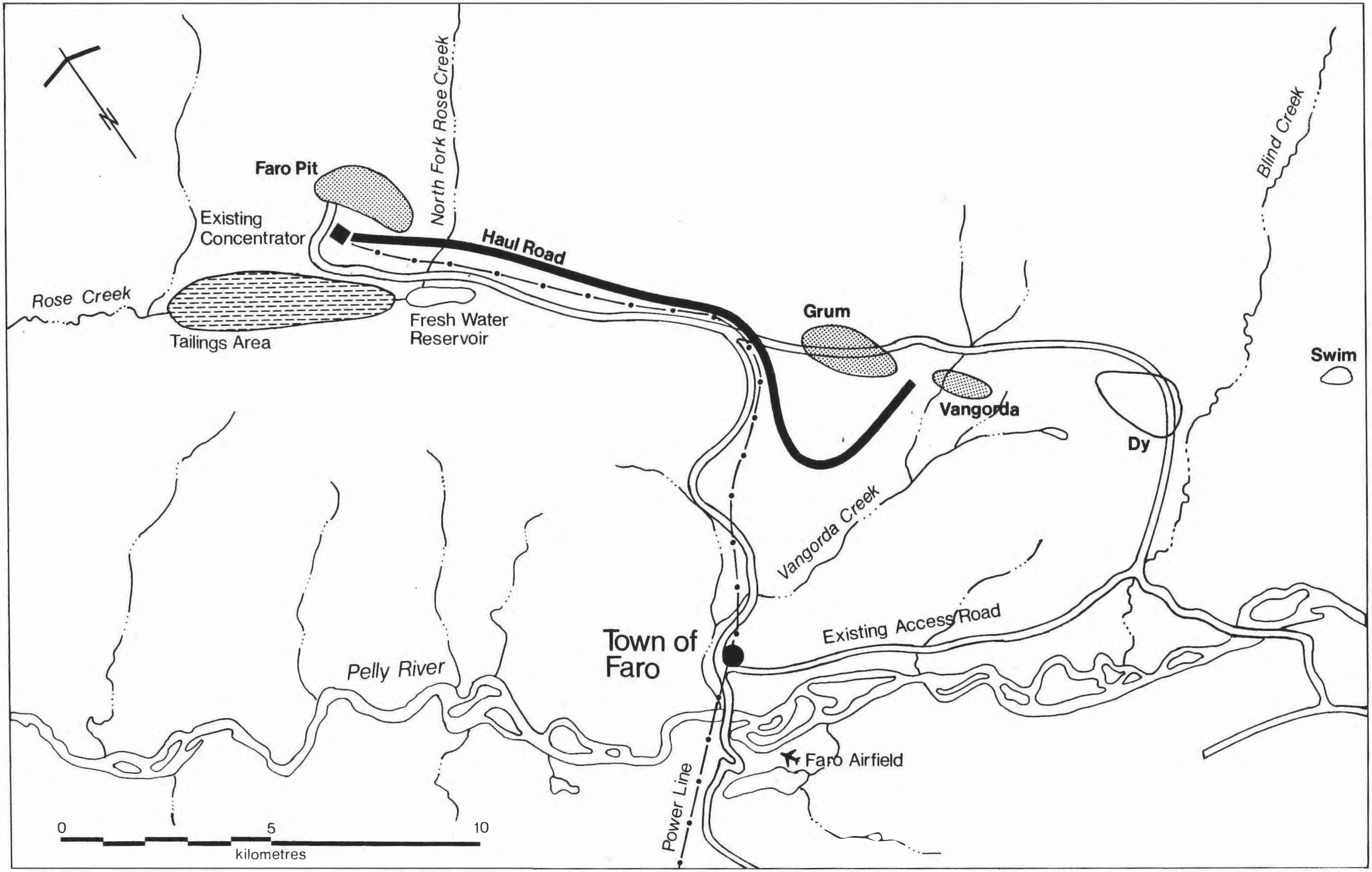


Figure 1.4-2

Curragh Resources
Faro, Vangorda and Grum
Ore Deposits and Haul Road

2

2.0 SUMMARY

2.0 SUMMARY

2.1 INTRODUCTION

Curragh proposes to extend the life of the Faro operations by mining the Vangorda and Grum deposits as well as the Faro Mine. It is planned to continue the operation to the year 1999 at which time all the known open-pit ore will be mined and processed. This will not exhaust the mineral inventory in the Faro area.

The mining concept incorporates the sequential mining of the three deposits by the open-pit method and the mining of underground remnants of the Faro deposit. Productivity rates for men and equipment are based on current experience in the Faro open pit.

2.2 ASSUMPTIONS

The principal assumptions used in the Study are:

Infrastructure	-	No planned changes;
Milling	-	Maximum average daily milling rate - 13,500 tonnes for Faro Ore and 11,000 tonnes for Vangorda/Grum Ore
	-	Maximum average daily concentrate production rate - 1,600 tonnes;
Productivity	-	Efficiency rates equivalent to 1987 budget.

2.3 DISCUSSION

The mining schedules and concepts described in this Report are one route which can be taken to profitably mine the deposits which are amenable to open-pit mining in the Faro area. The underground mining of the Faro remnants with the completion of the open-pit operations is a logical last step in completing the extraction of the ore in the deposit.

Throughout this Report there is discussion of additional work required. This work is necessary for optimization and detailed planning. The information is not required to make a decision to proceed. Further optimization of specific items in the schedule will enhance the value of the Project.

2.4 PRODUCTION SCHEDULE

Mining is scheduled to be completed by the end of 1997, after which time low grade stockpiles will be treated. Table 2.4-1 gives the mine ore production schedule, mill feed schedules and concentrate production. Table 2.4-2 (4 pages) gives the stockpile inventories and sources of mill feed by year. Figure 2.4-1 graphically shows the mining quantities by year.

2.5 SCHEDULE

A schedule for implementation of the Project as described in this Report is included as Figure 2.5-1. This schedule has some key event dates which require an early start on portions of the Project.

TABLE 2.4-1

FARO AREA DEPOSITS STUDY

MINED ORE, MILL FEED & CONCENTRATE PRODUCTION SUMMARY

	Apr - Dec 1987	1988	1989	1990	1991	1992	1993	1994	1995	1996	1997	1998	1999	2000	Total
Ore Summary															
Mined															
Tonnes	4 153 875	6 739 025	5 726 082	4 426 855	5 783 211	6 574 443	3 836 829	4 577 467	4 677 002	3 628 173	3 925 407	U/G	U/G	U/G	54 048 367
% Pb+Zn	7.65	7.01	7.76	8.53	7.10	7.40	7.34	8.39	8.70	8.24	8.40	NA	NA	NA	7.78
%Pb	3.19	2.86	3.15	3.62	2.77	2.65	2.68	3.18	3.22	3.14	3.15	NA	NA	NA	3.02
%Zn	4.46	4.15	4.61	4.91	4.33	4.75	4.66	5.22	5.48	5.09	5.26	NA	NA	NA	4.76
Ag g/t	43	39	44	51	38	32	44	53	55	54	56	NA	NA	NA	44.83
Au g/t	0.10	0.11	0.22	0.53	0.37	0.25	0.65	0.83	0.91	0.86	0.99	NA	NA	NA	0.48
Mill Feed															
Tonnes	3 533 005	4 791 321	4 324 747	3 997 068	4 879 716	4 466 176	4 374 324	4 015 000	3 800 000	4 088 273	4 015 000	4 147 856	4 018 403	0	54 450 887
% Pb+Zn	8.11	7.90	8.57	9.02	7.79	8.35	7.56	8.92	9.58	8.38	8.30	4.08	4.36	0.00	7.76
%Pb	3.35	3.22	3.49	3.83	3.09	3.01	2.77	3.37	3.53	3.18	3.10	1.57	1.69	0.00	3.01
%Zn	4.76	4.68	5.08	5.20	4.70	5.34	4.79	5.55	6.05	5.21	5.21	2.51	2.66	0.00	4.74
Ag g/t	42	42	47	54	41	37	43	56	60	54	55	26	26	0	44.49
Au g/t	0.09	0.07	0.22	0.53	0.38	0.30	0.54	0.85	0.96	0.79	0.96	0.50	0.11	0.00	0.48
Concentrate															
Lead Conc															
Tonnes	153 930	194 800	202 882	235 492	206 735	174 180	156 404	181 190	180 814	171 517	162 821	74 754	78 287	0	2 173 807
% Pb	61.06	60.67	58.67	54.73	57.42	58.83	59.51	60.00	60.00	59.92	60.00	58.82	56.68	0.00	58.98
Ag g/t	551	553	582	606	598	545	737	823	837	821	876	751	598	0	677.58
Au g/t	1.40	1.40	1.40	3.29	2.87	2.11	4.63	6.22	6.68	6.12	7.74	7.10	0.00	0.00	3.87
Zinc Conc															
Tonnes	269 269	355 206	345 732	311 243	350 245	378 787	319 620	345 518	361 435	326 566	318 959	140 516	152 975	0	3 976 070
% Zn	50.90	50.91	51.56	53.93	52.85	52.29	54.10	55.00	55.00	54.77	55.00	53.74	50.73	0.00	53.23
Total Conc															
Tonnes	420 020	550 006	548 615	546 735	556 980	552 967	476 023	526 708	542 249	498 083	481 781	215 270	231 262	0	6 146 699

Table:2.4-2
FARO AREA DEPOSIT STUDY
SUMMARY OF MINED ORE WITH OPENING AND CLOSING STOCKPILE INVENTORIES

	1987	1988	1989	1990	1991	1992	1993	1994	1995	1996	1997	1998	1999
Opening Stockpile													
Tonnes	402520	1023390	3571310	4372429	4802216	5705711	7813978	7276483	7838950	8715952	8255852	8166259	4018403
%Pb	1.87	2.70	2.81	2.90	2.97	2.92	2.85	2.84	2.86	2.90	2.91	2.92	2.92
%Zn	3.14	3.98	4.10	4.22	4.29	4.30	4.42	4.44	4.49	4.59	4.62	4.65	4.65
Ag g/t	26.18	36.56	38.12	39.60	40.68	39.96	37.74	38.16	39.22	40.78	41.44	42.09	42.09
Au g/t	0.06	0.08	0.10	0.13	0.17	0.22	0.23	0.25	0.29	0.36	0.38	0.41	0.41
Lead	8545	21682	92613	104889	113607	119830	166858	142878	154918	174839	153899	153041	78287
%Pb	47.81	56.24	58.77	58.66	58.34	58.25	58.40	58.48	58.60	58.76	58.82	58.87	58.87
Ag	497.74	547.97	565.72	572.13	574.82	580.79	565.78	577.41	596.55	623.87	634.15	644.33	644.33
Au	0.00	0.00	0.30	0.52	0.75	1.25	1.42	1.63	2.00	2.55	2.76	2.97	2.97
Zinc	18786	42519	158671	179844	192195	222336	333402	282342	304378	341515	297188	293491	152975
%Zn	50.34	50.68	51.00	51.12	51.32	51.68	51.81	51.99	52.21	52.51	52.62	52.71	52.71
Mined Ore													
Tonnes	4153875	7339241	5125866	4426855	5783211	6574443	3836829	4577467	4677002	3628173	3925407	0	0
%Pb	3.19	2.86	3.15	3.62	2.77	2.65	2.68	3.18	3.22	3.14	3.15	0.00	0.00
%Zn	4.46	4.15	4.61	4.91	4.33	4.75	4.66	5.22	5.48	5.09	5.26	0.00	0.00
Ag g/t	42.61	38.75	44.08	50.83	37.64	31.73	44.47	52.95	54.77	53.65	56.01	0.00	0.00
Au g/t	0.10	0.11	0.22	0.53	0.37	0.25	0.65	0.83	0.91	0.86	0.99	0.00	0.00
Lead	167067	265731	215159	244209	212959	221207	132424	193230	200736	150576	161964	0	0
%Pb	60.95	59.54	58.31	54.82	57.94	58.80	60.01	60.00	60.00	60.00	60.00	0.00	0.00
Ag	576.08	571.15	593.62	604.16	601.36	527.53	814.49	823.69	836.26	836.48	875.45	0.00	0.00
Au	0.00	0.39	1.25	3.33	2.95	1.84	5.92	6.40	6.89	6.71	7.82	0.00	0.00
Zinc	293002	471358	366905	323594	380385	489853	268560	367554	398572	282239	315262	0	0
%Zn	50.91	51.11	51.53	53.87	52.76	52.08	54.97	55.00	55.00	55.00	55.00	0.00	0.00

Table:2.4-2
FARO AREA DEPOSIT STUDY
SUMMARY OF MINED ORE WITH OPENING AND CLOSING STOCKPILE INVENTORIES

	1987	1988	1989	1990	1991	1992	1993	1994	1995	1996	1997	1998	1999
Pit to Mill													
Tonnes	3507146	4791321	4042122	3967068	4439760	4466176	3387324	4015000	3800000	3208962	3558106	0	0
%Pb	3.43	3.22	3.51	3.84	3.09	3.01	2.86	3.37	3.53	3.35	3.31	0.00	0.00
%Zn	4.79	4.68	5.11	5.21	4.75	5.34	4.93	5.55	6.05	5.46	5.56	0.00	0.00
Ag g/t	45.05	41.58	47.35	53.84	40.91	36.68	47.06	56.00	60.00	57.00	59.00	0.00	0.00
Au g/t	0.09	0.07	0.22	0.53	0.37	0.30	0.67	0.85	0.96	0.88	1.01	0.00	0.00
Lead	153163	194800	191048	234684	186024	174180	125739	181190	180814	143129	155509	0	0
%Pb	60.99	60.67	58.60	54.71	57.89	58.83	60.01	60.00	60.00	60.00	60.00	0.00	0.00
Ag	567.84	553.35	583.17	605.44	587.58	544.69	811.54	823.00	837.00	835.00	877.00	0.00	0.00
Au	0.00	0.05	1.32	3.28	2.73	2.11	5.76	6.22	6.68	6.47	7.64	0.00	0.00
Zinc	267701	355206	325028	309664	324392	378787	252653	345518	361435	269625	304042	0	0
%Zn	50.90	50.91	51.58	53.93	52.69	52.29	54.97	55.00	55.00	55.00	55.00	0.00	0.00
Pit to SP													
Tonnes	646729	2547920	1083744	459787	1343451	2108267	449505	562467	877002	419211	367301	0	0
%Pb	3.19	2.86	3.15	3.62	2.77	2.65	2.68	3.18	3.22	3.14	3.15	0.00	0.00
%Zn	4.46	4.15	4.61	4.91	4.33	4.75	4.66	5.22	5.48	5.09	5.26	0.00	0.00
Ag g/t	42.61	38.75	44.08	50.83	37.64	31.73	44.47	52.95	54.77	53.65	56.01	0.00	0.00
Au g/t	0.10	0.11	0.22	0.53	0.37	0.25	0.65	0.83	0.91	0.86	0.99	0.00	0.00
Lead	13904	70931	24111	9525	26935	47028	6685	12040	19922	7447	6455	0	0
%Pb	60.95	59.54	58.31	54.82	57.94	58.80	60.01	60.00	60.00	60.00	60.00	0.00	0.00
Ag	576.08	571.15	593.62	604.16	601.36	527.53	814.49	823.69	836.26	836.48	875.45	0.00	0.00
Au	0.00	0.39	1.25	3.33	2.95	1.84	5.92	6.40	6.89	6.71	7.82	0.00	0.00
Zinc	25301	116152	41877	13930	55994	111066	15907	22036	37137	12614	11220	0	0
%Zn	50.91	51.11	51.53	53.87	52.76	52.08	54.97	55.00	55.00	55.00	55.00	0.00	0.00

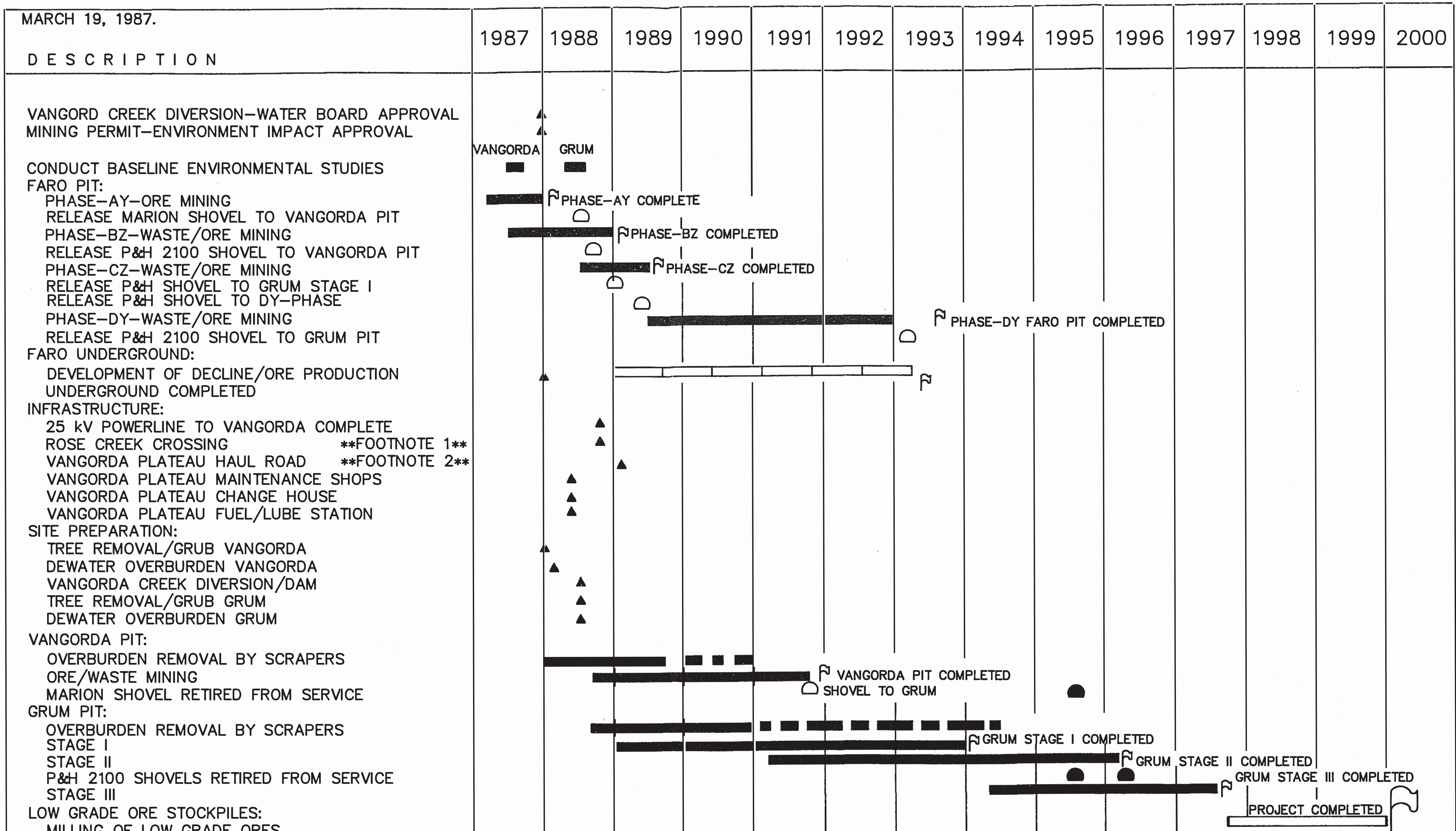
Table:2.4-2
FARO AREA DEPOSIT STUDY
SUMMARY OF MINED ORE WITH OPENING AND CLOSING STOCKPILE INVENTORIES

	1987	1988	1989	1990	1991	1992	1993	1994	1995	1996	1997	1998	1999
Stockpile to Mill													
Tonnes	25859	0	282625	30000	439956	0	987000	0	0	879311	456894	4147856	4018403
%Pb	2.41	0.00	3.24	2.20	3.07	0.00	2.47	0.00	0.00	2.55	1.44	1.57	1.69
%Zn	3.84	0.00	4.66	3.66	4.16	0.00	4.32	0.00	0.00	4.28	2.46	2.51	2.66
Ag g/t	34.00	0.00	42.11	36.00	43.85	0.00	27.12	0.00	0.00	40.98	25.87	26.37	25.63
Au g/t	0.06	0.00	0.12	0.48	0.49	0.00	0.09	0.00	0.00	0.48	0.59	0.50	0.11
Lead	767	0	11834	808	20712	0	30665	0	0	28388	7312	74754	78287
%Pb	62.00	0.00	59.77	60.00	53.26	0.00	57.45	0.00	0.00	59.49	60.00	58.82	56.68
Ag	371.00	0.00	558.88	808.00	614.31	0.00	429.91	0.00	0.00	751.96	861.74	750.78	598.47
Au	0.00	0.00	0.54	5.31	4.08	0.00	0.00	0.00	0.00	4.34	9.93	7.10	0.00
Zinc	1568	0	20704	1578	25853	0	66967	0	0	56941	14917	140516	152975
%Zn	51.00	0.00	51.25	55.00	54.91	0.00	50.82	0.00	0.00	53.69	55.00	53.74	50.73
Mill Feed Total													
Tonnes	3533005	4791321	4324747	3997068	4879716	4466176	4374324	4015000	3800000	4088273	4015000	4147856	4018403
%Pb	3.42	3.22	3.49	3.83	3.09	3.01	2.77	3.37	3.53	3.18	3.10	1.57	1.69
%Zn	4.78	4.68	5.08	5.20	4.70	5.34	4.79	5.55	6.05	5.21	5.21	2.51	2.66
Ag g/t	44.97	41.58	47.01	53.71	41.17	36.68	42.56	56.00	60.00	53.56	55.23	26.37	25.63
Au g/t	0.09	0.07	0.22	0.53	0.38	0.30	0.54	0.85	0.96	0.79	0.96	0.50	0.11
Lead	153930	194800	202882	235492	206735	174180	156404	181190	180814	171517	162821	74754	78287
%Pb	61.00	60.67	58.67	54.73	57.42	58.83	59.51	60.00	60.00	59.92	60.00	58.82	56.68
Ag	566.86	553.35	581.75	606.14	590.26	544.69	736.72	823.00	837.00	821.26	876.31	750.78	598.47
Au	0.00	0.05	1.28	3.29	2.87	2.11	4.63	6.22	6.68	6.12	7.74	7.10	0.00
Zinc	269269	355206	345732	311243	350245	378787	319620	345518	361435	326566	318959	140516	152975
%Zn	50.90	50.91	51.56	53.93	52.85	52.29	54.10	55.00	55.00	54.77	55.00	53.74	50.73

Table:2.4-2
FARO AREA DEPOSIT STUDY
SUMMARY OF MINED ORE WITH OPENING AND CLOSING STOCKPILE INVENTORIES

	1987	1988	1989	1990	1991	1992	1993	1994	1995	1996	1997	1998	1999
Closing SP													
Tonnes	1023390	3571310	4372429	4802216	5705711	7813978	7276483	7838950	8715952	8255852	8166259	4018403	0
%Pb	2.70	2.81	2.90	2.97	2.92	2.85	2.84	2.86	2.90	2.91	2.92	2.92	0.00
%Zn	3.98	4.10	4.22	4.29	4.30	4.42	4.44	4.49	4.59	4.62	4.65	4.65	0.00
Ag g/t	36.56	38.12	39.60	40.68	39.96	37.74	38.16	39.22	40.78	41.44	42.09	42.09	0.00
Au g/t	0.08	0.10	0.13	0.17	0.22	0.23	0.25	0.29	0.36	0.38	0.41	0.41	0.00
Lead	21682	92613	104889	113607	119830	166858	142878	154918	174839	153899	153041	78287	0
%Pb	56.24	58.77	58.66	58.34	58.25	58.40	58.48	58.60	58.76	58.82	58.87	58.87	0.00
Ag	547.97	565.72	572.13	574.82	580.79	565.78	577.41	596.55	623.87	634.15	644.33	644.33	0.00
Au	0.00	0.30	0.52	0.75	1.25	1.42	1.63	2.00	2.55	2.76	2.97	2.97	0.00
Zinc	42519	158671	179844	192195	222336	333402	282342	304378	341515	297188	293491	152975	0
%Zn	50.68	51.00	51.12	51.32	51.68	51.81	51.99	52.21	52.51	52.62	52.71	52.71	0.00

MARCH 19, 1987.



FOOTNOTE 1 ACCESS REQUIRED FOR SHOVEL MOVE FROM FARO TO VANGORDA PIT
FOOTNOTE 2 ROAD REQUIRED FOR VANGORDA ORE HAULAGE

▲ LATEST IMPLEMENTATION DATE

KILBORN

FIGURE: 2.5–1
DRAFT
SCHEDULE

CURRAGH RESOURCES
FARO–AREA DEPOSITS STUDY
TRUCK SHOVEL WITH SCRAPERS IN OVERBURDEN SCHEME
IMPLEMENTATION, EVENTS & ACTIVITIES

3

3.0 GEOLOGY AND GEOLOGICAL ORE RESERVES

3.0 GEOLOGY AND GEOLOGICAL ORE RESERVES

3.1 DISTRICT GEOLOGY

3.1.1 Introduction

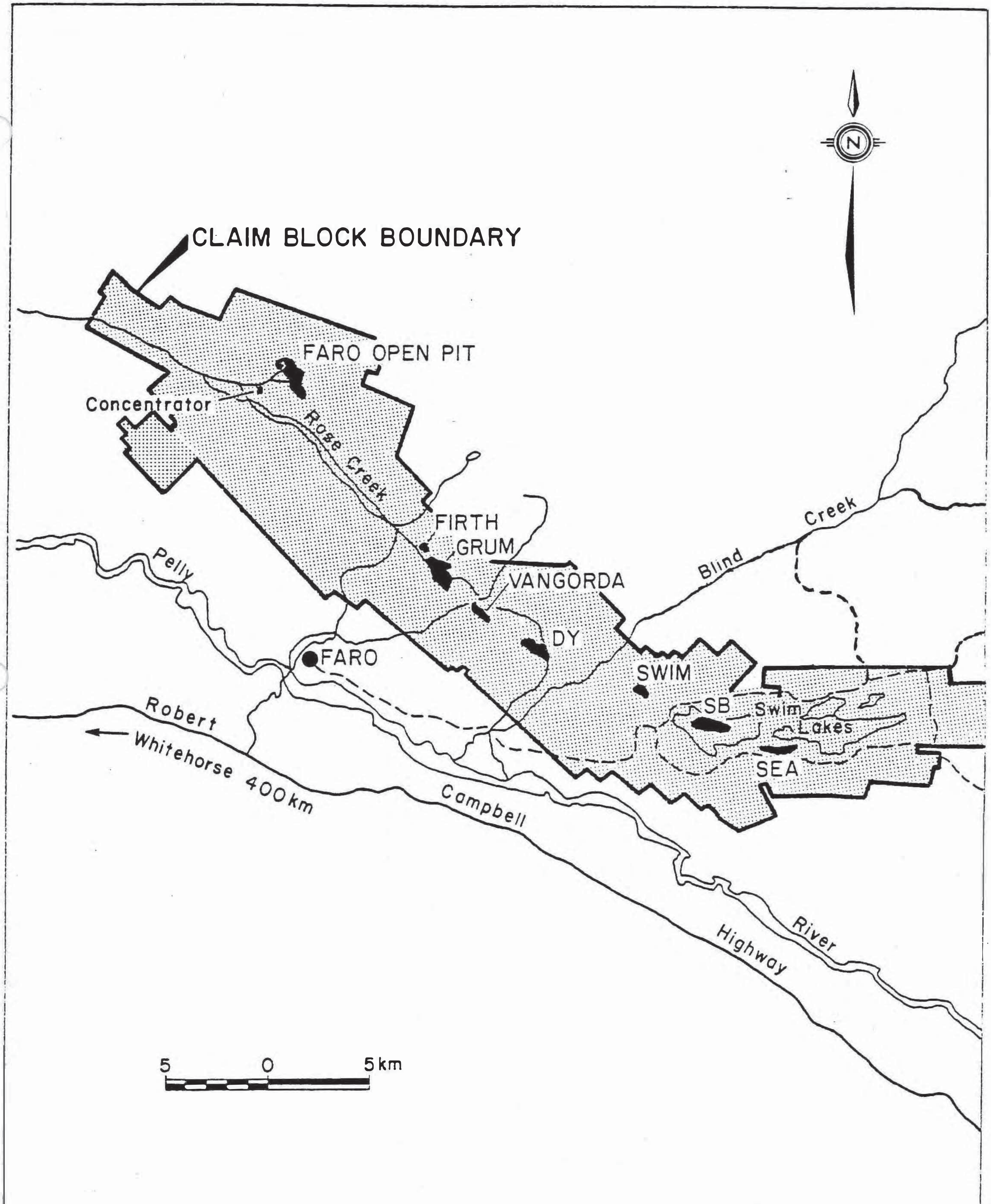
The Anvil Range Lead-Zinc-Silver District is located in the central Yukon Territory near the town of Faro. The district contains one of the world's largest reserves of lead and zinc in several deposits including the recently re-opened Faro Mine. Figure 3.1.1-1 shows the mineral holdings of Curragh Resources and the deposits which have been discovered to the present time.

3.1.2 Regional Geology

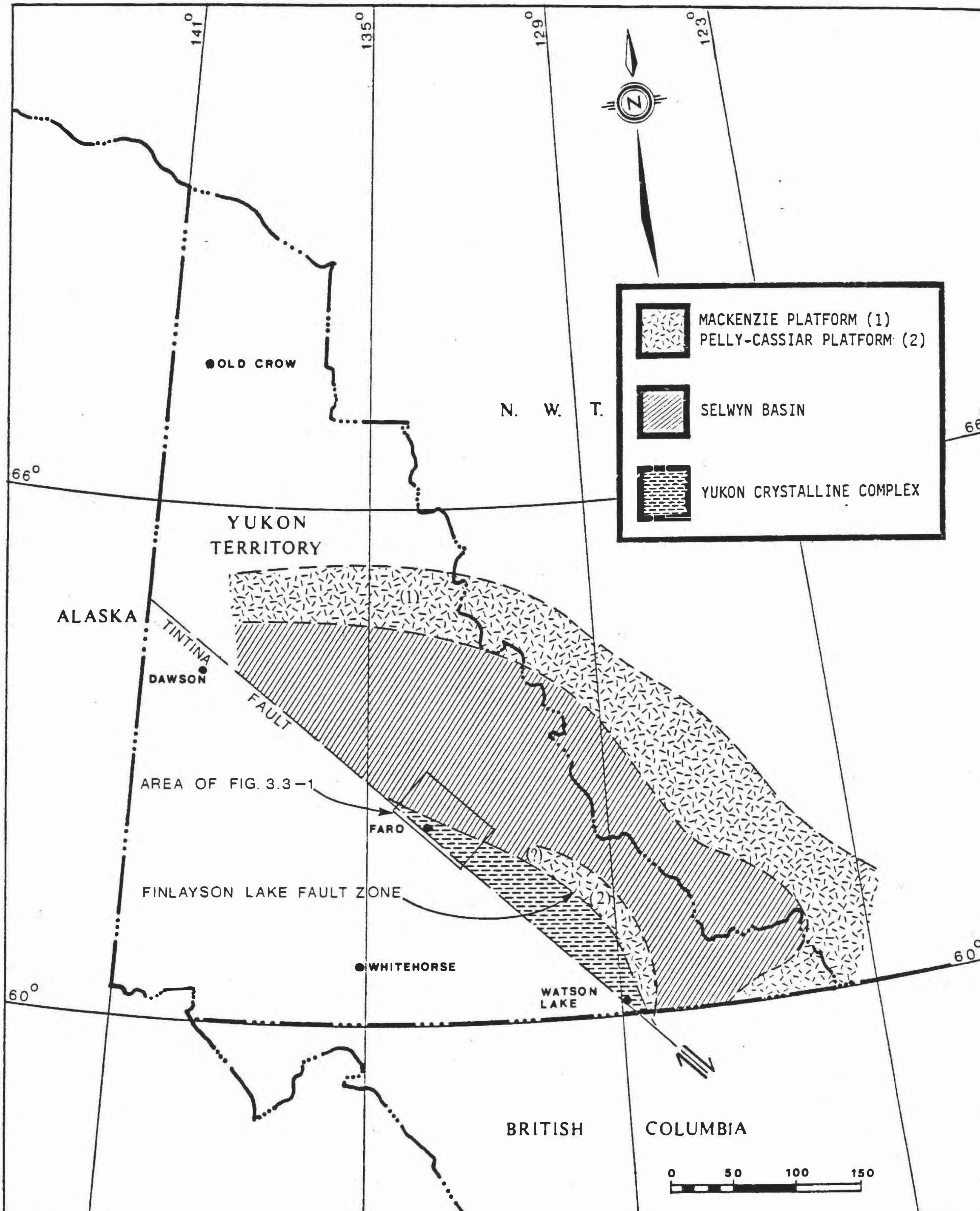
The Anvil District is part of the Selwyn Basin (Figure 3.1.2-1), a large area of the central Yukon where deep water shales accumulated along the ancient North American continental margin during the Paleozoic. The shales of the Selwyn Basin host most of Canada's large stratiform lead-zinc deposits, making it a metallogenic province of worldwide significance. Unlike the remainder of the Selwyn Basin, the rocks and ores of the Anvil District are metamorphosed thus 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 3.1.2-2). 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 laterial Cordilleran strike slip faults, passes just south of the district (Figures 3.1.2-1 and 3.1.2-2), but is not directly related to the ores.

