THE VANGORDA PLATEAU PROJECT

SUMMARY OF GEOLOGY, PIT DESIGN

METALLURGY AND INFRASTRUCTURE

CURRAGH RESOURCES

MARCH 1987

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VOLUME 1

1.0 INTRODUCTION

This volume summarizes the physical aspects that are the basis for planning the development of the Vangorda Plateau deposits. Included are sections on the geology and geological reserves, pit design and ore reserves, metallurgy and the infrastructure required. Not included are the actual mine plan, production schedules for the mine and mill, as well as manpower, equipment and capital schedules. The plan is currently being revised to accommodate pre production stripping by scrapers at Grum and Vangorda and a small underground mining operation at Faro. These aspects of the mine plan will be described in a report in preparation by Kilborn Engineering.

2.0 INTRODUCTION OF TERMS OF REFERENCE

Curragh Resources has successfully reactivated the Cyprus Anvil lead-zinc-silver mine located at Faro, Yukon. Mining operations commenced in January 1986, and milling of the ore began in June of that year.

The mine produces two concentrate products: lead, which also includes payable quantities of silver and gold, and zinc. The concentrate is hauled via road to Skagway, Alaska, where it is loaded on ships for markets in Europe and Asia.

Curragh Resources owns the rights to several additional orebodies in the Faro area. These deposits, located sufficiently close to the Faro concentrator, offer the opportunity to extend the life of the Faro operations.

Considerable engineering work has been done by the former owners, Cyprus Anvil Mining Corporation and Kerr Addison Mines, in assessing the Grum and Vangorda deposits, located on the Vangorda Plateau. Included in the studies were:

- An assessment of ore transportation alternatives
- Metallurgical testing of the ores
- Extensive exploration drilling and geological interpretation
- Preliminary mine plans
- Environmental baseline studies

Also, major renovations to the Faro concentrator were made in anticipation of finer grind requirements.

During 1986 the above work was reviewed and incorporated into a mine plan for the Grum and Vangorda pits, and the completed report, known as case "VP 1-1" was released in November of 1986 for internal distribution.

The following mine plan, called case "VP 1-5" was developed as a refinement of the VP 1-1 plan, and includes a detailed mine plan for the Faro pit, based on a start date of April 1, 1987. Included within the scope of this report is the following:

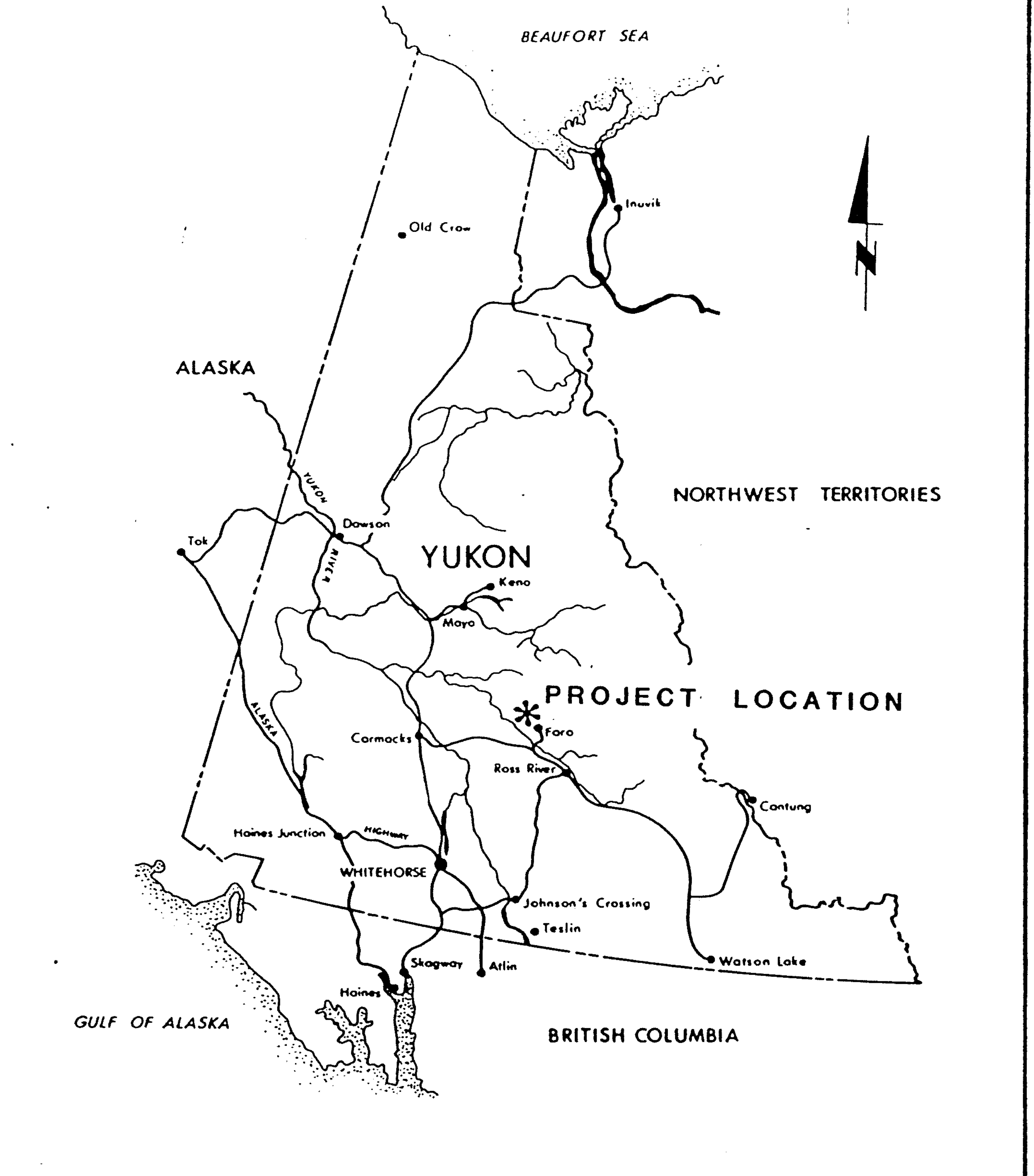
1. Deposit geology
2. Metallurgical response of the ores
3. Open pit mineable reserves, pit design, and pit planning
4. Detailed quantifying and sequencing of mining blocks
5. Detailed sequencing of mill feed and concentrate production
6. Infrastructure requirements
7. Operating equipment requirements
8. Manpower
9. Operating and capital costs and pretax revenues

Due to limitations in time and available information, some items are not within the scope of this report. These would include:

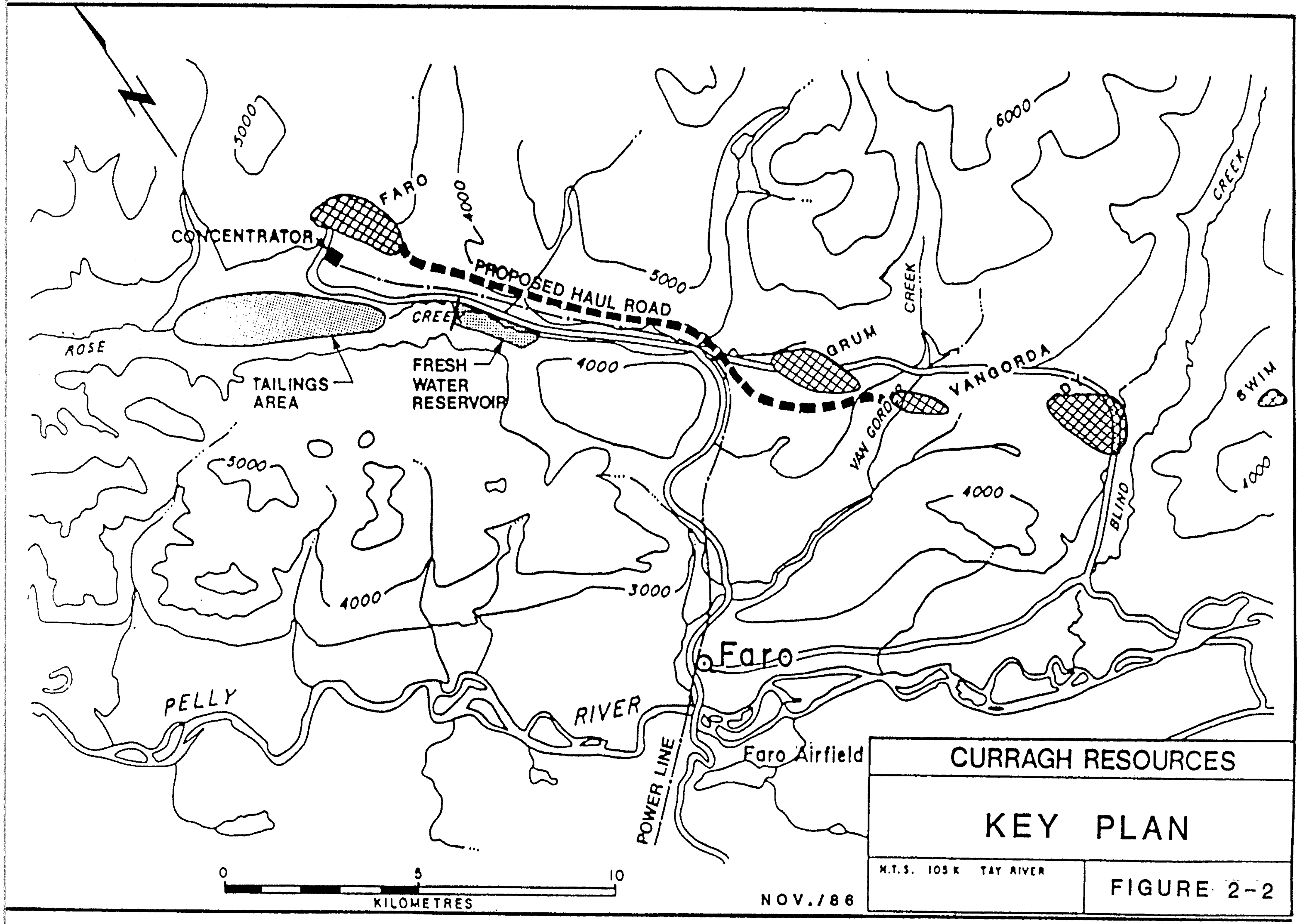
1. Alternate means of overburden stripping
2. Alternate haulage methods (waste and ore)
3. Assessment of the Champ Zone of the Grum deposit
4. Assessment of the 1986 Faro pit drilling program.

Additional engineering and development work is required to both optimize the current plan and validate key assumptions. Such work should include:

1. Geotechnical studies on overburden slope stability and dump stability in the Vangorda Plateau
2. Hydrogeological studies in the Vangorda Plateau
3. Geotechnical studies on foliation surfaces within the Vangorda and Grum pits
4. Optimization of the Grum open pit and its stages
5. Finalization of the Vangorda Haul Road alignment
6. Exploration drilling as required on the Vangorda Plateau.



CURRAGH RESOURCES	PREPARED BY: CGY
LOCATION PLAN	NOV./86
	FIGURE 2-1



CURRAGH RESOURCES

KEY PLAN

N.T.S. 105 K TAY RIVER

FIGURE 2-2

3. Geology and Reserves

3.1 District Geology

The Anvil Range contains four lead-zinc-silver deposits in addition to the Faro deposit. The aggregate geologic reserve of the five deposits is 91 million tonnes averaging 9.4% lead plus zinc and 58 grams/tonne silver.

The ore deposits of the Anvil Range are sediment hosted, stratiform, pyritic, massive sulphide deposits. They are hosted by lower Paleozoic metamorphosed shales. The deposits consist of one to five ore layers originally deposited parallel to bedding within a 150 metre thick stratigraphic section. The layers are generally 10 metres to 40 metres thick and are now contorted into complex fold structures.

The ore horizons are strongly zoned with respect to ore type and ore grade. The central and upper ore type in a given horizon is variably baritic, massive sulphide. The lower and peripheral ore type is hard, carbon rich quartzite containing disseminated sulphides. The zoning is significant to mine planning since the ore types have differing metallurgical performance, physical properties and grades. Different deposits are composed of different proportions of ore types thus ore in the Anvil Range cannot be assumed to be the same as Faro ore.

Strong metamorphism accompanied the deformations that folded the Anvil Range ore deposits. This metamorphism recrystallized the ores; it varies in intensity throughout the district adding further variety to the ore types and their metallurgical performance.

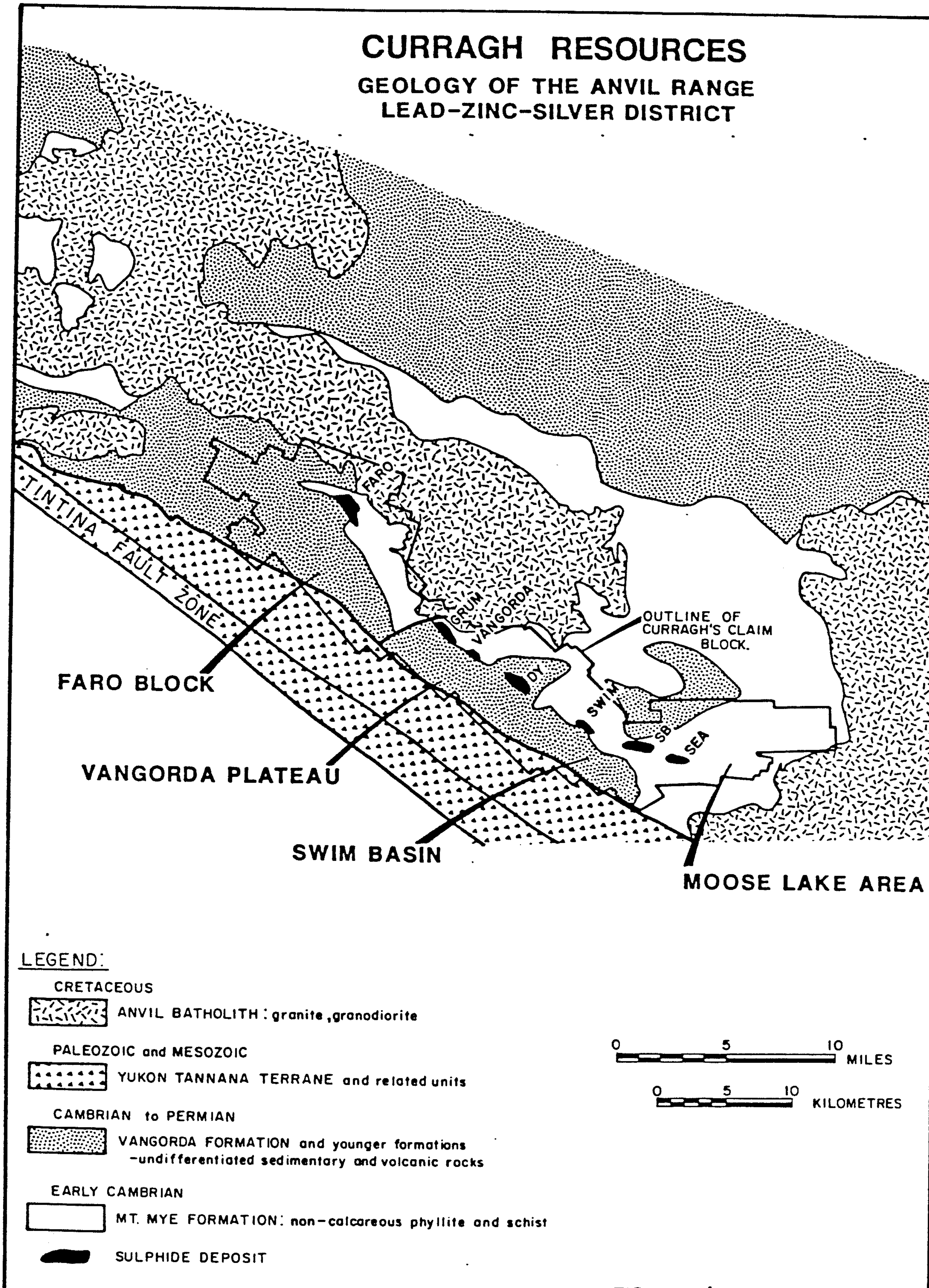
3.2 Faro Geology and Reserves

The Faro deposit is a flat lying to gently southwest dipping lens that varies from a few metres to 40 metres thick. In cross-section the deposit is highly assymetric with a thick northeast edge, tapering to a thin southwest edge.