3.1.3 District Stratigraphy

The stratigraphic sequence of Anvil District ranges in age from



TITLE: FARO AREA DEPOSITS MINERAL CLAIM MAP	SECTION:	
KILBORN ENGINEERING (B.C.) LTD.	AREA NO:	REV. NO:
CLIENT: CURRAGH RESOURCES FARO, YUKON — PROJECT NO: 3509-19	DRAWING NO: FIGURE 3.1.1-1	A
APPROVED: — DATE: MAR 3/87		



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KILBORN ENGINEERING (B.C.) LTD.	AREA NO:	REV. NO:
CLIENT: CURRAGH RESOURCES — PROJECT NO: 3509-19	DRAWING NO: FIG. 3.1.2-1	A
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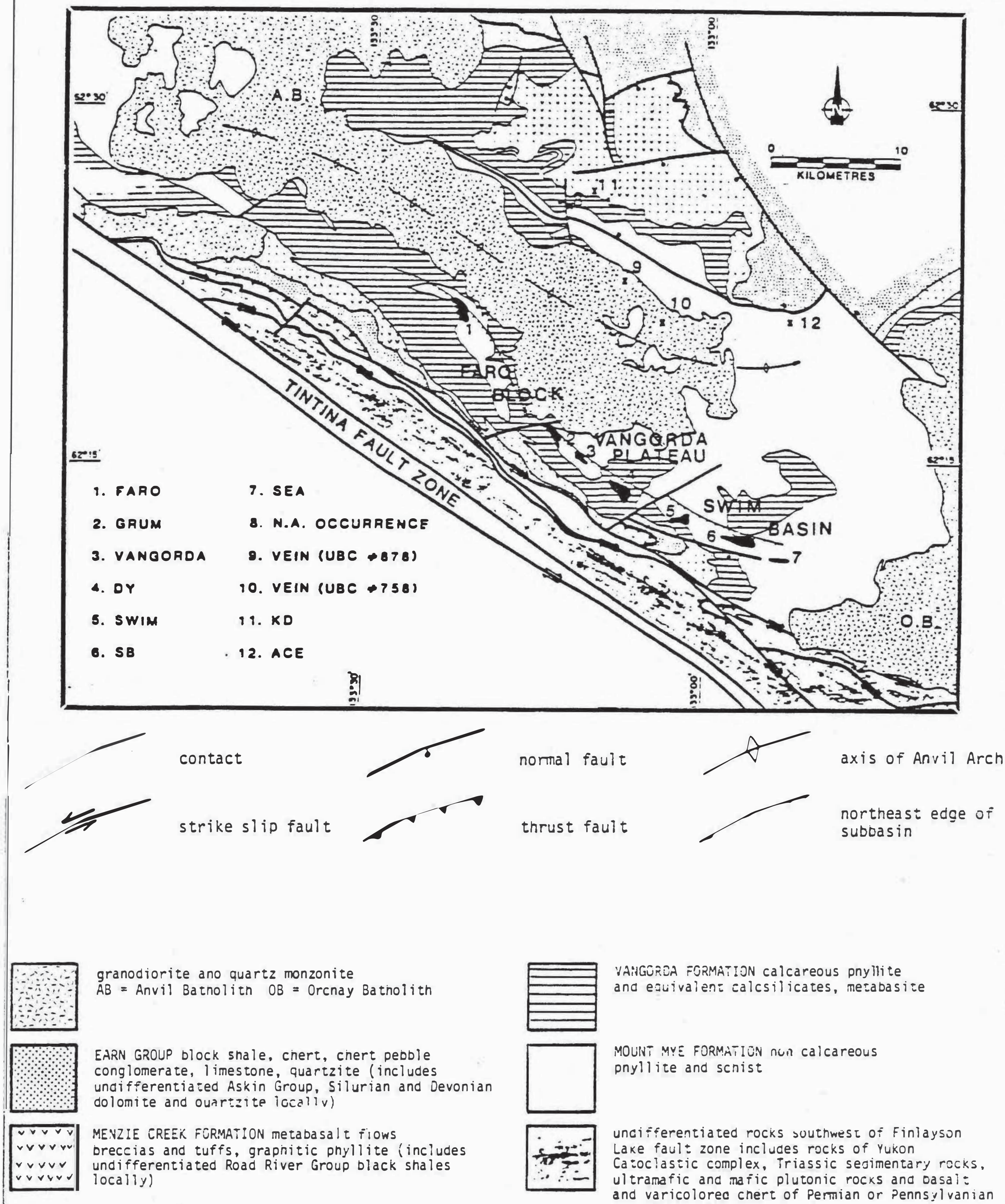

TITLE: FARO AREA DEPOSITS ANVIL RANGE GEOLOGY	SECTION:
KILBORN ENGINEERING (B.C.) LTD.	AREA NO:
CLIENT: CURRAGH RESOURCES — PROJECT NO: 3509-19	DRAWING NO: FIG. 3.1.2-2 — REV. NO: A
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latest Precambrian to Permian. Two major divisions or assemblages 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 3.1.3-1). 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 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, interlayed, 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 ±

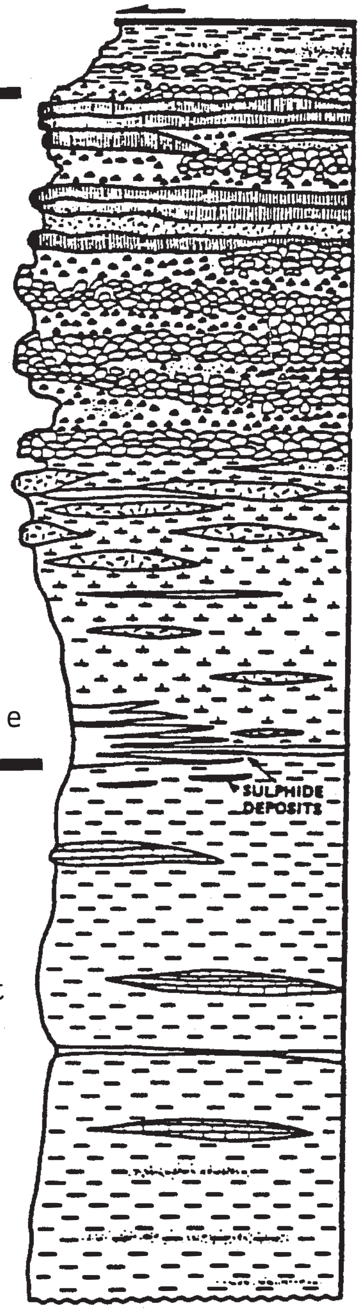
Vangorda
Formation

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:
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CLIENT: **CURRAGH RESOURCES** PROJECT NO: 3509-19	DRAWING NO: FIG. 3.1.3-1
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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 - 2 centimetres) interlayers of i) medium gray, non-calcareous, weakly carbonceous, 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 equi-granular, 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 Basic.

(d) Relation of Stratigraphy to Ore Deposits

The ore deposits of 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.

3.1.4 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 3.1.2-1). 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 3.1.2-2) 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.

3.1.5 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 3.1.2-2). From northwest to southeast, they include Faro, Grum, Vangorda, Dy and Swim. Additionally, two lead-zinc deficient sulphide occurrences, the SB and Sea, are also known.

The Faro area geological reserves and mineral inventory are summarized in Table 3.1.5-1.

TABLE 3.1.5-1

FARO AREA

GEOLOGICAL RESERVES AND MINERAL INVENTORY

Source	Cut-off % Pb+Zn	Class	Tonnes	% Pb+Zn	% Pb	% Zn	Ag g/t	Au g/t
Faro								
Total Deposit	4.0	Proven	29,251,000	8.2	3.1	5.0	41	N.A.
Grum								
Main Zone (61W-87W)	4.0	Proven	30,649,000	9.0	3.4	5.6	57	0.95
Champ Zone (51W-61W)	4.0	Probable	1,700,000	7.8	3.5	4.3	46	N.A.
N.W. Extension (87W-100W)	N.A.	Possible	8,000,000	10.0	N.A.	N.A.	N.A.	N.A.
Total Deposit	N.A.	N.A.	40,349,000	9.1	N.A.	N.A.	N.A.	N.A.
Vangorda								
Total Deposit	4.0	Probable	7,457,000	8.7	3.8	4.9	54	0.69
Dy								
Total Deposit	9.0	Possible	21,059,000	12.2	5.5	6.7	84	0.95
Swim								
Total Deposit	6.0	Possible	4,309,000	8.5	3.8	4.7	51	N.A.
TOTAL INVENTORY			102,425,000	9.4	N.A.	N.A.	N.A.	N.A.

NOTES: Two other deposits, SB and Sea, contain sulphide mineralization but have not been evaluated due to sub-marginal grades.

Effective Date January 1, 1986. During 1986, 1.8 million tonnes were mined from the Faro open pit.

There are six basic sulphide rock types as follows:

(a) Massive Pyritic Sulphides [Units 2E/2F]

The massive sulphides consist of banded to homogenous, usually weakly foliated and/or lineated, massive pyrite with lesser sphalerite and galena. Total sulphide content is at least 60 percent, generally greater than 80 percent and commonly nearly 100 percent. Gangue consists of quartz and/or barite and/or carbonates (calcite, dolomite, ankerite). Accessory minerals include pyrrhotite, chalcopyrite, magnetite, arsenopyrite and marcasite. At amphibolite facies metamorphic grade, this rock type commonly develops a buckshot porphyroblastic texture of pyrite in a matrix of dark reddish-brown to black lead-zinc sulphides. This texture usually is restricted to rocks with economic lead-zinc grades (Unit 2F). Hard, barren, massive pyrite, commonly with disseminated, black, magnetite porphyroblasts, is widespread at Faro, particularly in the northeast part of the deposit.

(b) Baritic, Massive Pyritic Sulphides [Unit 2G]

The baritic sulphides are strongly and thinly banded massive sulphide/sulphate rock consisting of pyrite, galena, sphalerite and commonly magnetic in a gangue of off-white barite and lesser carbonates (calcite, dolomite, ankerite and probably barytocalcite). The amount of barite may be as high as 50 percent; non-sulphidic, massive barite does not occur in the Anvil deposits. There is a complete gradation between this and the above facies with 10 percent visible barite by volume being the dividing line. The facies is usually quite high grade (10-15 percent combined lead-zinc). Sphalerite is characteristically honey coloured to reddish brown. Pyrrhotite is not commonly seen in the baritic facies except in the Faro deposit where overall pyrrhotite is more abundant.

(c) Carbonate-Bearing, Massive Pyritic Sulphides [Unit 2K]

The carbonate-bearing sulphides are similar to massive pyritic sulphides, but contain greater than 10 percent carbonate (calcite, dolomite, ankerite) either as interstitial gangue or as coarse patches and irregular blebs. This is a minor facies and is not known with certainty to always be an original composition variant. The most common occurrence of coarse, pinkish-beige to tan, ankerite patches may represent recrystallized original carbonate or re-worked pre/syn-metamorphic veins. This variant is generally lead-zinc poor. The variants with white interstitial gangue can be high grade, and locally they texturally resemble the baritic sulphides.

(d) Pyrrhotitic Massive Sulphides [Unit 2H]

This rock type consists of massive, finely crystalline, usually well foliated pyrrhotite with less than 50 percent pyrite porphyroblasts and highly variable amounts of sphalerite and galena. Minor chalcopyrite is characteristic of this relatively copper-rich facies. Rounded to angular, rotated, foliated quartzite or quartz-vein clasts 2 centimetres or less in diameter are typical. This is a minor facies and is not known with certainty to be primary as some pyrite in the massive facies may invert to pyrrhotite during regional metamorphism. At Faro, the pyrrhotitic facies is more volumetrically important than the other deposits. Pyrrhotite-rich ores are generally much finer grained than non-pyrrhotitic ores at Faro.

(e) Ribbon Banded, 'Graphitic', Pyritic Quartzite [Unit 2A]

This unit is a dark gray to black, well-banded, sulphide-bearing quartzite (metamorphic usage). Bands are: i) dark gray, very fine grained carbonaceous phyllitic quartzite to siliceous phyllite (presumed metachert), and ii) light gray, quartz-sulphide (pyrite-sphalerite-galena) bands. These bands are usually 2 millimetres to 2 centimetres thick. Total sulphide content of Unit 2A is usually between 10 to 30 percent, but ranges from 2 to 60 percent. Pyrite is usually the dominant species, but higher grade examples have subequal pyrite and lead-zinc sulphides. Lead-zinc dominant variants with little pyrite occur, but are not common unless total sulphide content is low. Strong sulphide species differentiation between bands, such that barren pyrite bands are adjacent to or near sphalerite or galena-rich bands, occurs but is not generally the case.

(f) Pyritic Quartzite [Units 2B, 2C, 2D]

The pyritic quartzites are light to medium gray, generally poorly banded, moderately to weakly foliated, micaceous quartzites with highly variable lead-zinc and pyrite contents. Pyrite contents are generally 10 to 40 percent ranging between 2 and 60 percent. Although there is a complete gradation from massive to quartzose ores, there is usually little problem in separating this facies from the massive pyritic sulphides as the vast majority of examples have less than 40 percent total sulphides. A minor variant of this facies (Unit 2B) shows low pyrite (< 5 percent) content with lead-zinc sulphides predominant. Barite in major amounts is uncommon in the quartzose facies; carbonate species are not typical but locally are abundant. Chalcopyrite, pyrrhotite and magnetite-bearing varieties are common. Sphalerite in the high grade examples is characteristically a vibrant reddish brown.

Both wallrocks and certain ore facies of the Anvil deposits are overprinted by a prominent, easily recognized, light beige, white mica dominant alteration assemblage (Units 2L and 1D4). This overprint facies is not a depositional unit and may form as a reaction product between wallrocks and deposit forming hydrothermal fluids, or as a metamorphic reaction envelope unrelated to ore forming fluids or as combination of these processes. In the multi-layered deposits, this alteration overprint appears discontinuous and often best developed in the footwall of a given lens or deposit as a whole. At Faro, a continuous envelope of this lithology encloses the entire deposit with local (especially Zone 1) best development in the hangingwall. The more intensely developed alteration assemblages can cause frothing problems in the mill since they contain talc or sericite that acts like talc.

In some cases, it is preferable to refer to a combination of sulphide rock types, particularly in a mining context. In such cases, the letters are combined with the dominant component listed first. Thus, 2EG would be mixed 2E and 2G; 2BCD refers to all non-carbonaceous quartzites regardless of sulphide species. Another common combination is 2CE which can refer both to mixed 2C and 2E and semimassive sulphides between 2C and 2E in character. At Faro, there are now three ore types mined: 2A, 2BG and 2H. In this case, 2BG means all the detailed sulphide types from B through G.

3.2 FARO GEOLOGY AND RESERVES

3.2.1 History

The Faro deposit was discovered in 1964 while drill testing airborne electromagnetic anomalies supported by other indications. Mining at Faro began in late 1969 and continued until 1982 when it was forced to close.

In November 1985, Curragh Resources bought the Faro Mine and other deposits in the Anvil Range. Waste removal from the Faro pit resumed in early 1986. The Faro concentrator resumed production in June 1986.

3.2.2 General Geology

Figure 3.2.2-1 and Figure 3.2.2-2 are schematic longitudinal and cross-section views of the Faro deposit.

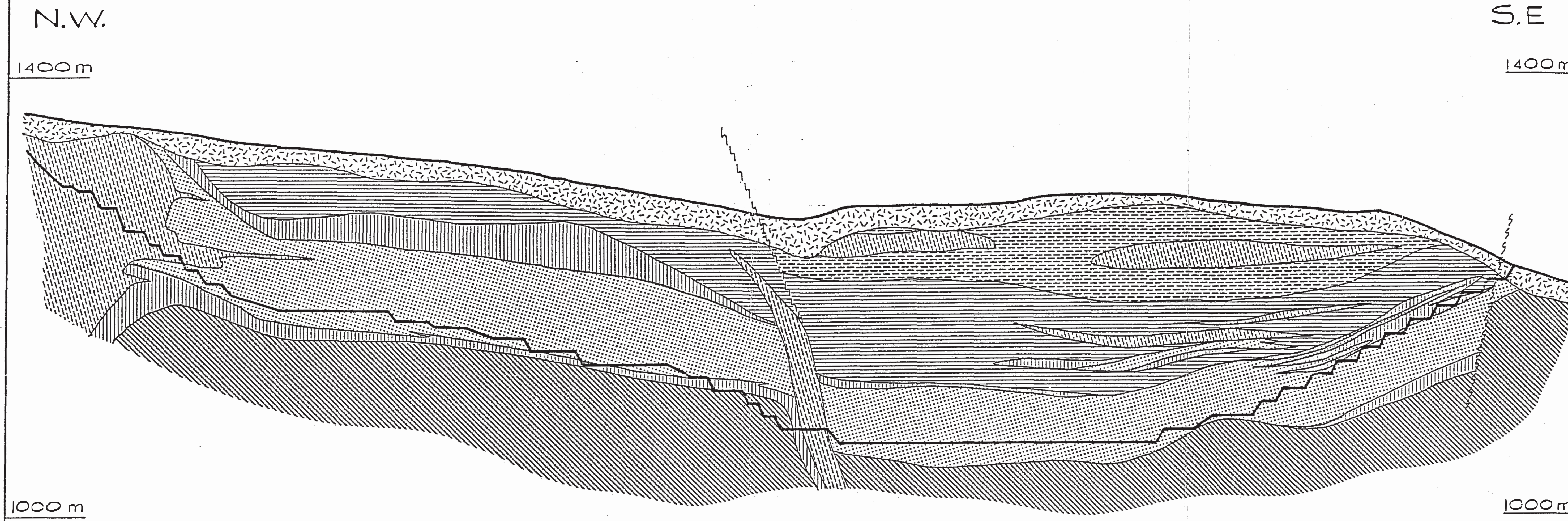
(a) Stratigraphy and Lithology

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) a equigranular to subporphyritic hornblende diorite to quartz diorite clan, and ii) a quartz-



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

0 300 600 900

SCALE IN FEET

REVISIONS	NO	DATE	BY			
ISSUED FOR TECHNICAL REVIEW	A	MAR 3/87		SCALE	DATE	CLIENT CURRAGH RESOURCES
				DESIGNED		LOCATION FARO, YUKON
				DRAWN PAH	MAR 3/87	KILBORN
				CHECKED		
				APPROVED	MAR 3/87	

FARO AREA DEPOSITS
FARO DEPOSIT
LONGITUDINAL SECTION

PROJ. NO. 3509-19 | DWG. NO. FIG. 3.2.2-1 | REV. A

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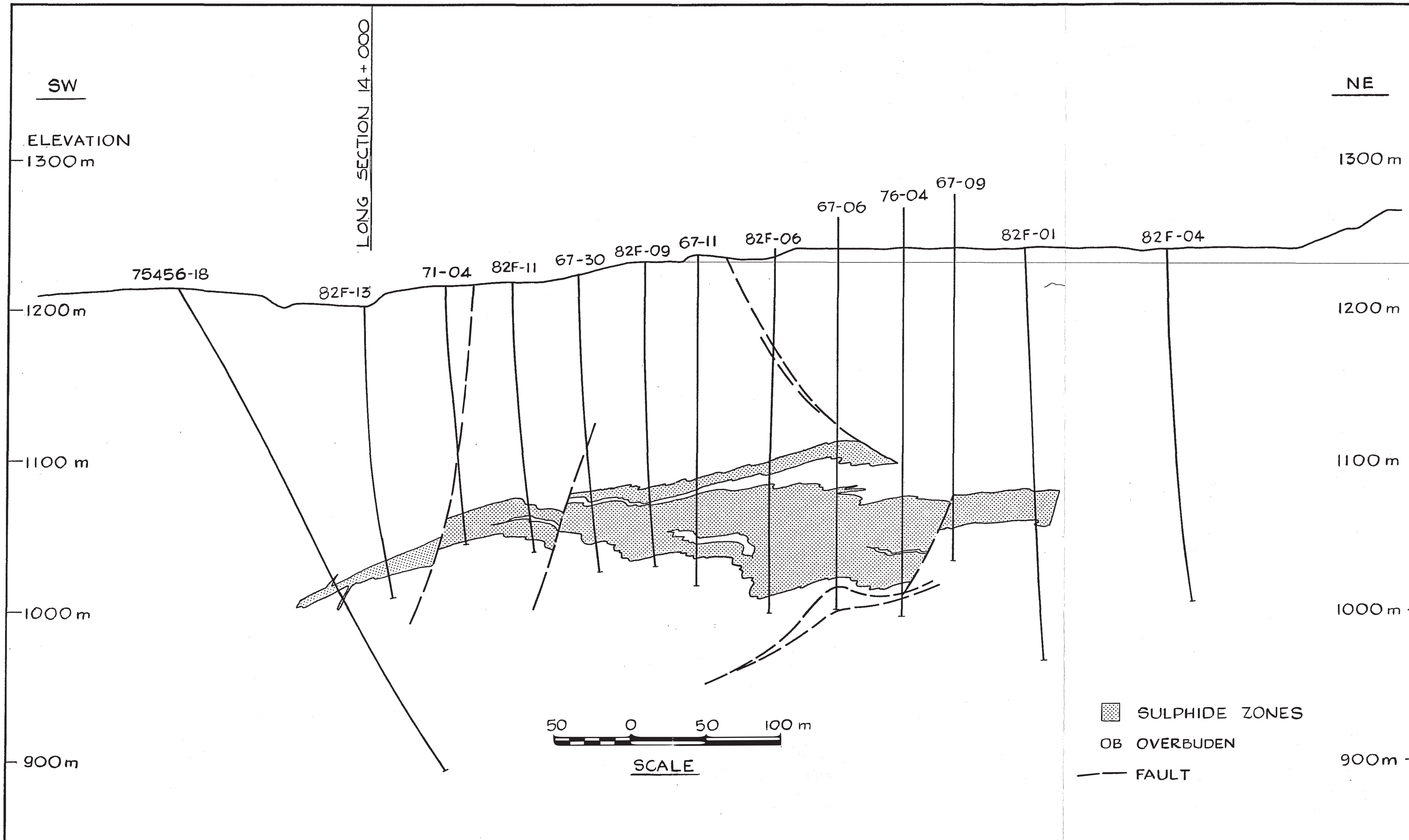


ISSUED FOR TECHNICAL REVIEW	A	MAR87	ZA	SCALE AS SHOWN	DATE	CLIENT CURRAGH RESOURCES	FARO AREA DEPOSITS
				DESIGNED J.B.F.	MAR87	LOCATION FARO, YUKON	FARO DEPOSIT
				DRAWN Z.A.	MAR87	KILBORN	X-SECTION (124 + 22)
REVISIONS	NO	DATE	BY	CHECKED			PROJ.NO 3509-19
				APPROVED			DWG.NO. FIG. 3.2.2-2
							REV. A

K.E.B.C.9

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 (see below), 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 becuase of its extreme hardness.

(b) 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 sign 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^{o}W striking and steeply west dipping set, and ii) an east or N60^{o}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 upthrown 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 butress the northeast pit wall as the pit deepens.

(c) Deposit Geology

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.

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, generally this is 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 using the current methods of ore sampling and delineation.

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 semi-massive 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.

3.2.3 Drilling Density

The Faro deposit was drilled off on 43 metre (141 foot) spaced sections with holes spaced nominally at 43 metres along the sections. Most holes were vertical. In some parts of the deposit, fill-in drilling to 43 metres has not been completed. In 1986, additional fill-in (to about 25 metres) was added to the northeast side of Zone 3 because of difficulties in making accurate short range projections with the available data. The results of this new drilling are still being processed for use in detailed mine production planning.

3.2.4 Reserve Calculation

(a) Method and Procedure - "FI" Model

The reserves used for Faro Mine planning are derived from the "FI" Mine Model, generated from October to December 1985 by Cyprus Anvil Mining Corporation. The FI Model is one of several computer block models of the Faro deposit; reference to some others will be made herein. The F3 and T3 date from 1981.

The F4 is more recent, but was never completed by Cyprus Anvil and thus is not useable for mine planning. The FI Model is an interim model that combined parts of the F3 and F4 and incorporated extensive drilling results postdating completion of the F3 and T3 Models. The 1985 Kilborn analysis of the Faro Mine Project used the T3 Model results.

Geological data will be continually updated to facilitate detailed production planning as the operation progresses.

(b) Block Geology and Drill Hole Information

The FI Model is a computer based block model with block size 15.25 metres by 15.25 metres by 6.1 metres high (50 feet by 50 feet by 20 feet high). The blocks are oriented North-South and East-West, at 45 degrees to the elongation of the ore deposit and geological sections. The Mintec Medsystem, release 10 software package, was used to generate the model and derive reserves.

The geological interpretive base was derived from two sources. In the southeast part of Zone 3, the geological interpretation is the most up to date possible (1983), and is the same as that used for the F4 Model. In the remainder of Zone 3, this new geological interpretation was not yet available, thus the interpretation used was that developed in 1981 for the F3 and T3 Models. This did not take advantage of 1984 drilling. The interpretations will differ only in the relative importance of folds and faults which result in differences in bench to bench geology. For an overall section through the deposit, the cross-section area, and hence the volume, will not be very different.

Block geological code assignment was based on 6 metre (20 foot) spaced bench plans of the geology. A block whose area is underlain by more than 50 percent sulphide rock type was coded as an ore type, otherwise it was coded as waste. The actual block geological code was defined by the unit occupying the maximum plan view area within the block. One rock code was assigned to each block and that code was assumed to apply uniformly to the entire block.

The drill hole database used includes all holes in the deposit to the time of model construction; thus, all holes prior to Curragh's acquisition of Faro, but none of Curragh's 1986 holes, are included. These will be included in subsequent geological updates.

All drill holes were relogged to a common standard between 1982 and 1984 and extensive checking of the assay and survey data for consistency was carried out by Cyprus Anvil's staff. Some holes have not been surveyed for downhole deviation. In these cases, an average deviation based on nearby surveyed holes was used. There are now known to be errors in the collar locations of some holes on the order of 50 feet laterally. This is apparently due to a small angular error in re-establishing survey control during the late 1960's and cannot be corrected now. The error is largest in the more southeasterly holes. Apparently, not all holes are affected. Holes postdating the mid-1970's are thought to be consistent with the current survey control.

(c) Composite Calculation

Drill hole assays were composited on a 6 metre (20 foot) bench basis. Assays were weighted by length within the bench and specific gravity of the constituent samples. High assay values were rolled back to the 95th percentile level before compositing. Internal waste (3 metre or 10 feet thick or less) was included in the composites at zero grade. Composites were again clipped to the 95th percentile before interpolation. External waste and waste bands greater than a one-half bench height (3 metres or 10 feet) were not included in the composite intervals resulting in a composite shorter than 6 metres (20 feet) long. This was carried out on the premise that waste or ore thicker than half a bench height could be separated during mining. There is some debate as to the validity of the assumption since this method of composite calculation could lead to grades that require a higher dilution in order to quote mill feed than a calculation that averages an entire bench regardless of material type. Part of the rationale in this method of compositing is that a given composite will be used on more than one bench, thus a composite from the margin of the deposit will be used to estimate the grade of the interior of the deposit, and in that case it would not be appropriate to have averaged in a large amount of unmineralized material. All previous Anvil District geological models have followed exactly the same compositing scheme. Thus, this is not an explanation of differences between model reserves.

The geological coding of the composites was changed and improved over previous models (particularly the T3). Each composite was checked manually to ensure that it was coded consistently with the sectional geology rather than machine coded by detailed logged geology. Since large interpreted units often encompass several smaller intervals of different geology, this procedure ensured that the composite would be used to interpolate only relevant units. The implications of this coding are discussed in the next section.

(d) Interpolation

Interpolation search volume was 69 metres (225 feet) along strike, 46 metres (150 feet) along dip and 8 metres (25 feet) vertically. Composites were selected for interpolation on the basis of block geology being equivalent to composite geology coding. No composite less than 2.4 metres (8 feet) long was used in the interpolation to avoid biasing large blocks with small data points. One composite per drill hole was allowed for interpolation to minimize vertical averaging across banding in the stratiform deposit.

Interpolation was carried out in five passes starting with strict matching requirements, and a small search volume then gradually loosening the restrictions to interpolate values into blocks missed on previous passes without affecting the values already assigned. The search volume was enlarged to as much as 76 metres (250 feet) by 53 metres (175 feet) by 32 metres (105 feet) high.

Where more than one composite was available to estimate a given block, they were weighted isotropically by the inverse square of distance to the point being estimated as well as by the length of the composite. The length weighting of composites was carried out to avoid biasing large volumes of ore with assays representative of only a small amount of material. This procedure is different from all previous Faro deposit models except the little known F4, and probably explains why the F4 Model differs from other calculations in marginal benches.

The implications of the geological matching requirement during interpolation have not been tested; however, some statements can be made in light of the rock type - grade correlations and grade zoning described above. Since massive ores tend to be higher grade than disseminated ores and massive ores are more central and higher in the deposit than disseminated ores, there will be a tendency to average grade both by rock type and in space if there is not matching required. The use of matching will tend to make massive ores higher grade and disseminated ores lower grade than would be the case without matching. Because of the geometry of the deposit and its zoning, the higher grade ore will be more central and higher in elevation in the case of matching than it would be without matching. When a cutoff is applied to the block values computed through geological code matching, the tonnage above cutoff will be lower and the average grade higher than a reserve computed without a matching requirement; furthermore, the limits of ore will be higher in elevation and closer to the centre of the deposit. Because of these rock type and zoning characteristics of the Faro deposit, it is considered essential to use matching to produce an accurate picture of grade distribution. It is, however, inescapable that a reserve computed through geology matching will require a higher dilution factor than one computed without matching. It is, however, considered more realistic for dilution calculations to be made after the model is constructed rather than during interpolation when it will occur in uncontrollable and unpredictable fashion.

Specific gravity (SG) was treated as an assay and interpolated into blocks. This is due to the variability of SG by rock type and by grade as well as regional variations of SG within one rock type. Sulphide blocks outside of interpolation range were assigned an average SG based on rock type. The SG values used for interpolation were the pulp SG, not the whole rock SG, thus the value did not reflect porosity in the insitu intact material. A number of comparative SG tests have been carried out on Anvil district ore to determine the degree of overstatement of SG. In light of the results of these tests, block SG values were reduced by 5 percent for quartzose ore types or 10 percent for massive ore types to correct for porosity in the insitu whole rock. Previous Faro models have not had this correction made if pulp SG was used. Cyprus Anvil's practice in quoting model results as mill feed predictions was generally to reduce the grade by 5 percent, but not to adjust the tonnage; since the tonnage was already overstated by use of the pulp SG value, this was nearly the same as adding 5 percent dilution. Curragh's approach is to attempt to estimate the insitu tonnage and apply an appropriate dilution factor later rather than attempt to make two sets of corrections at once.

(e) Reserve Reporting

Reserves were computed for six ore types: 2A, 2BCD, 2CE, 2EF, 2GE and 2HE. Geological reserve computation was by a weighted average of all blocks in the model below topography that exceed a certain arbitrary lead and zinc content. Pit reserves are reported by computing the weighted average of all blocks above cutoff lying between two surfaces gridded on the same block grid as the block model. The two surfaces are an upper surface representing topography or the previous phase bottom, and a lower surface representing the current phase bottom. Blocks partly above or below the surface are multiplied by the fraction of the block that is between the surface elevations for that block when computing the weighted average.

3.2.5 Results

Geological reserves calculated from the FI Model are given in Table 3.2.5-1 below, along with some previous figures of a comparable nature. The Dome manual calculation covers a larger area than the F1 Model thus cannot be compared directly. Since nearest neighbour sectional calculations, such as the Dome one, tend to report higher grades than inverse distance squared interpolated models, the difference in grade may not be significant.

TABLE 3.2.5-1

GEOLOGICAL RESERVES FOR THE FARO DEPOSIT ZONE 3

[all values are undiluted and unadjusted]

Grade Category	Ore (tonnes)	Pb (%)	Zn (%)	Pb+Zn (%)	Ag (g/t)
FI Computer Model					
+5% Pb+Zn	22,793,000	3.28	5.14	8.41	41.5
4-5% Pb+Zn	3,770,000	1.72	2.78	4.50	27.3
+4% Pb+Zn	26,563,000	3.06	4.81	7.86	39.5
TOTAL METAL AT +4% Pb+Zn CUTOFF = 2,088,000 TONNES					
Dome Sectional Manual Calculation					
+4% Pb+Zn	29,251,000	3.13	5.03	8.16	40.8
TOTAL METAL AT +4% Pb+Zn CUTOFF = 2,387,000 TONNES					
Cyprus Anvil (F3 Computer Model ?)					
+4%()?) Pb+Zn	33,000,000	3.0	4.6	7.6	35.7
TOTAL METAL AT +4%(?) Pb+Zn CUTOFF = 2,508,000 TONNES					

3.2.6 Additional Work Required

In order to provide improved estimates of reserves for short-term production planning, it will be necessary to drill additional fill-in holes on an ongoing basis. To fill in the current pattern to 43 metres (141 feet) minimum on the main 43 metre (141 foot) spaced sections, will require 32 additional holes totalling 4,270 metres (14,000 feet). Of this total, 20 holes totalling 2,300 metres (7,500 feet) are in the AY and early BY phases and have higher priority.

Blasthole cuttings assays will continue to be used for day-to-day forecasting of mill feed grades and geological updating.

A new modelling technique has been devised that treats blocks not as homogenous single geologic entities but as the sum of two or more material types. This modelling technique will be implemented soon and should significantly improve the prediction of low grade and waste dilution.

3.3 GRUM GEOLOGY AND RESERVES

3.3.1 History

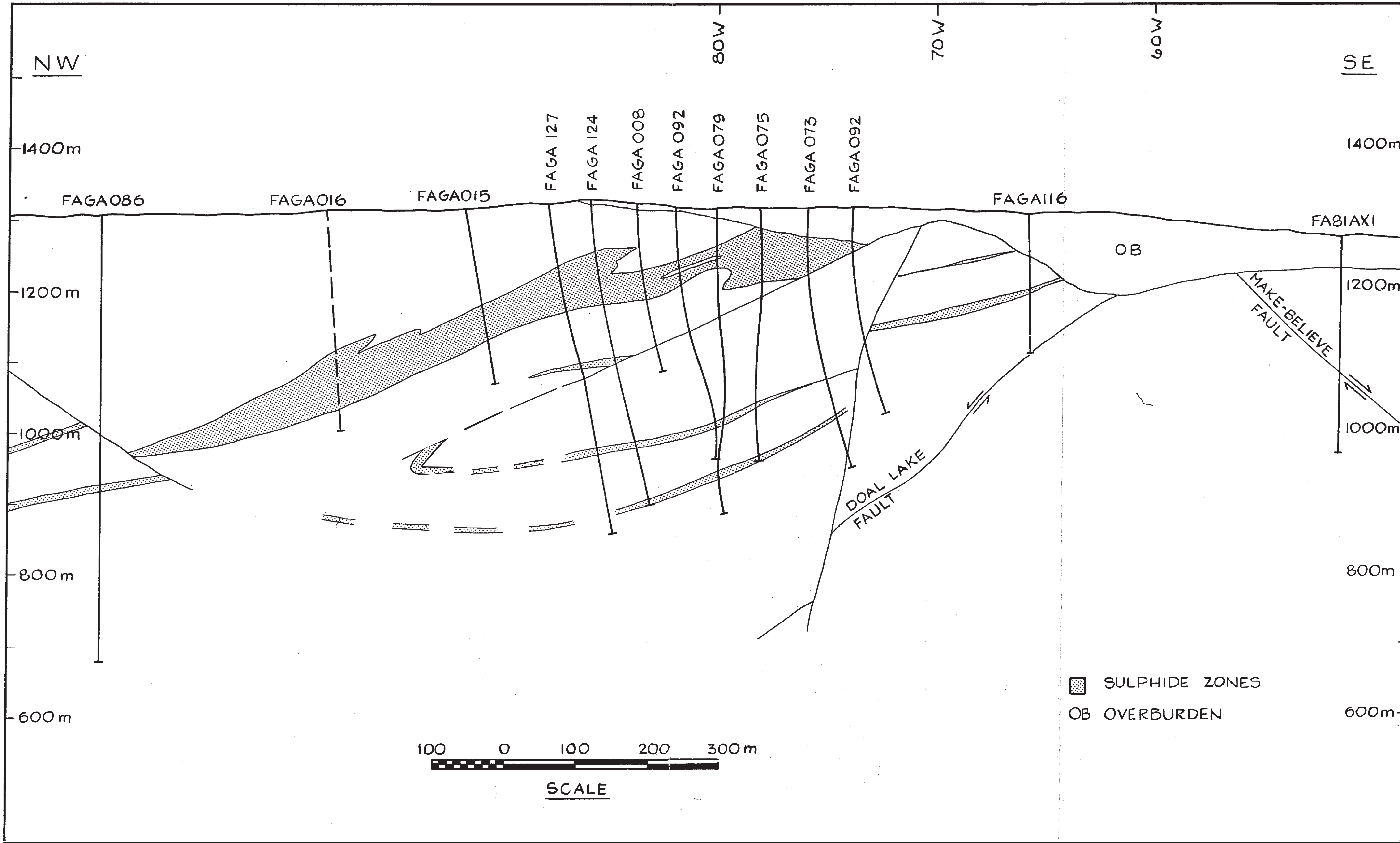
The Grum deposit was discovered in 1973 by AEX Minerals in joint venture with Kerr Addison Mines. Discovery was as the result of drill testing a gravity anomaly in an area down fold plunge from the Vangorda deposit along what was then a, as yet, poorly defined favourable trend.