Based on metallurgical performance there are currently three ore types recognized at Faro.

- BG: massive to disseminated pyrite, spalerite and galena in a quartz or barite gangue
- A: Disseminated pyrite, sphalerite and galena in a carbonaceous quartz gangue
- H: massive, fine grained pyrrhotite with galena and sphalerite

The northeast part of the deposit is low grade because it contains a high proportion of nearly barren massive and semi-massive sulphides interbanded with thin high grade "BG" zones. The southwestern part of the deposit is higher grade and largely the "BG" ore type. The "H" type is erraticly distributed in the



CURRAGH RESOURCES
GEOLOGY OF THE ANVIL RANGE
LEAD-ZINC-SILVER DISTRICT
TINTINA FAULT ZONE
FARO
GRUM
VANGORDA
DY
SWIM
SB
SEA
OUTLINE OF CURRAGH'S CLAIM BLOCK.
FARO BLOCK
VANGORDA PLATEAU
SWIM BASIN
MOOSE LAKE AREA
LEGEND:
CRETACEOUS
ANVIL BATHOLITH : granite, granodiorite
PALEOZOIC and MESOZOIC
YUKON TANNANA TERRANE and related units
CAMBRIAN to PERMIAN
VANGORDA FORMATION and younger formations -undifferentiated sedimentary and volcanic rocks
EARLY CAMBRIAN
MT. MYE FORMATION: non-calcareous phyllite and schist
SULPHIDE DEPOSIT
FAULT
0 5 10 MILES
0 5 10 KILOMETRES

FIGURE 4

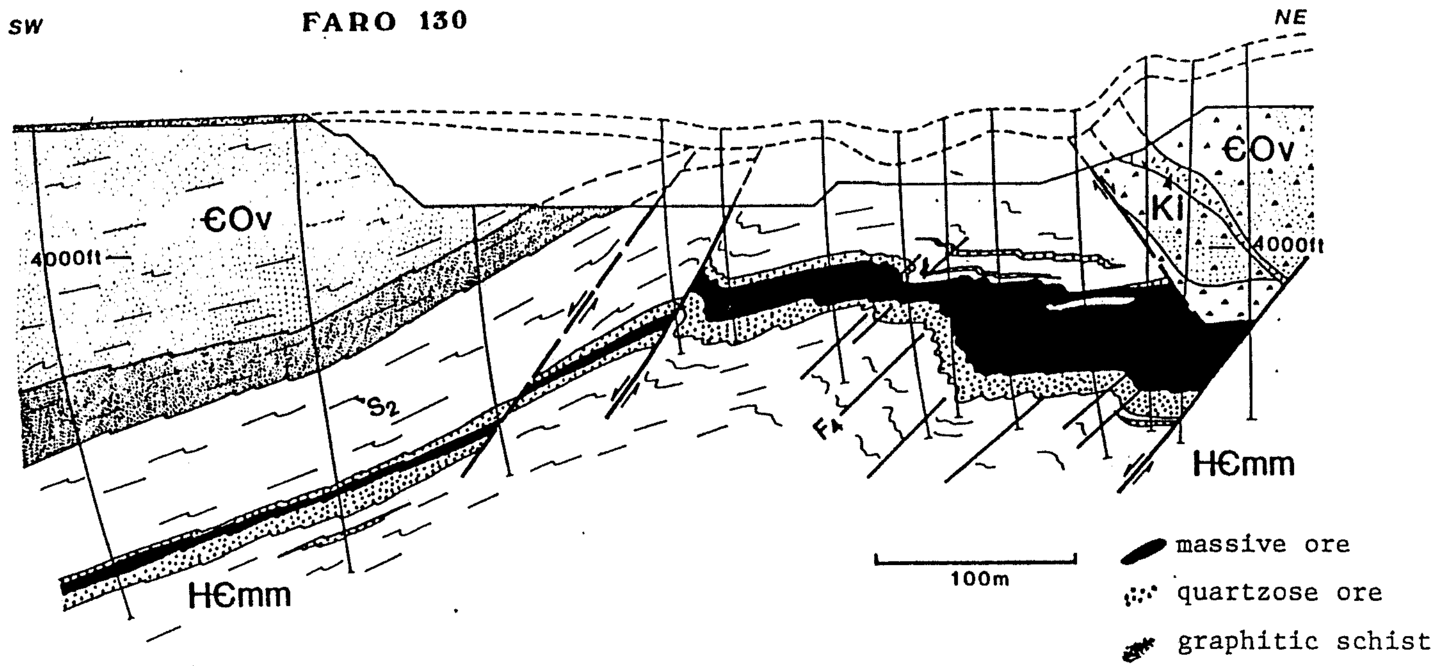


Figure 5. Cross section 130 through the southeast end of Zone 3. The pit outline is as of the June 1982 suspension of mining. The deep sulphides at the southwest end of the section are part of the area of proposed underground mining.

southwestern part of the deposit. A basal unit of "A" type carbonaceous quartzite with disseminated sulphides occurs throughout the deposit; it is generally low grade but in the northeast is sulphide rich and some very high grade zones are present.

The shallowly dipping orebody is strongly layered parallel to its tabular dimension. The layering is discontinuous and layers change thickness eratically. The deposit is offset by a number of significant normal faults adding to the complexities of tracing ore. External contacts of the orebody are sharp and visually distinct against barren phyllite. Dilution from this material is easily controlled. Internal to the sulphide mass, ore grade mineralization cannot be separated visually from sulphide waste thus dilution from this material can only be controlled by blasthole assays and dilution is high where high grade layers are thin.

Geologic reserves at the start of 1986 were 29.3 million tonnes at 3.13% lead 5.03% zinc and 40.8 grams/tonne silver (undiluted at a 4% lead plus zinc cutoff), 1.8 million tonnes of which was mined in 1986. The deposit is, for the most part, sufficiently well drilled off that the reserves can be considered proven.

Southwest of the present open pit area is a part of the deposit too deep to mine by open pit methods. Geologic reserves in this area are estimated to be 2.6 million tonnes at 5.03% lead 7.72% zinc and 67.3 grams/tonne silver. Part of this geologic reserve overlaps that given above thus they are not additive. Further drilling is needed in this area to delineate the reserves in detail.

2.3 Grum Geology and Reserves

The Grum deposit is structurally and stratigraphically more complex than the Faro deposit. Grum consists of three to five separate sulphide horizons with appreciable thicknesses of intervening barren phyllite. The ore horizons are contorted into a complex polyphase fold structure. The fold plunges northwest about 11°. The upplunge end of the deposit has been truncated by erosion but is buried beneath thick glacial overburden thus Grum is essentially a blind deposit. This extensive cover by glacial overburden or bedrock has protected the Grum ores from oxidation.

Grum consists of the same ore types as Faro but the A type is much more abundant and higher grade. The A type carbonaceous sulphide bearing quartzites comprise 35% of the Grum Reserves. The massive sulphide ores at Grum are finer grained and have more complex mineral intergrowth textures than Faro ores. This neccessiatates a finer grind than Faro ores. The Faro concentrator has previously been modified to accomodate this grind but the tonnage throughput must be lower than for Faro ores to achieve the finer grind.

At a given cutoff grade Grum ores are higher grade that Faro ores, this is particularly true for precious metals. Gold at Grum is 8 to 10 times higher than in Faro ores. Because of the volume of production, the Anvil Range will be one of the largest gold

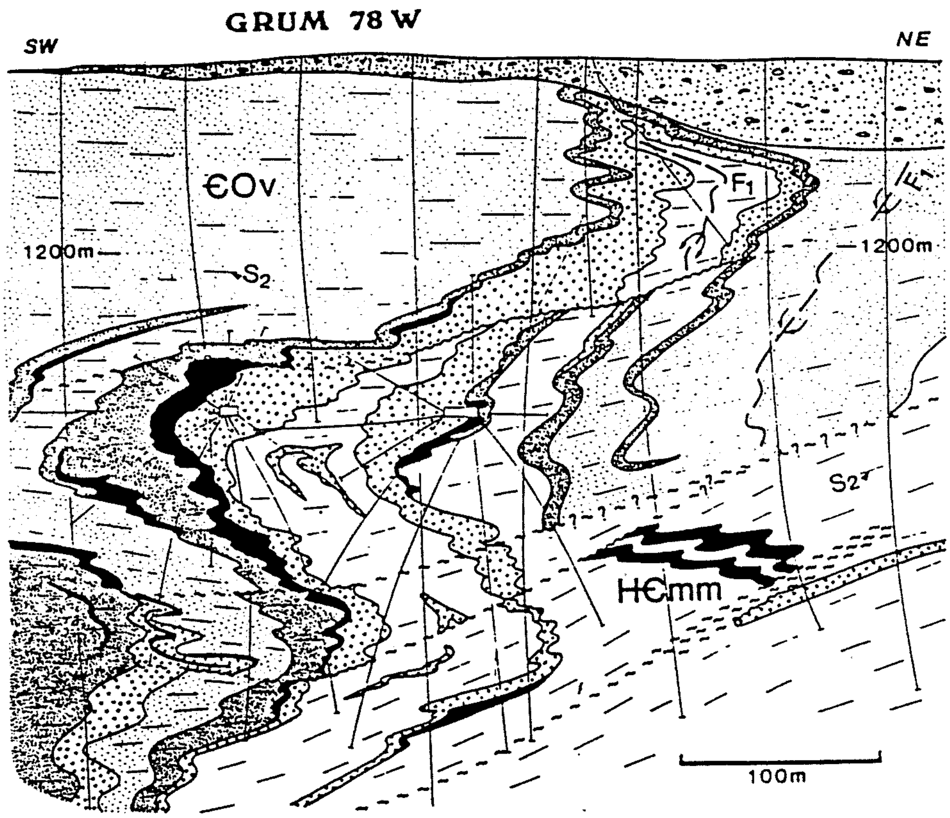


Figure 6. Cross section 78 W through the Grum deposit. The section is close to the northwest limit beyond which the high grade massive sulphides near the underground workings cannot be mined by open pit methods because of high stripping requirements. This ore would be left for underground followup not included in the current mine plan.

massive ore

quartzose ore

graphitic phyllite

producing districts in the Yukon when the Vangorda Plateau deposits come on stream.

Geologic reserves for the main part of the deposit have been newly calculated from base principles by Curragh staff. At a 4% lead plus zinc cutoff grade the geologic reserves are 30.6 million tonnes averaging 3.4% lead, 5.6% zinc, 57 grams/tonne silver and 0.95 grams/tonne gold (undiluted). These reserves are well defined by surface drilling, underground sampling and underground diamond drilling and are considered proven reserves.

In addition to the main deposit is the Champ Zone, adjoining to the southeast, which contains probable reserves of 1.7 million tonnes averaging 3.5% lead 4.3% zinc and 46 grams/tonne silver.

To the northwest of the main deposit, down the fold plunge, there is potential for a further five to ten million tonnes of high grade sulphides in areas not yet drilled off. These sulphides are too deep for open pit mining. The deep extension, when added to the high grade proven ore beneath the current pit design, indicates a potential for underground followup similar to that planned for Faro, but on a larger scale.

2.4 Vangorda Geology and Reserves

The Vangorda Deposit is similar to Grum but smaller and shallower. The deposit cosists of one major ore horizon which is contorted into a subhorizontal to slightly northwest plunging fold. Rock and glacial overburden cover is not great thus the stripping ratio for Vangorda is much lower than Grum. Because of the shallow depth of burial there is a possibility that oxidation of Vangorda ores may be significant, particularly in the southeast part of the deposit.

Most of the high grade ore at Vangorda is barite bearing massive sulphide. There is relatively little of the A type ore at Vangorda. Like Grum ores, the Vangorda ores are finer grained than Faro and richer in precious metals.

Geologic reserves for Vangorda have been newly calculated based on a intensive geologic re-interpretation of existing data by Curragh staff. The geologic reserves are 7.5 million tonnes averaging 3.9% lead, 4.9% zinc, 53 grams/tonne silver and 0.69 grams/tonne gold (based on a 4% lead plus zinc cutoff grade and no dilution). More drilling is required to delineate the reserves in greater detail. Results of this drilling will not only enhnace the reliability of the bench by bench reserves but also provide data from metallurgical testing for oxidation limits.

A relatively gold rich zone of more than 2 million tonnes averaging approximately 0.9 grams/tonne gold and 22 grams/tonne silver with minor lead and zinc underlies the lead-zinc rich massive sulphides. Some of this material will be moved to gain access to the lead-zinc zone. The amenability of this material to treatment will be studied.

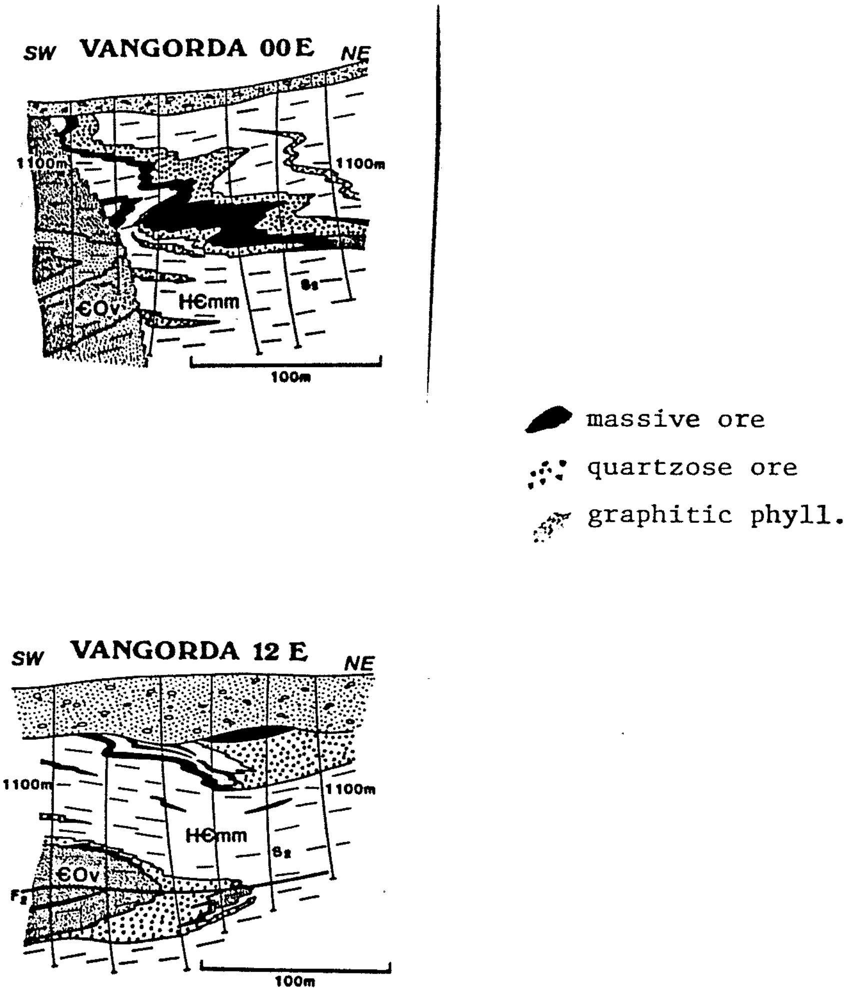


Figure 7. Cross sections through the Vangorda deposit. Note the difference in size and depth of burial between Vangorda and the other deposits

2.5 Other Reserves and Potential in District

The current mine plan does not exhaust all reserves in the district. In addition to the Champ zone of the Grum deposit, further Faro underground and Grum underground, there are two known deposits, DY and Swim. Further anomalies exist that indicate other possible reserve potential.

The DY deposit has geologic reserves of 21.0 million tonnes averaging 5.5% lead, 6.7% zinc, 84 grams/tonne silver and 0.95 grams/tonne gold (at a 9% lead plus zinc cutoff and no dilution).

DY is very deep (500m to the top) and thus far has been drilled from the surface only. The deposit is open to the southwest and southeast. Because of the sparse drilling pattern a major underground exploration program will eventually be required at DY.

Most DY ore is high grade massive sulphide. The deposit will be similar to Grum in other respects.

The Swim deposit has geologic reserves of 4.3 million tonnes averaging 3.8% lead, 4.7% zinc and 51 grams/tonne silver. (at a 6% lead plus zinc cutoff, undiluted). Further work is required to bring the confidence of these reserves up to a level required for a production decision.

Swim is similar in most characteristics to Grum but smaller and lower grade.

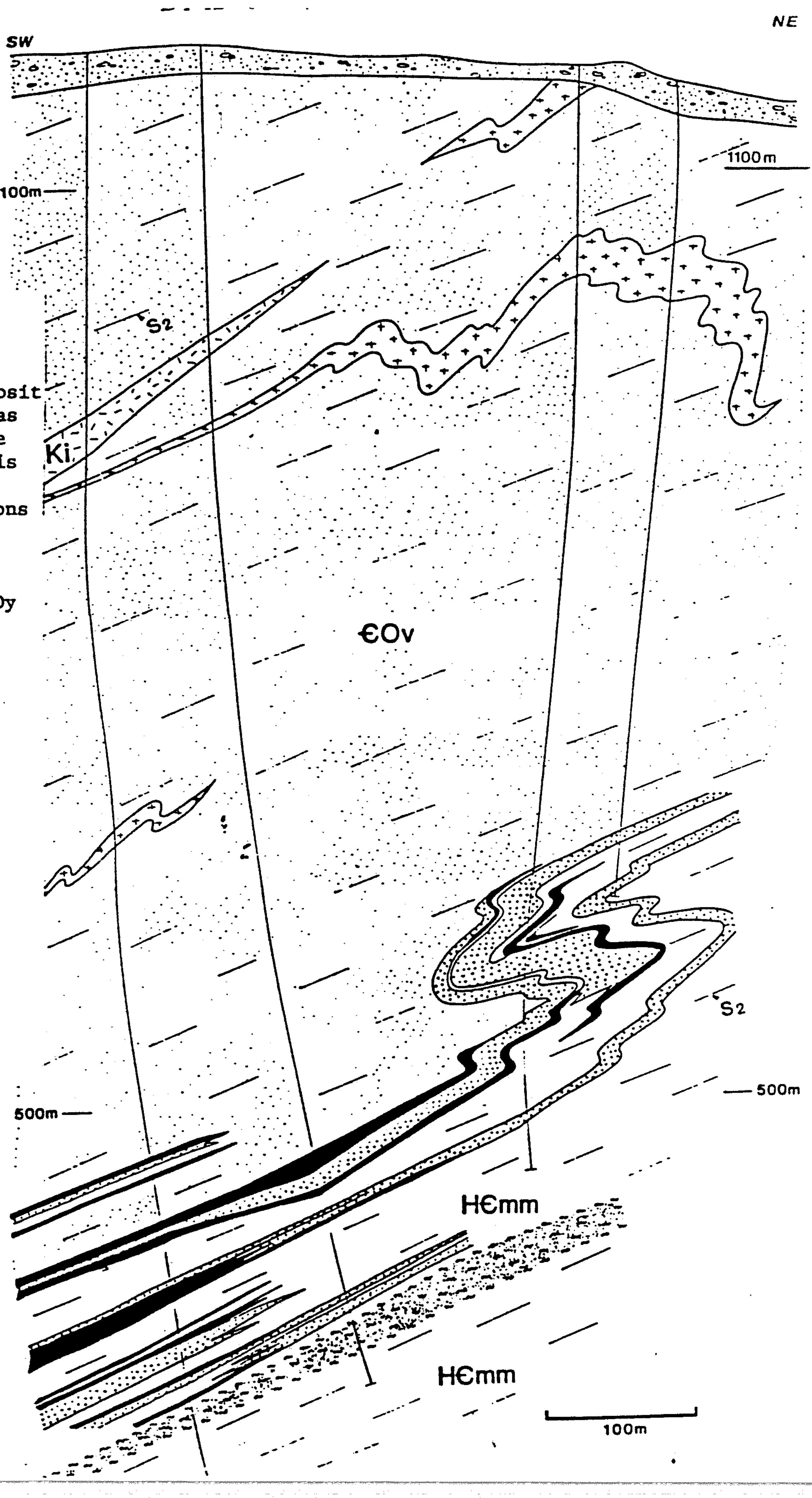
Exploration potential still exists on the Vangorda Plateau where blind fold noses could contain another Vangorda sized deposit and extensions to Grum and DY are indicated.

In the Swim basin there is potential for open pit ore remaining as well as deeper possibilities.

Northwest of Faro, exploration is needed along the favourable trend of the existing deposits.

Figure 8

Schematic section through the DY deposit at the same scale as the sections of the other deposits. This is not one of the best drilled sections of the deposit but it illustrates the relative paucity of information at Dy and the difficulty of obtaining more.



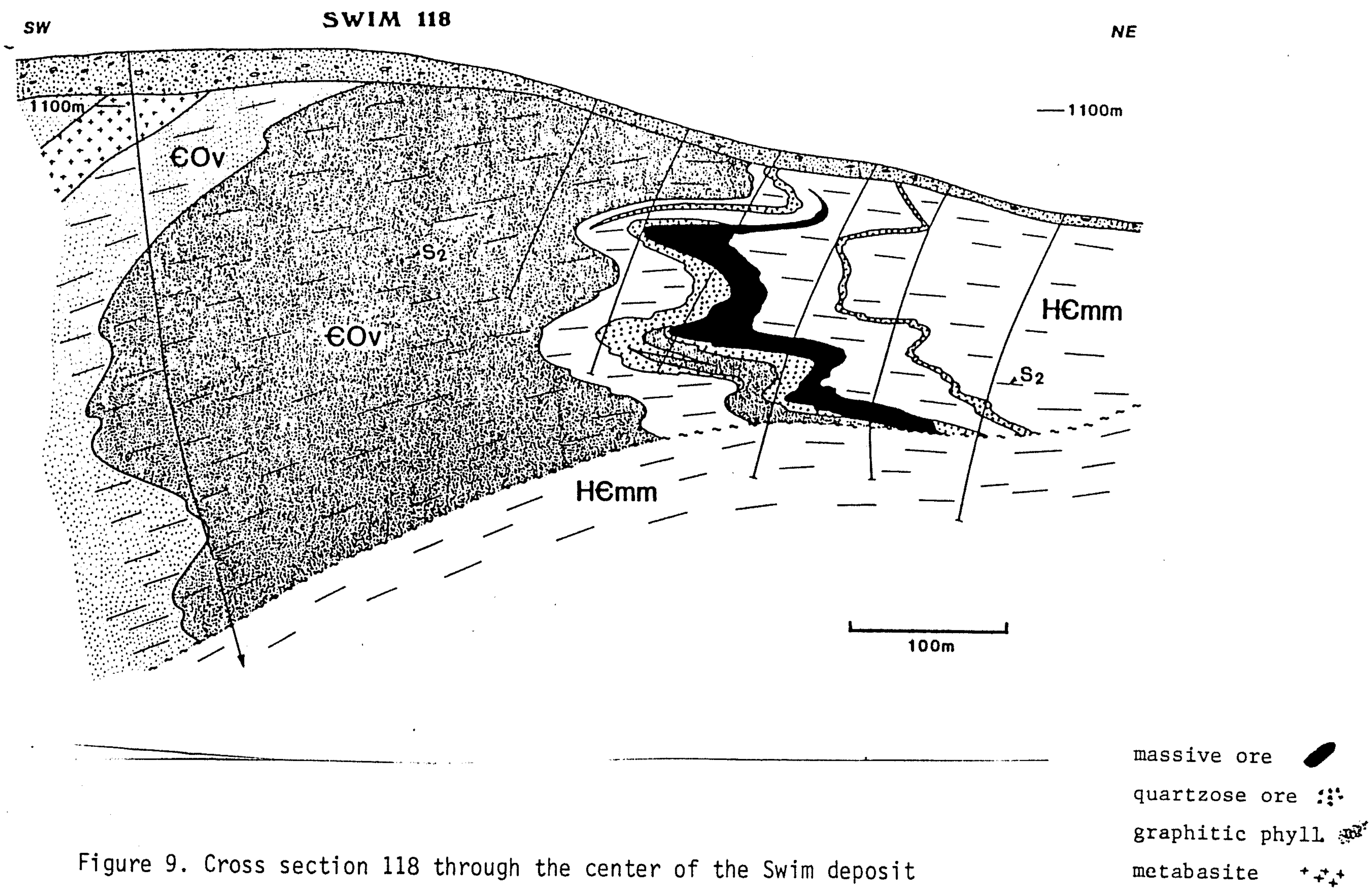


Figure 9. Cross section 118 through the center of the Swim deposit

4.0 METALLURGY

Metallurgical parameters used in the preparation of this report come from Curragh Resources' operating experience for the Faro orebody. Vangorda Plateau metallurgy is taken from the CAMC report on the development of the Vangorda Plateau. For a detailed review of the metallurgies, appropriate sections of the CAMC report have been included as Appendix C to this report.

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

4.1 Faro Metallurgy

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

For planning purposes, the Faro orebody has been 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 predicted metallurgical response is as shown in Table 4-1 below:

	Pb Conc			Zn Conc	
Ore Type	Pb rec.	% Pb	Ag rec.	Zn rec.	% Zn
2BG	78%	62	55%	82%	51
2H	76%	60	53%	80%	50
2A	70%	40	45%	78%	50

TABLE 4-1
Predicted Metallurgical Response

Although concentrate shipped to date has contained minor quantities of payable gold, no estimate of gold recovery has been made.

Much of the low grade ore is planned to be stockpiled, some for as long as 4 years. Although oxidation of this ore is almost certain to have a detrimental effect on recovery, no allowance for oxidization has been made in this plan.

4.2 Vangorda Metallurgy

The Vangorda orebody has been categorized into three ore types: baritic, pyritic, and quartzitic. The predicted metallurgical response as determined by CAMC is shown in Table 4.2-1 below.

	Pb Conc.				Zn Conc.	
Ore Type	Pb rec.	Ag rec.	Au rec.	% Pb	Zn rec.	% Zn
Baritic	84%	65%	40%	53	80%	55
Pyritic	77%	50%	50%	49	72%	51
Quartzitic	81%	55%	15%	48	79%	53

TABLE 4.2-1
Predicted Vangorda Metallurgy

4.3 Grum Metallurgy

Although the Grum orebody is composed of several different ore types, metallurgical parameters are based on an "average" ore. The metallurgical response has been determined primarily by pilot plant testwork on a bulk sample obtained by the underground exploration. This sample did not provide representative quantities of each specific ore type.

Of particular concern to the Grum metallurgy will be the response of the graphitic A type ore. Experience with A type ore at Faro suggests that the predicted response may be optimistic, and further testwork by Curragh Resources personnel is required to provide a reliable forecast. The A type ore makes up about 35 % of the geologic reserves at Grum.

The predicted metallurgical response as determined by CAMC is shown in Table 4.3-1 below:

	Pb Conc.				Zn Conc.	
Ore Type	Pb rec.	Ag rec.	Au rec.	% Pb	Zn rec.	% Zn
"Average"	80%	65%	33%	60	83%	55%

TABLE 4.3-1
Predicted Grum Metallurgy

Much of the low grade ore is expected to be stockpiled for a number of years. No allowance has been made for the effect of oxidation of these ores on metallurgical recovery.