Surface drilling in 1973 and 1974 indicated a significant deposit; in 1975 and 1976, an underground sampling and drilling program, along with further surface diamond drilling, was carried out to further define it.

Kerr Addison Mines sold the deposit, along with Vangorda and Swim, to Cyprus Anvil Mining Corporation in 1979. From 1980 to 1982, Cyprus Anvil drilled additional holes in and around the deposit and relogged all existing holes in it. All available sulphide intersections were resampled and reassayed at that time.

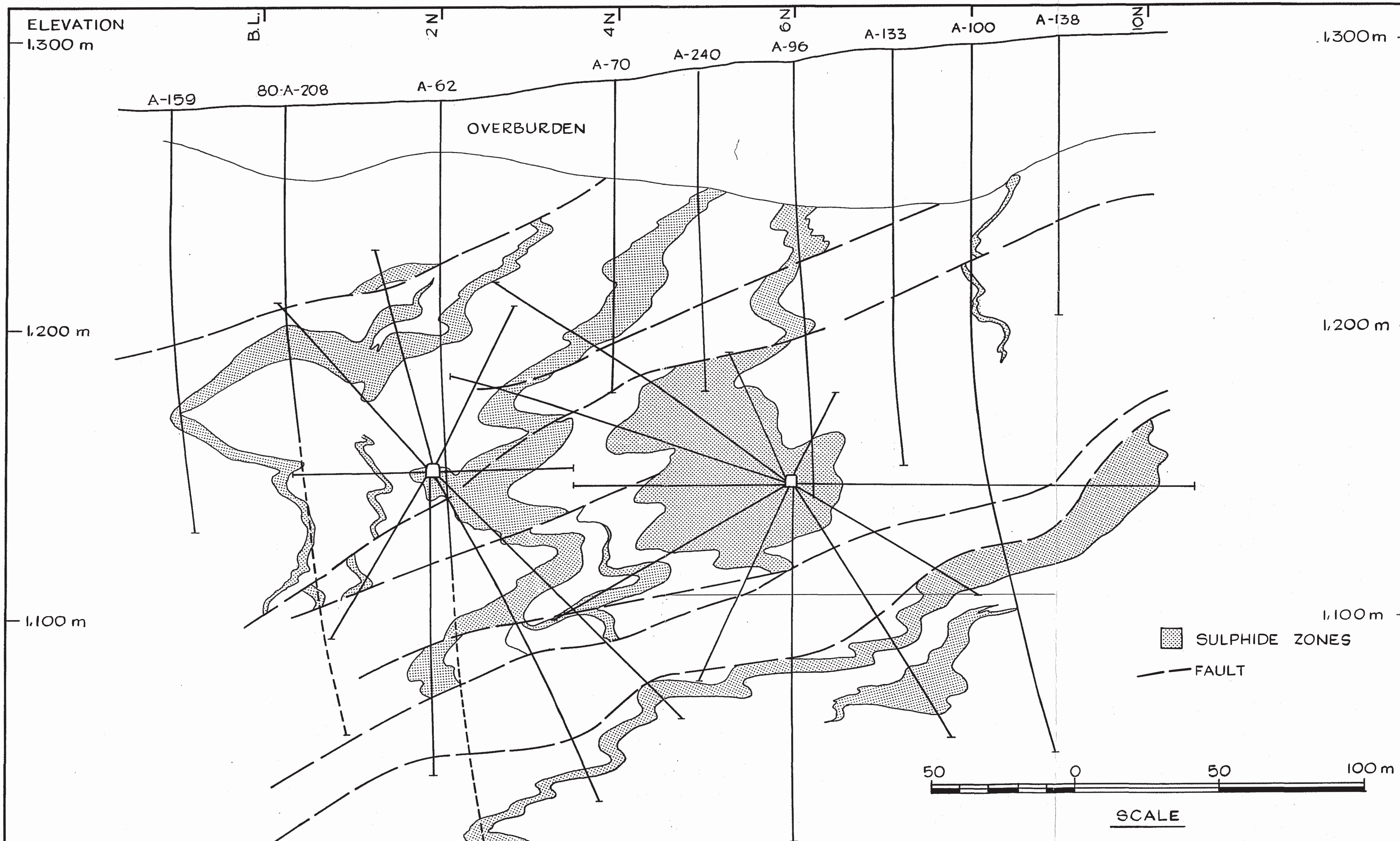
3.3.2 General Geology

Figures 3.3.2-1 and 3.3.2-2 are respectively longitudinal and cross-sectional view of the Grum deposit.



				SCALE 1:5000	DATE	CLIENT CURRAGH RESOURCES	FARO AREA DEPOSITS
				DESIGNED J.B.F.	MAR.87		GRUM DEPOSIT
				DRAWN Z.A.	MAR.87	LOCATION FARO, YUKON	LONGITUDINAL SECTION
ISSUED FOR TECHNICAL REVIEW	A	MAR87	ZA	CHECKED		KILBORN	PROJ. NO 3509-19 DWG. NO FIG. 3.3.2-1 REV. A
REVISIONS	NO	DATE	BY	APPROVED			

KE.B.C.9



			SCALE AS SHOWN	DATE
			DESIGNED JBF	MAR.87
			DRAWN ZA	MAR.87
ISSUED FOR TECHNICAL REVIEW	A	MAR.87	ZA	CHECKED	
REVISIONS	NO	DATE	BY	APPROVED	

CLIENT CURRAGH RESOURCES

LOCATION FARO, YUKON

KILBORN

FARO AREA DEPOSITS
GRUM DEPOSIT
X-SECTION (70W)

PROJ. NO 3509-19 | DWG. NO. FIG. 3.3.2-2 | REV. A

KEBC.9

(a) 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.

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 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 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 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 significantly affected it.

(b) 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 a 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.

(c) Surficial Geology

The subcrop of the ore deposit is covered by up to 100 metres of morainal material (tills) and better sorted galciofluvial 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. Dewatering in advance of stripping may help increase stability substantially as well as simplify operations in the pit.

(d) Ore 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, 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.

Grum ores have a finer grain size and more complex material intergrowth than Faro ores, necessitating finer grinding.

At a given lead-zinc cutoff 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 and zinc.

3.3.3 Drilling Definition and Information Base

The Grum deposit extends from section 52W in the southest to section 112W in the northwest. The deposit has been most densely drilled between 62W and 86W, and it is this portion of the deposit for which proven geologic reserves are reported.

Most of the deposit southeast of 88W has been drilled from the surface on at least a 61 metre by 30.5 metre (200 foot by 100 foot) pattern. Most surface holes are vertical.

Between sections 62W and 86W, the deposit has also been explored by 15,000 metres of underground drilling in fans from a pair of parallel inclines following the deposit trend. The strike length of the deposit examined from underground is 700 metres, underground workings, now flooded, total 2,900 metres.

The fans are most complete on even numbered sections (i.e. spaced 61 metres apart); on the odd numbered sections, inbetween, some fill-in drilling has also been carried out from underground. The overall density of drilling is on the order of 15 metres by 30 metres with local areas being much in excess of that.

In the southeast part of the deposit, additional fill-in drilling was carried out by Cyprus Anvil in 1980-82 from the surface to more closely define shallow ore for early production.

Total drilling at Grum is 67,200 metres of which 15,000 metres is underground drilling and 52,200 metres is surface drilling. Between 62W and 86W, there is a total of 53,600 metres of drilling in 372 drill holes (154 surface and 218 underground) of which 344 are used in the current model. The remainder, not included in the model, are underground holes that are at high angles to the geological sections and some short holes that did not intersect ore.

There are 9,000 samples in the Grum deposit assay database. Assay intervals generally average 1.5 metres in length and are normally cross-referenced to sulphide rock types. Assaying was completed by Kamloops Research and Assay Laboratories. All assays were determined using a set of Anvil District ore type standards for control. Rejects and N_2 purged pulps (by now somewhat oxidized) have been retained for additional analytical work.

For most samples, assays were determined for lead, zinc, copper and silver. For two-thirds of these samples, determinations were also made for insoluble iron, soluble (in hot concentrated HCL) iron, gold and pulp SG. For approximately 10 percent of the total samples from the property, assays were determined for lead, zinc and silver only. There are no barium or manganese assays available, nor systematic data for mercury, arsenic, cadmium or any other elements.

3.3.4 Reserve Calculation

(a) Introduction

To evaluate the Grum ore reserves, a new block model, the G8606 Model, was constructed by Curragh in June and July 1986. New reserves were calculated for the deposit in two portions: one from surface (1,336 metre maximum elevation) to 1,088.5 metre elevation, and a second from 1,088.5 to 868.0 metre elevation. This was due to software and hardware limitations brought about by a low bench height (4.5 metres) and correspondingly larger number of benches.

The PC Mine software package was used for grade interpolation and reserve calculation. The block geology and composites had been previously calculated using Mintec's Medsystem release 10. The results of the calculation are outlined in Section 3.3.5 of this Report.

Geological data will be continually updated during production to facilitate detailed bench-by-bench mine planning.

(b) Block Geology and Drill Hole Information

The reserves are calculated from a computer-based 3D Block Model based on a set of cross-sections produced by Cyprus Anvil geologists in 1982. The sections are parallel to the columns in the mine model and perpendicular to the elongation of the deposit. The cross-sections are 61 metres apart and provide the geologic control for the mine model. These sections were used for the sectional calculations by Cyprus Anvil in 1983, and for recalculations by Dome in 1984. All sections are available in a supporting document available at Curragh's Toronto and Whitehorse offices.

The logging and drill hole orientation data used were the most current available for the deposit. Most of the deposit is so densely drilled that there is little scope for variations in geologic interpretation to change the volume of the deposit significantly. However, the details of ore distribution on a given bench can change. For the purposes of annual projections of production, the current geological model is considered adequate.

The block geology was generated manually by laying a grid over the geologic sections and hand coding the rock types. Block dimensions are 4.5 metres high, 8.0 metres across deposit trend, and 15.0 metres along deposit trend. These block sizes provide a reasonable approximation of the complex structure of Grum. Codes were assigned by visual estimation of the most abundant rock types in the area. If the block was more than 50 percent sulphides, it was coded as a sulphide type, otherwise the block was considered waste. One code is assigned per block and that code is assumed to apply homogenously to the entire block. The sectional codes were plotted, checked and edited for each section. The blocks in overburden or air were assigned from interpolated grids representing topography and bedrock surfaces based on digitized contour maps. Blocks more than 50 percent above topography were coded as air, and more than 50 percent between topography and bedrock as overburden. One generic waste code was carried for the remainder of the model not coded as air, overburden or sulphide.

Sectionally assigned codes were applied to two columns of blocks on either side of the section. Since the Grum deposit plunges to the northwest, this assignment would create a stairstep appearance in long section. By coincidence, the diagonal of two blocks in long section is parallel to the deposit plunge, thus a "plunge correction" was made by raising the first column of blocks one level (southeast of the section) and lowering the fourth column of blocks one level (northwest of the section). The second and third columns of blocks are kept the same. This has no affect on deposit reserves. All this block coding was carried out outside PC Mine either using the Mintec Medsystem release 10 software package or by manual means. The block codes were reformatted to suit PC Mine, then imported.

DDH data and composites were imported directly to PC Mine from reformatted Mintec output files.

(c) Composite Calculation

Composites were calculated by Medsystem on a 4.5 metre bench basis for holes steeper than 45 degrees. For holes shallower than 45 degress, composites were based on 4.5 metre horizontal intervals from the drill hole collar. Composite intervals can range from 4.5 metres to 6.5 metres depending on borehole orientation. Waste intervals less than one-half bench height (2.25 metres) were considered internal waste and included in the composite interval. Intervals greater than one-half bench height were external waste and not included. This procedure was intended to accurately represent the grade of ore in blocks of all settings, but did not automatically include all dilution in marginal composites. Such dilution adjustments must be made separately and are addressed in Section 4.0.

Drill hole assay data were clipped to the 95th percentile levels to avoid assigning unusually high assays to large blocks. These levels are listed in Table 3.3.4-1. Intervals with no measured SG were assigned an SG depending on rock type as listed in Table 3.3.4-2. These were based on statistical analysis of the measured data.

Composite calculation was carried out for the mineralized sections using this modified assay data and weighting by length and specific gravity. The length of the composite within a mineralized band was carried as well as the values since only the length of the mineralized part of an interval was composited.

The final composites used were as short as 0.1 metres but only if a small part of a mineralized band was within a composite interval. After calculation, the composites were also clipped to the 95th percentile level as outlined in Table 3.3.4-3. Every composite was manually checked against the cross-sections to ensure that the codes applied to sectional units were consistent with the composite codes.

(d) Interpolation

The geostatistical analysis carried out was not adequate to use kriging as an interpolation method, thus inverse square distance weighting was used following precedent set at Faro. The search volume was an ellipsoid with major axis of 150 metres parallel to the deposit plunge and with diameter in cross-section of 106 metres. The ellipsoid centered on the block being interpolated, thus the maximum distance a sample can be used to weight a block is 75 metres. A horizontal and vertical anisotopy of 1.41 was used. This results in samples along trend being weighted twice as heavily as those across trend (with an anisotopy of 1.41, a sample 53 metres across strike is weighted the same as one 75 metres along plunge because the sample is treated as if it is 53 by 1.41 or 75 metres from the block centre, once this apparent distance is squared the factor of 2 [= 1.41^2] appears). A search volume radius much larger than the range was used in order to ensure that the blocks in the less intensely drilled part of the deposit would get grades assigned.

TABLE 3.3.4-1

MAXIMUM PERMITTED ASSAY VALUES AND SG VALUES

Pb	11.0 %
Zn	20.0 %
Ag	175.0 g/tonne
Au	2.8 g/tonne
Cu	0.4 %
Pulp SG	5.0

TABLE 3.3.4-2

SG VALUES ASSIGNED FOR EACH MAJOR ORE TYPE

IN CASE OF MISSING ANALYTICAL DATA

Ore Type	SG
4A0	3.23
4A4/4AE	3.31
4B	3.00
4C	3.45
4D	3.53
4E	4.32
4G	4.42
4H	3.86
4J	3.87
4K	3.84
4L0	3.11
4L4/4LE	3.29

TABLE 3.3.4-3

MAXIMUM PERMITTED ASSAY AND SG

FOR DDH COMPOSITES

Pb	9.00 %
Zn	17.00 %
Ag	150.00 g/tonne
Au	2.30 g/tonne
Cu	0.34 %
Pulp SG	4.80

In the model, the most important test that a sample within the search volume must pass before being used to interpolate a block is the equivalence of geological codes. This is important in Anvil District deposits because of the strong ore type zoning and coupled grade zoning. The implications of this restrictive code matching for use of the model are very significant.

The minimum number of composites required to interpolate a block was set at two, the maximum at eight. The minimum limit was set to avoid the possibility that one very short composite could bias an entire block, the possibility that two very short composites could be used cannot be excluded; however, in most cases, a very short composite would be near a longer one.

Specific gravity was interpolated in the same fashion as assays. Uninterpolated sulphide blocks were assigned an SG of 2.7. Pulp SG was reduced by 5 percent in the final model in order to convert the pulp SG to insitu whole rock SG. The average SGs for the Grum deposit are given with the geological reserves in Tables 3.3.4-4 and 3.3.4-5.

(e) Geological Reserve Reporting

Reserves were calculated by the weighted average of block values for all blocks that exceed an arbitrary percent lead plus percent zinc cutoff value. Geologic reserves are the sum of all blocks in the model below topography, but irrespective of any pit outlines.

Since there are two partial Grum models to cover the whole deposit, the results from the two models were combined by a spreadsheet.

Sectional geologial reserves were also computed for each cross-section by reporting the reserves within a plan view polygon representing the area of influence of each section ($\pm$ 30.5 metres from the section line). The sectional reserves were needed to compare to previous manual calculations.

3.3.5 Results

(a) Geological Reserves

The geological reserves calculated for Grum are summarized in Table 3.3.4-4 for the two constituent models and for the entire deposit. Sectional geological reserves are summarized in Table 3.3.4-5 for the entire deposit.

The Champ Zone southeast of section 62W and separate from Grum, is estimated to contain an additional 1.7 million tonnes of geological reserves averaging 3.5 percent lead, 4.3 percent zinc, and 46 grams per tonne of silver. These figures are based on sectional calculations and quoted at a 4 percent lead-zinc cutoff. Northwest of 86W and part of the Grum deposit, preliminary drilling indicates that there may be an additional 5 to 10 million tonnes of deep mineralization.

(b) Reliability of Reserves

The current level of geostatistical knowledge of the deposit has precluded determination of block estimation variance. Thus, quantifiation of overall deposit variance is not yet possible. However, the density of drilling is sufficient to limit the possibility of major changes in deposit volume due to variance in interpretation. Volume ranges of plus or minus 10 percent would be possible.

TABLE 3.3.4-4

G8606 MODEL GEOLOGICAL RESERVES FOR THE TWO CONSTITUENT MODELS AND FOR THE ENTIRE DEPOSIT

Grade Category	Volume (bcm)	SG	Ore (tonnes)	Lead (%)	Zinc (%)	Pb+Zn (%)	Silver (g/t)	Gold (g/t)
Above Grum (1336.0 m to 1088.5 m)								
+ 6%	4,677,480	3.37	15,765,300	3.84	6.50	10.34	64.0	0.92
4 - 6%	1,942,920	3.12	6,065,990	1.90	3.10	4.99	33.0	0.77
+ 4%	6,620,400	3.30	21,831,290	3.30	5.56	8.85	55.4	0.88
Under Grum (1088.5 m to 868.0 m)								
+ 6%	1,880,820	3.75	7,057,100	4.04	6.36	10.40	68.5	1.18
4 - 6%	522,720	3.37	1,760,770	2.12	2.76	4.88	35.5	0.99
+ 4%	2,403,540	3.67	8,817,870	3.66	5.64	9.30	61.9	1.14
Total Deposit (1336.0 m to 868.0 m)								
+ 6%	6,558,300	3.48	22,822,400	3.90	6.46	10.36	65.4	1.00
4 - 6%	2,465,640	3.17	7,826,760	1.95	3.02	4.97	33.6	0.82
+ 4%	9,023,940	3.40	30,649,160	3.40	5.58	8.98	57.2	0.95

TABLE 3.3.4-5

GEOLOGICAL RESERVES BY CROSS-SECTION FROM G8606 COMPUTER MODEL, 1986

TOTAL DEPOSIT - 4% Pb+Zn CUTOFF GRADE

Sec.	Volume (bcm)	SG	Ore (tonnes)	Lead (%)	Zinc (%)	Silver (g/t)	Gold (g/t)	Pb+Zn (%)	Total Metal (tonnes)
86W	684,180	3.09	2,116,210	2.60	4.99	46.0	0.44	7.59	160,592
84W	584,820	3.16	1,849,610	2.80	4.90	47.6	0.60	7.70	142,432
82W	787,860	3.40	2,680,540	3.19	5.55	56.5	0.84	8.73	234,069
80W	1,297,080	3.35	4,340,270	3.10	4.85	51.8	1.05	7.95	345,081
78W	975,780	3.27	3,195,500	3.16	5.50	54.7	1.04	8.66	276,762
76W	907,200	3.36	3,048,860	3.49	5.80	58.6	1.07	9.29	283,135
74W	1,013,040	3.45	3,498,640	3.72	6.20	62.3	1.05	9.92	347,148
72W	748,440	3.45	2,584,690	3.64	5.77	61.2	1.02	9.41	243,256
70W	694,980	3.52	2,449,940	3.86	6.57	63.6	1.04	10.43	255,415
68W	355,320	3.70	1,314,260	4.10	6.27	66.7	1.00	10.37	136,297
66W	442,260	3.69	1,632,190	4.14	5.75	66.7	0.99	9.90	161,507
64W	301,860	3.61	1,091,070	3.79	5.52	61.2	0.99	9.30	101,486
62W	178,200	3.69	657,640	3.39	5.03	54.8	1.10	8.42	55,347
	8,971,020	3.40	30,459,410	3.41	5.60	57.3	0.95	9.00	2,742,527

Possible variance due to calculation methods is also not quantified, but changes of a few percent would be possible through adjustment of interpolation parameters such as anisotopy, weighting scheme and range, and the geology matching scheme.

How well the reported geological reserves will relate to mill feed depends on the type and degree of dilution allowed. The need to use a higher dilution than the historic 5 percent used for Anvil District deposits in the past is brought on largely by the restrictive geology matching used during interpolation. This matching was not carried out previously, thus the models tended to dilute themselves in unpredicable ways during interpolation. The modelling technique was changed in order to present a more accurate portrayal of grade distribution, especially with respect to grade averaging into otherwise barren sulphides, particularly footwall sulphides. As in the case for Faro, dilution will be specific to each ore zone, taking into account the complexity of the geological structure and the nature of the surrounding waste rock.

3.3.6 Additional Work Required

The complex geology of the Grum deposit and its multi-horizon nature creates some difficulty in producing an interpretation that is consistent from section to section. Horizons are difficult to distinguish from one another in drill core, thus the iterative process of section by section interpretation and comparison must be used. The current interpretation requires further refinement in this regard. Additionally, 60.5 metre spaced sections do not fully utilize of all available drill hole information. A revised detailed interpretation, based on thirty metre spaced sections, is being prepared for short range production planning.

The Champ Zone will be included in the Grum deposit models since mining that area might influence the overall pit economics by lowering the main pit exit. Considering both zones together offers the possibility of enhancing the economics of both.

Due to the wide spacing of drill holes in most Anvil District deposits, geostatistical analyses have been difficult. At Grum, the drill pattern is dense enough to produce meaningful analysis once the basic geologic data is organized into the thirty metre spaced interpretation. Such analysis should allow interpolation methods other than the simple inverse square method to be used.

Further geotechnical data is required at Grum in order to design the final pit slopes. A program of oriented core drilling will be undertaken. These holes will also sample the overburden to evaluate its clay content and hydrogeology. If the overbruden is amenable to dewatering and can be kept dry, then pit slopes steeper than those currently designed may be possible.

The foundation conditions of the proposed waste dump sites will be investigated.

It may be necessary to carry out fill-in drilling to further delineate and detail the reserves in early production areas. This work could be coordinated with the collection of metallurgical samples.

3.4 VANGORDA GEOLOGY AND RESERVES

3.4.1 History

Vangorda was the initial discovery in the Anvil Range. The deposit was drill tested from 1953 to 1955 by Prospector Airways, a predecessor to Kerr Addison Mines. This drilling showed a significant deposit existed, but a production decision was not warranted at that time. Minor additional drilling was carried out by Kerr Addison, largely for metallurgical sampling. The deposit was sold to Cyprus Anvil in 1979. Cyprus Anvil geologists examined the available drill core and concluded that it would be necessary to redrill the deposit to provide adequate material to re-evaluate it.

In 1979, the portion of the deposit from 2W to 12E was redrilled with NQ core holes. Scattered core holes were put down in the southeast part of the deposit. Because of anticipated poor recoveries in this area, it was judged advisable to drill this part of the deposit with rotary methods. This fill-in drilling was carried out in 1981. Since 1981, no additional drilling has been carried out.

3.4.2 General Geology

Figure 3.4.2-1 is a cross-sectional view of the Vangorda deposit.

(a) 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. 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.

(b) Structure

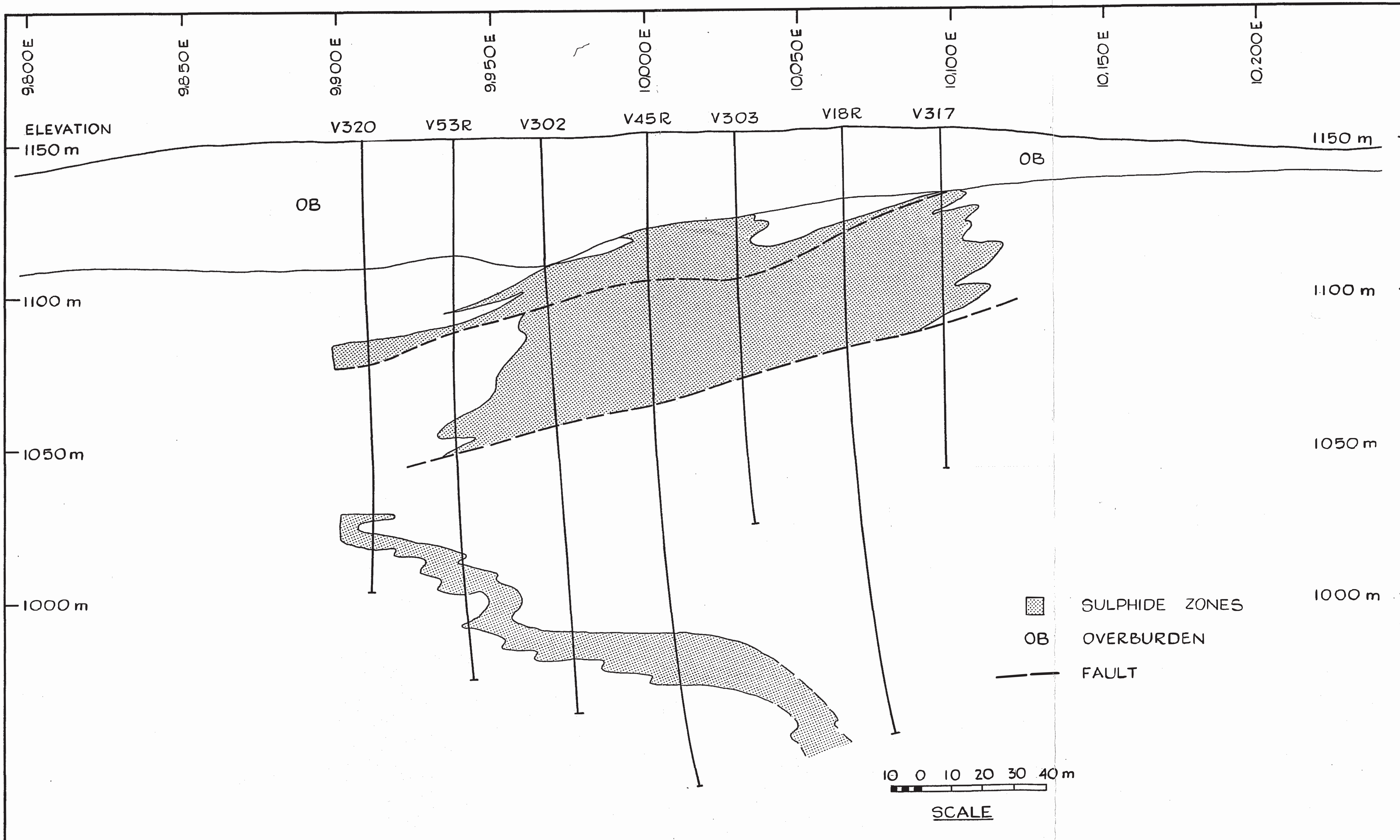
The Vangorda deposit occurs in the hinge of a large second phase fold. Overall, the deposit has the shape of a reclining "M" or a "3" in cross-section; however, there is some uncertainty in the details of fold morphology. The deposit is elongate in the northwest-southwest direction parallel to fold axes. 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 a 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 of small gouge zones that parallel it.

(c) Deposit Geology

The deposit is quite shallow; in most places, subcroping 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.



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			DRAWN Z.A.	MAR.87
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REVISIONS	NO	DATE	BY	APPROVED

CLIENT CURRAGH RESOURCES

LOCATION FARO, YUKON

KILBORN

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

PROJ. NO 3509-19 | DWG. NO. FIG. 3.4.2 - 1 | REV. A

KEBC.9

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 semi-massive 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 (see Table 3.4.2-1) The quartzite is similar to the semi-massive 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 (see Table 3.4.2-1).

TABLE 3.4.2-1

GEOLOGICAL RESERVES FOR VANGORDA DEPOSIT FROM V8607 MODEL

NO DILUTION

Rock Type		% of Ore Type	Tonnes (x1000)	Density (tn/bcm)	Pb (%)	Zn (%)	Pb+Zn (%)	Ag (g/t)	Au (g/t)
PLUS 6% Pb+Zn									
4A	1	7.1	382.33	2.81	3.17	4.66	7.83	39.74	0.68
4C	2	0.0	1.42	3.50	6.40	1.01	7.41	57.51	1.77
4EC	3	0.4	21.41	3.65	2.97	3.79	6.75	46.65	0.80
4E	4	0.1	4.16	3.43	2.57	4.12	6.69	46.50	0.63
4EG	5	90.8	4,917.88	3.96	4.51	5.83	10.34	64.03	0.75
4EH	6	1.7	90.23	3.65	6.82	5.05	11.87	83.15	0.54
TOTAL		100.0	5,417.43	3.88	4.45	5.72	10.17	62.55	0.74
4% TO 6%									
4A	1	58.6	1,194.40	2.83	1.86	3.02	4.88	26.48	0.46
4C	2	9.5	194.12	3.27	2.34	2.08	4.42	25.82	0.79
4EC	3	14.1	286.99	3.68	2.28	2.40	4.68	33.18	0.85
4E	4	10.2	207.58	3.76	1.90	2.87	4.77	37.94	0.14
4EG	5	6.7	137.28	3.75	2.47	2.76	5.23	36.96	0.76
4EH	6	0.9	19.13	3.78	2.43	2.95	5.38	34.31	0.41
TOTAL		100.0	2,039.51	3.16	2.01	2.81	4.83	29.30	0.53
MINUS 4%									
4A	1	36.2	3,721.08	2.92	0.90	1.44	2.34	12.48	0.33
4C	2	42.8	4,393.21	3.27	0.79	1.00	1.79	14.50	0.65
4EC	3	17.9	1,842.51	3.67	1.33	1.34	2.67	21.57	0.94
4E	4	2.9	299.17	3.65	1.50	1.40	2.90	17.48	0.28
4EG	5	0.1	5.88	3.23	0.67	0.86	1.53	11.66	0.65
4EH	6	0.1	9.09	2.99	0.39	0.49	0.88	5.95	0.08
TOTAL		100.0	10,270.94	3.23	0.95	1.23	2.18	15.12	0.57
TOTAL DEPOSIT (ALL GRADES)									
4A	1	29.9	5,297.81	2.89	1.28	2.03	3.31	17.60	0.38
4C	2	25.9	4,588.75	3.27	0.86	1.04	1.90	15.00	0.66
4EC	3	12.1	2,150.90	3.67	1.47	1.51	2.98	23.37	0.93
4E	4	2.9	510.92	3.69	1.67	2.02	3.69	26.03	0.23
4EG	5	28.5	5,061.05	3.96	4.45	5.74	10.19	63.24	0.75
4EH	6	0.7	118.46	3.62	5.62	4.36	9.98	69.33	0.48
TOTAL		100.0	17,727.88	3.42	2.14	2.78	4.92	31.24	0.62
PLUS 4%									
4A	1	21.1	1,576.72	2.83	2.18	3.42	5.60	29.69	0.51
4C	2	2.6	195.54	3.27	2.37	2.07	4.44	26.05	0.80
4EC	3	4.1	308.40	3.68	2.32	2.50	4.82	34.11	0.84
4E	4	2.8	211.75	3.75	1.91	2.90	4.81	38.11	0.15
4EG	5	67.8	5,055.17	3.96	4.45	5.74	10.20	63.30	0.75
4EH	6	1.5	109.36	3.67	6.05	4.68	10.74	74.60	0.52
TOTAL		100.0	7,456.94	3.68	3.78	4.92	8.71	53.46	0.69

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 observed by the writer 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.

3.4.3 Drilling Definition and Information Base

In the portion of the deposit from 2W to 12E, there are 53 diamond drill holes in a 60 metre by 30 metre pattern. All holes are vertical and all are NQ diameter. Collar locations have been surveyed, checked and appear accurate. Assay intervals are 1.5 to 2 metres long and, where possible, are confined to one rock type.

In the remainder of the deposit, there are forty-five rotary holes and eight recent NQ and HQ diamond drill holes. The pattern is also 60 metres by 30 metres. There appears to have been a sampling problem with some of the rotary holes that resulted in unuseable assays which have cast doubt on the assay results from the remainder of the rotary holes. For this reason, where a Prospector Airways hole was also available and good recovery was obtained, the information from the older hole has been used in preference to that from a rotary hole.

For most assay intervals, copper, lead, zinc, silver, gold, soluble (in hot HCl) iron, insoluble iron, and barium assays were determined. Old diamond drill holes were only assayed for lead, zinc and silver. The newer assays are the same as those described for Grum. The lead assays from the older holes are suspect because no correction for barium interference was made.

3.4.4 Reserve Calculation

(a) Introduction

The Vangorda deposit reserves were generated using a computer-based 3D Block Model, the V8607 Model, constructed by Curragh during June and July 1986. The PC Mine software package was used for all stages of model construction.

(b) Block Geology and Drill Hole Information

The model was generated using geologic control provided by 60 metre spaced cross-sections. The cross-sections were newly interpreted for this model in order to provide a uniform structural concept for the entire deposit. The assumptions used in cross-section construction were:

i) The deposit must fit the overall structural setting of this part of the Vangorda Plateau as defined by surface mapping and deep drilling nearby. This work shows that the deposit is in the hinge region of a large recumbent second phase fold.

ii) Stratigraphic facings implied by Anvil cycles were followed wherever possible. Of particular importance was the barren pyritic quartzite which shows strong indications of being a footwall facies.

iii) Symmetric suphide intersections were taken to imply folds.

iv) The dominant deformation eposide would be the second phase implying a deposit shape similar to Grum, but different from Faro.

Assumption i) and ii) together require highly attenuated folds in order to produce a consistent interpretation. The resulting fold shapes are consistent with the deformation style observed in nearby outcrops on a small scale and with the inferred shapes of more closely controlled folds at Grum. All sections are included in supporting documents at Curragh's Toronto and Whitehorse offices.

The density of drilling at Vangorda is sufficient to produce geological interpretations and a global estimate of reserves with reasonable confidence (changes in volume of plus or minus 10 percent to 15 percent may be possible due to interpretation).

The rotary drilling results in the southeast part of the deposit are more difficult to interpret because the rock type logging was not as precise or reliable as the diamond drilled part of the deposit. As a result, there is more uncertainty in the geologic sections of this part of the deposit.

For computerized reserves calculation, the cross-sections were digitized and block assignments were made. The logic used for assignment of the geological rock type code for each block was based on the geology at the centre of that block. The block size used was 4.5 metres high, 4.5 metres across strike, and 10.0 metres along strike. This was essentially the smallest block size practical that allowed maximum resolution of geologic detail. Rows of blocks were arranged in the model parallel to the geologic sections. The same geological codes were applied to three rows of blocks on either side of a section, for a strike length of 60 metres, but each 10 metre segment was treated separately for purposes of interpolation. There are 45 levels in the model extending from 1209.5 to 1007.0 metre elevation (1979 Cyprus Anvil Datum).

Drill hole data was imported as ASCII files into the mine modelling system from the Hewlett Packard HP3000 based database for Vangorda drill hole information.