5.0 PIT DESIGN

5.1 Faro Pit

The pit design and mining plan used in this plan is essentially that presented in the 1985 Kilborn mining plan. This mining plan has the Faro pit being mined in 4 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.

5.1.1 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.

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 in-situ 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 (in sita ore tonnes)

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

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

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

= (in situ ore tonnes) x (recovery) x (1 + dilution)

The dilution is assumed to contain zero metal:

Recovered ore grade = in situ ore grade

Dilution grade = 0.0

Mined ore grade

$$= \frac{\text{(recovered ore grade) x (recovered tonnes)}}{\text{(recovered tonnes + dilution tonnes)}}$$

$$= \frac{\text{(in situ ore grade) x (recovered tonnes)}}{\text{(recovered tonnes) x (1 + dilution)}}$$

$$= \frac{\text{(in situ ore grade)}}{\text{(1 + dilution)}}$$

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

Table 5.1-1 summarizes the April 1, 1987 mining reserves, based on 4 percent and 6 percent cutoffs. Table 5.1-2 lists the mining reserves by category. Appendix C lists the detailed mining reserves by block.

	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 5.1-1
Mining Reserves
April 1, 1987 Status

In addition to the pit reserves above, a pit stockpile of previously mined ore exists. These stockpile reserves are listed - Table 5.3-3 below.

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

CURRAGH RESOURCES

Reserve listing: 87 Budget Reserves (10% dilution, 95% recovery)

Material	Grade	Volume BCY	Tonnes	HEAD GRADE %Pb+Zn	%Pb	%Zn	Ag g/t	Au g/t	CONCENTRATE Pb DMT	%Pb	Ag g/t	Au g/t	Zn DMT	%Zn	% RECOVERY Pb	Zn	Ag	Au
Waste		21204410.	43051330.															
Calc-Silic		1842640.	3781097.															
Sulph Wast		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

TABLE 5.1-2

MINING RESERVES

5.2 Vangorda Pit

5.2.1 Geotechnical

Geotechnical parameters for the design of the Vangorda pit were adapted from an October 1980 report prepared by CAMC (ref.4).

5.2.1.1 Geology

Surficial deposits in the pit and waste dump areas consist mainly of a glacial till, with thickness ranging from 2 to 30 metres The till is relatively consistent and is composed of sandy silt with some clay and gravel. Permafrost has not been encountered in any test pit or drill hole.

The deposit itself is associated with a graphitic phyllite occuring at a broad vertical facies change between calcareous pelitic phyllites of the Vangorda formation above and non-calcareous pelitic phyllites of the Mt. Mye formation below.

The deposit area has been deformed by four phases of folding and one phase of faulting. The dominant structural feature is the S_2 foliation. Average orientation of the S_2 foliation is 130^0 strike, dipping 28^0 SW, but local variations in the orientation exist.

5.2.1.2 Wall Design

Pit wall 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 S_2 foliation dipping into the pit, has been designed at a shallower slope than the other walls. Slope design parameters are summarized in the table below:

Wall	Slope (overall, not including roads)
SE	45°
NE	40°
SW	45°
NW	45°

Overburden slopes have been designed at 35°.

More accurate determination of the orientation of the S_2 foliation is required; such information could have a major effect on the design of the northeast wall.

5.2.2 Economic Modeling - Vangorda Pit

An economic model for the Vangorda pit was generated using PC-Mine software. The economic model generated is used as a tool for pit design only and is not used for operating cost estimates. The economic model represents the net value of each model block, based on all operating costs and an estimated revenue. Source of operating cost data is the 1987 Curragh Resources Operating Budget. The economic model is based on a 5% (combined lead plus zinc) cutoff grade. This cutoff grade is somewhat arbitrary and is primarily based on Faro experience; more work is required to determine an economic cutoff.

5.2.2.1 Costs

Costs for the economic model are divided into three categories: volumetric mining costs, variable haulage costs, and ore based costs.

Volumetric mining costs apply to all model blocks and include costs for drilling, blasting, loading, "fixed" haulage (road maintenance etc.), and mining services. All of these costs were taken from the 1987 Curragh Resources Operating Budget, except that drilling and blasting costs for the unconsolidated overburden have been reduced by 90%, as this material is not expected to require blasting.

Variable haulage costs for a block are calculated as a function of the block location and grade. This is to allow for increased haulage costs as the pit deepens, and also is to allow for the long ore haul.

Ore based costs are calculated for all blocks grading over 5% (combined lead plus zinc) and include: processing cost, Faro G and A, Whitehorse G and A, and Toronto G and A, but do not include concentrate transport or smelting fees.

Cost parameters are summarized in tables 5.2-1 and 5.2-2.

5.2.2.2 Revenue

Revenue is calculated as payment for a percentage of contained metal in the concentrate less smelting and refining fees less concentrate handling costs. Smelting and concentrate handling costs are taken from the 1987 Curragh Resources Operating Budget.

Contained metal in the concentrate is calculated as a function of head grade, concentrate grade, and rock type.

A metallurgical recovery factor is defined for each ore type. This base recovery is then adjusted by a recovery factor, which is based on current Faro milling experience, to allow for the "head grade effect". The head grade effect equations are 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 ln (Wt. % to Pb conc.) + 0.733
Au factor = 0.18 x ln (Wt. % to Pb conc.) + 0.733

Table 5.2-3 shows the calculation of revenue estimates for entry into the PC-MINE system; tables 5.2-4 nd 5.2-5 show the PC-MINE referenced metallurgical and revenue parameters and table 5.2-6 shows a sample calculation for one ore block.

5.2.3 Pit Design

The Vangorda ultimate pit design is based on sectional representations of the economic model. Sections of the economic model were constructed at 60 metre intervals through the orebody. These sections show the net economic value of each block on the section. For each section, an optimum pit was determined, based on the economic model combined with the geotechnical wall parameters. No allowances for ramps were made at this time.

After the sectional representations were complete, the information was transferred to plan. At this point the walls were adjusted to give section to section continuity, and the haul ramp was included. This defines the ultimate pit.

Because the northeast wall is the least stable geotechnically, the haul road has been designed to avoid that wall. In this way, should a failure of the northeast wall occur, mining can continue while stablization operations are underway.

The Vangorda ultimate pit floor is at the 1060 metre elevation.

Figure 5.2-1 shows the ultimate pit.

5.2.4 Ore Reserves

Ore reserves have been calculated on 4.5 metre model benches.

Mining reserves are adjusted to reflect a 95% mining recovery and a 15% waste dilution. Recovery is defined as the tonnes of ore mined, expressed as a percentage of the in-situ 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(in-situ ore tonnes)
Dilution tonnes = (recovered ore tonnes)x(dilution)
Mined ore tonnes = Recovered ore tonnes + dilution tonnes
= (recovered ore tonnes)x(1 + dilution)
= (in-situ ore tonnes)x(recovery)x(1 + dilution)

The dilution is assumed to contain zero metal:

Recovered ore grade = in-situ ore grade
Dilution grade = zero

$$\text{Mined ore grade} = \frac{(\text{recovered ore grade})\times(\text{recovered tonnes})}{(\text{recovered tonnes} + \text{dilution tonnes})}$$

$$= \frac{(\text{in-situ ore grade}) \times (\text{recovered tonnes})}{(\text{recovered tonnes}) \times (1 + \text{dilution})}$$

$$= \frac{(\text{in-situ ore grade})}{(1 + \text{dilution})}$$

Ore reserves are calculated based on a 5% (combined lead plus zinc) cutoff grade, with quantities grading between 4% and 5% also being reported.

Tables 5.2-7 details the mining reserves for the pit. In addition, a detailed bench by bench listing is given in appendix E.

5.2.5 Waste Dumps

Mining of the Vangorda Pit will release about 9 million BCM of waste. A waste dump for the material has been located southwest of the pit, into the Vangorda Creek valley.

Because Vangorda Creek is to be diverted around the pit, placing the dump in the creek bed, upstream of where the flow is to be returned to the creek, minimizes the impact on the natural drainage.

This design does not incorporate a separate overburden dump, however a separate dump may prove practical for dump stability reasons.

The dump will initially be built at the elevation of 1120 metres, later being built up to 1140.

The dump location is shown in figure 5.2-2.

PRINTOUT OF ROCK-TYPE INFORMATION FOR RECORDS [1] TO [10]

MINING COST DATA

REC	STAT	ROCK CODE	DESCRIPTION	COST DATA [CDN $ PER bcm] DRILLING	BLASTING	LOADING	FIXED HAULAGE	MINING SERVICES
1	1	1	4A Ribbon banded graphitic pyritic quartzite	.1748	.3865	.4018	.3533	.9817
2	1	2	4C Pyritic quartzite	.1748	.3865	.4018	.3533	.9817
3	1	3	4C Quartz rich massive or nearly massive sulphides	.1748	.3865	.4018	.3533	.9817
4	1	4	4E Pyritic Massive Sulphides	.1748	.3865	.4018	.3533	.9817
5	1	5	4EG Variably baritic pyritic massive sulphides	.1748	.3865	.4018	.3533	.9817
6	1	6	4H Pyrrhotitic massive sulphides	.1748	.3865	.4018	.3533	.9817
7	1	10	Waste	.1748	.3865	.4018	.3533	.9817
8	1	11	Overburden	.0175	.0387	.4018	.3533	.9817
9	1	0	Air	.0000	.0000	.0000	.0000	.0000
10	1	12	Partially Above Topography	.0175	.0387	.4018	.3533	.9817

TABLE 5.2.1

VOLUMETRIC MINING COSTS - VANGORDA PIT

PRINTOUT OF COSTS INFORMATION FOR RECORDS [2] to [2]

DETAILED PRINTOUT FOR RECORD [2]

DATE CAPTURED : 4/11/1986

RECORD DESCRIPTION :November 86 Model

GENERAL COST INFORMATION

MINE CALL FACTOR	:	90.00 PERCENT
ORE/STOCKPILE CUT-OFF GRADE	:	5.000
STOCKPILE/WASTE CUT-OFF GRADE	:	4.000
ORE PROCESSING COST	:	6.52
MINE ADMIN COST	:	1.56
HEAD OFFICE ADMIN COST	:	.86
SPARE COST # 1	:	.93
SPARE COST # 2	:	.00

HAULAGE INFORMATION

	PIT EXIT ELEVATIONS [m]	AVERAGE IN-PIT HAULAGE [m]	AVERAGE SURFACE HAULAGE [m]	HORIZONTAL HAULAGE COST [CDN $ /bcm/100 m]	UPWARDS VERTICAL HAULAGE COST [CDN $ /bcm/10 m]	DOWNWARDS VERTICAL HAULAGE COST [CDN $ /bcm/10 m]
ORE	1120.00	800.00	17000.00	.0189	.0624	.0368
STOCKPILE	1120.00	800.00	1000.00	.0189	.0624	.0368
WASTE	1120.00	800.00	1000.00	.0189	.0624	.0368

TABLE 5.2.2

ORE BASED COSTS AND HAULAGE COSTS - VANGORDA PIT

Revenue Calculations   Vangorda Pit

Deductions:	$/tonne		
Highway Freight	51.9329		
Ocean Freight	24.0394	$US/$CDN	1.39
Smelting	201.3746		
Total:	277.3469		

	$ US	$ CDN	Smelter Pays
Lead	$0.20	0.278	95.00%
Zinc	$0.42	0.5838	85.00%
Silver	$5.50	7.645	95.00%
Gold	$400.00	556	95.00%

Lead; per tonne of ore

Head	Recovery Factor	Conc Grade	Conc Tonnes	Metal Pounds	Revenue
2	0.892459	50	0.035698	39.35073	0.491692
4	1.017919	50	0.081433	89.76511	1.121626
6	1.091308	50	0.130957	144.3553	1.803738
8	1.143378	50	0.182940	201.6575	2.519735
10	1.183767	50	0.236753	260.9761	3.260929
12	1.216768	50	0.292024	321.9017	4.022202
14	1.244669	50	0.348507	384.1636	4.800173
16	1.268838	50	0.406028	447.5696	5.592438
18	1.290157	50	0.464456	511.9757	6.397201
20	1.309227	50	0.523691	577.2705	7.213067

Zinc; per tonne of ore

Head	Recovery Factor	Conc Grade	Conc Tonnes	Metal Pounds	Revenue
2	0.845289	52.8	0.032018	37.27090	9.614692
4	0.957579	52.8	0.072543	84.44407	21.78385
6	1.023265	52.8	0.116280	135.3547	34.91717
8	1.069869	52.8	0.162101	188.6926	48.67663
10	1.106018	52.8	0.209473	243.8354	62.90168
12	1.135554	52.8	0.258080	300.4163	77.49775
14	1.160527	52.8	0.307715	358.1934	92.40238
16	1.182159	52.8	0.358230	416.9944	107.5711
18	1.201240	52.8	0.409513	476.6906	122.9708
20	1.218308	52.8	0.461480	537.1821	138.5757

Silver; per tonne of ore

Head	Recovery Factor	Metal Grams	Metal Ounces	Revenue
300	1	300	9.646302	70.05868

Gold; per tonne of ore

Head	Recovery Factor	Metal Grams	Metal Ounces	Revenue
50	1	50	1.607717	849.1961

TABLE 5. 2- 3

REVENUE CALCULATIONS - VANGORDA PIT

PRINTOUT OF ROCK-TYPE INFORMATION FOR RECORDS [1] TO [10]

METALLURGICAL DATA

ROCK CODE	DESCRIPTION	TYPE	PRIMARY MINERAL	CUT OFF GRADES O - S/P	CUT OFF GRADES S/P - W	RECOVERIES [PERCENT] %Pb+Zn	%Pb	%Zn	Ag g/t	Au g/t
1	4A Ribbon banded graphitic pyritic quartzite	ore	%Pb+Zn	5.000	4.000	.0	81.0	79.0	55.0	15.0
2	4C Pyritic quartzite	ore	%Pb+Zn	5.000	4.000	.0	77.0	72.0	50.0	50.0
3	4C Quartz rich massive or nearly massive sulphides	ore	%Pb+Zn	5.000	4.000	.0	77.0	72.0	50.0	50.0
4	4E Pyritic Massive Sulphides	ore	%Pb+Zn	5.000	4.000	.0	77.0	72.0	50.0	50.0
5	4EG Variably baritic pyritic massive sulphides	ore	%Pb+Zn	5.000	4.000	.0	81.0	76.0	57.0	45.0
6	4H Pyrrhotitic massive sulphides	ore	%Pb+Zn	5.000	4.000	.0	76.0	80.0	53.0	50.0
10	Waste	waste								
11	Overburden	waste								
0	Air	air								
12	Partially Above Topography	waste								

TABLE 5.2.4

METALLURGICAL PARAMETERS - VANGORDA PIT

PRINTOUT OF COSTS INFORMATION FOR RECORDS [2] to [2]

DETAILED PRINTOUT FOR RECORD [2]

REVENUE / HEAD GRADE INFORMATION

NO OF REVENUE / HEAD GRADE CURVES : 5

DESCRIPTION : Pb + Zn (no revenue)

POINT	HEAD GRADE [%Pb+Zn]	REVENUE [CDN $ /%Pb+Zn]
1	.000	.000
2	100.000	.000

DESCRIPTION : Lead Revenue

POINT	HEAD GRADE [%Pb]	REVENUE [CDN $ / %Pb]
1	.000	.000
2	2.000	.492
3	4.000	1.122
4	6.000	1.804
5	8.000	2.520
6	10.000	3.261
7	12.000	4.022
8	14.000	4.800
9	16.000	5.592
10	20.000	7.213

DESCRIPTION : Zinc Revenue

POINT	HEAD GRADE [%Zn]	REVENUE [CDN $ / %Zn]
1	.000	.000
2	2.000	9.615
3	4.000	21.784
4	6.000	34.917
5	8.000	48.677
6	10.000	62.902
7	12.000	77.498
8	14.000	92.402
9	16.000	107.571
10	20.000	138.576

DESCRIPTION : Silver Revenue

POINT	HEAD GRADE [Ag g/t]	REVENUE [CDN $ /Ag g/t]
1	.000	.000
2	300.000	70.059

TABLE 5.2.5

REVENUE/HEAD GRADE INFORMATION - VANGORDA PIT

PRINTOUT OF COSTS INFORMATION FOR RECORDS [2] to [2]

DETAILED PRINTOUT FOR RECORD [2]

DESCRIPTION : Gold Revenue

POINT	HEAD GRADE [Au g/t]	REVENUE [CDN $ /Au g/t]
1	.000	.000
2	50.000	849.196

TABLE 5.2.5 (CONTINUED)

REVENUE/HEAD GRADE INFORMATION - VANGORDA PIT

PRINTOUT OF ROCK-TYPE INFORMATION FOR RECORDS [1] TO [13]

MINING COST DATA

REC	STAT	ROCK CODE	DESCRIPTION	COST DATA [CDN $ PER bcm] DRILLING	BLASTING	LOADING	FIXED HAULAGE	MINING SERVICES
1	1	0	Air	.0000	.0000	.0000	.0000	.0000
2	1	1	A Type / 2nd code A, A4, C, E, L	.1748	.3865	.4018	.3533	.9817
3	1	2	A4 Type	.1748	.3865	.4018	.3533	.9817
4	1	3	C Type	.1748	.3865	.4018	.3533	.9817
5	1	4	D Type	.1748	.3865	.4018	.3533	.9817
6	1	5	E Type	.1748	.3865	.4018	.3533	.9817
7	1	6	E4 Type	.1748	.3865	.4018	.3533	.9817
8	1	7	G Type	.1748	.3865	.4018	.3533	.9817
9	1	8	H Type	.1748	.3865	.4018	.3533	.9817
10	1	9	L Type	.1748	.3865	.4018	.3533	.9817
11	1	10	Waste	.1748	.3865	.4018	.3533	.9817
12	1	11	Unconsolidated Overburden	.0175	.0387	.4018	.3533	.9817
13	1	12	Partially Above Topography	.0175	.0387	.4018	.3533	.9817

TABLE 5.3.1

VOLUMETRIC MINING COSTS - GRUM PIT

PRINTOUT OF COSTS INFORMATION FOR RECORDS [1] to [1]

DETAILED PRINTOUT FOR RECORD [1]

DATE CAPTURED : 4/11/1986

RECORD DESCRIPTION :November 86 Model

GENERAL COST INFORMATION

MINE CALL FACTOR	:	90.00 PERCENT
ORE/STOCKPILE CUT-OFF GRADE	:	5.000
STOCKPILE/WASTE CUT-OFF GRADE	:	4.000
ORE PROCESSING COST	:	6.52
MINE ADMIN COST	:	1.56
HEAD OFFICE ADMIN COST	:	.86
SPARE COST # 1	:	.93
SPARE COST # 2	:	.00

HAULAGE INFORMATION

	PIT EXIT ELEVATIONS [m]	AVERAGE IN-PIT HAULAGE [m]	AVERAGE SURFACE HAULAGE [m]	HORIZONTAL HAULAGE COST [CDN $ /bcm/100 m]	UPWARDS VERTICAL HAULAGE COST [CDN $ /bcm/10 m]	DOWNWARDS VERTICAL HAULAGE COST [CDN $ /bcm/10 m]
ORE	1260.00	800.00	14700.00	.0189	.0624	.0368
STOCKPILE	1260.00	800.00	1000.00	.0189	.0624	.0368
WASTE	1260.00	800.00	1000.00	.0189	.0624	.0368

TABLE 5.3.2

ORE BASED COSTS AND HAULAGE COSTS - GRUM PIT

PRINTOUT OF ROCK-TYPE INFORMATION FOR RECORDS [1] TO [13]

METALLURGICAL DATA

ROCK CODE	DESCRIPTION	TYPE	PRIMARY MINERAL	CUT OFF GRADES O - S/P	CUT OFF GRADES S/P - W	RECOVERIES [PERCENT] %Pb+Zn	%Pb	%Zn	Ag g/t	Au g/t
0	Air	air								
1	A Type / 2nd code A, A4, C, E, L	ore	%Pb+Zn	5.000	4.000	.0	80.0	83.0	65.0	33.0
2	A4 Type	ore	%Pb+Zn	5.000	4.000	.0	80.0	83.0	65.0	33.0
3	C Type	ore	%Pb+Zn	5.000	4.000	.0	80.0	83.0	65.0	33.0
4	D Type	ore	%Pb+Zn	5.000	4.000	.0	80.0	83.0	65.0	33.0
5	E Type	ore	%Pb+Zn	5.000	4.000	.0	80.0	83.0	64.0	33.0
6	E4 Type	ore	%Pb+Zn	5.000	4.000	.0	80.0	83.0	65.0	33.0
7	G Type	ore	%Pb+Zn	5.000	4.000	.0	80.0	83.0	65.0	33.0
8	H Type	ore	%Pb+Zn	5.000	4.000	.0	80.0	83.0	65.0	33.0
9	L Type	ore	%Pb+Zn	5.000	4.000	.0	80.0	83.0	65.0	33.0
10	Waste	waste								
11	Unconsolidated Overburden	waste								
12	Partially Above Topography	waste								

TABLE 5.3.4

METALLURGICAL PARAMETERS - GRUM PIT

PRINTOUT OF COSTS INFORMATION FOR RECORDS [1] to [1]

DETAILED PRINTOUT FOR RECORD [1]

REVENUE / HEAD GRADE INFORMATION

NO OF REVENUE / HEAD GRADE CURVES : 5

DESCRIPTION : Pb + Zn (no revenue)

POINT	HEAD GRADE [%Pb+Zn]	REVENUE [CDN $ /%Pb+Zn]
1	.000	.000
2	100.000	.000

DESCRIPTION : Lead Revenue

POINT	HEAD GRADE [%Pb]	REVENUE [CDN $ / %Pb]
1	.000	.000
2	2.000	2.142
3	4.000	4.886
4	6.000	7.857
5	8.000	10.976
6	10.000	14.205
7	12.000	17.521
8	14.000	20.910
9	16.000	24.361
10	20.000	31.420

DESCRIPTION : Zinc Revenue

POINT	HEAD GRADE [%Zn]	REVENUE [CDN $ / %Zn]
1	..000	.000
2	2.000	9.970
3	4.000	22.589
4	6.000	36.207
5	8.000	50.480
6	10.000	65.226
7	12.000	80.361
8	14.000	95.816
9	16.000	111.545
10	20.000	143.695

DESCRIPTION : Silver Reveune

POINT	HEAD GRADE [Ag g/t]	REVENUE [CDN $ /Ag g/t]
1	.000	.000
2	300.000	70.059

TABLE 5.3.5

REVENUE/HEAD GRADE INFORMATION - GRUM PIT

PRINTOUT OF COSTS INFORMATION FOR RECORDS [1] to [1]

DETAILED PRINTOUT FOR RECORD [1]

DESCRIPTION : Gold Revenue

POINT	HEAD GRADE [Au g/t]	REVENUE [CDN $ /Au g/t]
1	.000	.000
2	50.000	849.196

TABLE 5.3.5 (CONTINUED)

REVENUE/HEAD GRADE INFORMATION - GRUM PIT

TRACE BLOCK IN COLUMN [52] ROW [64] LEVEL [19] ROCK-TYPE CODE : 5
MATERIAL TYPE (0=AIR 1=WASTE 2=ORE) : 2
PRIMARY MINERAL : %Pb+Zn
PRIMARY GRADE : 8.095 [%Pb+Zn]
DENSITY : 3.86 [tn/bcm]
BLOCK VOLUME : 202.50 [bcm]
BLOCK TONNAGE : 781.65

ORE-STOCKPILE CUT-OFF GRADE : 5.000 [%Pb+Zn]
STOCKPILE-WASTE CUT-OFF GRADE : 4.000 [%Pb+Zn]

LABEL:	UNITS:	GRADE:	REVENUE:	RECOVERY:	BLOCK REVENUE:	CUM. REVENUE:
1	%Pb+Zn	8.10	.00	.00	.00	.00
2	%Pb	3.28	.89	81.00	509.19	509.19
3	%Zn	4.82	27.16	76.00	14522.09	15031.28
4	Ag g/t	40.00	9.34	57.00	3745.68	18776.96
5	Au g/t	.50	8.56	45.00	2709.79	21486.75

VOLUMETRIC MINING COSTS :

UNIT DRILLING COST :	.175	TOT DRILLING COST :	35.40
UNIT BLASTING COST :	.387	TOT BLASTING COST :	78.27
UNIT LOADING COST :	.402	TOT LOADING COST :	81.36
UNIT SERVICES COST :	.982	TOT SERVICES COST :	198.79
UNIT FIXED HAUL COST :	.353	TOT FIXED HAUL COST :	71.54

VARIABLE HAULAGE COSTS :

UNIT VERTICAL HAULAGE COST : .037
VERTICAL HAULAGE DISTANCE : -4.00
TOT VERTICAL HAULAGE COST : 2.98
UNIT HORIZONTAL HAULAGE COST : .019
HORIZONTAL HAULAGE DISTANCE : 17800.00
TOT HORIZONTAL HAULAGE COST : 681.61

ORE BASED MINING COSTS :

UNIT PROCESSING COST :	6.519	TOT PROCESSING COST :	5095.50
UNIT MINE ADMIN COST :	1.558	TOT MINE ADMIN COST :	1217.81
UNIT HEAD OFFICE ADMIN COST :	.856	TOT HEAD OFFICE ADMIN COST :	668.78
UNIT SPARE COST # 1 :	.930	TOT SPARE COST # 1 :	726.62
UNIT SPARE COST # 2 :	.000	TOT SPARE COST # 2 :	.00

TOTAL BLOCK REVENUE : 21486.75

TOTAL VOLUMETRIC MINING COST : 1149.96 -
TOTAL ORE BASED MINING COST : 7708.71 -

TOTAL BLOCK MINING COST : 8858.67

BLOCK ECONOMIC VALUE : 12628.08

TABLE 5.2-6

SAMPLE CALCULATION

Mining Reserves
Vangorda Pit

	Tonnes
Rock	8593620
Overburden	9069372
Sulphide Waste	3824375
All Waste	18636702

			%Pb+Zn	%Pb	%Zn	Ag g/t	Au g/t
Baritic	4.0-5.0	13181.	3.95	1.92	2.02	29.	0.88
Baritic	5.0 +	4750546.	9.08	3.96	5.12	57.	0.63
Pyritic	4.0-5.0	138091.	3.84	1.92	1.92	28.	0.50
Pyritic	5.0 +	163821.	7.39	3.94	3.44	57.	0.20
Quartzitic	4.0-5.0	482793.	3.92	1.61	2.31	25.	0.48
Quartzitic	5.0 +	910336.	5.69	2.30	3.38	31.	0.53