(c) Composite Calculation

The assays were composited on the basis of geology with an attempt to conform to a 4 to 6 metre length. This composite coding ensured that an interpreted geologic unit would only be assigned an assay from a length of drill core that was actually used to define the unit. In some cases, these defined units actually contained two or more different geological types, but the single assignment was necessary in order to produce units that would be reasonable from a mining point of view with the minimum number of necessary ore types.

In general, these mixed types are related. The minimum composite length was about one metre. Internal waste was included in the composites at a zero percent assay value. Unlike most previous models, except the F4, bench composites were not used.

Assays were weighted by length of the sample, but not by its specific gravity (SG). Since composite intervals tend to be restricted to one ore type, SG weighting would have relatively little effect. There was no "clipping" of assay values to arbitrary maxima prior to compositing calculations.

(d) Interpolation

Interpolation was carried out in essentially the same fashion as the Grun G8606 Model described previously. Experimental variograms calculated from these composites showed only nugget effect due to the relatively large drill hole spacing. For this reason, the anisotopy developed for Grum was used at Vangorda with a slightly larger search volume. Search volume parameters were:

Elongation	-	Model 090^{o} (i.e. along deposit trend)
Plunge	-	-10^{o} (i.e. 10^{o} northwest plunge)
Horizontal Anisotopy	-	1.41
Vertical Anisotopy	-	1.41
Maximum Range	-	90 m (1.5 x section spacing)
Minimum Number Composites Used for a Block	-	2
Maximum Number Composites Used for a Block	-	8

Composites were weighted by the inverse square of the distance from the composite centre to the block centre. A composite could be used to interpolate a block only if its geologic code matched the code of the block being estimated.

SG was treated as an assay and interpolated into blocks based on measured pulp SG of composite intervals. These SGs were determined by air pycnometer on assay pulps, and consequently do not account for void space. For tight quartzose ore types, these SGs are high by about 5 percent and for vuggy, porous, massive ore types, they are high by as much as 10 percent based on comparison of whole rock and pulp SGs carried out on Grum, Faro and Dy ores. In order to recast the SG data in terms of whole rock values, the pulp SG was reduced by 5 percent in the final model.
Geological reserves were computed by weighted average of block values between certain lead plus zinc grades for all levels below topography.

3.4.5 Results

(a) Geological Reserves

Geological reserves at 4 percent and 6 percent lead plus zinc cutoff grades are summarized in Table 3.4.2-1. The results quoted are for all mineralization regardless of potential pit outlines. There has been no allowance made for dilution or mining recovery in the geological reserves.

(b) Comparison of Geological Reserves

Two manual calculations have been carried out for the entire Vangorda deposit. Both are based on the old drilling and assays for the deposit. Table 3.4.5-1 below compares these results to the present computer model.

TABLE 3.4.5-1

EARLIER GEOLOGICAL RESERVE ESTIMATES FOR THE ENTIRE DEPOSIT COMPARED TO THE V8707

	Tonnes (x1000)	Density (tn/bcm)	Pb (%)	Zn (%)	Pb+Zn (%)	Ag (g/t)	Au (g/t)	Total Metal (tonnes)
PROSPECTORS AIRWAYS / CHISHOLM ET AL [Extent unknown - may be 4W to 40E]								
High Grade (+ 4%?)	8,528	4.0 ?	3.16	4.96	8.12	60.33	0.69	692,474
Low Grade	11,431	------------not determined------------						
TOTAL DEPOSIT	19,959							
KERR ADDISON MINES / PAXTON [Recalculated to 3W to 29E]								
High Grade (+ 4%?)	6,942	4.0 ?	----nd----		8.67	----nd-----		601,871
Low Grade	9,139	3.5 ?	----not determined-----------					
TOTAL DEPOSIT	16,081							
THIS CALCULATION [3W to 29E]								
High Grade (+ 4%)	7,457	3.68	3.78	4.92	8.71	53.46	0.69	649,224
Low Grade (- 4%)	10,271	3.23	0.95	1.23	2.18	15.12	0.57	
TOTAL DEPOSIT	17,728	3.42	2.14	2.78	4.92	31.24	0.62	

The oldest one by Prospector Airways is based on the triangular calculation method. The comparison to the present model is reasonably close. The large variance in lead grade is expected as it is known that the older lead assays were low probably because a correction for barium was not made. The areal extent of this calculation is not known, but it may extend as far as 40E. The amount of ore in the far southeast sections (between 30E and 40E) is slight but this detracts from the comparison since the V8607 Model only extends to 29E.

The Kerr Addison reserves are based on a sectional calculation by J. Paxton using sections between 4W and 30E in 1966. These sections show unconnected enechelon pods of ore, thus the tonnage should be lower than the current model where the assumptions outlined above imply the pods should be connected into fold patterns.

(c) Reliability of Geological Reserves

As a check on model computations, the areas of geological units output during digitizing of unit outlines was used to compute the volume of the overall deposit at the zero percent combined cutoff grade. The average SGs of the ore types were used to calculate tonnes of mineralization. This calculation gave a total deposit tonnage of 18,921,000 tonnes compared to 18,660,930 from the block model (all these tonnages are prior to SG reduction), or compared to 18,732,000 if the 97 uninterpolated blocks are included.

During test interpolation, increasing the degree of anisotopy tended to increase the spread in extreme block values. The effect on the average is not known but, above a given cutoff, the average grade would probably increase.

The reliability of the Vangorda geological reserves is within the range of plus or minus 10 percent to plus or minus 20 percent.

In calculating Vangorda mill feed, mining dilution must be considered in a manner similar to that described previously for Grum in Section 3.3.5 (b). Mining recovery and dilution are quantified in Section 4.3.1.

3.4.6 Additional Work Required

More diamond drilling will be required at Vangorda to facilitate detailed mine production planning. In particular, fill-in holes will be necessary where previously drilled holes deviated from their planned courses and left information gaps, in the area explored by rotary drilling, and in likely areas of oxidation.

It is likely that a staggered triangular pattern will be diamond drilled to supplement the information provided from the current 100 ft by 200 ft rectangular pattern.

4

4.0 PIT DESIGN AND MINING RESERVE

4.0 PIT DESIGN AND MINING RESERVE

4.1 GENERAL

There are three deposits which are scheduled to be mined by open-pit mining methods. These include the Faro, Grum and Vangorda deposits. Both the Faro and Grum deposits have additional underground reserves.

4.2 FARO PIT

The pit design and mining plan used in this plan is a modification of that presented in the 1985 plan prepared by Kilborn. This mining plan has the Faro pit being mined in four phases from northwest to southeast, these phases being referred to as AY, BZ, CZ and DY. Two additional subphases, known as JB and the Ramp Zone, will have been mined out by the time that this mine plan starts.

4.2.1 Faro Pit Reserves

Open-pit mineable reserves have been calculated from the FI Model. Reserves are calculated from mining blocks which are laid out on bench plans. The area of each mining block is digitized and the block reserve is then calculated.

Pit geometry is as discussed in Section 5.6.

Waste quantities have been categorized into three types: "Sulphide waste" is composed of sulphides grading less than 4 percent combined lead and zinc, plus any material grading greater than 4 percent but not recovered as ore; "Calc silicate waste" is quantified separately; and all other waste is quantified under the generic term "waste".

Ore quantities have been categorized into the three main ore types, 2A, 2H and 2BG, and have been further subdivided into three grade intervals, 4-5 percent, 5-6 percent and plus 6 percent.

Mining reserves are adjusted to reflect a 95 percent mining recovery and a 10 percent waste dilution. Recovery is defined as the tonnes of ore mined, expressed as a percentage of insitu tonnes. Dilution is defined as the tonnes of waste (at zero grade) mined with the ore and reported as ore, expressed as a percentage of recovered ore tonnes.

Adjustments to the ore quantities are as follows:

Recovered ore tonnes = (Recovery) x (insitu ore tonnes)

Dilution tonnes = (Recovered ore tonnes) x (dilution)

Mined ore tonnes = (Recovered ore tonnes) + (dilution tonnes)

= (Recovered ore tonnes) x (1 + dilution)

= (Insitu ore tonnes) x (recovery) x (1 + dilution)

The dilution is assumed to contain zero metal:

Recovered ore grade = Insitu ore grade

Dilution grade = 0.0

Mined ore grade:

$$= \frac{\text{(Recovered Ore Grade) x (Recovered Tonnes)}}{\text{(Recovered Tonnes + Dilution Tonnes)}}$$

$$= \frac{\text{(Insitu Ore Grade) x (Recovered Tonnes)}}{\text{(Recovered Tonnes) x (1 + Dilution)}}$$

$$= \frac{\text{(Insitu Ore Grade)}}{\text{(1 + Dilution)}}$$

Cutoff grades are defined with respect to the insitu ore grade, not the mined ore grade.

Table 4.2.1-1 summarizes the April 1, 1987 mining reserves, based on 4 percent and 6 percent cutoffs.

TABLE 4.2.1-1

MINING RESERVES - APRIL 1, 1987 STATUS

	4% Cutoff	6% Cutoff
Tonnes	20,582,530	14,380,466
% Pb+Zn	7.12	8.24
% Pb	2.81	3.26
% Zn	4.31	4.98
Ag g/t	35.4	39.2
Au g/t	0.10	0.09

Table 4.2.1-2 (see page 4-4) lists the mining reserves by category.

In addition to the pit reserves above, a pit stockpile of previously mined ore exists. These stockpile reserves are listed in Table 4.2.1-3 below:

TABLE 4.2.1-3

STOCKPILE RESERVES

	4% Cutoff
Tonnes	402,521
% Pb+Zn	5.01
% Pb	1.87
% Zn	3.14
Ag g/t	26
Au g/t	0.06

TABLE 4.2.1-2

RESERVE LISTING: 1987 BUDGET RESERVES (10% DILUTION, 95% RECOVERY

				HEAD GRADE					CONCENTRATE						% RECOVERY			
Mat'l.	Grade	Volume BCY	Tonnes	% Pb+Zn	%Pb	%Zn	Ag g/t	Au g/t	Pb DMT	%Pb	Ag g/t	Au g/t	Zn DMT	%Zn	Pb	Zn	Ag	Au
Waste		21204410	43051330															
Calc-Silic		1842640	3781097															
Sulph-Waste		2676316	6815670															
2BG	4.0-5.0	905917	2370695	4.11	1.68	2.43	26	.12	43399	62.00	654	.00	81268	51.00	67.6	72.0	46.9	.0
2BG	5.0-6.0	973573	2603726	4.98	1.98	3.01	28	.11	58012	62.00	616	.00	114876	51.00	69.8	74.9	48.6	.0
2BG	6.0+ .0	4289278	12100230	8.16	3.23	4.94	38	.10	484700	62.00	514	.00	957464	51.00	76.9	81.7	54.6	.0
2H	4.0-5.0	9599	24349	4.23	1.91	2.33	35	.09	520	60.00	747	.00	789	50.00	67.2	69.6	46.2	.0
2H	5.0-6.0	51708	146066	4.93	1.91	3.02	30	.06	3133	60.00	646	.00	6427	50.00	67.3	73.0	46.0	.0
2H	6.0+ .0	534271	1502319	8.61	3.59	5.02	49	.06	68539	60.00	580	.00	120401	50.00	76.3	79.8	53.6	.0
2A	4.0-5.0	285633	628026	4.06	1.31	2.75	22	.09	11758	40.00	451	.00	24218	50.00	57.2	70.0	38.2	.0
2A	5.0-6.0	189726	429202	4.83	1.57	3.26	26	.07	10051	40.00	438	.00	20212	50.00	59.6	72.2	40.2	.0
2A	6.0+ .0	324294	777917	8.63	3.18	5.45	39	.07	42603	40.00	329	.00	67298	50.00	68.9	79.4	46.1	.0

4.3 VANGORDA PIT

The pit design and mining plan used in this report have been developed by Curragh personnel based on the Vangorda Geological model discussed in Section 3.4. The pit is to be mined in its entirety as a single phase operation with each lift being mined to ultimate pit limits in sequence. Pit geometry is as given in Section 5.7.

4.3.1 Economic Modeling

An economic model for the Vangorda pit was generated by Curragh using PC-Mine software. This economic model represents the net value of each model block based on an estimation of all operating costs and revenue. An arbitrary minimum grade for processing of a block was chosen at 5% (Pb + Zn). This is partially based on Faro experience.

The economic modelling method is done on a sectional basis where each section is evaluated to obtain a two dimensional optimization of the section. For this model sections were developed every 60 metres along the strike of the deposit. In this process no provision was made for pit ramps but only pit slope parameters.

After modelling by section was completed the information was transferred to plans, walls were smoothed and a ramp access added to define the ultimate pit. This ultimate pit was digitized. Contained material within the digitized pit were tabulated on 4.5 metre benches. These reserves were adjusted to reflect a 95 percent mining recovery and 15% waste dilution.

4.3.2 Model Economic Parameters

Operating costs used in the value assigned to each block are:

Mining		
Drilling	-	$ .1748/BCM
Blasting	-	.3865/BCM
Loading	-	.4018/BCM
Mine Services	-	.9817/BCM
Fixed Haulage	-	.3533/BCM
Variable Haulage		
Horizontal	-	.0189/BCM/100M horizontal
Upward Vertical	-	.0624/BCM/10M vertical
Downward Vertical	-	.0368/BCM/10M vertical
Ore Processing	-	$6.52/tonne
Mine Administration	-	$1.56/tonne
Head Office	-	$ .86/tonne
Unassigned	-	$ .93/tonne

Revenue assignment to each block are based on:

Highway Freight	$51.9329/tonne conc.
Ocean Freight	$24.0394/tonne conc.
Smelting Cost	201.3746/tonne conc.
Concentrate Grades	
Zinc	52.8%
Lead	50%

Mill revenues as discussed in Section 7.3 (mill recovery formulas)

Exchange rate	$1.39:1.00 Canada/U.S.
Metal Prices	
Lead	$0.20 U.S./lb
Zinc	$0.42 U.S./lb
Silver	$5.50 U.S./oz troy
Gold	400.00 U.S./oz troy

Smelter Payment for combined metal:

Lead	95%
Zinc	85%
Silver	95%
Gold	95%

4.3.3 Ore Reserves

The mining reserves for the Vangorda open pit are:

Waste

Rock	8,593,620 tonnes
Overburden	9,069,372 tonnes
Sulphide Waste	3,824,375 tonnes
All Waste	21,487,367 tonnes

Ore

Type	Grade Range % (Pb + Zn)	Quantity tonnes	Pb+Zn %	Pb %	Zn %	Ag g/t	Au g/t
Baritic	4.0 - 5.0	13,181	3.95	1.92	2.02	29	0.88
Baritic	5.0 +	4,750,546	9.08	3.96	5.12	57	0.63
Pyritic	4.0 - 5.0	138,091	3.84	1.92	1.92	28	0.50
Pyritic	5.0 +	163,821	7.39	3.94	3.44	57	0.20
Quartzitic	4.0 - 5.0	482,793	3.92	1.61	2.31	25	0.48
Quartzitic	5.0 +	910,336	5.69	2.30	3.38	31	0.53
Total Ore		6,458,768	8.05	3.50	4.55	50	0.59

4.4 GRUM PIT

The pit design and mining plan, for the Grum open pit, used in this report have been developed by Curragh personnel based on the Grum Geological Model discussed in Section 3.3. The pit is to be mined in three phases. Only the ultimate phase was modelled for economic parameters. Pit phases and geometry are as given in Section 5.8.

4.4.1 Economic Modelling

The Grum pit was modelled using the same method and economic parameters discussed in for the Vangorda deposit.

4.4.2 Ore Reserves

The reserves within the various phase of the Grum pit are shown on Table 4.4.2-1.

TABLE 4.4.2-1

Mining Reserves

	Quantity Tonnes	Grades %(Pb+Zn)	% Pb	% Zn	Ag g/t	Au g/t
GRUM STAGE ONE PIT						
Waste						
Waste Rock	39207270					
Overburden	15067690					
Sulphide Waste	1254449					
All Waste	55529409					
Ore						
4 - 5% combined	1244585	3.91	1.28	2.63	24	0.47
+ 5% combined	7262196	7.25	2.59	4.66	43	0.66
GRUM STAGE TWO PIT						
Waste						
Waste Rock	84432130					
Overburden	12258714					
Sulphide Waste	1834150					
All Waste	98524994					
Ore						
4 - 5% combined	1408768	3.89	1.56	2.33	26	0.66
+ 5% combined	8927805	9.46	3.55	5.91	60	0.93
GRUM STAGE THREE PIT						
Waste						
Waste Rock	71302610					
Overburden	4254012					
Sulphide Waste	1050922					
All Waste	76607544					
Ore						
4 - 5% combined	599910	3.86	1.55	2.31	28	0.78
+ 5% combined	5550125	8.80	3.34	5.46	58	0.94
GRUM TOTAL PIT						
Waste						
Waste Rock	194942010					
Overburden	31580416					
Sulphide Waste	4139521					
All Waste	230661947					
Ore						
4 - 5% combined	3253263	3.89	1.45	2.44	26	0.61
+ 5% combined	21740126	8.55	3.17	5.38	54	0.84
Total Ore	24993389	7.94	2.95	5.00	50	0.81

5

5.0 OPEN-PIT MINING

5.0 OPEN-PIT MINING

5.1 GENERAL

Open-pit mining at the Faro pit commenced in 1969 and continued to cessation of milling in June 1982. Waste stripping was carried out between June 1983 and October 1984. The property was shut down and remained idle until the operation was acquired by Curragh in November 1985.

The facilities were reactivated in December 1985, and waste stripping in the Faro pit started in January 1986 for a mill start-up in June 1986.

The Faro pit will be completely mined out in 1992. Future open-pit production after completion of the Faro pit will be provided by the Grum and Vangorda deposits.

5.2 CURRENT CONDITIONS

The Faro open-pit is in operation at a nominal 100,000 tonnes of material moved per calendar day. Material containing 5 percent lead plus zinc is shipped to the mill. Material from 4.0 to 5.0 percent lead plus zinc is stockpiled for later treatment.

5.3 EXISTING EQUIPMENT

Major mine operating equipment at the site consists of the following:

Equipment	Number	Description
Blasthole Drill	2	M4 Marion Drill (electric)
Electric Shovel	4	3 P&H 2100 1 Marion 191M 4 15-cubic yard buckets
Haulage Truck (170-Ton Capacity)	8 4	Euclid R170 Unit Rig Electrahaul
Haulage Truck (120-Ton Capacity)	22	Wabco 120
Dozer - Track Type	4	1 Caterpillar D9H 2 Caterpillar D8L 1 Caterpillar D8K
Dozer - Rubber-Tired	2	Caterpillar 824
Motor Grader	3	2 Caterpillar 16G 1 Caterpillar 14G
Front-End Loader	2	Le Tourneau L800 12-cubic yard bucket

In addition, there is a fleet of service vehicles required for servicing, maintenance and employee transport.

5.4 OPERATIONS

Reporting directly to the General Manager - Faro Operations, the Mine Manager is responsible for the safe and efficient operation, maintenance, technical support and control of the open-pit. The General Foreman - Operations, General Foreman - Maintenance, and Chief Engineer all report to the Mine Manager.

The open-pit operates on two 12 hour shifts per day, 7 days per week.

5.5 MAINTENANCE

The Faro Mine has a well-equipped facility for repair and maintenance of the equipment on site. No physical modifications or additions will be required to handle the maintenance for planned future operations other than those discussed in Section 5.9 of this Report.

Maintenance philosophy for future operations takes into consideration the following:

(a) Light vehicles will be maintained on site, under contract, by one of the local garages.

(b) Major engine and component rebuilds will be carried out by equipment suppliers in their facilities or their agent's facilities.

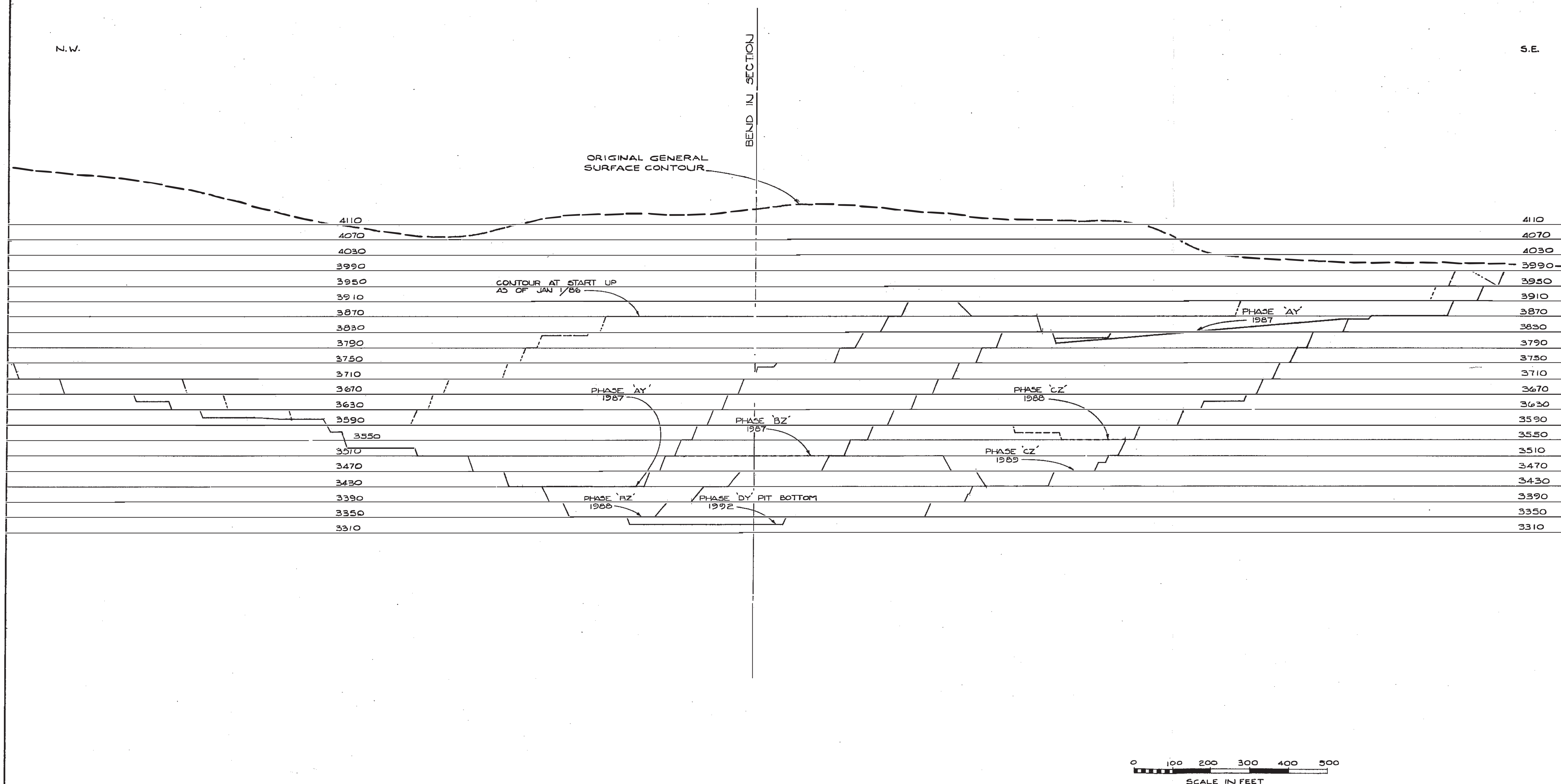
(c) Maintenance personnel will, of necessity, be of flexible skills.

5.6 FARO OPEN-PIT

5.6.1 Introduction

The main Faro open-pit will completely mine the open-pit ore in Zones 1 and 3 of the Faro deposit. Zone 2 was mined out by a small open-pit now completed and partially backfilled. Zone 1 was the initial ore source and has been mined out to the 3,590 foot level. It is the northwestern portion of the ultimate pit.

The current mining plan consists of a four-phase mining approach in which the remainder of the ore will be mined in four phases shown in Figure 5.6.1-1. Mining will progress from the Faro No.1 Zone, southeast in successive phases until mining is complete. Table 5.6.1-1 shows annual material movement.



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APPROVED BY: JBF

CLIENT: CURRAGH RESOURCES
LOCATION: FARO, YUKON

KILBORN

TITLE
FARO AREA DEPOSITS
FARO PIT
LONGITUDINAL SECTION

PROJECT No. 3509 DIVISION No. 19
DRAWING NUMBER FIG. 5.6.1-1
REV. A

TABLE 5.6.1-1

ANNUAL MATERIAL MOVEMENT

FARO OPEN PIT

(Tonnes Stated In Thousands)

	Waste	Mill Grade Ore (> 6% Pb + Zn)						Stockpile Grade Ore (> 4% < 6% Pb + Zn)					
Year	Tonnes	Tonnes	% Pb+Zn	% Pb	% Zn	Ag g/t	Au g/t	Tonnes	% Pb+Zn	% Pb	% Zn	Ag g/t	Au g/t
1987	20,375	3,172	8.62	3.60	5.02	47	.09	981	4.53	1.86	2.67	29	.10
1988	15,794	4,864	7.99	3.26	4.73	42	.07	1,875	4.54	1.81	2.73	30	.10
1989	5,512	2,449	8.27	3.33	4.94	43	.11	888	4.59	1.84	2.75	32	.12
1990	7,833	32	6.77	2.62	4.15	38	.30	90	4.55	2.14	2.41	31	.31
1991	4,966	1,294	7.55	2.83	4.72	27	.14	1,138	4.42	1.62	2.80	22	.11
1992	2,342	2,569	8.57	3.02	5.55	26	.09	1,231	4.56	1.62	2.94	19	.10

5.6.2 Mining Criteria

The Faro pit is based on the following criteria:

(a) Material In-Place Specific Gravities

Waste Rock	-	Type 1D - 2.60
		Type 3D - 2.75
		Type 3DBX - 2.71
	-	Average - 2.70
Ore (all types for equipment sizing)	-	3.93
Ore (for production calculation)	-	Measured less 5 or 10 percent of voids depending on ore type
Average - 2A	-	3.19
- 2BCD	-	3.48
- 2CE	-	3.77
- 2EF	-	4.33
- 2G	-	4.44
- 2H	-	4.02

(b) Bench Heights - Current practice

Waste	-	12.12 metres
Ore	-	6.06 metres

(c) Pit Slope Geometry - Current practice

Southwest Wall	-	45^{o} slope
Northeast Wall	-	39^{o} slope
Berms	-	8 meters

(d) Drilling and Blasting - Current practice

(e) Loading - Operation of all shovels until stripping is advanced sufficiently to permit permanent reduction in shovel usage with the pit.

(f) Haulage - Operate all 170-ton trucks on waste. Operate 120-ton trucks on ore and waste. Schedule 170-ton units is preference to 120-ton units.

5.6.3 Pit Design

(a) Material Description

The waste material within the confines of the ultimate open-pit consists of an assemblage of gneiss, schists, quartzite and metamorphic rocks with a minor amount of quartz diorite and diorite intrusives which form the majority of the overburden, and 6 sulphide mineralized facies which may or may not be ore subject to the lead and zinc content.

Hardness and in-place density of the various materials are variable.

(b) Pit Slope Design

Pit slopes at the Faro pit are based on a Geotechnical Study undertaken by D.R. Piteau & Associates Ltd., Geotechnical Consultants, dated January 1976. This evaluation was based on detailed structural mapping of all accessible benches on the east side of the pit. Their recommendations are shown in Table XI excerpted from their Report.

TABLE XI

SLOPE DESIGN RECOMMENDATIONS

[Excerpt from D.R. Piteau & Associates Ltd. Report]

Sector	Water	Watered	Blasting	Design	Angle	Water	Watered
A	39°	44°	35°-37°	53°	70°	34 1/2	26 1/2
B	36 1/2°	39 1/2°	35°-37°	49°-75°	70°	39	34
C	39°	41°	35°-37°	49°-75°	70°	34 1/2	31
D (North)	38 1/2	43°	35°-37°	-	70°	35 1/2	28
D (South)	43°	47°	41°-42°	30°-80°	80°	36	30
E	N/A	47°	N/A	±80	80°	N/A	30
F	N/A	45°	N/A	72°-74°	70°	N/A	25
G	N/A	45°	N/A	72°-74°	70°	N/A	25

N/A = Not Available.

Existing plans are based on final slopes approximating the average for high groundwater and dewatered conditions assuming controlled blasting on the final pit wall. To achieve these slopes, water control measures and blasting control will be required.

5.6.4 Mining Plan Concept

The Faro pit is being mined in four main phases from northwest to southeast. An additional subphase at the southern end of the pit was mined soon after commencement of mining activities.

Stripping operations commenced with one shovel operating in January 1986 and increased to operation of all shovels in June 1986. All shovels will be used until the end of 1987 after which time they will be operated on a diminishing scale as stripping requirements are reduced.

All materials with a combined grade above 5.0 (Pb+Zn) percent will be delivered to the crusher or to temporary stockpiles near the crusher. Material with grades between 4.0 and 5.0 (Pb+Zn) percent will be stockpiled and treated after open-pit mining is complete.

5.6.5 Mining Plan

(a) Phases [Figure 5.6.1-1]

The Faro open-pit has been mined out northwest of the Faro Fault. The longitudinal section shows the remaining portion of the open-pit which will be mined. Phases AY, BZ, CZ and DY are shown, as are the portions which will be mined in each time period. The drawings associated with each time period show major ramps as they exist at the end of the time period.

Table 5.6.1-1 indicates the material movement by time period.

(b) Current Condition [Figure 5.6.5-1]

The Faro open-pit, as of April 1, 1987, has been stripped to ore in the AY phase and ore production is from this phase. Stripping of the BZ phase has started.

(c) Mining - 1987 [Figure 5.6.5-2]

During the period April 1, 1987 to end of 1987, four shovels will be operating in the Faro pit. Ore production will come from the AY phase of the pit, from the 3,570 through 3,430 benches. During this period, the BZ phase will be stripping from 3830 through 3510 benches and will be producing ore at year-end. The CZ phase stripping will be started.

SECTION: MINING	CLIENT: CURRAGH RESOURCES	TITLE
SCALE: 1" = 100'		FARO AREA DEPOSITS
DESIGNED BY: J.B.F. MAR.87	LOCATION. FARO, YUKON	FARO DEPOSIT OPEN PIT PLAN MARCH 31, 1987
DRAWN BY: ZA MAR.87	KILBORN	PROJECT No. 3509 DIVISION No. 19
CHECKED BY:		DRAWING NUMBER FIG. 5.6.5-1
APPROVED BY: JBF		REV. A

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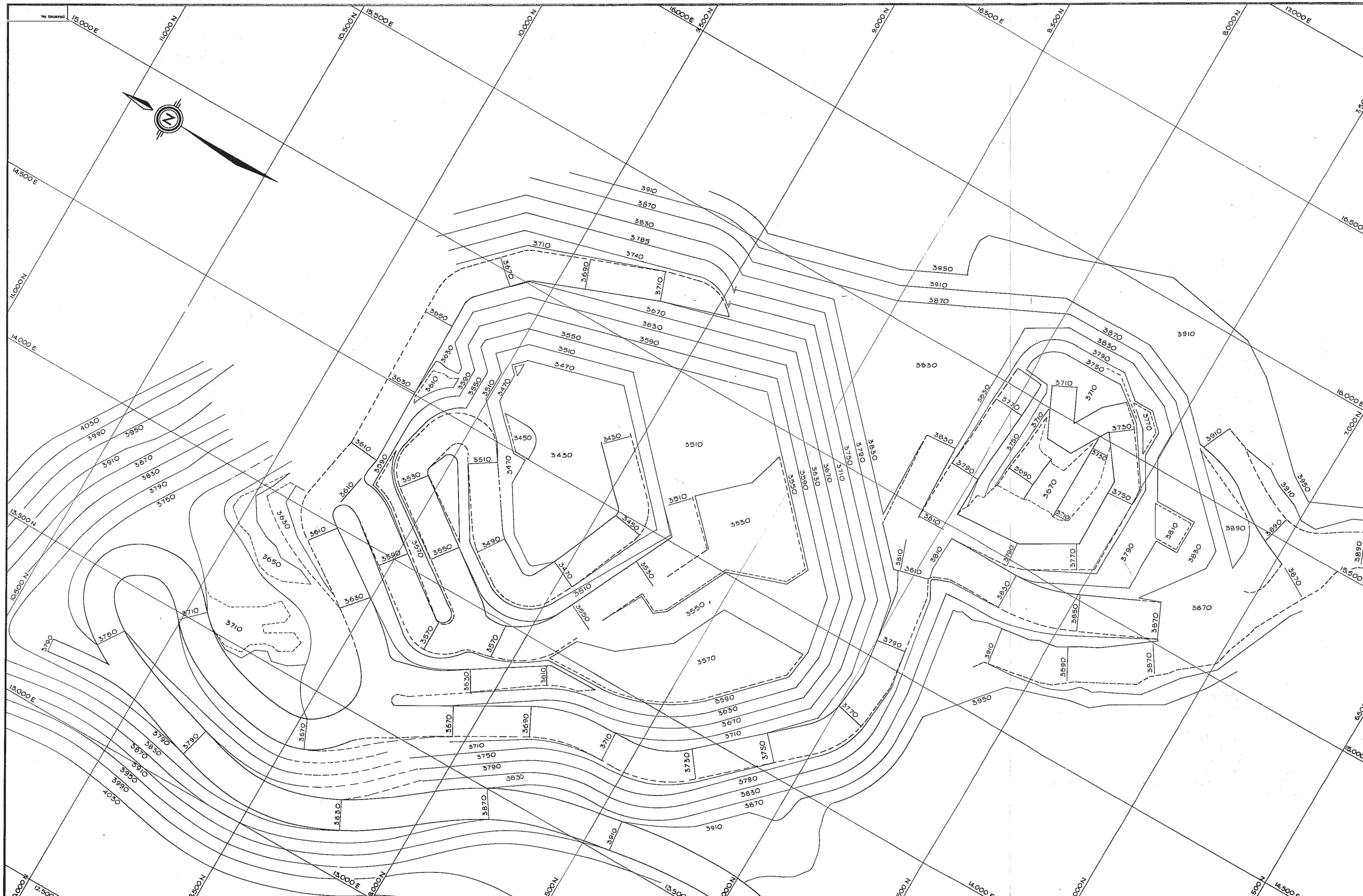

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LOCATION: FARO, YUKON

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TITLE
FARO AREA DEPOSITS
FARO DEPOSIT
OPEN PIT PLAN
END OF 1987

PROJECT No. 3509 | DIVISION No. 19
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(d) Mining - 1988 [Figure 5.6.5-3]

During the initial part of the year, three shovels will be operating in the Faro pit. Ore production will come from the BZ phase while stripping of the CZ phase is in progress. The BZ phase will be completed in late 1988 at which time the CZ phase will be producing the ore, and stripping will have commenced on the DY phase. At year-end, there will be only two shovels operating in the Faro pit.

(e) Mining - 1989 [Figure 5.6.5-4]

Two shovels will be operating in the Faro pit at the start of 1989. Ore production will come from the CZ phase until it is completed in mid-1989, at which time there will be one remaining shovel operating in the Faro pit stripping waste from the DY phase.

(f) Mining - 1990 [Figure 5.6.5-5]

Waste stripping of the DY phase will continue throughout the year with only minor amounts of ore released.

(g) Mining - 1991 [Figure 5.6.5-6]

Ore production will come from the DY phase. One shovel will be operating in the Faro pit during the year.

(h) Mining - 1992 [Figure 5.6.5-7]

The Faro open-pit will be completed by the end of the year.