Strip Ratio 2.9

TABLE 5.2-7
VANDORDA PIT RESERVES

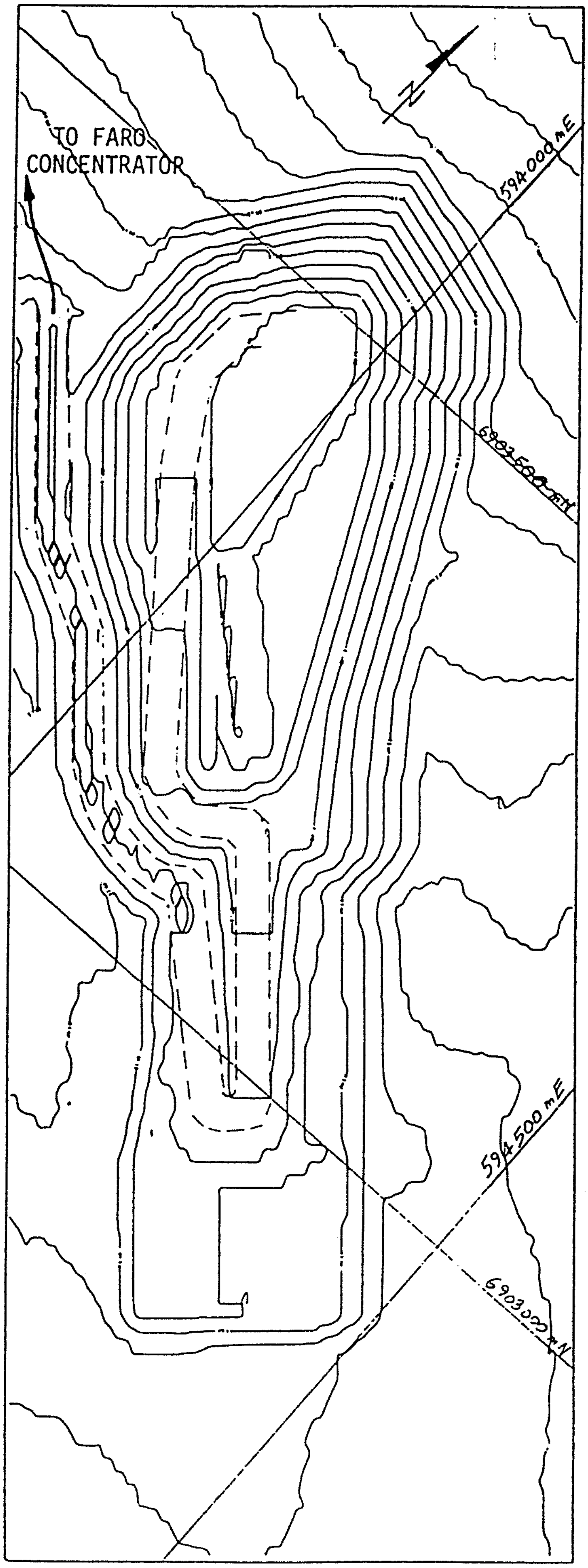
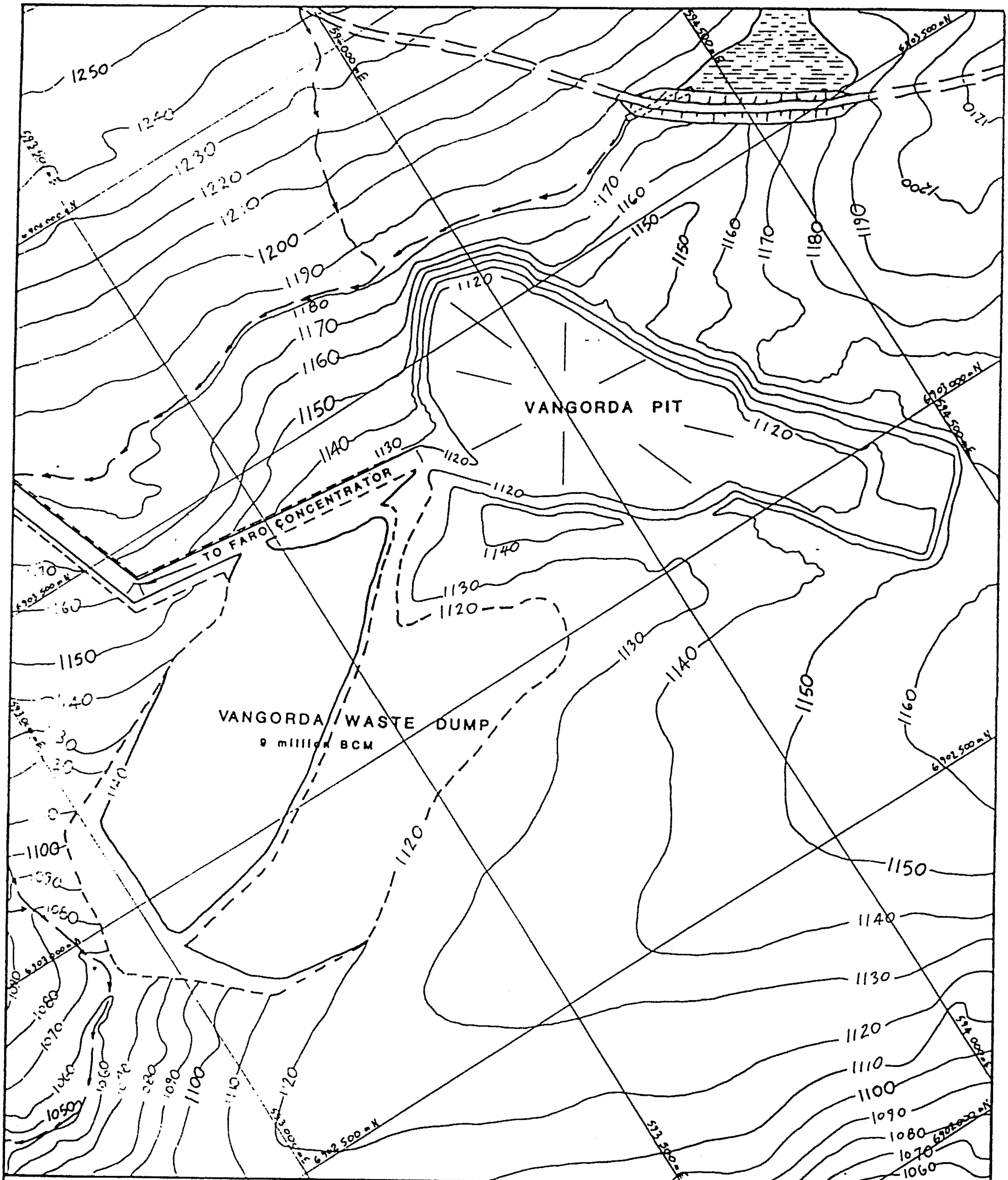


FIGURE 5.2 - 1

VANGORDA ULTIMATE PIT



0 100 200 300 400 500 METRES

CURRAGH RESOURCES

VANGORDA WASTE DUMP
GENERAL AREA

DR.BY: CGY
NOV./86
5.1 - 2

5.3 Grum Pit

5.3.1 Geotechnical

Geotechnical parameters for the design of the Grum pit are adapted from a December 1979 draft report prepared by Montreal Engineering Co. Ltd. for CAMC (ref. 3).

5.3.1.1 Geology

Surficial deposits within the mine and waste dump areas consist primarily of morainal and glaciofluvial deposits, with depths ranging from a minimum of 0-10 metres in the northern region of the deposit to over 100 metres in the south end. These deposits contain tills, uniform silts, sands and gravels. Pockets of permafrost do exist in the area, but permafrost has not been found 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 in the Grum deposit occurs within the phyllites.

The overall structure of the ore zone is that of a broad syncline. Dips on the northwest limb range from 30° to 40°, while the structure of the south limb is more complex, with a large "S" shaped fold apparently overturing the zone. The structure is further complicated by a number of faults which cut through the deposit.

The primary structural control on sulphide deposition appears to be the original bedding. The main foliation has been folded into a broad syncline with the Grum deposit located on the north limb. It is this foliation that is the most important feature affecting pit stability. Strike and dip of this foliation are not well defined, but on average strikes north-south (true) and dips to the west.

5.3.1.2 Wall design

Pit wall 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 southeast and northeast walls, with the foliation dipping into the pit, have been designed at a shallower slope than the southwest and northwest walls. Slope design parameters are summarized in the table below:

Wall	Slope (overall, not including roads)
SE	40°
NE	40°
SW	45°
NW	45°

Overburden slopes have been designed at 35°.

Geotechnical studies still required include:

- more accurate definition of the orientation surfaces.
- detailed investigation of the overburden strength parameters.
- investigation of the need for overburden dewatering.

5.3.2 Economic Modeling - Grum Pit

An economic model for the Grum pit was generated using PC-Mine software. The economic model generated is used as a tool for pit design only and is not used for operating cost estimates. The economic model represents the net value of each model block, based on all operating costs and an estimated revenue. Source of operating cost data is the 1987 Curragh Resources Operating Budget. The economic model is based on a 5% (combined lead plus zinc) cutoff grade. This cutoff grade is somewhat arbitrary and is based on Faro experience; more work is required to determine an economic cutoff.

5.3.2.1 Costs

Costs for the economic model are divided into three categories: volumetric mining costs, variable haulage costs, and ore based costs.

Volumetric mining costs apply to all model blocks and include costs for drilling, blasting, loading, "fixed" haulage (road maintenance etc.), and mining services. All of these costs were taken from the 1987 Curragh Resources Operating Budget, except that drilling and blasting costs for the unconsolidated overburden have been reduced by 90%, as this material is not expected to require blasting.

Variable haulage costs for a block are calculated as a function of the block location and grade. This is to allow for increased haulage costs as the pit deepens, and also is to allow for the long ore haul.

Ore based costs are calculated for all blocks grading over 5% (combined lead plus zinc) and include: processing cost, Faro G and A, Whitehorse G and A, and Toronto G and A, but do not include concentrate transport or smelting fees.

Cost parameters are summarized in tables 5.3-1 and 5.3-2.

5.3.2.2 Revenue

Revenue is calculated as payment for a percentage of contained metal in the concentrate less smelting and refining fees less concentrate handling costs. Smelting and concentrate handling costs are taken from the 1987 Curragh Resources Operating Budget.

Contained metal in the concentrate is calculated as a function of head grade, concentrate grade, and rock type.

A metallurgical recovery factor is defined for each ore type. This base recovery is then adjusted by a recovery factor, which is based on current Faro milling experience, to allow for the "head grade effect". The head grade effect equations are 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 ln (Wt % to Pb conc.) + .733
Au factor = 0.18 x ln (Wt % to Pb conc.) + .733

Table 5.3-3 shows the calculation of revenue estimates for entry into the PC-MINE system; tables 5.3-4 nd 5.3-5 show the PC-MINE referenced metallurgical and revenue parameters and table 5.3-6 shows a sample calculation for one ore block.

5.3.3 Pit Design

The Grum ultimate pit design is based on sectional representations of the economic model. Sections of the economic model were constructed at 60 metre intervals through the orebody. These sections show the net economic value of each block on the section. For each section, an optimum pit was determined, based on the economic model combined with the geotechnical wall parameters. No allowances for ramps were made at this time.

After the sectional representations of the economic pit were complete, the information was transferred to plan. At this point the walls were adjusted to give section to section continuity, and the haul ramp was included. This defines the ultimate pit.

Because the northeast wall is the least stable geotechnically, the haul road has been designed to avoid that wall. In this way, should a failure of the northeast wall occur, mining can continue while stablization operations are underway.

A potential addition to the main orebody, known as the Champ Zone, has been identified at the southern extremity of the orebody. This zone has not yet been quantified and is not included in the ultimate pit.

In order to both reduce the stripping necessary to sustain ore production, and to protract the stripping over as long a period as possible, the pit has been designed in three stages.

The first stage of the pit, shown in figure 5.3-1, goes to the 1180 metre elevation. Concurrent with the mining of the Stage One pit, stripping will begin for the second stage of the pit shown in figure 5.3-2. This will involve pushing back both the southeast and northeast walls, with the bulk of the stripping being done on the southeast wall. Stripping will continue to the 1180 metre elevation before any appreciable ore is released. The Stage Two pit floor is at the 1090 metre elevation.

Stage Three stripping will take place as ore is being mined from Stage Two. This will involve pushing back the walls again, in order to mine deeper into the pit. The floor of the Stage Three pit will be at an elevation of 1030 metres. The Stage Three pit is shown in figure 5.3-3.

It must be recognized that both the ultimate pit and the staging sequence presented here is little better than a "first pass" at an optimum pit. Much work is required to refine the pit design. Such work would be expected to both reduce the overall strip ratio, and provide a better sequence of stages.

5.3.4 Pit Reserves

Pit reserves for each stage have been calculated on 4.5 metre model benches. In addition, Stage Two stripping quantities have been broken out into north and south side stripping.

Mining reserves are adjusted to reflect a 95% mining recovery and a 15% waste dilution. Recovery is defined as the tonnes of ore mined, expressed as a percentage of the in-situ 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 = (mining recovery)x(in-situ ore tonnes)
Dilution tonnes = (recovered ore tonnes)x(dilution)
Mined ore tonnes = Recovered ore tonnes + dilution tonnes
= (recovered ore tonnes)x(1 + dilution)
= (in-situ ore tonnes)x(recovery)x(1 + dilution)

The dilution is assumed to contain zero metal:

Recovered ore grade = in-situ ore grade
Dilution grade = zero

$$\text{Mined ore grade} = \frac{\text{(recovered ore grade) (recovered tonnes)}}{\text{(recovered tonnes + dilution tonnes)}}$$

$$= \frac{\text{(in-situ ore grade) (recovered tonnes)}}{\text{(recovered tonnes) (1 + dilution)}}$$

$$= \frac{\text{(in-situ ore grade)}}{\text{(1 + dilution)}}$$

Ore reserves are calculated based on a 5% (combined lead plus zinc) cutoff grade, with quantities grading between 4% and 5% also being reported.

Table 5.3-7 details the mining reserves for each stage of the pit.

5.3.5 Waste Dumps

Mining of the Grum Pit will see the removal of some 90 million BCM of waste. A waste dump has been selected at the southeast corner of the pit, near the pit entrance.

The dump will be constructed in a small valley south of the pit initially at the 1260 metre elevation, later being built up to the 1300 metre elevation.

The dump location was chosen for several reasons:

- drainage down this valley will be diverted as a result of the pit excavation, so that placing the dump here minimizes the impact on local drainage patterns.

- the dump is close to the pit exit, minimizing haulage distance.

- the valley provides the greatest potential dump volume while minimizing the affected surface area.

A separate dump for the overburden has not been designed but such a dump may prove practical for reclamation purposes or for dump stability considerations. Such a dump could be located to the west of the main dump.

The location of the waste dump is shown in figure 5.3-5.

PC-MINE VERSION 1.10
SERIAL NO : 20320
19/11/1986

Curragh Resources
Grum Data Geological Model

SOFTWARE BY GEMCOM SERVICES INC
MODULE 1.04
PAGE 1

PRINTOUT OF ROCK-TYPE INFORMATION FOR RECORDS (1) TO (13)

MINING COST DATA

REC	STAT	ROCK CODE	DESCRIPTION	COST DATA [CDN $ PER bcm] DRILLING	BLASTING	LOADING	FIXED HAULAGE	MINING SERVICES
1	1	0	Air	.0000	.0000	.0000	.0000	.0000
2	1	1	A Type / 2nd code A, A4, D, E. .	.1748	.3865	.4018	.3533	.9817
3	1	2	A4 Type	.1748	.3865	.4018	.3533	.9817
4	1	3	C Type	.1748	.3865	.4018	.3533	.9817
5	1	4	D Type	.1748	.3865	.4018	.3533	.9817
6	1	5	E Type	.1748	.3865	.4018	.3533	.9817
7	1	6	E4 Type	.1748	.3865	.4018	.3533	.9817
8	1	7	G Type	.1748	.3865	.4018	.3533	.9817
9	1	8	H Type	.1748	.3865	.4018	.3533	.9817
10	1	9	L Type	.1748	.3865	.4018	.3533	.9817
11	1	10	Waste	.1748	.3865	.4018	.3533	.9817
12	1	11	Unconsolidated Overburden	.1748	.3865	.4018	.3533	.9817
13	1	12	Partially Above Topography	.0175	.0387	.4018	.3533	.9817
				.0175	.0387	.4018	.3533	.9817

TABLE 5.3 - 1

VOLUMETRIC MINING COSTS - GRUM PIT

PC-MINE VERSION 1.10
SERIAL NO : 20320
19/11/1986

Curragh Resources
Grum [illegible] Geological Model

SOFTWARE BY GEMCOM SERVICES INC
MODULE 1.06
PAGE 1

PRINTOUT OF COSTS INFORMATION FOR RECORDS [1] to [1]

DETAILED PRINTOUT FOR RECORD [1]

DATE CAPTURED : 4/11/1986

RECORD DESCRIPTION :November 86 Model

GENERAL COST INFORMATION

MINE CALL FACTOR	:	90.00 PERCENT
ORE/STOCKPILE CUT-OFF GRADE	:	5.000
STOCKPILE/WASTE CUT-OFF GRADE	:	4.000
ORE PROCESSING COST	:	6.52
MINE ADMIN COST	:	1.56
HEAD OFFICE ADMIN COST	:	.36
SPARE COST # 1	:	.50
SPARE COST # 2	:	.00

HAULAGE INFORMATION

	PIT EXIT ELEVATIONS [m]	AVERAGE IN-PIT HAULAGE [m]	AVERAGE SURFACE HAULAGE [m]	HORIZONTAL HAULAGE COST [CDN $ /bcm/100 m]	UPWARDS VERTICAL HAULAGE COST [CDN $ /bcm/10 m]	DOWNWARDS VERTICAL HAULAGE COST [CDN $ /bcm/10 m]
ORE	1260.00	800.00	[illegible]	.0187	.0624	.0368
STOCKPILE	1260.00	800.00	[illegible]	.0187	.0624	.0368
WASTE	1260.00	800.00	[illegible]	.0187	.0624	.0368

TABLE 5.3 - 2

ORE BASED COSTS AND HAULAGE COSTS - GRUM PIT

Revenue Calculations Grum Pit

Deductions:	$/tonne		
Highway Freight	51.9329		
Ocean Freight	24.0394	$US/$CDN	1.39
Smelting	201.3746		
Total:	277.3469		

	$ US	$ CDN	Smelter Pays
Lead	$0.20	0.278	95.00%
Zinc	$0.42	0.5838	85.00%
Silver	$5.50	7.645	95.00%
Gold	$400.00	556	95.00%

Lead; per tonne of ore

Head	Recovery Factor	Conc Grade	Conc Tonnes	Metal Pounds	Revenue
2	0.892459	60	0.029748	39.35073	2.141831
4	1.017919	60	0.067861	89.76511	4.885849
6	1.091308	60	0.109130	144.3553	7.857159
8	1.143378	60	0.152450	201.6575	10.97607
10	1.183767	60	0.197294	260.9761	14.20474
12	1.216768	60	0.243353	321.9017	17.52087
14	1.244669	60	0.290422	384.1636	20.90974
16	1.268838	60	0.338356	447.5696	24.36088
18	1.290157	60	0.387047	511.9757	27.86646
20	1.309227	60	0.436409	577.2705	31.42041

Zinc; per tonne of ore

Head	Recovery Factor	Conc Grade	Conc Tonnes	Metal Pounds	Revenue
2	0.845289	55	0.030737	37.27090	9.969902
4	0.957579	55	0.069642	84.44407	22.58864
6	1.023265	55	0.111628	135.3547	36.20717
8	1.069869	55	0.155617	188.6926	50.47497
10	1.106018	55	0.201094	243.8354	65.22555
12	1.135554	55	0.247757	300.4163	80.36087
14	1.160527	55	0.295406	358.1934	95.81613
16	1.182159	55	0.343900	416.9944	111.5453
18	1.201240	55	0.393133	476.6906	127.5139
20	1.218308	55	0.443021	537.1821	143.6953

Silver; per tonne of ore

Head	Recovery Factor	Metal Grams	Metal Ounces	Revenue
300	1	300	9.646302	70.05868

Gold; per tonne of ore

Head	Recovery Factor	Metal Grams	Metal Ounces	Revenue
50	1	50	1.607717	849.1961

TABLE 5. 3- 3

RVEENUE CALCULATIONS - GRUM PIT

PC-MINE VERSION 1.10
SERIAL NO : 20020
13/11/1986

Curragh Resources
[illegible] Geological Model

SOFTWARE BY GEMCOM SERVICES INC
MODULE 1.04

PRINTOUT OF ROCK-TYPE INFORMATION FOR RECORDS [illegible]

METALLURGICAL DATA

ROCK CODE	DESCRIPTION	TYPE	PRIMARY MINERAL	CUT OFF GRADES 0 - S/F	CUT OFF GRADES S/F - W	RECOVERIES (PERCENT) %Pb+Zn	%Pb	%Zn	Ag g/t	Au g/t
0	Air	air								
1	A Type / 2nd code A, AA, C, E, [illegible]	ore	%Pb+Zn	5.000	4.000	.0	80.0	83.0	65.0	33.0
2	AA Type	ore	%Pb+Zn	5.000	4.000	.0	80.0	83.0	65.0	33.0
3	C Type	ore	%Pb+Zn	5.000	4.000	.0	80.0	83.0	65.0	33.0
4	[illegible] Type	ore	%Pb+Zn	5.000	4.000	.0	80.0	83.0	65.0	33.0
5	E Type	ore	%Pb+Zn	5.000	4.000	.0	80.0	83.0	64.0	33.0
6	[illegible] Type	ore	%Pb+Zn	5.000	4.000	.0	80.0	83.0	65.0	33.0
7	B Type	ore	%Pb+Zn	5.000	4.000	.0	80.0	83.0	65.0	33.0
8	H Type	ore	%Pb+Zn	5.000	4.000	.0	80.0	83.0	65.0	33.0
9	L Type	ore	%Pb+Zn	5.000	4.000	.0	80.0	83.0	65.0	33.0
10	Waste	waste								
11	Unconsolidated Overburden	waste								
12	Partially Above Topography	waste								

TABLE 5.3 - 4

METALLURGICAL PARAMETERS - GRUM PIT

PRINTOUT OF COSTS INFORMATION FOR RECORDS [1] to [1]

DETAILED PRINTOUT FOR RECORD [1]

REVENUE / HEAD GRADE INFORMATION

NO OF REVENUE / HEAD GRADE CURVES : 5

DESCRIPTION : Pb + Zn (no revenue)

POINT	HEAD GRADE [%Pb+Zn]	REVENUE [CDN $ /%Pb+Zn]
1	.000	.000
2	100.000	.000

DESCRIPTION : Lead Revenue

POINT	HEAD GRADE [%Pb]	REVENUE [CDN $ / %Pb]
1	.000	.000
2	2.000	2.142
3	4.000	4.886
4	6.000	7.857
5	8.000	10.976
6	10.000	14.205
7	12.000	17.521
8	14.000	20.910
9	16.000	24.361
10	20.000	31.420

DESCRIPTION : Zinc Revenue

POINT	HEAD GRADE [%Zn]	REVENUE [CDN $ / %Zn]
1	.000	.000
2	2.000	9.970
3	4.000	22.587
4	6.000	36.207
5	8.000	50.480
6	10.000	65.226
7	12.000	80.361
8	14.000	95.816
9	16.000	111.545
10	20.000	[illegible]

DESCRIPTION : Silver Revenue

POINT	HEAD GRADE [Ag g/t]	REVENUE [CDN $ /Ag g/t]
1	.000	.000
2	300.000	70.059

TABLE 5.3 - 5

REVENUE/HEAD GRADE INFORMATION - GRUM PIT

PRINTOUT OF COSTS INFORMATION FOR RECORDS [1] to [1]

DETAILED PRINTOUT FOR RECORD [1]

DESCRIPTION : Gold Revenue

POINT	HEAD GRADE (Au g/t)	REVENUE (CDN $ / Au g/t)
1	.000	.000
2	50.000	547.196

TABLE 5.3 - 5 (CONTINUED)

REVENUE/HEAD GRADE INFORMATION - GRUM PIT

TRACE BLOCK IN COLUMN [40] ROW [63] LEVEL [36]   ROCK-TYPE CODE :   6
MATERIAL TYPE (0=AIR 1=WASTE 2=ORE) : 2
PRIMARY MINERAL : %Pb+Zn
PRIMARY GRADE : 12.583 [%Pb+Zn]
DENSITY : 4.31 [tn/bcm]
BLOCK VOLUME : 540.00 [bcm]
BLOCK TONNAGE : 2329.02

ORE-STOCKPILE CUT-OFF GRADE : 5.000 [%Pb+Zn]
STOCKPILE-WASTE CUT-OFF GRADE : 4.000 [%Pb+Zn]

LABEL:	UNITS:	GRADE:	REVENUE:	RECOVERY:	BLOCK REVENUE:	CUM. REVENUE:
1	%Pb+Zn	12.58	.00	.00	.00	.00
2	%Pb	4.47	5.58	80.00	9361.47	9361.47
3	%Zn	8.11	51.32	83.00	89285.78	98647.25
4	Ag g/t	71.00	16.58	65.00	22590.62	121237.90
5	Au g/t	1.24	21.09	33.00	14591.14	135829.00

VOLUMETRIC MINING COSTS :

UNIT DRILLING COST :	.175	TOT DRILLING COST :	94.39
UNIT BLASTING COST :	.387	TOT BLASTING COST :	208.71
UNIT LOADING COST :	.402	TOT LOADING COST :	216.97
UNIT SERVICES COST :	.982	TOT SERVICES COST :	530.12
UNIT FIXED HAUL COST :	.353	TOT FIXED HAUL COST :	190.78

VARIABLE HAULAGE COSTS :

UNIT VERTICAL HAULAGE COST : .062
VERTICAL HAULAGE DISTANCE : 108.50
TOT VERTICAL HAULAGE COST : 365.84
UNIT HORIZONTAL HAULAGE COST : .019
HORIZONTAL HAULAGE DISTANCE : 15500.00
TOT HORIZONTAL HAULAGE COST : 1582.77

ORE BASED MINING COSTS :

UNIT PROCESSING COST :	6.519	TOT PROCESSING COST :	15182.65
UNIT MINE ADMIN COST :	1.558	TOT MINE ADMIN COST :	3628.61
UNIT HEAD OFFICE ADMIN COST :	.856	TOT HEAD OFFICE ADMIN COST :	1992.71
UNIT SPARE COST # 1 :	.930	TOT SPARE COST # 1 :	2165.06
UNIT SPARE COST # 2 :	.000	TOT SPARE COST # 2 :	.00

TOTAL BLOCK REVENUE : 135829.00

TOTAL VOLUMETRIC MINING COST : 3189.58 -
TOTAL ORE BASED MINING COST : 22969.03 -

TOTAL BLOCK MINING COST : 26158.61

BLOCK ECONOMIC VALUE : 109670.40

TABLE 5. 3- 6

Sample Calculation

Mining Reserves
Grum Stage One Pit

	Tonnes	%(Pb+Zn)	% Pb	% Zn	Ag g/t	Au g/t
Waste Rock	39207270					
Overburden	15067690					
Sulphide Waste	1254449					
All Waste	55529409					
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
Strip Ratio	6.5					

Mining Reserves
Grum Stage Two Pit South Side Stripping

	Tonnes	%(Pb+Zn)	% Pb	% Zn	Ag g/t	Au g/t
Waste Rock	47445630					
Overburden	2776652					
Sulphide Waste	99787					
All Waste	50322069					
4 - 5 % combined	69419	4.05	1.96	2.09	35	0.98
+ 5 % combined	219664	11.21	4.85	6.36	76	0.94
Strip Ratio	174.1					

Mining Reserves
Grum Stage Two Pit North Side Stripping

	Tonnes	%(Pb+Zn)	% Pb	% Zn	Ag g/t	Au g/t
Waste Rock	13486850					
Overburden	9447329					
Sulphide Waste	29778					
All Waste	22963957					
4 - 5 % combined	47084	4.13	1.42	2.71	27	0.63
+ 5 % combined	355281	11.12	4.34	6.78	73	0.84
Strip Ratio	57.1					

TABLE 5.3 - 7

GRUM PIT RESERVES

Mining Reserves
Grum Stage Two Pit Core

	Tonnes	%(Pb+Zn)	% Pb	% Zn	Ag g/t	Au g/t
Waste Rock	23499650					
Overburden	34733					
Sulphide Waste	1704585					
All Waste	25238968					
4 - 5 % combined	1292265	3.87	1.54	2.33	26	0.64
+ 5 % combined	8352860	9.34	3.48	5.86	59	0.93
Strip Ratio	2.6					

Mining Reserves
Grum Stage Two Pit Total

	Tonnes	%(Pb+Zn)	% Pb	% Zn	Ag g/t	Au g/t
Waste Rock	84432130					
Overburden	12258714					
Sulphide Waste	1834150					
All Waste	98524994					
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
Strip Ratio	9.5					

Mining Reserves
Grum Stage Three Pit

	Tonnes	%(Pb+Zn)	% Pb	% Zn	Ag g/t	Au g/t
Waste Rock	71302610					
Overburden	4254012					
Sulphide Waste	1050922					
All Waste	76607544					
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
Strip Ratio	12.5					

TABLE 5.3 - 7 (CONTINUED)

Mining Reserves
Grum Total Pit

	Tonnes					
Waste Rock	194942010					
Overburden	31580416					
Sulphide Waste	4139521					
All Waste	230661947					
		%(Pb+Zn)	% Pb	% Zn	Ag g/t	Au g/t
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
Strip Ratio	9.2					

TABLE 5.3 - 7 (CONTINUED)