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APPROVED BY:

CLIENT: CURRAGH RESOURCES
LOCATION: FARO, YUKON

KILBORN

TITLE
FARO AREA DEPOSITS
FARO DEPOSIT
OPEN PIT PLAN
END OF 1988

PROJECT No. 3509
DIVISION No. 19
DRAWING NUMBER FIG. 56.5-3
REV. A

A	ISSUED FOR TECHNICAL REVIEW — MAR. 87 — ZA
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DRAWN BY:	Z.A — MAR. 87
CHECKED BY:	
APPROVED BY:	JBF

CLIENT: CURRAGH RESOURCES

LOCATION: FARO, YUKON

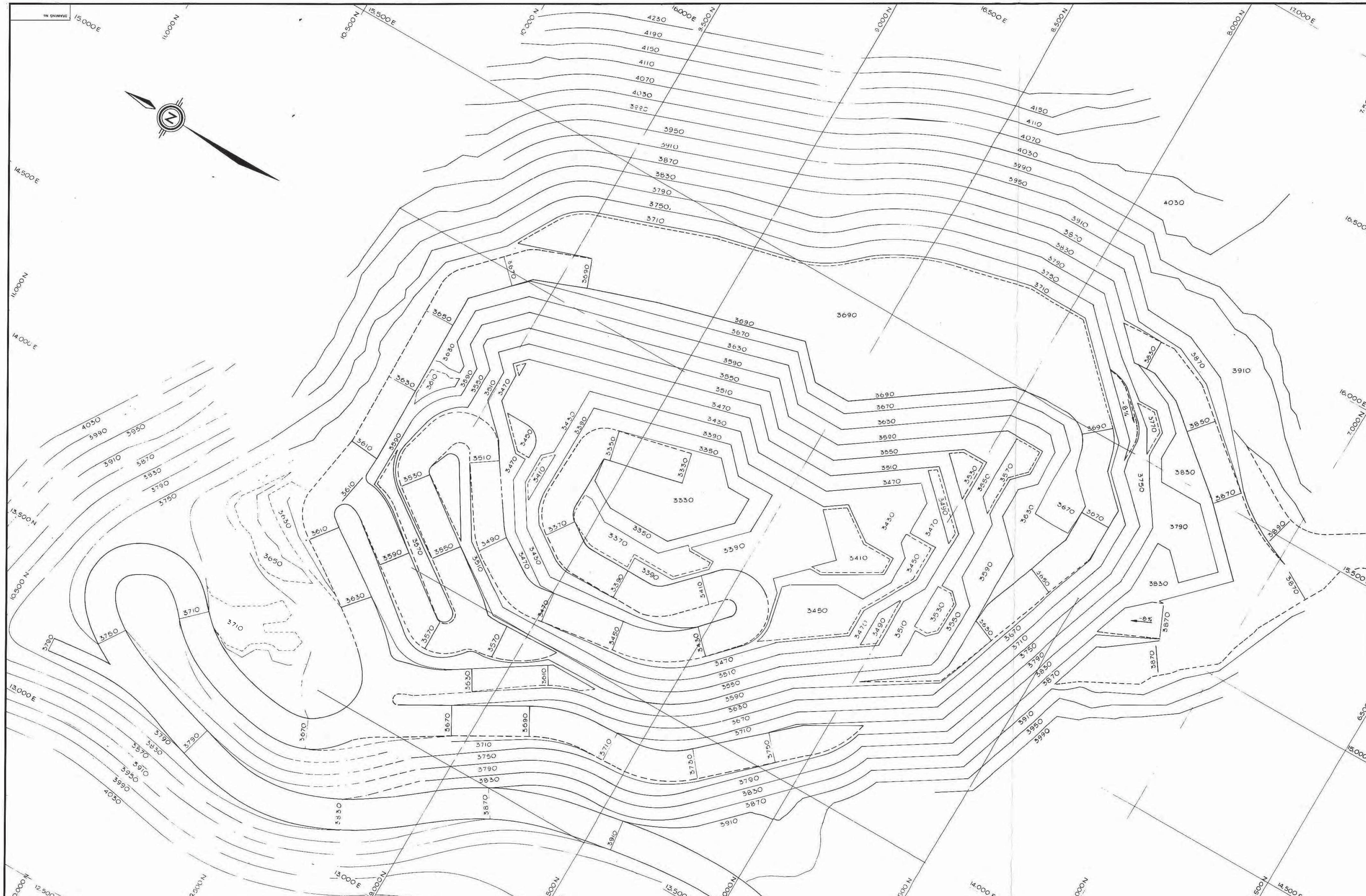
KILBORN

TITLE

FARO AREA DEPOSITS

FARO DEPOSIT
OPEN PIT PLAN
END OF 1989

S.O.M. No.	PROJECT No.	DIVISION No.	DRAWING NUMBER	REV.
	3509	19	FIG. 5.6.5-4	A



No.	DESCRIPTION			CLIENT	PROJ. ENG.	CHECK
A	ISSUED FOR TECHNICAL REVIEW	MAR 87	ZA			

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DWG. NO. | REFERENCE DRAWINGS

SECTION: MINING
SCALE: 1" = 100'
DESIGNED BY: J.B.F. — MAR. 87
DRAWN BY: Z.A. — MAR. 87
CHECKED BY:
APPROVED BY: JBF

CLIENT: CURRAGH RESOURCES
LOCATION: FARO, YUKON

KILBORN

TITLE
FARO AREA DEPOSITS
FARO DEPOSIT
OPEN PIT PLAN
END OF 1990

B.O.M. No.
PROJECT No. 3509 — DIVISION No. 19
DRAWING NUMBER FIG.5.6.5-5 — REV. A

DWG. NO.	REFERENCE DRAWINGS			

No.	DESCRIPTION			
A	ISSUED FOR TECHNICAL REVIEW	MAR. 87	ZA	

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SECTION: MINING
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CHECKED BY:
APPROVED BY: JBF

CLIENT: CURRAGH RESOURCES
LOCATION: FARO, YUKON

KILBORN

TITLE: FARO AREA DEPOSITS
FARO DEPOSIT
OPEN PIT PLAN
END OF 1991

B.O.M. No.
PROJECT No.: 3509 — DIVISION No.: 19
DRAWING NUMBER: FIG. 5.6.5-6 — REV. A

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No.	DESCRIPTION		CLIENT / PROJ. ENG. / CHECK
	REVISIONS		
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No.	DESCRIPTION		CLIENT / PROJ. ENG. / CHECK
	REVISIONS		

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SCALE: 1" = 100'	DATE	
DESIGNED BY: J.B.F.	FEB 87	
DRAWN BY: Z.A.	FEB 87	
CHECKED BY:		
APPROVED BY: JBF		

CLIENT: CURRAGH RESOURCES

LOCATION: FARO, YUKON

KILBORN

TITLE: FARO AREA DEPOSITS

FARO DEPOSIT
OPEN PIT PLAN
END OF 1992 - ULTIMATE

B.O.M. No.	
PROJECT No.	DIVISION No.
3509	19
DRAWING NUMBER	REV.
FIG. 5.6.5-7	A

5.7 VANGORDA OPEN-PIT

5.7.1 Introduction

The Vangorda deposit will be the first of the Vangorda Plateau deposits developed and mined. It is intended to prestrip and mine the deposit over a period of four years from 1988 to 1991 inclusive. Table 5.7.1-1.

There is a total of 27,946,000 tonnes of material to be moved, of which 6,459,000 is considered ore. The effective stripping ratio is 3.33 to 1.

5.7.2 Mining Criteria

The Vangorda pit is based on the following criteria:

(a) Material In-Place Specific Gravities

Unconsolidated Overburden	-	2.1
Waste Rock	-	2.70
Ore (For production calculation)	-	Measured SG less 5 percent for voids
Average - 4A	-	2.83
- 4C	-	3.27
- 4EC	-	3.68
- 4E	-	3.75
- 4EG	-	3.96
- 4EH	-	3.67

(b) Bench Height

Waste	-	13.5 metres
Ore	-	4.5 metres

TABLE 5.7.1-1

ANNUAL MATERIAL MOVEMENT

VANGORDA OPEN PIT

(Tonnes Stated In Thousands)

	Waste	Mill Grade Ore (> 5% Pb + Zn)						Stockpile Grade Ore (> 4% < 5% Pb + Zn)					
Year	Tonnes	Tonnes	% Pb+Zn	% Pb	% Zn	Ag g/t	Au g/t	Tonnes	% Pb+Zn	% Pb	% Zn	Ag g/t	Au g/t
1988	6,666												
1989	6,029	1,595	7.81	3.33	4.48	48	.55	242	3.88	1.62	2.26	28	.41
1990	8,132	3,217	8.91	3.87	5.04	54	.61	320	3.91	1.67	2.24	23	.49
1991	661	1,013	8.31	3.78	4.53	54	.66	72	3.92	1.95	1.97	27	.76

(c) Pit Slope Geometry

Southeast Wall	- 45^{o} slope
Northeast Wall	- 40^{o} slope
Southeast Wall	- 45^{o} slope
Northwest Wall	- 45^{o} slope
Overburden	- 35^{o} slope

(d) Pit Ramps

Width	- 30 metres
Maximum Grade	- 8 percent

(e) Drilling and Blasting - Current Faro practice

(f) Loading - Current practice

(g) Haulage - All materials in 170-ton trucks

5.7.3 Pit Design

(a) Material Description

Surficial deposits in the pit area consist mainly of glacial till. Thicknesses of overburden range from 2 to 30 metres. The till is relatively consistent and is composed of primarily sandy silt with some sand and gravel. No permafrost was encountered in any test pits or drill holes.

The waste rock within the pit consists primarily of non-calcareous phyllites. Most phyllites, especially in the deposit footwall, are bleached, locally silicified and/or chloritic and sulphide-bearing.

(b) Pit Slope Design

Pit slopes stability will be governed primarily by the orientation of fault and foliation surfaces. Possible failure modes include plane failure on foliation surfaces and wedge failure on intersecting fault and foliation surfaces. The northeast wall, with the foliation dipping into the pit, will probably require a flatter wall. Further study will be required to determine pit slopes; however, the pit slopes contained within the criteria are realistic approximations.

5.7.4 Mining Plan Concept

The Vangorda pit will be mined as one phase. Benches will be taken to ultimate walls as the pit is mined to ultimate depth.

Overburden will be removed with scrapers. Rock stripping operations will start as soon as overburden stripping is sufficiently advanced and necessary facilities are available.

All ore will be stockpiled outside the pit and rehandled into haulage trucks to feed the crusher at the Faro Mill. Material having a combined grade above 5.0 percent (Pb+Zn) will be delivered to the crusher as required. Material with grades between 4.0 and 5.0 percent (Pb+Zn) will be left in stockpile and treated after open-pit mining is complete.

Waste from the pit will be disposed of in a single waste dump outside the pit area (see Figure 5.9.1-2).

5.7.5 Mining Plan

(a) Mining Plan Description [Figure 5.7.5-1]

The Vangorda open-pit will be approximately 1,000 metres in length, 400 metres in width at the widest point and 110 metres in depth. The longitudinal axis of the pit is approximately northwest/southeast with the deepest portion to the northwest end of the pit. Access to the pit will be a ramp located on the southwest pit wall which will exit the pit on the northwestern end of the pit.

(b) Current Condition

At the present, no preproduction work has been undertaken at the pit site. Site clearing, water diversion ditching and pit area dewatering have not been started. No facilities are available at site.

(c) Mining - 1987

Work at the Vangorda pit, during 1987, will consist of site clearing, water diversion and a start on pit area dewatering.

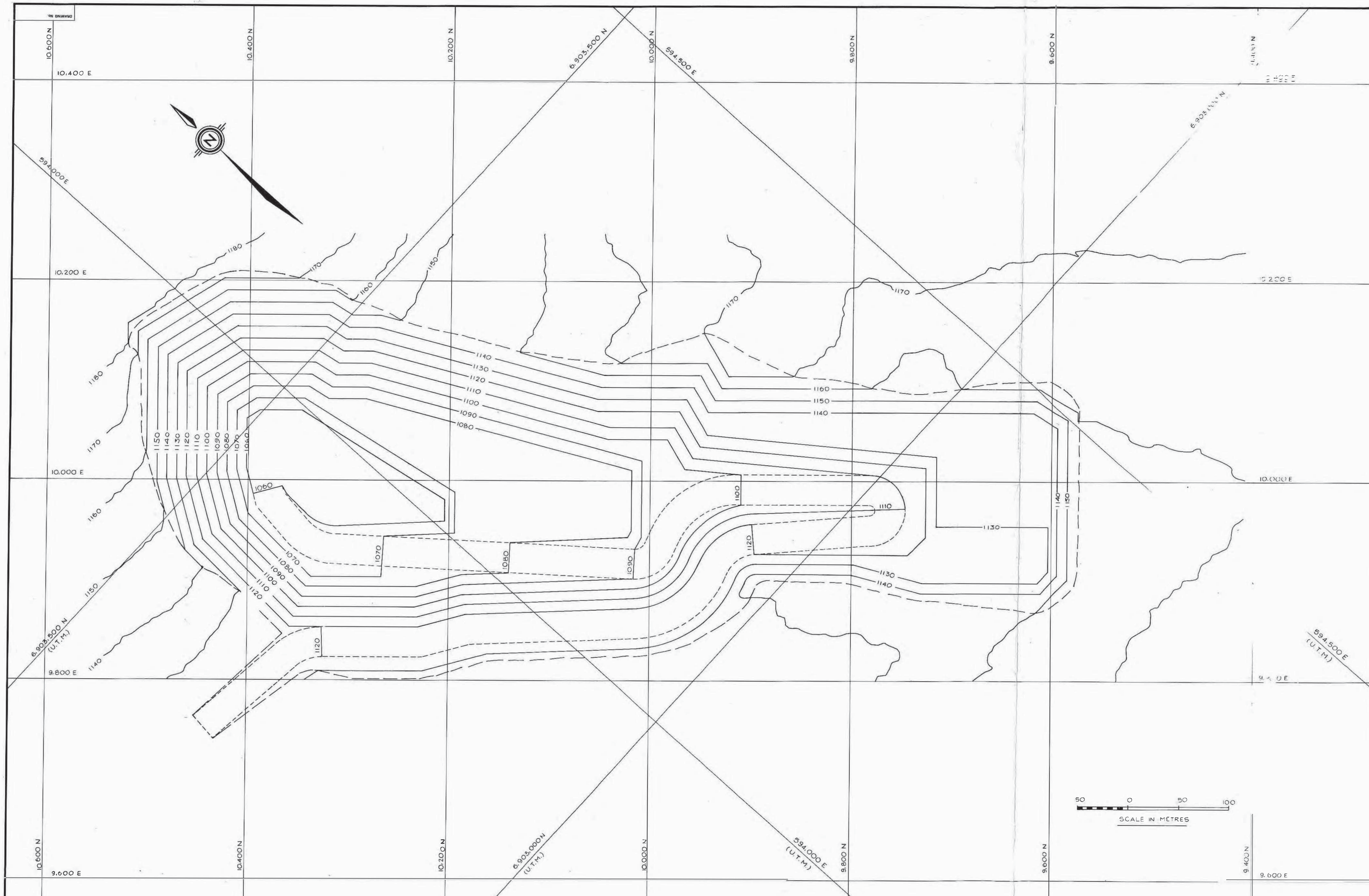
(d) Mining - 1988 [Figure 5.7.5-2]

Mining will commence at the 1,170 metre elevation at the north end of the pit. During the early part of the year, the Vangorda Creek diversion will be constructed. Some of the unconsolidated overburden will be used for the diversion dam.

By year-end, the pit will be mined down to the 1,130 metre elevation and 6.7 million tonnes of waste will have been removed.

During the year, one shovel will have been operating in this pit.

Pit status at the end of the year is shown in Figure 5.7.5-2.



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No.	DESCRIPTION				

REVISIONS

DWG. NO. | REFERENCE DRAWINGS

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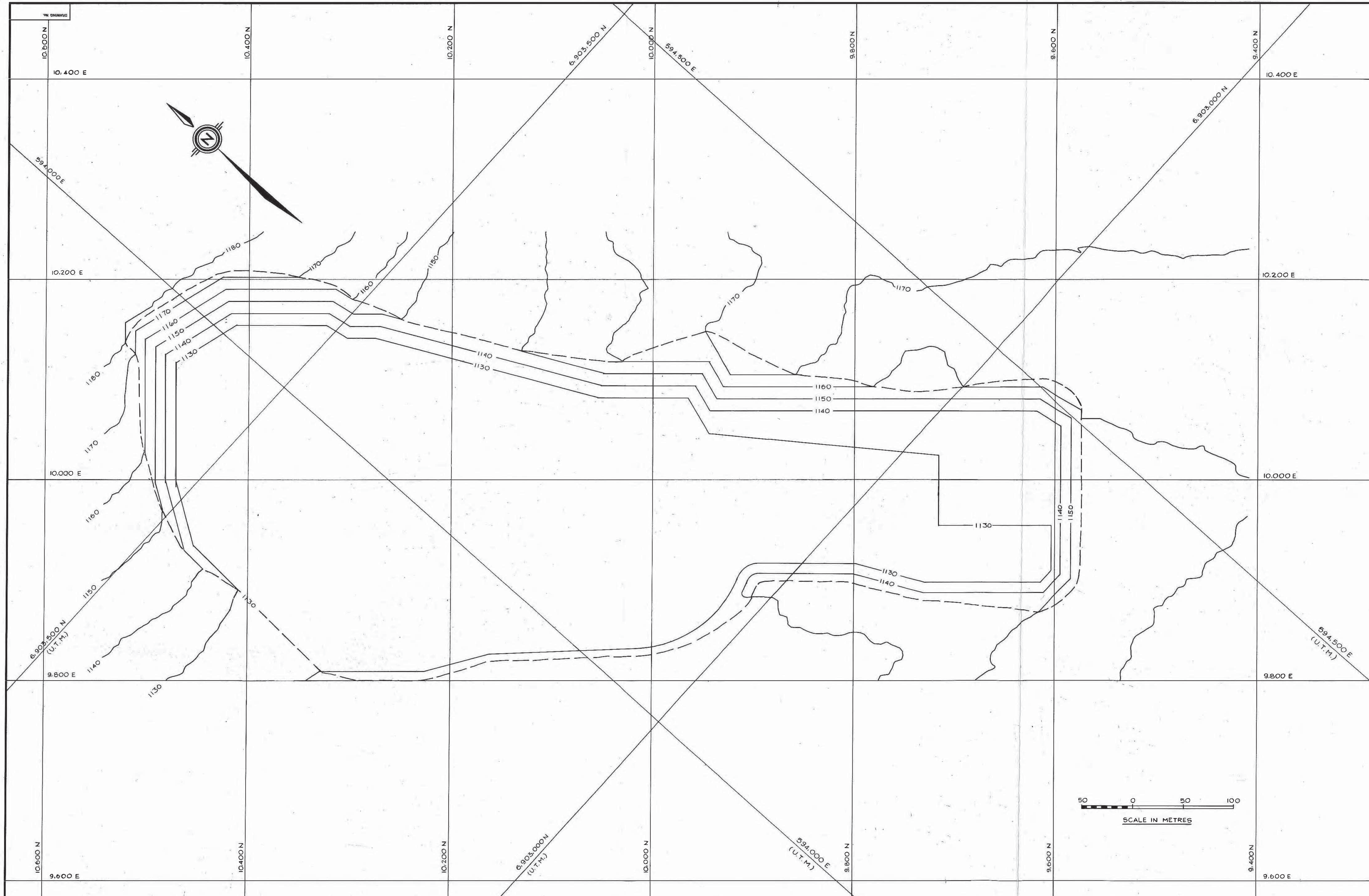
DESIGNED BY: J.B.F.	FEB. 87	KILBORN	
DRAWN BY: Z.A.	FEB. 87		
CHECKED BY:			
APPROVED BY: J.E.F.			

B.O.M. No.	
PROJECT No.	DIVISION No.
3509	19
DRAWING NUMBER	REV.
FIG. 5.7.5-1	A



DWG. NO.	REFERENCE DRAWINGS	A	ISSUED FOR TECHNICAL REVIEW	MAR.87	ZA			
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			REVISIONS					

SECTION:		CLIENT: CURRAGH RESOURCES	TITLE	B.O.M. No.
SCALE: 1:1200	DATE		FARO AREA DEPOSITS	PROJECT No. 3509 / DIVISION No. 19
DESIGNED BY: J.B.F.	FEB.87	LOCATION: FARO, YUKON	VANGORDA DEPOSIT OPEN PIT PLAN END OF 1988	DRAWING NUMBER
DRAWN BY: Z.A.	FEB.87	KILBORN		FIG. 5.7.5-2 REV. A
CHECKED BY:				
APPROVED BY: J.B.F				

(e) Mining - 1989 [Figure 5.7.5-3]

During 1989, approximately 1.8 million tonnes of ore and 6.0 million tonnes of waste will be removed from the Vangorda open-pit. Pit bottom at year-end, as shown in Figure 5.7.5-3, will be 1,110 metres elevation.

Up to 500,000 tonnes of near surface ore may be oxidized to some degree.

(f) Mining - 1990 [Figure 5.7.5-4]

During 1990, approximately 3.5 million tonnes of ore and 8.1 million tonnes of waste will be removed from the Vangorda open-pit. In order to meet these production requirements, a second shovel will be moved into the pit early in 1990.

Pit status at the end of 1990 is shown in Figure 5.7.5-4.

(g) Mining - 1991 [Figure 5.7.5-1]

During early 1991, mining of the Vangorda open-pit will be complete. Production during this period will be 1.0 million tonnes of ore and 0.7 million tonnes of waste.

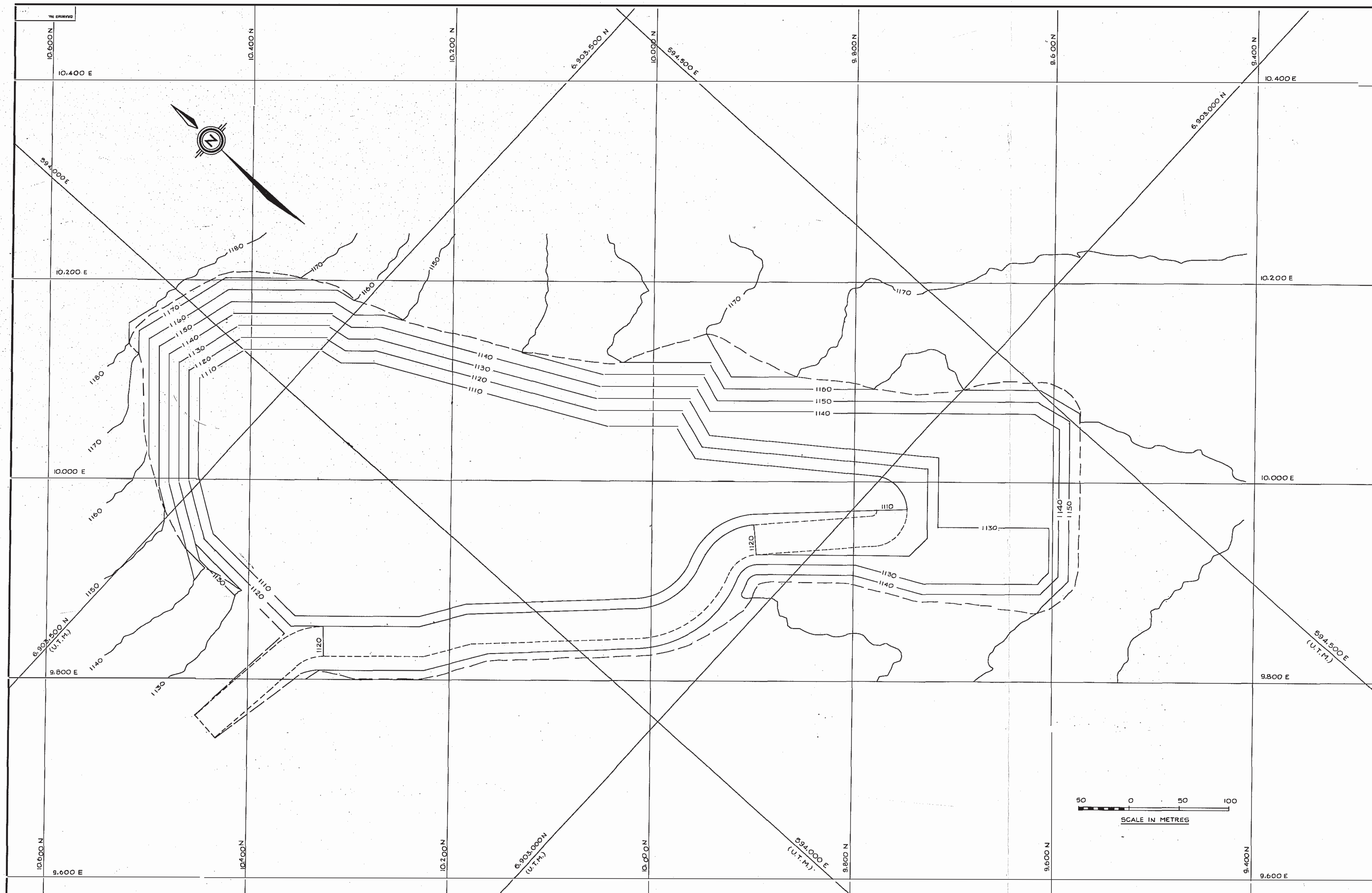
Pit status at end of mining in Vangorda is shown in Figure 5.7.5-1.

5.8 GRUM OPEN-PIT

5.8.1 Introduction

The Grum deposit will be the second of the Vangorda Plateau deposits to be developed and mined. It is intended to pre-strip and mine the deposit over the period 1989 through 1997 inclusive.

The mining plan consists of three phases which are described later in this Report.



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REVISIONS				

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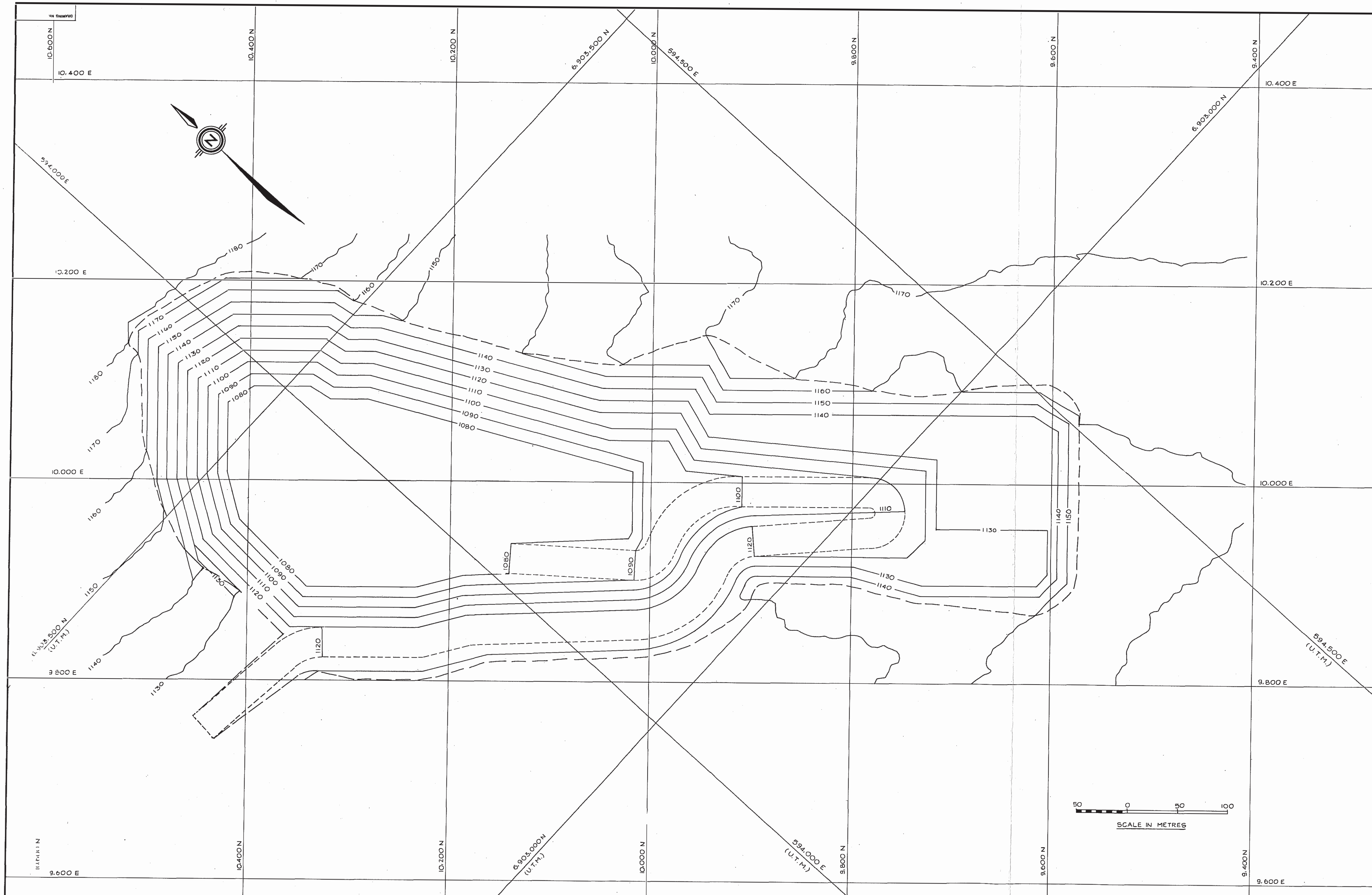
DESIGNED BY: J.B.F.	FEB.87	KILBORN	
DRAWN BY: Z.A.	FEB.87		
CHECKED BY:			
APPROVED BY: J.B.F.			

B.O.M. No.	
PROJECT No.	DIVISION No.
3509	19
DRAWING NUMBER	REV.
FIG.5.7.5-3	A



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DRAWN BY: Z.A.	FEB.87
CHECKED BY:	
APPROVED BY: J.B.F.	

CLIENT: CURRAGH RESOURCES

LOCATION: FARO, YUKON

KILBORN

TITLE

FARO AREA DEPOSITS

VANGORDA DEPOSIT
OPEN PIT PLAN
END OF 1990

B.O.M. No.	
PROJECT No.	DIVISION No.
3509	19
DRAWING NUMBER	REV.
FIG. 5.7.5-4	A

No.	DESCRIPTION		CLIENT	PROJ. ENG.	CHECK
A	ISSUED FOR TECHNICAL REVIEW	MAR.87	ZA		

REVISIONS

DWG. NO. | REFERENCE DRAWINGS

A total of 256,315,000 tonnes will be removed from the pit during a 10 year period. This total is comprised of 40,645,000 tonnes overburden, 190,677,000 tonnes waste and 24,993,000 tonnes ore. Table 5.8.1-1 gives the annual mining quantities.

5.8.2 Mining Criteria

The Grum open-pit is based on the following criteria:

(a) Material In-Place Specific Gravities

Unconsolidated Overburden	-	2.10
Waste Rock	-	2.70
Ore (for production calculation)	-	Measured SG less 5 percent for voids
Average - 4A0	-	3.23
- 4A4/4AE	-	3.31
- 4B	-	3.00
- 4C	-	3.45
- 4D	-	3.53
- 4E	-	4.32
- 4G	-	4.42
- 4H	-	3.86
- 4J	-	3.87
- 4K	-	3.84
- 4L0	-	3.11
- 4L4/4LE	-	3.29

TABLE 5.8.1-1

ANNUAL MATERIAL MOVEMENT

GRUM OPEN PIT

(Tonnes Stated In Thousands)

	Waste	Mill Grade Ore (> 5% Pb + Zn)						Stockpile Grade Ore (>4% < 5% Pb + Zn)					
Year	Tonnes	Tonnes	% Pb+Zn	% Pb	% Zn	Ag g/t	Au g/t	Tonnes	% Pb+Zn	% Pb	%Zn	Ag g/t	Au g/t
1988	1,334												
1989	21,500	52	5.86	2.20	3.66	36	.48						
1990	20,208	218	5.62	1.96	3.66	33	.66	51	3.81	1.27	2.54	24	.77
1991	29,190	1,377	7.37	2.66	4.71	44	.66	389	3.92	1.28	2.64	25	.53
1992	31,684	1,897	7.25	2.59	4.66	42	.62	378	3.85	1.27	2.58	24	.40
1993	34,277	3,374	7.77	2.85	4.92	47	.67	450	3.98	1.35	2.63	25	.50
1994	32,168	4,097	8.92	3.37	5.55	56	.85	481	3.91	1.54	2.37	27	.65
1995	29,423	3,958	9.58	3.53	6.05	60	.96	719	3.87	1.53	2.34	26	.63
1996	27,832	3,209	8.81	3.35	5.46	57	.88	419	3.86	1.57	2.29	28	.73
1997	3,707	3,558	8.87	3.31	5.56	59	1.01	367	3.88	1.55	2.33	27	.78

(b) Bench Height

Waste	- 13.5 metres
Ore	- 4.5 metres

(c) Pit Slope Geometry

Southeast Wall	- 40^{o} slope
Northeast Wall	- 40^{o} slope
Southwest Wall	- 45^{o} slope
Northwest Wall	- 45^{o} slope
Overburden	- 35^{o} slope

(d) Pit Ramps

Width	- 30 metres
Maximum Grade	- 8 percent

(e) Drilling and Blasting - Current Faro practice

(f) Loading - Current practice

(g) Haulage - All material in 170-ton trucks

5.8.3 Pit Design

(a) Material Description

Surficial deposits within the pit area consists primarily of morainal and glaciofluval deposits. Depths of this overburden material range from zero to 10 metres in the northern portion of the deposit to over 100 metres in the southern end. This material contains tills, uniform silts and gravels. Some pockets of permafrost have been encountered, but do not appear to be extensive. The overburden is known to be water-saturated in some areas, but the extent of saturation has not been defined.

Regional bedrock geology contains biotite-muscovite schists, calc-silicate gneiss and biotite-muscovite phyllites. Mineralization occurs within the phyllites.

The overall structure of the deposit is that of a broad syncline, complicated by a number of faults.

(b) Pit Slope Design

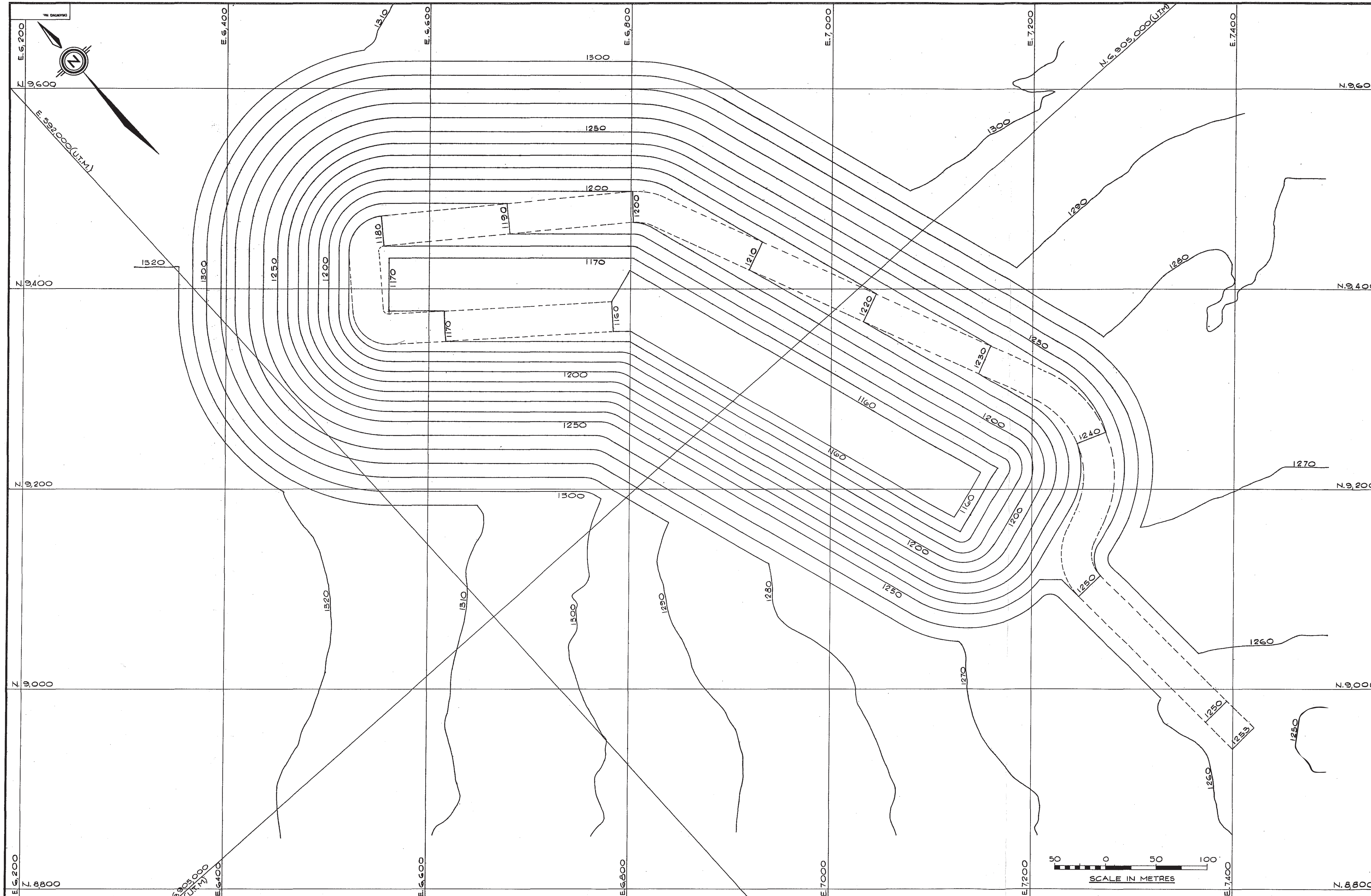
Pit wall stability will be governed primarily by the orientation of fault and foliation surfaces, and potential wedge failure on intersecting fault and foliation surfaces. The foliation on the northeast and southeast walls dips into the pit, and these walls will of necessity be of flatter slopes than the others. Slopes selected are given previously in Section 5.8.2 of this Report.

5.8.4 Mining Plan Concept

The Grum pit will be mined in three overlapping stages. The first stage of the pit, shown in Figure 5.8.4-1, will go to the 1,160 metre elevation. Concurrent with the mining of the stage one pit, stripping will begin on the second stage, shown in Figure 5.8.4-2.

The second stage will involve pushing back both the southeast and northeast walls of the stage one pit. The majority of the material will be removed from the southeast wall. The stage two pit bottom will be 1,090 metres elevation.

The stage three pit (ultimate pit) is shown in Figure 5.8.4-3. The stripping of this stage will start during the mining of stage two and will take the ultimate pit to the 1,030 metre level.



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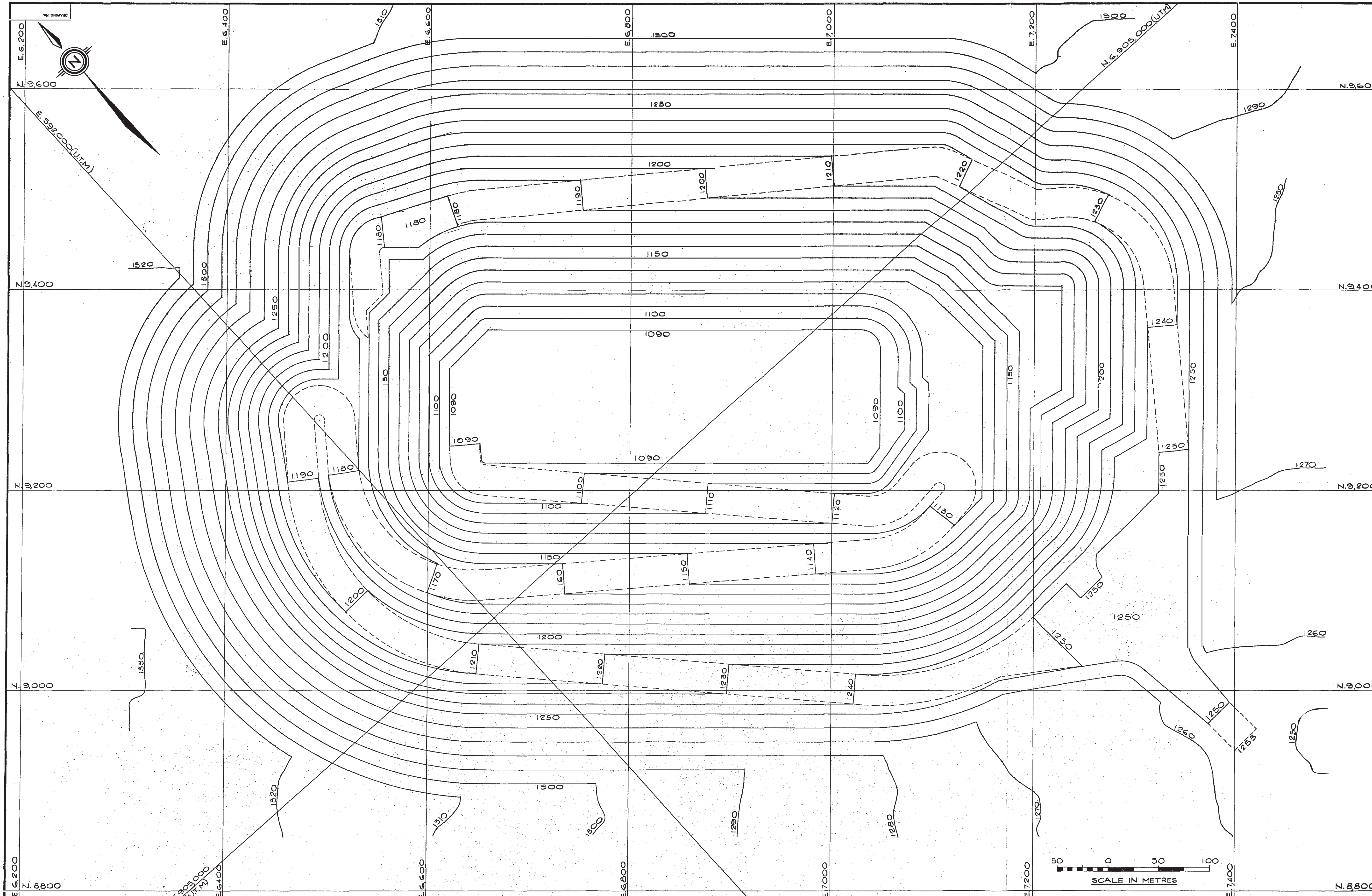
DESIGNED BY: J.B.F. FEB.87	KILBORN	GRUM DEPOSIT OPEN PIT PLAN STAGE I	DRAWING NUMBER
DRAWN BY: P.A.H MAR.87			FIG.5.8.4-1 REV. A
CHECKED BY:			
APPROVED BY: JBF			



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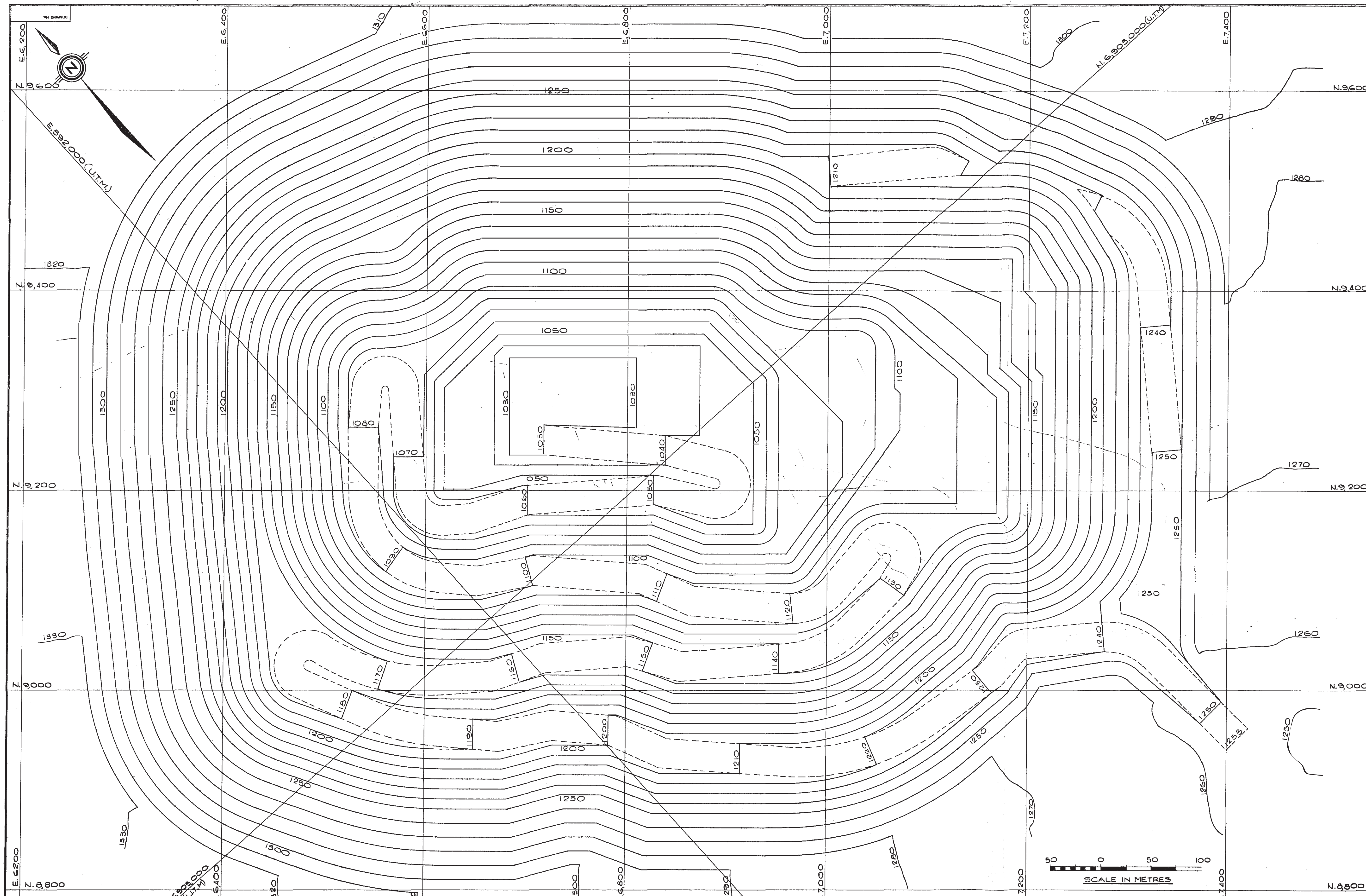
CHECKED BY:
APPROVED BY: JBF

CLIENT: CURRAGH RESOURCES
LOCATION: FARO, YUKON

KILBORN

TITLE: FARO AREA DEPOSITS
GRUM DEPOSIT
OPEN PIT PLAN
STAGE II

B.O.M. No.
PROJECT No. 3509
DIVISION No. 19
DRAWING NUMBER: FIG.5.8.4-2
REV. A



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CHECKED BY:		
APPROVED BY: J.B.F		

CLIENT: CURRAGH RESOURCES

LOCATION: FARO, YUKON.

KILBORN

TITLE: FARO AREA DEPOSITS

GRUM DEPOSIT
OPEN PIT PLAN
STAGE III - ULTIMATE

PROJECT No.	DIVISION No.
3509	19

DRAWING NUMBER: FIG. 5.8.4-3 REV. A

A ISSUED FOR TECHNICAL REVIEW MAR.3.87

No. DESCRIPTION REVISIONS

DWG. NO. REFERENCE DRAWINGS

5.8.5 Mining Plan

(a) Mining Plan Description

The Grum open-pit will be ultimately 1,150 metres in length, 850 metres in width and 300 metres in depth. The longitudinal axis of the pit is approximately northwest/southeast. The ramp exit from the pit will be on the southwest corner, 220 metres above the ultimate pit bottom. Mining activities in the Grum open-pit will start in late 1988 and continue until late 1997. A total of 25 million tonnes of ore and 231 million tonnes of waste will be mined.

(b) Mining - 1988

Mining will begin in late 1988 with scraper removal of overburden. Stripping of the stage one pit will be started at the 1,310 metre elevation. During the year, 1.3 million tonnes of material will be removed.

(c) Mining - 1989 [Figure 5.8.5-1]

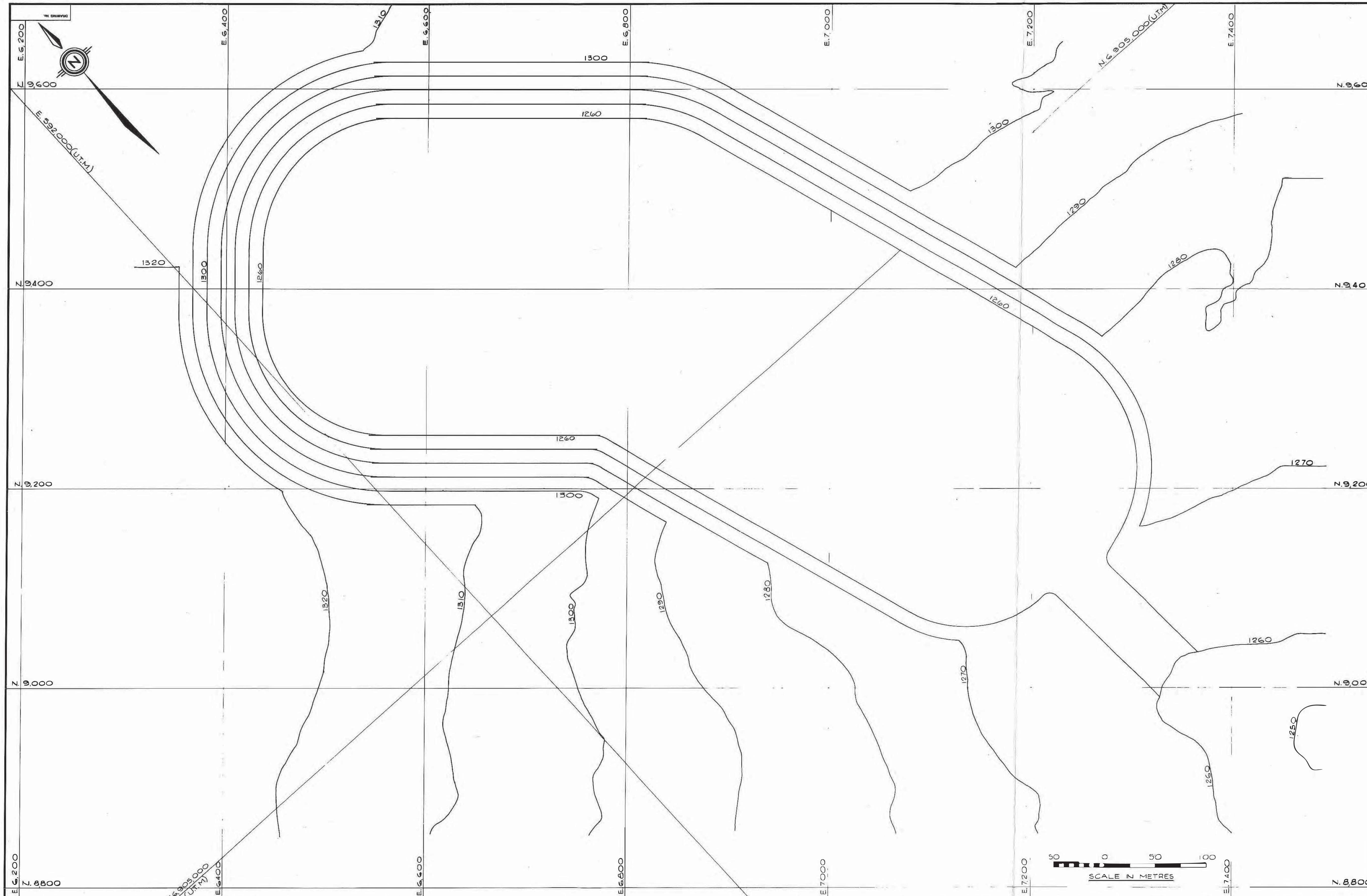
Early in 1989, two shovels will be available to the Grum pit. Stage one stripping will continue during the year with scrapers and shovels operating. Approximately 21.5 million tonnes of waste will be removed. A minor amount of ore (52,000 tonnes) will be mined and stockpiled.

Figure 5.8.5-1 shows the pit status at the end of the year.

(d) Mining - 1990 [Figure 5.8.5-2]

One shovel will be removed from the Grum pit for use at the Vangorda pit early in 1990 and replaced in the latter part of the year. By year-end, two shovels will be operating in the Grum pit. Approximately 270,000 tonnes of ore will be mined and stockpiled. Waste removal will amount to 20.2 million tonnes.

Pit status at the end of 1990 is shown in Figure 5.8.5-2.



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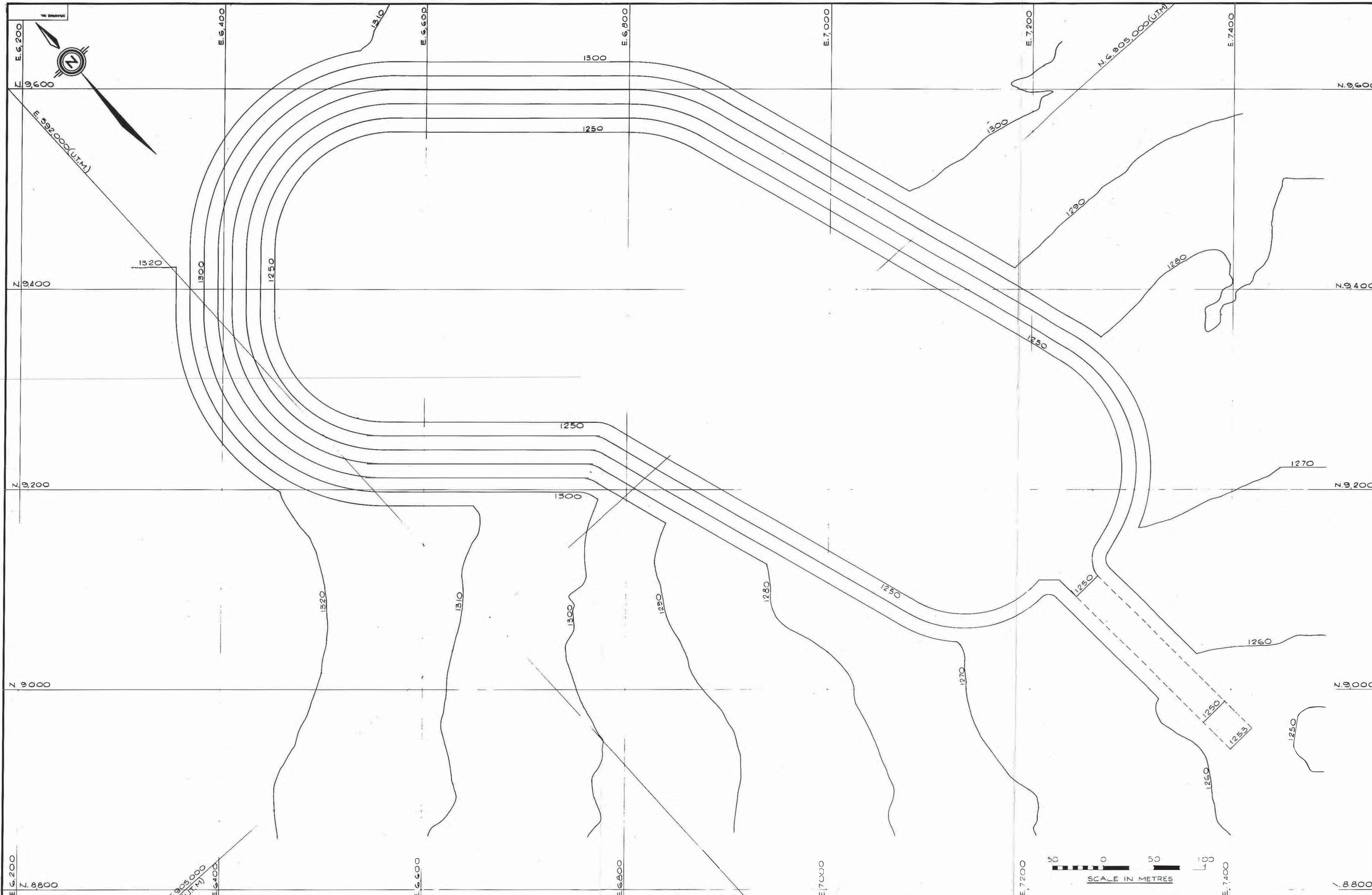
SCALE 1:1200 DATE
DESIGNED BY J.B.F. FEB.87
DRAWN BY P.A.H. MAR 87
CHECKED BY
APPROVED BY JEF

CLIENT CURRAGH RESOURCES
LOCATION FARO YUKON
KILBORN

TITLE
FARO AREA DEPOSITS
GRUM DEPOSIT
OPEN PIT PLAN
END OF 1989

B.O.M. No.
PROJECT No. 3509
DIVISION No. 19
DRAWING NUMBER FIG.5.8.5-1
REV. A



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REVISIONS

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CHECKED BY
APPROVED BY J.B.F.

CLIENT CURRAGH RESOURCES
LOCATION FARO YUKON

KILBORN

TITLE
FARO AREA DEPOSITS
GRUM DEPOSIT
OPEN PIT PLAN
END OF 1990

B.O.M. No
PROJECT No. 3509
DIVISION No. 19
DRAWING NUMBER FIG.5.8.5-2
REV. A

(e) Mining - 1991 [Figure 5.8.5-3]

During 1991, 1.7 million tonnes of ore will be produced from stage one of the Grum pit along with 29.2 million tonnes of waste from stages one and two. At year-end, three shovels will be operating in the Grum pit.
Pit status at the end of 1991 is shown in Figure 5.8.5-3.

(f) Mining - 1992 [Figure 5.8.5-4]

Ore production will continue to come from the stage one pit with 2.3 million tonnes of ore being released. Stripping will continue on the stage two pit.

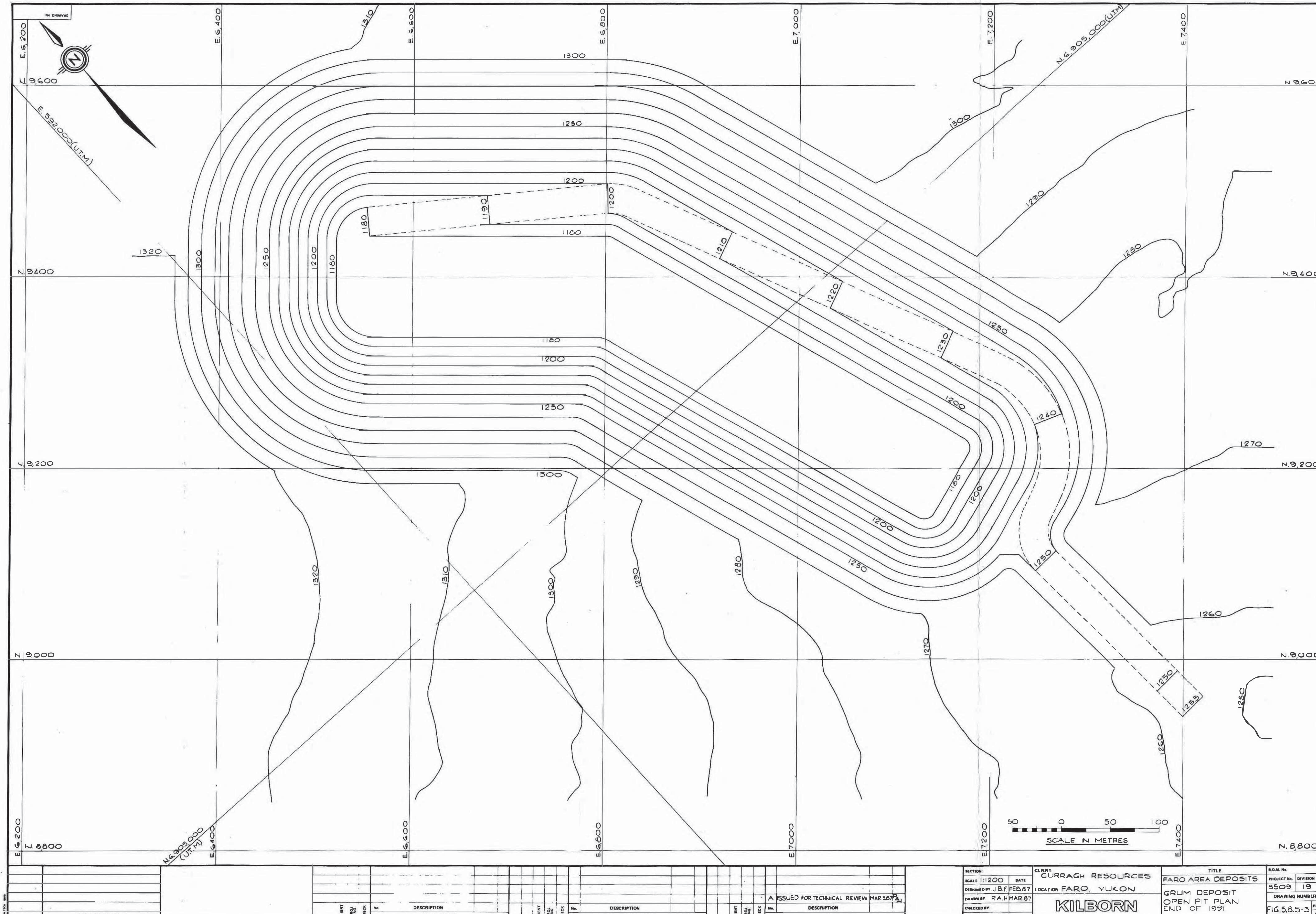
A total of 31.7 million tonnes of waste will be mined during the year. At completion of the Faro pit, a fourth shovel will be moved into the Grum pit.
Pit status at the end of 1992 is shown in Figure 5.8.5-4.

(g) Mining - 1993 [Figure 5.8.5-5]

During the year, ore production will come primarily from stage one with only a small portion liberated in stage two. Stripping of stage two will continue. A total of 3.8 million tonnes of ore and 34.3 million tonnes of waste will be mined.
Pit status at the end of 1993 is shown in Figure 5.8.5-5.

(h) Mining - 1994 [Figure 5.8.5-6]

During early 1994, stage one will be completed and stage two will become the primary source of ore production. During the second quarter of the year, stripping will start on stage three. A total of 4.6 million tonnes of ore and 32.2 million tonnes of waste will be mined.
Pit status at the end of 1994 is shown in Figure 5.8.5-6.



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1310
1290
1280
1270
1260
1210
1220
1230
1240
1253
50 0 50 100
SCALE IN METRES
A ISSUED FOR TECHNICAL REVIEW MAR.3.87
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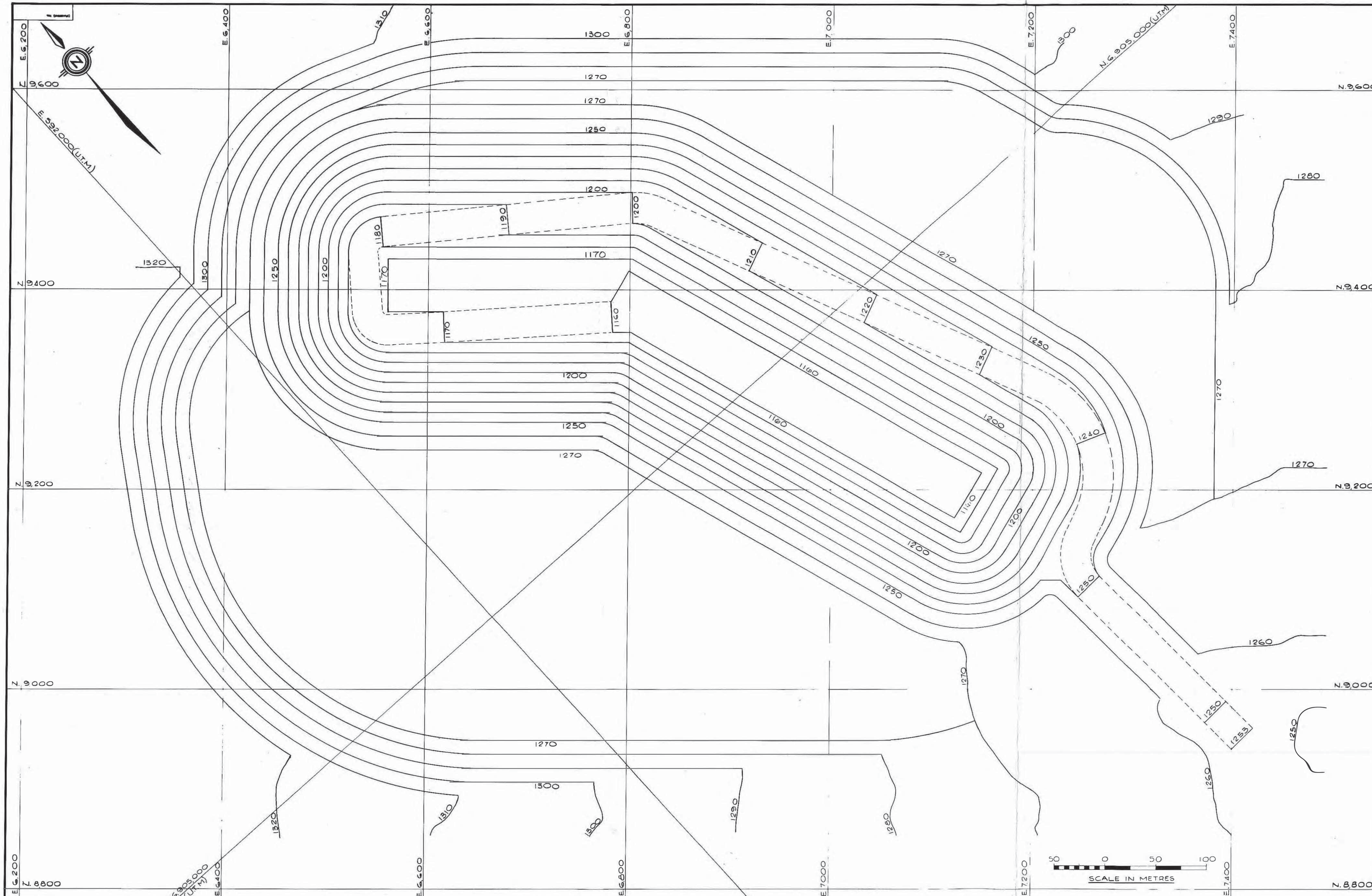
DRAWN BY. P.A.H MAR.87
CHECKED BY:
APPROVED BY. JBF
CLIENT: CURRAGH RESOURCES
LOCATION. FARO, YUKON
KILBORN
TITLE
FARO AREA DEPOSITS
GRUM DEPOSIT
OPEN PIT PLAN
END OF 1991
B.O.M. No.
PROJECT No. 3509
DIVISION No. 19
DRAWING NUMBER
FIG.5.8.5-3
REV. A



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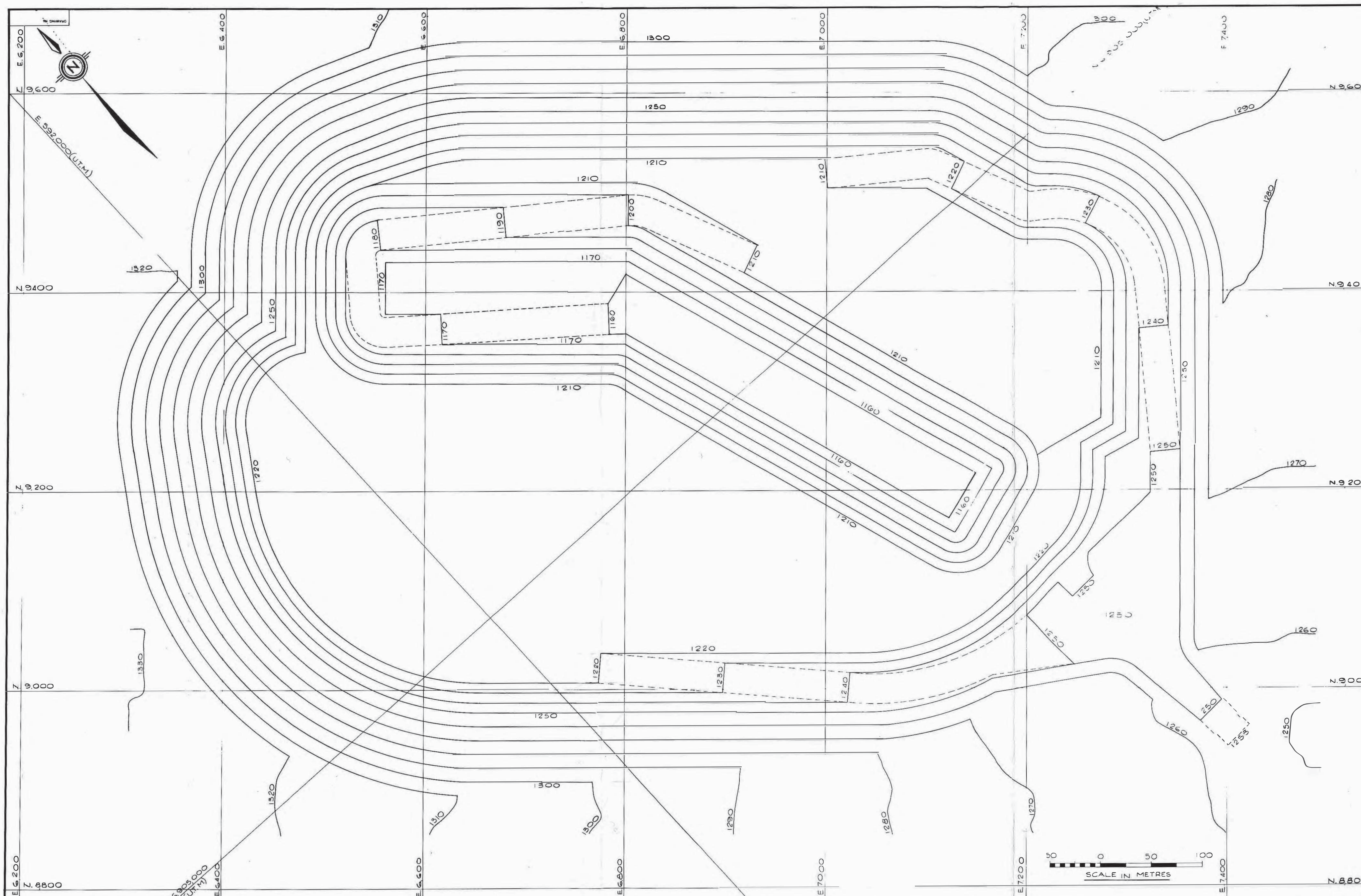
CLIENT: CURRAGH RESOURCES
LOCATION: FARO YUKON

KILBORN

TITLE
FARO AREA DEPOSITS
GRUM DEPOSIT
OPEN PIT PLAN
END OF 1992

B.O.M. No.	
PROJECT No.	DIVISION No.
3509	19
DRAWING NUMBER	REV.
FIG.5.8.5-4	A



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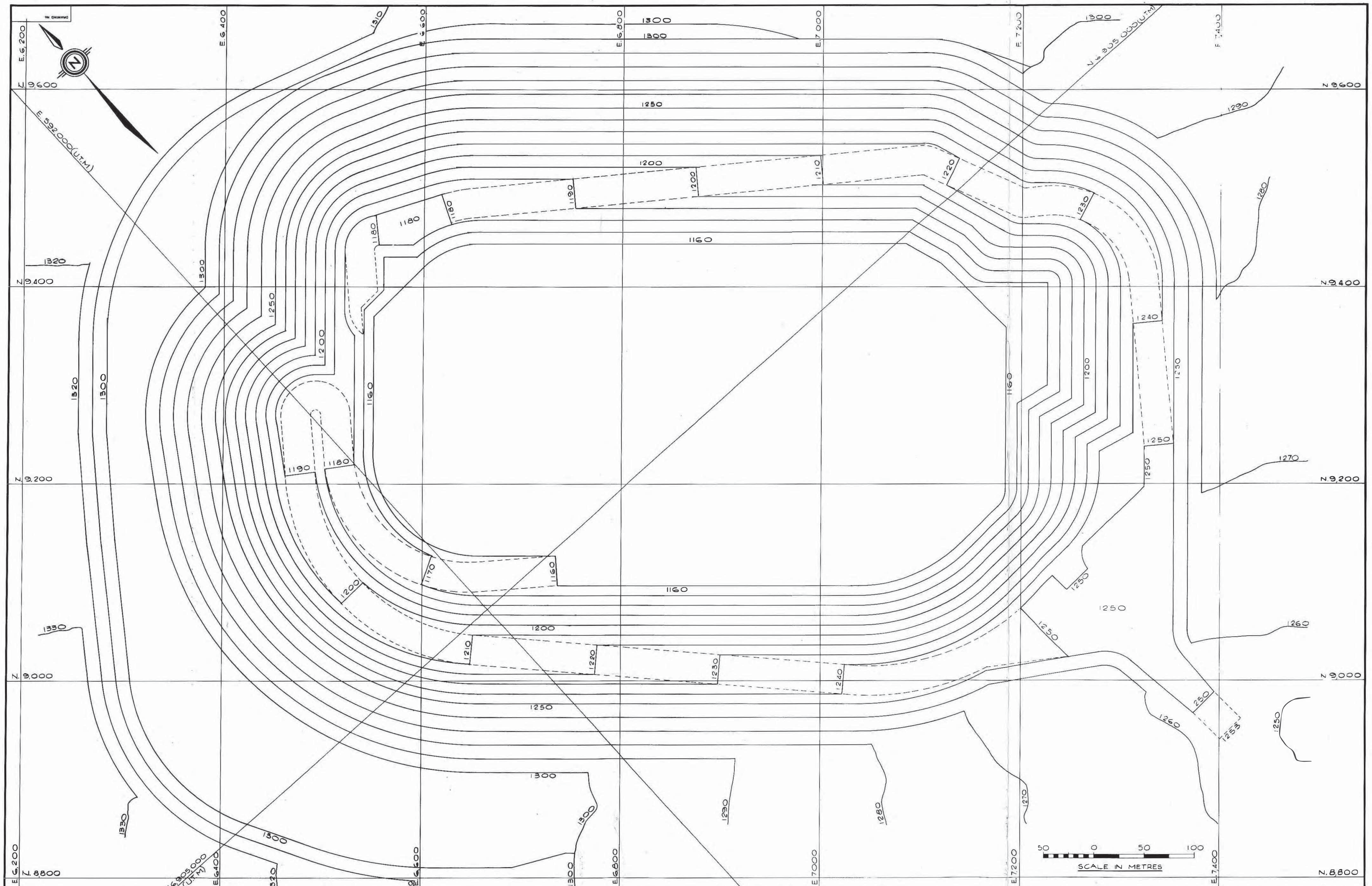
CLIENT: CURRAGH RESOURCES
LOCATION: FARO, YUKON

KILBORN

TITLE: FARO AREA DEPOSITS
GRUM DEPOSIT
OPEN PIT PLAN
END OF 1993

PROJECT No. 3509 | DIVISION No. 19
DRAWING NUMBER FIG.5.8.5-5 | REV. A



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APPROVED BY JEF

A ISSUED FOR TECHNICAL REVIEW MAR.3 87

CLIENT: CURRAGH RESOURCES
LOCATION: FARO, YUKON
KILBORN

TITLE: FARO AREA DEPOSITS
GRUM DEPOSIT
OPEN PIT PLAN
END OF 1994

PROJECT No. 3509 DIVISION No. 19
DRAWING NUMBER FIG.5.8.5-6 REV. A

(i) Mining - 1995 [Figure 5.8.5-7]

Ore production will be 4.7 million tonnes from the stage two pit during the year. Associated waste from the stage two pit and stripping in the stage three pit will produce 29.4 million tonnes of waste during 1995. During the third quarter, a shovel will be retired from service and three shovels will be operating in the pit at the end of the year.
Pit status at the end of 1995 is shown in Figure 5.8.5-7.

(j) Mining - 1996 [Figure 5.8.5-8]

The stage two pit will be completed during the year and the stage three pit will become the ore source. A total of 3.6 million tonnes of ore and 27.8 million tonnes of waste will be mined. During the second quarter of the year, operating shovels will be reduced from three to two.
Pit status at the end of 1996 is shown in Figure 5.8.5-8.

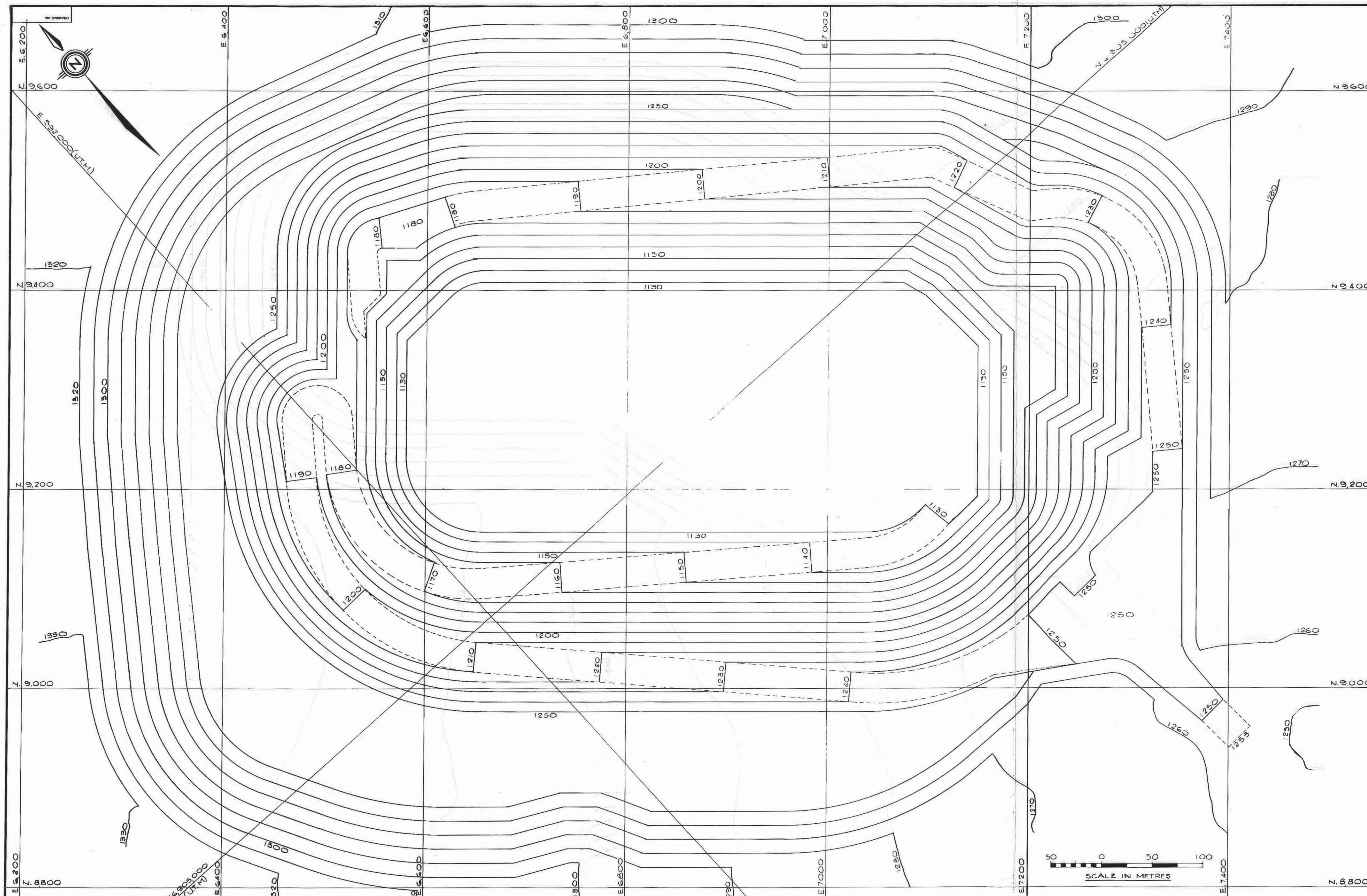
(k) Mining - 1997 [Figure 5.8.5-9]

The Grum pit will be completed during 1997. Production will be 3.9 million tonnes of ore and 3.7 million tonnes of waste. Pit status at the end of open pit mining in Grum is shown in Figure 5.8.5-9.

5.9 VANGORDA/GRUM SITE FACILITIES

5.9.1 General

The site development for the mining of the Vangorda and Grum deposit will consist primarily of road access, water diversion, site clearing, electrical power supply, changehouse and a shop for equipment servicing. The site plan showing pit and dump locations is given in Figure 5.9.1-1. The ore haulage road is shown on Figure 5.9.1-2.



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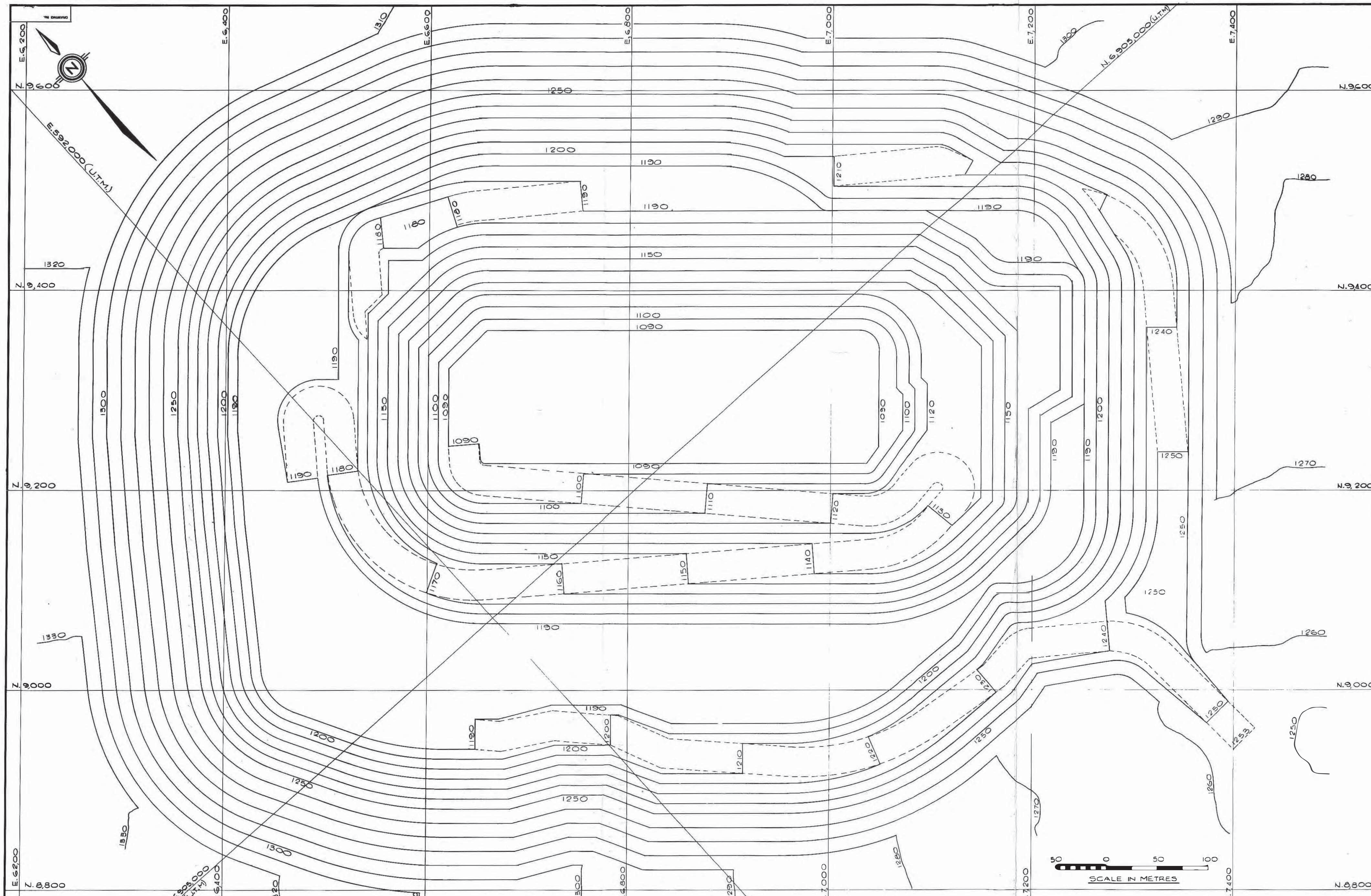
APPROVED BY	JBF
CLIENT	CURRAGH RESOURCES
LOCATION	FARO, YUKON
	KILBORN
TITLE	FARO AREA DEPOSITS GRUM DEPOSIT OPEN PIT PLAN END OF 1995
PROJECT No.	3509
DIVISION No.	19
DRAWING NUMBER	FIG. 5.8.5-7
REV.	A



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DESIGNED BY: J.B.F.	FEB 87	KILBORN	CRUM DEPOSIT OPEN PIT PLAN END OF 1996	
DRAWN BY: P.A.H.	FEB.87			FIG.5.8.5-8
CHECKED BY:				REV. A
APPROVED BY: J.B.F.				

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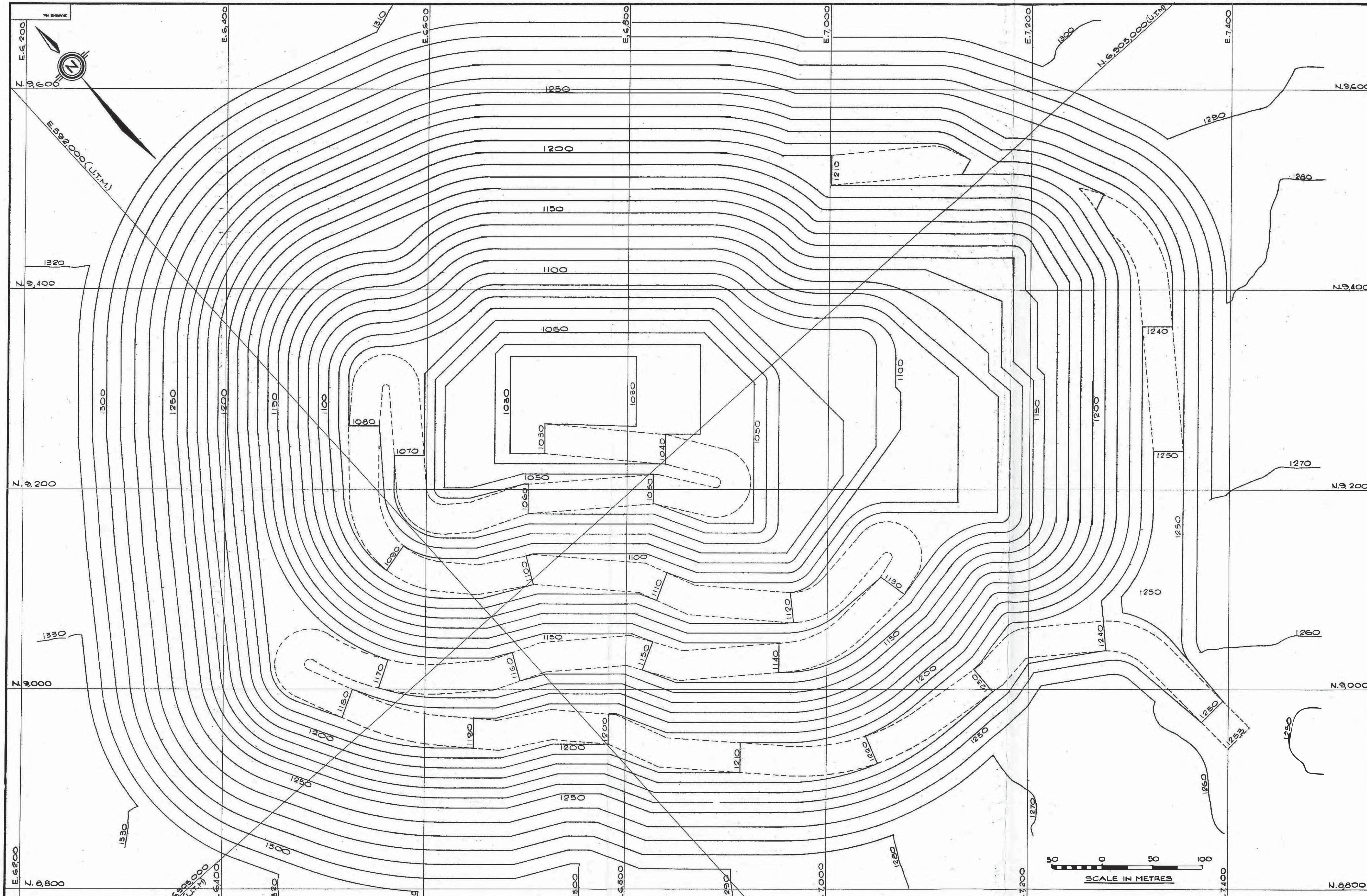

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APPROVED BY: JBF	

CLIENT: CURRAGH RESOURCES

LOCATION: FARO, YUKON.

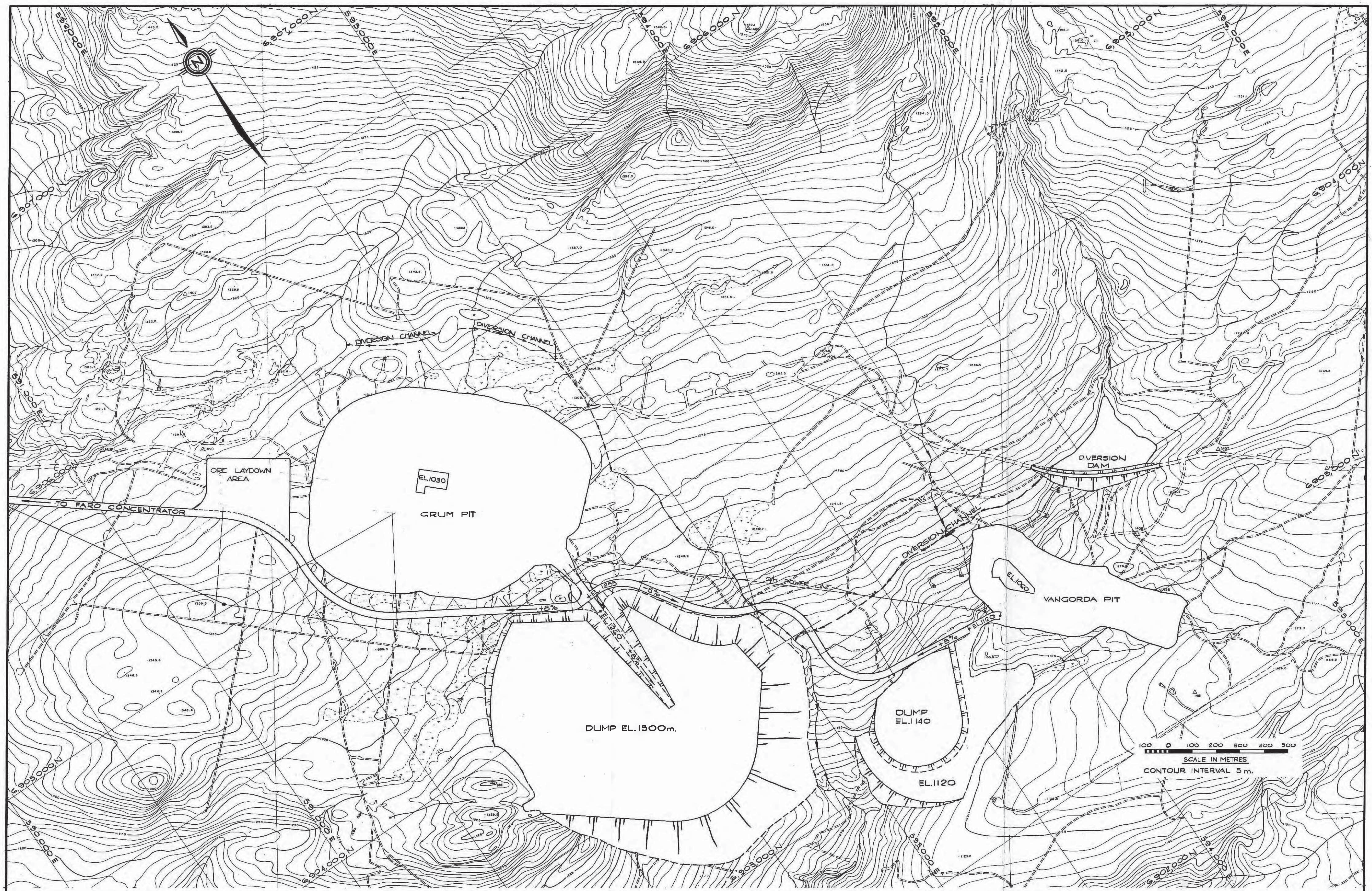
KILBORN

TITLE

FARO AREA DEPOSITS

GRUM DEPOSIT
OPEN PIT PLAN
MAY 1997 - ULTIMATE

B.O.M. No.	
PROJECT No.	DIVISION No.
3509	19
DRAWING NUMBER	REV.
FIG. 5.8.5-9	A



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REVISIONS

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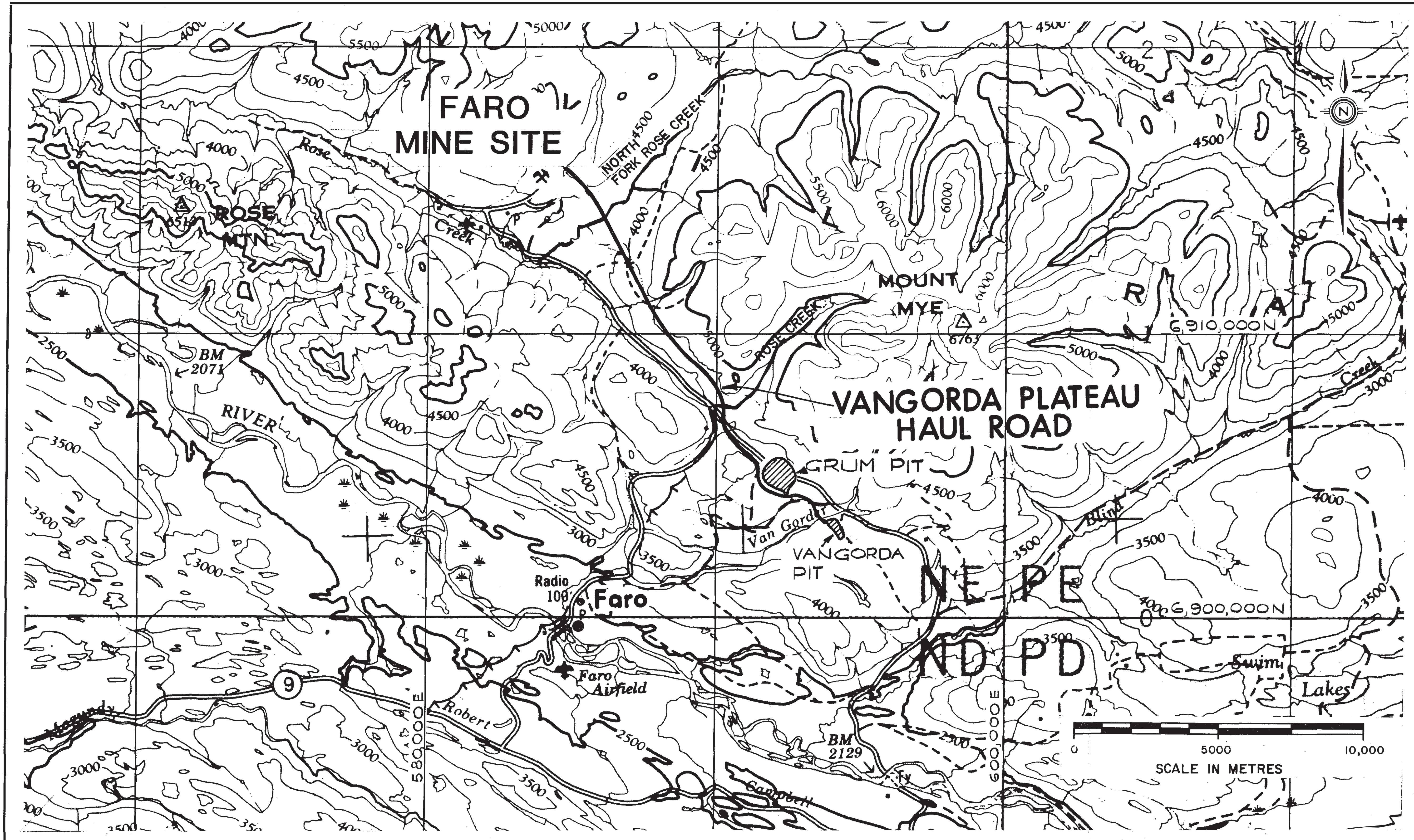
APPROVED BY: JBF		

CLIENT: CURRAGH RESOURCES
LOCATION: FARO, YUKON

KILBORN

TITLE
FARO AREA DEPOSITS
VANGORDA PLATEAU SITE PLAN

B.O.M. No.
PROJECT No. 3509 DIVISION No. 19
DRAWING NUMBER FIG.5.9.1-1 REV. A



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				DRAWN P.A.H.	FEB.87
ISSUED FOR TECHNICAL REVIEW	A	MAR87	PAH	CHECKED	
REVISIONS	NO	DATE	BY	APPROVED	

CLIENT CURRAGH RESOURCES

LOCATION FARO, YUKON

KILBORN

FARO AREA DEPOSITS
VANGORDA PLATEAU
HAUL ROAD

PROJ. No 3509-19 | DWG. No. FIG 5.9.1-2 | REV. A

K.E.B.C.9

5.9.2 Site

Site clearing will be undertaken during 1987. The areas to be cleared are lightly to moderately forested. There is existing road access to the site. Start of work is governed only by weather, environmental permits and mobilization time.

The two waste dump locations and final open pits are shown on Figure 5.9.1-1. Their location necessitates the redirection of a number of water courses. Ditching will be required around the upper slopes of the pits to divert water from existing water courses.

5.9.3 Site Facilities

The planned operation takes into consideration the continued use of maintenance facilities, warehouse and offices at the Faro site. Minimal facilities will be provided at the Vangorda/Grum site.

A two bay running maintenance and lubrication shop with a small attached ready-use warehouse. This building will be a pre-engineered structure without crane facilities. Shop dimensions will be 23 metres by 20 metres. The warehouse and office will be a 5 metre by 24 metre lean-to attached to the shop.

A prefabricated changehouse will be provided for the mining crew. Facilities will be provided for both men and women. Total locker capacity will be for 100 people.

5.9.4 Pit Dumps

Two separate waste dumps will be provided in one common watershed. One dump will be provided for each pit. The dump sites are near the pit exits and will fill two ravines. Dump locations permit a short flat haul for waste from the pit edge.

5.9.5 Ore Storage and Haulage

Ore from the Vangorda and Grum pit will be stockpiled in the pit area and hauled to the Faro Mill as required. The haulage road will be 14 kilometres in length with maximum grades of 2 percent against the load.

5.9.6 Power Supply

Electrical power will be provided by a short line from the main NCPC power line to the Faro Site. The power line will be 5,000 metres in length. Power will be transmitted transformed down from 138 kv to 34.5 kv and transmitted to the mine site at 34.5 kv. In the pit area, it will be further transformed to 4.16 kv for pit use.

6

6.0 FARO UNDERGROUND MINING

6.0 FARO UNDERGROUND MINING

6.1 GENERAL

Southwest of the ore being mined in the Faro Open pit is a zone of high grade lead zinc ore which will be mined by underground methods.

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

a) The mineralized tabular zones dip at 15 to 25 degrees;
b) The rock comprising the footwall and the hanging wall is mineralized and the grades similar to the material being mined within the open pit.

The dip of the zones is well below the angle of repose of broken rock and gravity cannot be used to move broken ore. The dip is too steep to allow the use of rubber tired equipment within the stopes.

The competent rocks, however, allow the use of a mining method requiring relatively large areas of exposed, unsupported back.

Open stoping methods will be used. Open stopes will be mined up dip and pillars left between stopes. These pillars will be partially removed during final mining retreat to give a final overall extraction of 75 percent.

The thickness which will be mined are normally greater than 3 metres. A minimum mining height of 2.1 metres has been used.

Mine water inflows are not expected to be a major problem. The wet areas encountered within the Faro open pit appear to be associated with faults and most of the rocks have low permeability.

The mine will be operated on a two - ten hour shift, five day per week basis. Production per operating day will be 2000 tonnes. Access will be provided by a decline. Hydraulic jumbos and trackless loaders and trucks will be used to drive the decline and the access drifts to the stoping areas. Stope mining operations will be conducted with short hole, portable drills and electric slushers.

6.2 ORE RESERVES

The ore zone shown on Figure 6.2-1 is an extension of the orebody being mined by the Faro open pit. The reserves are based on surface drill holes as indicated in the figure. A geolocial reserve was calculated by the polygon method using the following parameters:

a) Minimum insitu grade 9 percent Pb + Zn.
b) Minimum mining height 2.1 metres.
c) Maximum radius of influence 46 metres.
d) Minimum pillar between underground reserves and final open pit wall is 15 metres.

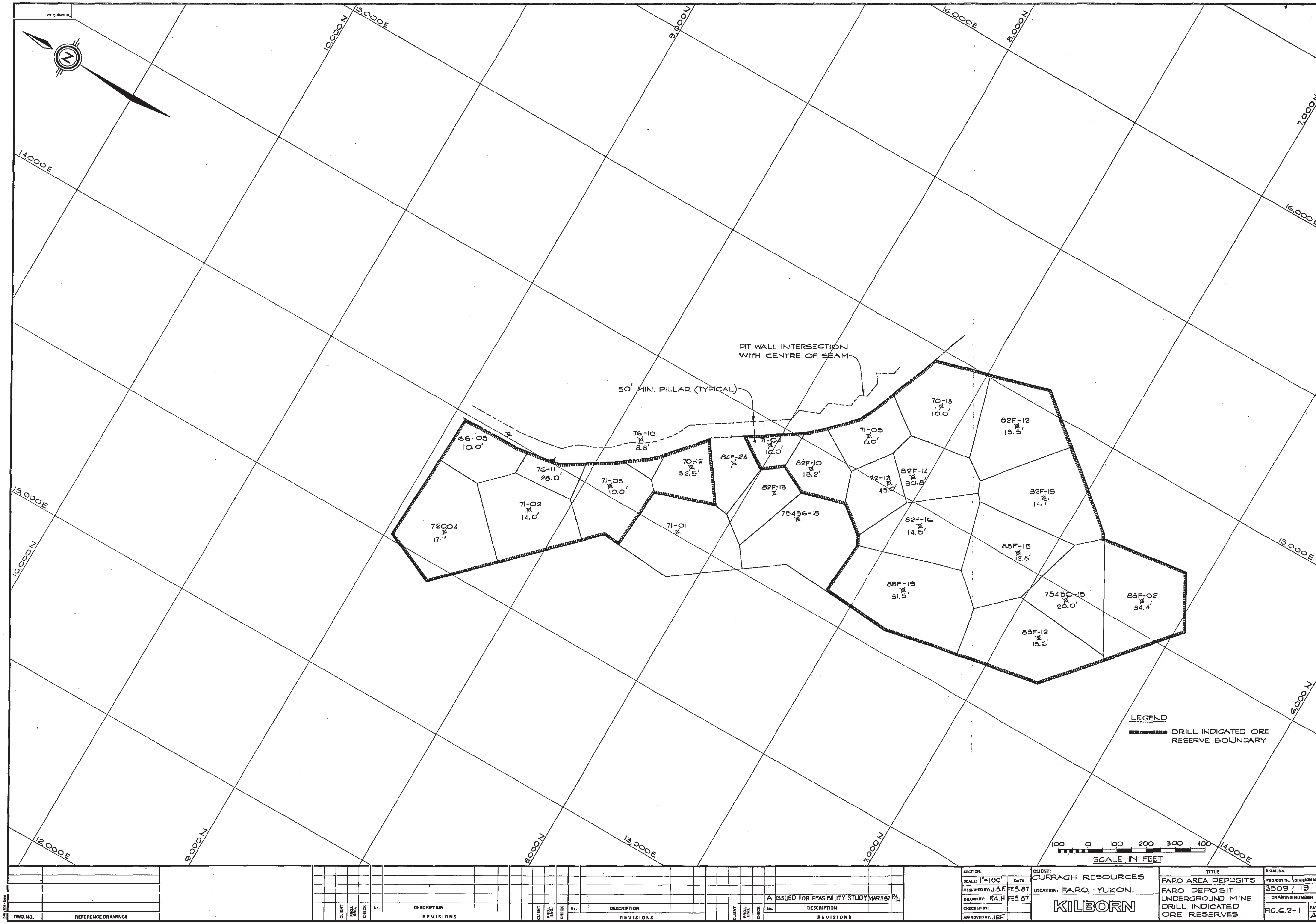
These reserves are considered to be in the probable catagory.

Geological reserves are given in table 6.2-1.

Mining reserves are calculated from geological reserves on the following parameters:

a) Dilution - 10 percent at zero grade.
b) Mining Recovery - 75 percent of inplace reserves.

Mining Reserves are given in table 6.2-1.



PIT WALL INTERSECTION WITH CENTRE OF SEAM
50' MIN. PILLAR (TYPICAL)
66-05 10.0'
76-10 8.8'
76-11 28.0'
71-03 10.0'
71-02 14.0'
72004 17.1'
70-12 32.5'
84F-24
71-04 10.0'
82F-10 13.2'
82F-13
71-01
75456-18
71-05 10.0'
70-13 10.0'
72-13 45.0'
82F-14 30.8'
82F-16 14.5'
82F-12 13.5'
82F-15 14.7'
83F-15 12.8'
83F-19 31.5'
75456-15 20.0'
83F-02 34.4'
83F-12 15.6'
LEGEND
DRILL INDICATED ORE RESERVE BOUNDARY
SCALE IN FEET
A ISSUED FOR FEASIBILITY STUDY MAR.387 P.A.H.
SCALE: 1"=100'
DESIGNED BY: J.B.F. FEB.87
DRAWN BY: P.A.H FEB.87
APPROVED BY: JBF
CLIENT: CURRAGH RESOURCES
LOCATION: FARO, YUKON.
KILBORN
FARO AREA DEPOSITS
FARO DEPOSIT UNDERGROUND MINE DRILL INDICATED ORE RESERVES
PROJECT No. 3509
DIVISION No. 19
DRAWING NUMBER FIG.6.2-1
REV. A

Table 6.2-1

Geological Reserves

Ore Type	Lead Percent	Zinc Percent	Silver gm/tonne	Quantity tonnes
2A	1.98	3.88	13.86	58,000
2BG	5.09	7.69	67.51	2,288,000
2H	5.21	9.26	82.17	264,000
TOTAL	5.04	7.76	67.90	2,610,000
Mining Reserve				
TOTAL	4.59	7.00	61.29	2,014,000

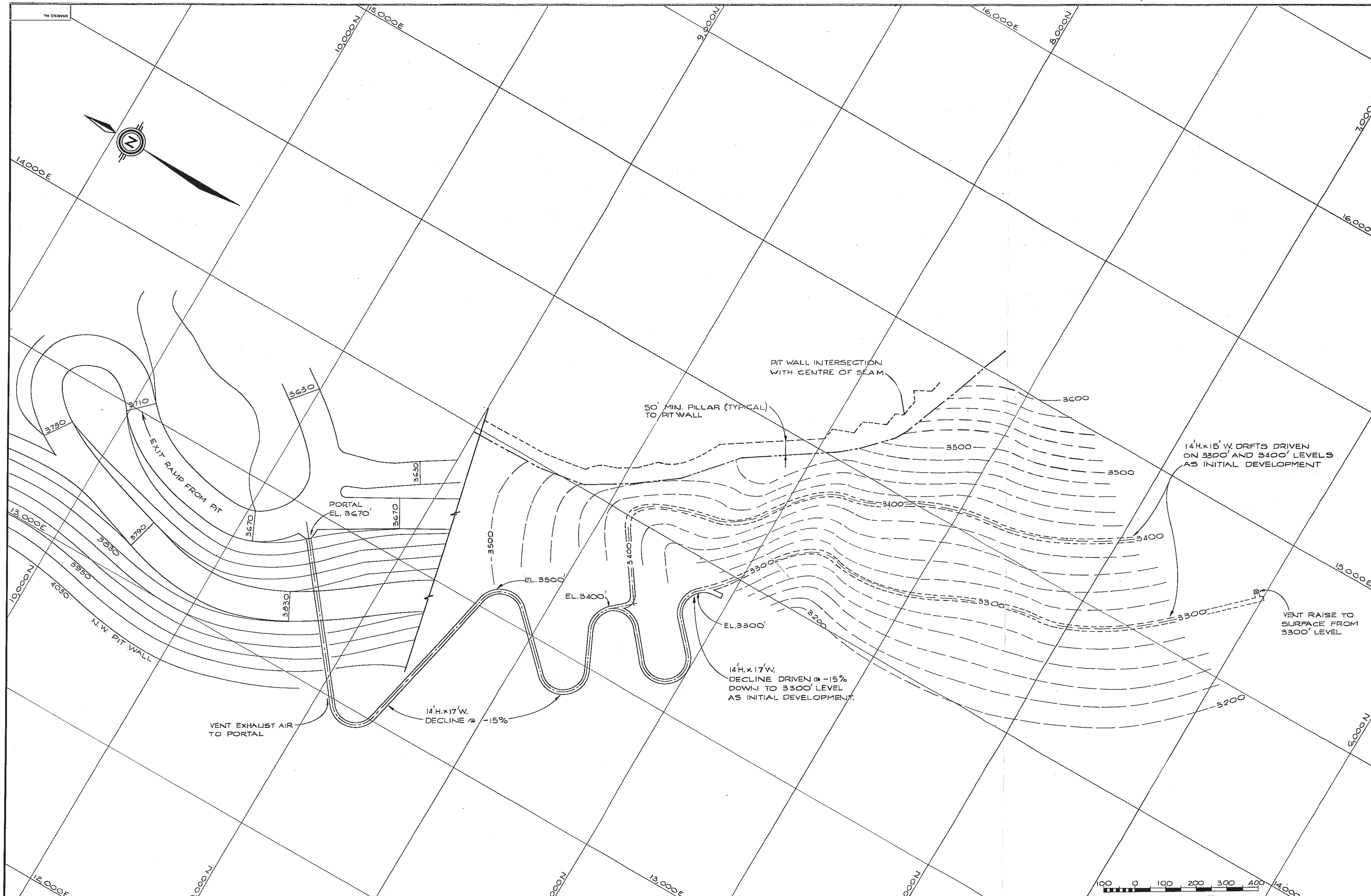
6.3 MAJOR DEVELOPMENT

Major development prior to the start of production is shown on Figure 6.3-1 and comprises 3 main areas:

- Main access decline;
- Mining levels;
- Ventilation Raise.

The main access decline will be collared on the west wall of the existing open pit at 3670 ft elevation. The grade of the decline will be 15 percent. Dimensions of the decline will be 4.3 metres high by 5.2 metres wide. This decline will be positioned to permit intersection of the bottom of the ore zone at 30 metres vertical intervals. Prior to start of production 1219 metres of decline will be driven. Two main level drifts will be driven as part of the preproduction development at 3300 and 3400 ft elevations. This major drifting will amount to 1128 metres of which 579 metres will be at the 3400 ft elevation. The remaining 550 metres of pre-production development will be at the 3200 ft elevation. These drifts will have the same dimensions as the decline.

The main ventilation downcast raise will be driven from the end of the 3300 ft level to surface in a distance of 245 metres. This raise will be a 2.4 metre diameter bored raise fitted with an emergency escape manway.



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CLIENT: CURRAGH RESOURCES

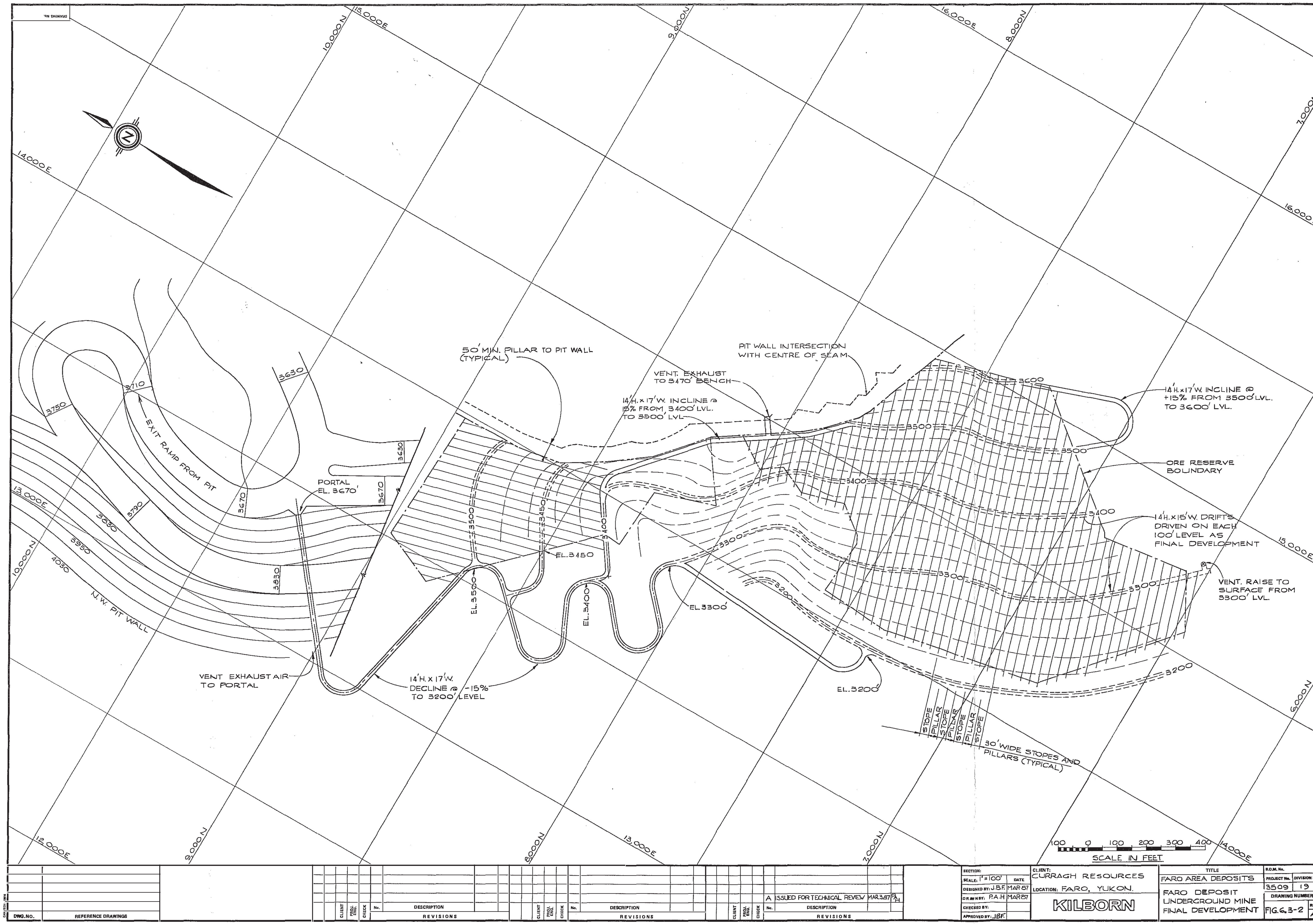
LOCATION: FARO, YUKON.

KILBORN

TITLE: FARO AREA DEPOSITS

FARO DEPOSIT
UNDERGROUND MINE
INITIAL DEVELOPMENT

B.O.M. No.	
PROJECT No.	DIVISION No.
3509	19
DRAWING NUMBER	REV.
FIG. 6.3-1	A



N
15,000E
16,000E
14,000E
13,000E
12,000E
10,000N
9,000N
8,000N
7,000N
6,000N
3710
3750
3630
3670
3630
3790
3830
3950
4030
EXIT RAMP FROM PIT
N.W. PIT WALL
PORTAL EL. 3670'
50' MIN. PILLAR TO PIT WALL (TYPICAL)
VENT. EXHAUST TO 3470' BENCH
PIT WALL INTERSECTION WITH CENTRE OF SEAM
14'H.x17'W. INCLINE @ 15% FROM 3400' LVL. TO 3500' LVL.
14'H.x17'W. INCLINE @ +15% FROM 3500' LVL. TO 3600' LVL.
3600
3500
3400
3300
3200
ORE RESERVE BOUNDARY
14'H.x15'W. DRIFTS DRIVEN ON EACH 100' LEVEL AS FINAL DEVELOPMENT
VENT. RAISE TO SURFACE FROM 3300' LVL.
EL.3450
EL.3500
EL.3400
EL.3300'
EL.3200
VENT EXHAUST AIR TO PORTAL
14'H.x17'W. DECLINE @ -15% TO 3200' LEVEL
STOPE
PILLAR
STOPE
PILLAR
STOPE
PILLAR
STOPE
30' WIDE STOPES AND PILLARS (TYPICAL)
100 0 100 200 300 400
SCALE IN FEET
DWG. NO.
REFERENCE DRAWINGS
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DESCRIPTION
REVISIONS
A ISSUED FOR TECHNICAL REVIEW MAR.3.87
SECTION:
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DESIGNED BY: J.B.F. MAR.87
DRAWN BY: P.A.H. MAR.87
CHECKED BY:
APPROVED BY: JBF
CLIENT: CURRAGH RESOURCES
LOCATION: FARO, YUKON.
KILBORN
TITLE
FARO AREA DEPOSITS
FARO DEPOSIT UNDERGROUND MINE FINAL DEVELOPMENT
PROJECT No. 3509
DIVISION No. 19
DRAWING NUMBER FIG.6.3-2
REV. A

Total major development for the life of the mine is shown on Figure 6.3-2 and includes further drifting on the 3200 ft., 3400 ft. and 3500 ft. elevations. This additional ongoing development amounts to 1433 metres in length.

6.4 STOPE DEVELOPMENT

Stope development will be similar for both stopes in thick ore zones and stopes in thinner ore zones. This development for a stope, which can be seen on Figure 6.4-1, includes:

- slusher station with access raise
- drawpoint
- stope raise

Stope development will be positioned in ore to minimize the excavation of waste.

6.5 STOPING METHOD

The stoping method which will be used is up-dip open-stoping with air leg pneumatic drills and electric powered slusher/scrapers for ore removal to the stope drawpoint. Typical stopes are shown on Figure 6.4-1.

The proceedure will be as follows:

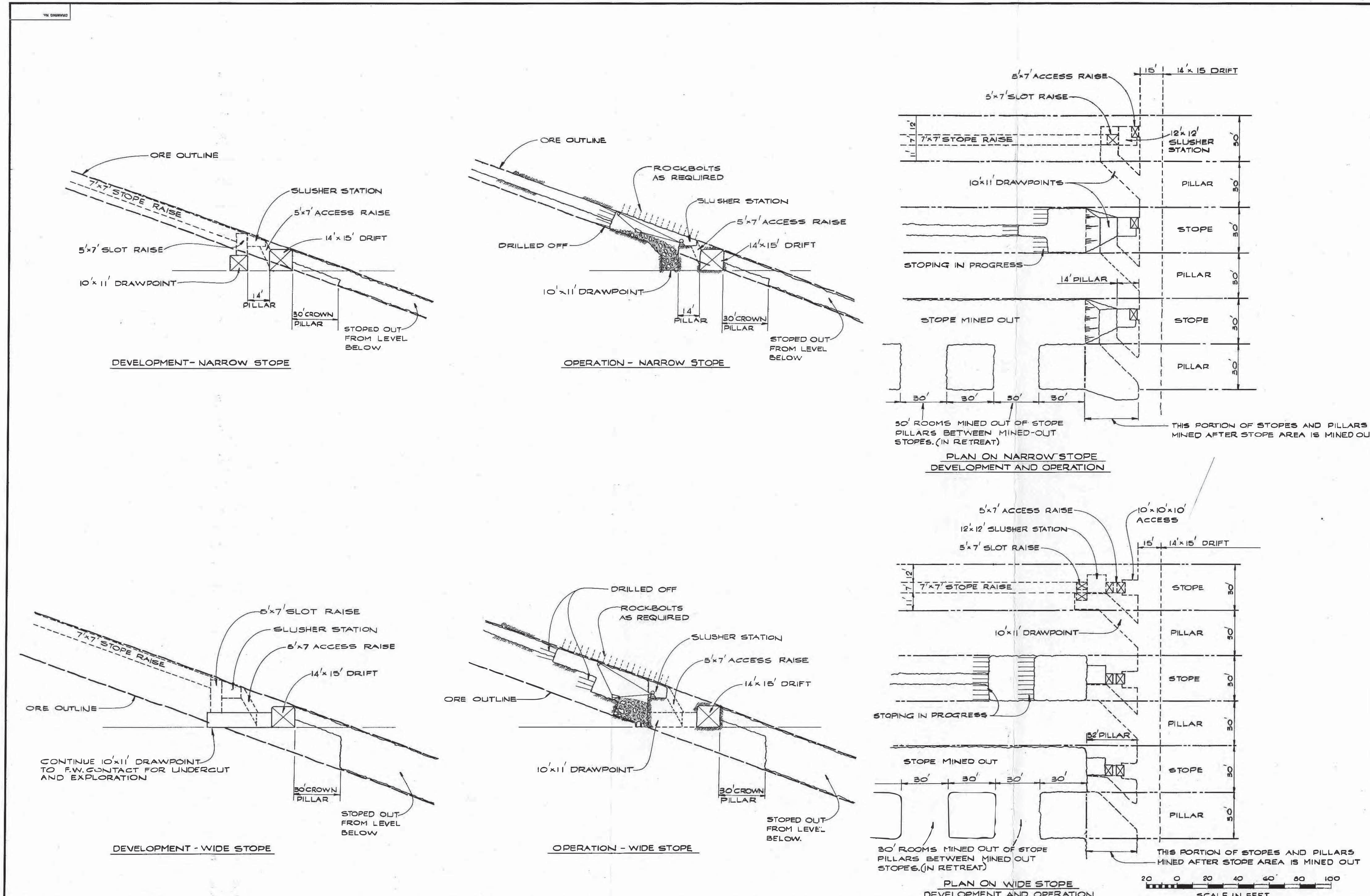
a) A slot raise will be excavated from the top of the drawpoint drift to the top of the ore.

b) An access raise from the main level drift to the slusher hoist station and the slusher hoist station will be excavated.

c) The slusher hoist and scraper will be installed.

d) A 2.1 metre slot raise will be driven along the back of the stope from the top of the slot raise to the level above.

e) The ore body will be mined up dip from the slot raise to the crown pillar. Drill holes will be drilled up dip as shown on Figure 6.4-1. Hole length will be 10 feet. Initial mining height will be 3 metres or ore thickness if less than 3 metres. The stope back will be rockbolted as required.



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APPROVED BY: J.B.F.	

CLIENT: CURRAGH RESOURCES
LOCATION: FARO, YUKON.

KILBORN

TITLE: FARO AREA DEPOSITS
FARO DEPOSIT
UNDERGROUND MINE
STOPING DETAILS

PROJECT No.	DIVISION No.
3509	19

DRAWING NUMBER: FIG. 6-4-1 REV. A

f) If the ore is over 3 metres in thickness successive lifts of up to 3 metres will be mined from the stope.

g) Broken ore will be removed to the drawpoint with the scraper as required for mining.

h) After all ore is broken any remaining broken ore within the stope will be slushered to the drawpoint.

i) After mining of two primary stopes, the pillar will be partially mined to leave square 9.1 metres by 9.1 metres pillars and give a 75 percent mining extraction.

A total of 10 stopes operating on a two shift per day schedule will be active at any one time.

6.6 LOADING AND HAULAGE

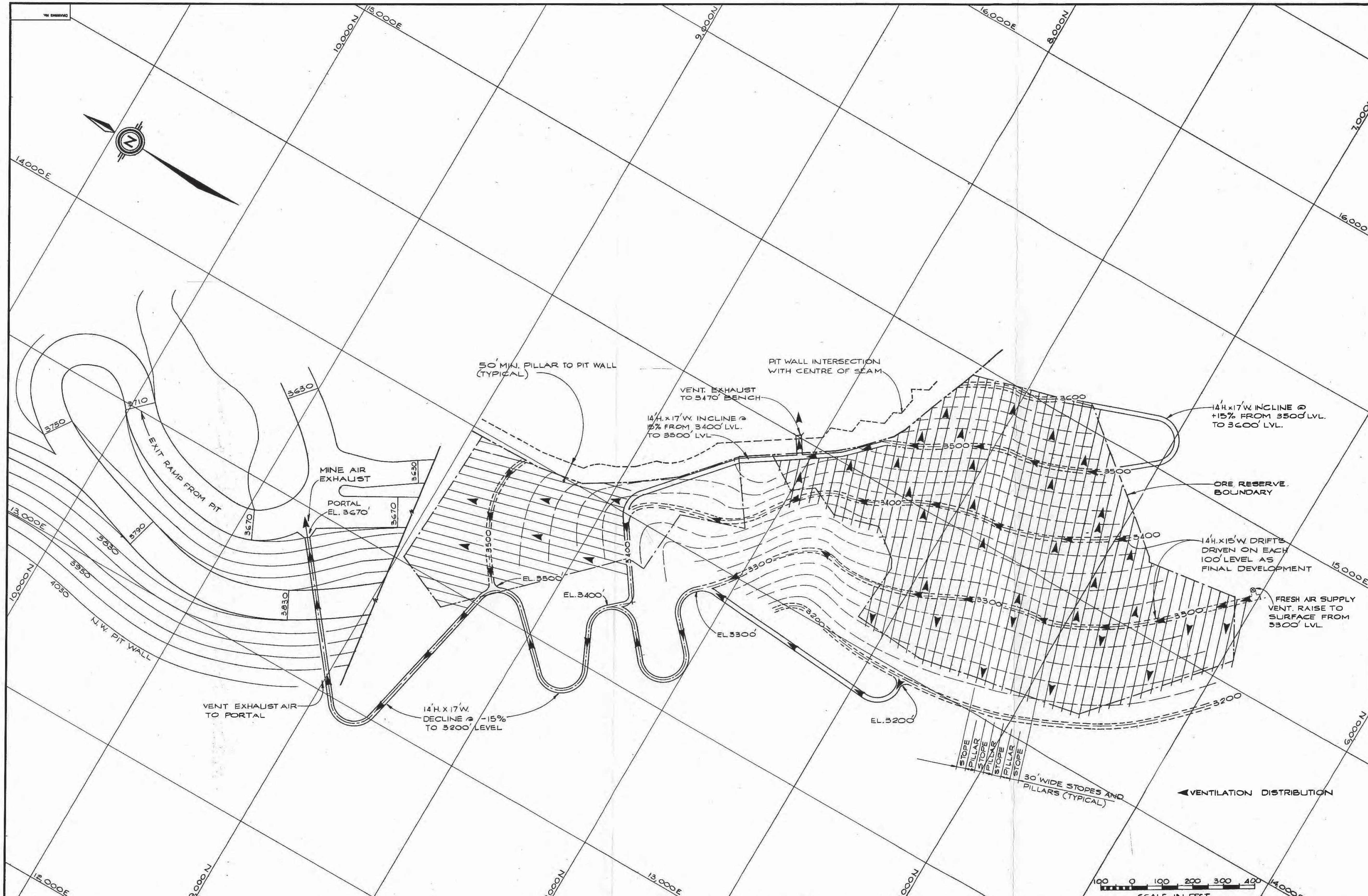
Ore will be removed from the stope drawpoints with 4.6 m^3 LHDs and 3 35-ton trucks will be required to handle the ore produced in stoping and the material from ongoing development.

The ore will be hauled to a stockpile area immediately outside the mine portal from where it will be loaded into open pit haulage trucks and hauled to the crusher as needed.

6.7 VENTILATION AND MINE AIR HEATING

The main ventilation system, as shown on Figure 6.7.3-1, will be down the main ventilation raise from surface to the 3300 ft level and then upwards and downwards through the stoping blocks and hence exiting the mine through the decline system and exhaust drifts into the open pit.

Ventilation requirements are controlled primarily by the amount of diesel equipment operating within the mine. An allowance of 300,000 cfm is considered adequate for this operation.



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DRAWN BY: P.A.H. MAR 87
CHECKED BY:
APPROVED BY: JBF

CLIENT: CURRAGH RESOURCES
LOCATION: FARO, YUKON.

KILBORN

TITLE: FARO AREA DEPOSITS
FARO DEPOSIT
UNDERGROUND MINE
VENTILATION
DISTRIBUTION

PROJECT No. 3509 DIVISION No. 19
DRAWING NUMBER FIG. 6.7.3-1 REV. A

The main ventilation fans will be located at the top of the ventilation raise and will pressurize the mine workings. Within the mine, air flows to the various working areas will be controlled with auxiliary fans and bulkheads.

Mine air will require heating during sub zero weather to ensure that waterlines and drainage systems within the mine do not freeze. Direct-fire propane heaters will be provided for this service. A propane storage system will be required near the ventilation raise.

6.8 UNDERGROUND SERVICES

Temporary underground services will be carried into the mine through the decline and level drifts until the workings are connected to the ventilation raise. When the ventilation raise is complete, mine services will be brought down the ventilation raise to service the mine. Services which will be provided include electrical power, compressed air and water. The mine has been designed with the majority of the energy requirements being for electrical and diesel powered equipment. Compressed air usage has been minimized.

Mine water drainage will be handled in sumps located at the junctions of the levels and the decline. The level drifts will be driven at a positive one percent grade to provide drainage. Unsettled water will be pumped to surface for disposal with the water from the open pit. Mine water discharge lines will be located in the decline.

Mine communications will be provided by a mine mobile radio system operating in conjunction with coaxial antenna located in the main levels and declines. A stench warning system will be provided in the mine ventilation system and compressed air system for emergencies.

Mobile equipment will be taken to the main surface shop for major maintenance. Routine maintenance will be undertaken in small shop areas within the mine workings.

6.9 RELATED SURFACE FACILITIES

The surface facilities required for the underground mine will be minimal.

Office and changehouse facilities are presently in existance and no increase in capacities will be required.

Maintenance shop facilities are available and equipped for the open pit equipment. These facilities will be used to maintain the underground equipment.

A compressor plant to provide compressed air to the mine at 100 psi will be required. Compressed air requirements are estimated to be 3500 cfm. To permit maintenance and to cover peaks 4500 cfm will be installed. A compressor house of 12.2 metres by 9.1 metres will be required to house this equipment.

Ore will be stockpiled in open piles near the decline collar. This material will be rehandled with open pit equipment.

6.10 MINE PERSONNEL

The total mine compliment is:

Supervision & Technical	14
Operating Labour	72
Maintenance Labour	14
TOTAL	100

The detailed listing are shown on Tables:

6.10-1

6.10-2

6.10-3

Table 6.10-1

Supervision & Technical

Underground Superintendent	1
U/G General Foreman	1
Shift Foreman	4
Maintenance Foreman	1
Sr. Mine Engineer	1
Mine Engineer	1
Mine Technician	1
Surveyor	1
Survey Helper	1
Geologist	1
Geological Technician	1
TOTAL	14

Table 6.10-2

Underground Operating Labour

Stope Miner	40
Development Miner	6
LHD Operator	4
Truck Driver	6
Timberman	2
Timber Helper	2
Utility	6
U/G Labour	6
TOTAL	72

Table 6.10-3

Maintenance Labour

Lead Mechanic	2
Lead Electrician	1
Mechanic	3
Electrician	2
Drill Doctor	1
Oiler	1
Maintenance Helper	4
TOTAL	14

6.11 MAJOR MINE EQUIPMENT

Major underground equipment for the mine includes:

Item	Numbers
2 Boom Hydraulic Jumbo	2
Airleg and Stoper Drills	50
LHD Units - 4.6 cu.m. Capacity	4
Trucks - 35 ton Capacity	3
Slusher Hoists c/w Scrapers	
25 Horsepower Electric	12
Service Vehicles	2
Mine Dewatering Pumps	
Various Horsepowers & Heads	5
Auxiliary Fans	15
Pumps	Lot

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7.0 MILLING

7.0 MILLING

7.1 GENERAL

The Faro Concentrator incorporates the unit processes of crushing, grinding, froth flotation, thickening, filtering and drying to produce lead and zinc concentrates from raw pit ore feed.

The design throughput rate of the plant was initially 5,000 tonnes per day. Substantial changes in all areas except the crushing plant has increased the mill capacity to the present rate which is 13,500 tonnes per day for Faro ore, and 11,000 tonnes per day for the finer grained Vangorda and Grum ores.

7.2 METALLURGY

The metallurgical characteristics of the Anvil District ores are profoundly influenced by the following factors:

a) The host deposit from which the ore is extracted;
b) The mineralogical constitution of the ore;
c) The fineness of grind.

The Anvil District deposits consist of a number of mineralogically different ore species, each of which exhibits its own 'metallurgical profile'. Further, it has been established that the response of any particular ore type to differential flotation techniques can vary when samples of the ore are selected from the various deposits.

Metallurgical parameters used in the preparation of this study are based on Curragh Resources operating experience for the Faro orebody. Vangorda Plateau metallurgy is taken from the CAMC report on the development of the Vangorda Plateau.

Experience at the Faro concentrator has established a definite relationship between the zinc content of the lead concentrate and the mercury content of the lead concentrate. This relationship has been used as a guide in the predicted mercury contents of the Vangorda and Grum concentrates.

For all orebodies, recoveries and concentrate grades have been determined for the various ore types. To further refine the predicted recoveries, a factor to compensate for the effect of head grade on recovery is applied. This recovery factor has been derived from current Faro mill operations and is listed below:

Pb factor = 0.181 x ln (pb head grade, %) + 0.767
Zn factor = 0.162 x ln (Zn head grade, %) + 0.733
Ag factor = 0.18 x ln (Wt % to Pb conc) + 0.733
Au factor = 0.18 x ln (Wt % to Pb conc) + 0.733

Silver recoveries of between 54 and 65 percent in the lead concentrates are relatively low. The possibility of recovering more of this silver as well as some gold values should be investigated. One approach might be to preconcentrate the precious metals remaining in the tailings by flotation of the pyrite followed by cyanidation to dissolve the gold and silver values.

7.2.1 Faro Metallurgy

Metallurgical parameters used in this plan are based on current operating experience at the Faro Concentrator.

For planning purposes, the Faro orebody is categorized into three ore types: graphitic ore known as "2A", pyrrhotitic ore known as "2H", and all other ore types referred to collectively as "2BG".

The average predicted metallurgical response is as shown in the following Table 7.2-1:

Table 7.2-1

Predicted Metallurgical Response

Concentrate	Assays				Recovery - %			
	%Pb	%Zn	g/t Au	g/t Ag	Pb	Zn	Au	Ag
Lead	61	6.4	0.9	550	77	-	-	54
Zinc	-	50	-	45	-	81	-	-

Lead concentrate shipped to date has contained quantities of payable gold.

It is planned to stockpile much of the low grade ore, some for as long as 4 years. During 1986, oxide ore from the Faro pit was milled after being stockpiled since the 1970's. Although reagent consumptions were higher than for fresh ore, metallurgical results were acceptable. Therefore no allowance for oxidation is made in this plan.

7.2.2 Vangorda Metallurgy

The Vangorda orebody has been categorized into three ore types: baritic, pyritic, and quartzitic. The predicted average metallurgical response is shown in Table 7.2-2.

Table 7.2-2

Predicted Vangorda Metallurgy

Concentrate	Assays				Recovery - %			
	%Pb	%Zn	g/t Au	g/t Ag	Pb	Zn	Au	Ag
Lead	53	10.6	4.0	604	83	-	35	62
Zinc	-	55	-	-	-	79	-	-

Low grade ore will be stockpiled for a number of years. As for the Faro stockpiled ores, no allowance has been made in this plan for the effect of oxidation on the milling and concentrating of these stockpiled ores.

7.2.3 Grum Metallurgy

Although the Grum orebody is composed of several different ore types, metallurgical parameters in this plan are based on an "average" ore. The metallurgical response has been determined primarily by pilot plant testwork on a bulk sample obtained by underground exploration and bench scale work carried out at Kamloops in 1982. Further metallurgical work will be required on the specific ore types that will be treated year by year in this plan.

The overall predicted metallurgical response is shown in Table 7.2-3.

Table 7.2-3

Predicted Grum Metallurgy

Concentrate	Assays				Recovery - %			
	%Pb	%Zn	g/t Au	g/t Ag	Pb	Zn	Au	Ag
Lead	60	11	3.5	750	80	-	33	65
Zinc	2.5	55	-	-	-	83	-	-

Much of the low grade ore is expected to be stockpiled for a number of years. No allowance has been made in this plan for the effect of oxidation on the milling and concentrating of these stockpiled ores.

7.3 MILL PRODUCTION CRITERIA

The principal criteria applied in the development of this long range production forecast are summarized in Table 7.3-1.

Table 7.3-1

Operating Days Per Year	365
Plant Availability	93 percent
Average Mill Throughputs	
Faro Ore	13,500 tonnes per day
Vangorda and Grum Ore	11,000 tonnes per day
Concentrate Production	
Controlled by Filter Capacity	1,600 tonnes per day

7.4 MILL OPERATIONS

The mill was reactivated in June 1986. To allow for the June 1986 startup, the plant was demothballed and refurbished where required. The process plant was not substantially changed from previous operations but modifications were made in the loadout area to incorporate a new concentrate loading system and reduce cross contamination of the concentrates caused by inadvertent mixing.

8

8.0 SERVICES

8.0 SERVICES

8.1 GENERAL

The present operation has adequate facilities and services for the Faro operation. Site facilities for the Vangorda Plateau are discussed in Section 5.9.

8.2 ELECTRICAL POWER

The Northern Canada Power Commission (NCPC) electrical power generating stations and distribution grid provides power to the Faro minesite.

The Commission generates hydro electric power at Aishihik and Whitehorse. Diesel generation capacity is located at Whitehorse and Faro. NCPC hydro generation capacity is 79.0 MVA and diesel generation capacity is 29.7 MVA. Figure 8.2-1 shows the location of generating facilities and major 138 kV transmission line.

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

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

The current average total mine site energy consumption is 14 GWh per month.

It is anticipated that no increase in average consumption rate will result from the mining of Vangorda Plateau ores.

8.3 TAILINGS DISPOSAL

Tailings facilities for the present operations are located in Rose Creek Valley. The tailings disposal site was expanded in 1980-1981 to increase the capacity sufficiently to permit the mining and treatment of the Vangorda Plateau ores. If all tailings were deposited at this site, two additional lifts would be required on the impoundment structure.

The proposed plan is to place one additional lift on the tailings dam to increase the capacity sufficiently to handle all tailings until the Faro pit is complete and then to deposit tailings into the mined out Faro open pit.

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9.0 GENERAL AND ADMINISTRATION

9.0 GENERAL AND ADMINISTRATION

Curragh functions out of three locations. There is a small Executive Office in Toronto, Ontario. An office, located in Whitehorse, is responsible for concentrate shipping, government liaison, long range 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 Vice President/General Manager. Reporting to him are the Mine Manager, Mill Manager, Comptroller, Materials Manager and Personnel Manager. The Mine Manager and Mill Manager are responsible for all operation, maintenance and associated technical support for their particular areas.

Accommodation for employees at the Faro operations is in the town of Faro. Curragh does not provide housing for its employees. Adequate housing is available for future operations within the town.

10

10.0 TRANSPORTATION

10.0 TRANSPORTATION

10.1 GENERAL

Curragh Resources have 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 20,000 to 35,000 tonne bulk freighters.

10.2 OVERLAND TRANSPORT

Through agreements with the Yukon and Alaska governments, concentrates are trucked under contract to the port of Skagway. Concentrate is shipped in 12.5 tonne containers with 2 containers per trailer for a total of 4 containers per B-train. The concentrate trucking system is based on a design used by Boliden A.B. in northern Sweden.

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.

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

The existing B-Train fleet has the capacity to transport in excess of 600,000 tonnes of concentrates per year, and has up to ten times the required backhaul freight capacity.

10.3 PORT FACILITIES

The concentrate storage and ship loading facilities at Skagway are leased by Curragh Resources from White Pass and Yukon Route. These facilities can handle well in excess of 600,000 tonnes of concentrates annually.

10.4 OCEAN FREIGHT

Concentrates will be shipped from Skagway to the Orient or Europe in 20,000 to 35,000 tonne capacity bulk cargo ships.