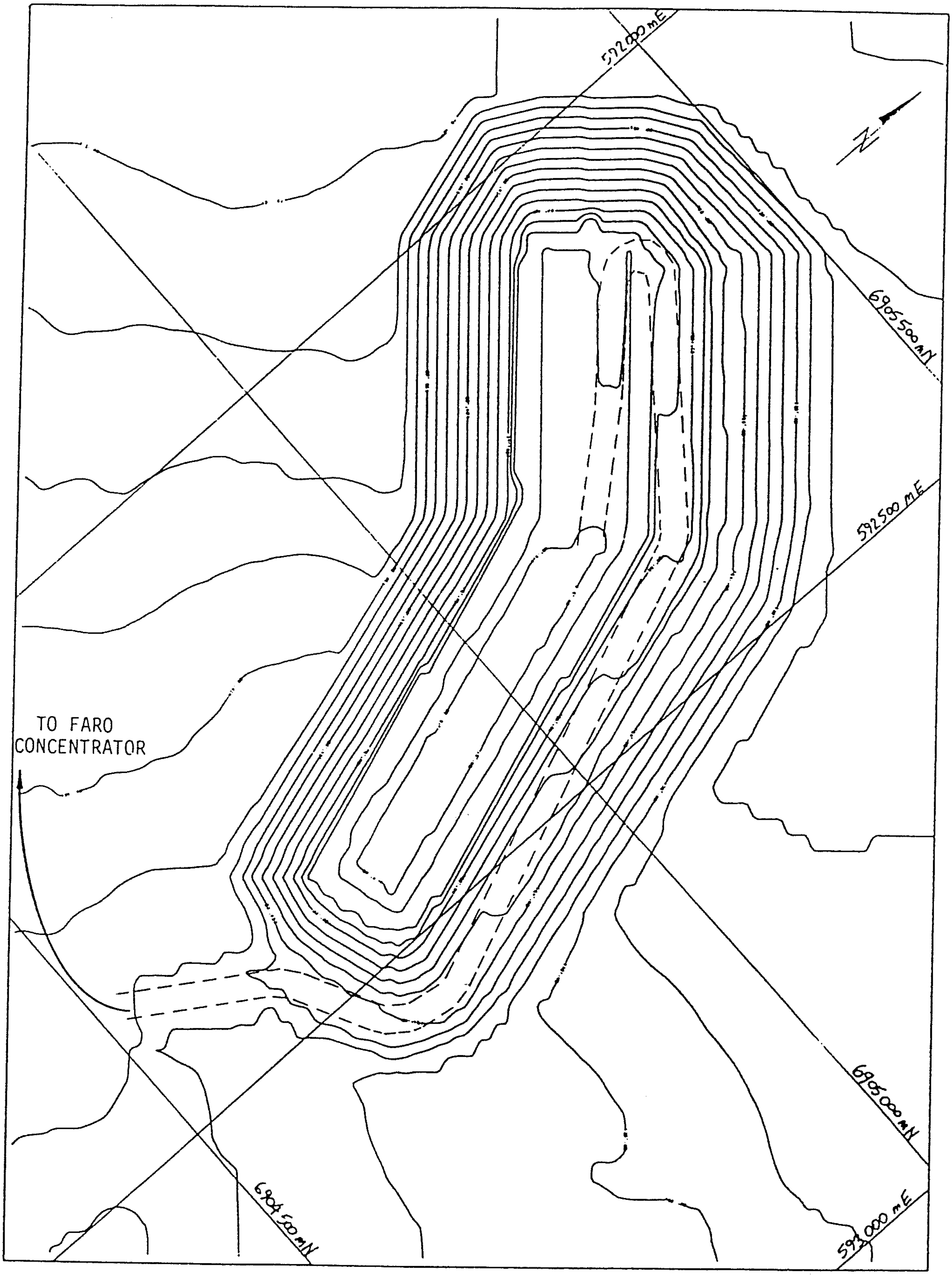


FIGURE 5.3 - 1

GRUM STAGE ONE PIT

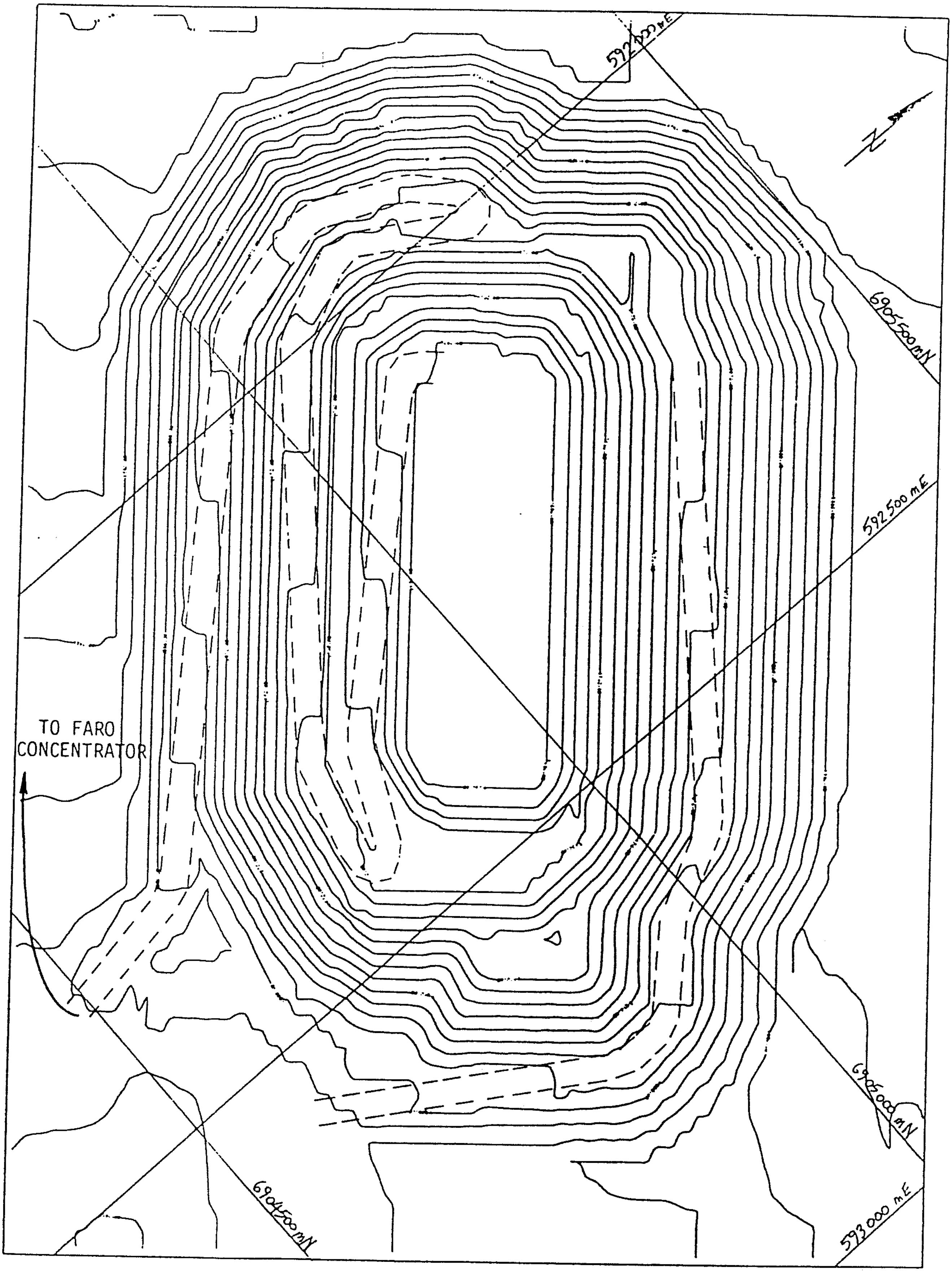


FIGURE 5. 3- 2

GRUM STAGE TWO PIT

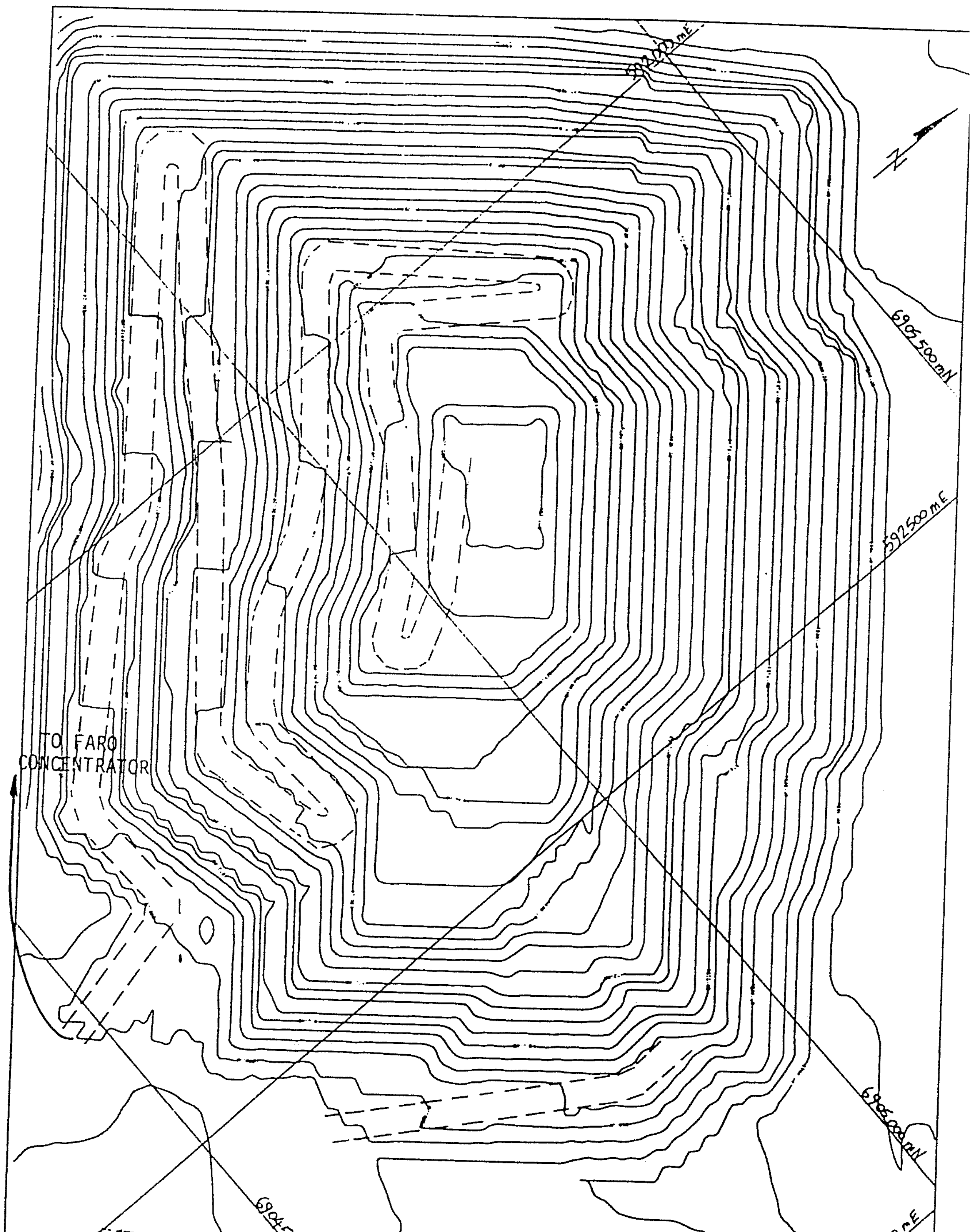
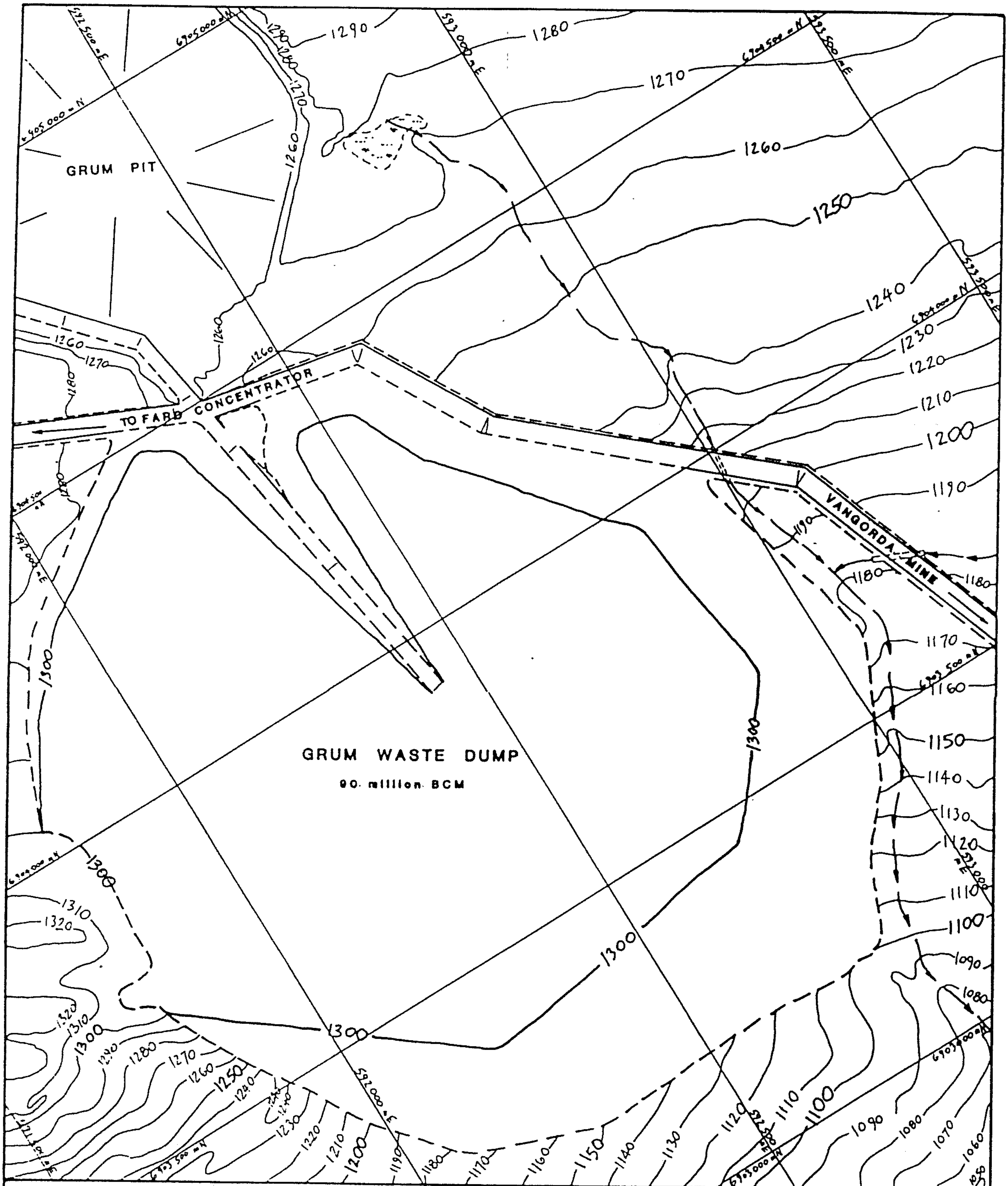


FIGURE 5.3 - 3

GRUM STAGE THREE PIT



0 100 200 300 400 500 METRES

CURRAGH RESOURCES

GRUM WASTE DUMP
GENERAL AREA

DR.BY: CGY
NOV./86
5.2-4

6.1 Power

Power to the Vangorda Plateau will be provided by a new 5000 metre powerline, to be constructed as a branch off the existing 138kV powerline. A substation will be located near the Grum Pit, and from this substation power will be distributed to the two pits.

An alternative currently being studied is to run a new 35 kV line from the Faro Pit to the Vangorda Plateau. This may provide a capital cost advantage in that the substations required would cost less.

Since no electrical equipment in excess of current usage is being planned, the existing power supply from NCPC is expected to be adequate.

6.2 Vangorda Haul Road

Ore is to be hauled to the Faro concentrator via conventional haulage trucks. Trucks loaded with ore at the shovel face will haul to a stockpile located near the pit exits. The ore will then be rehandled and hauled to the Faro concentrator on a haul road yet to be completed.

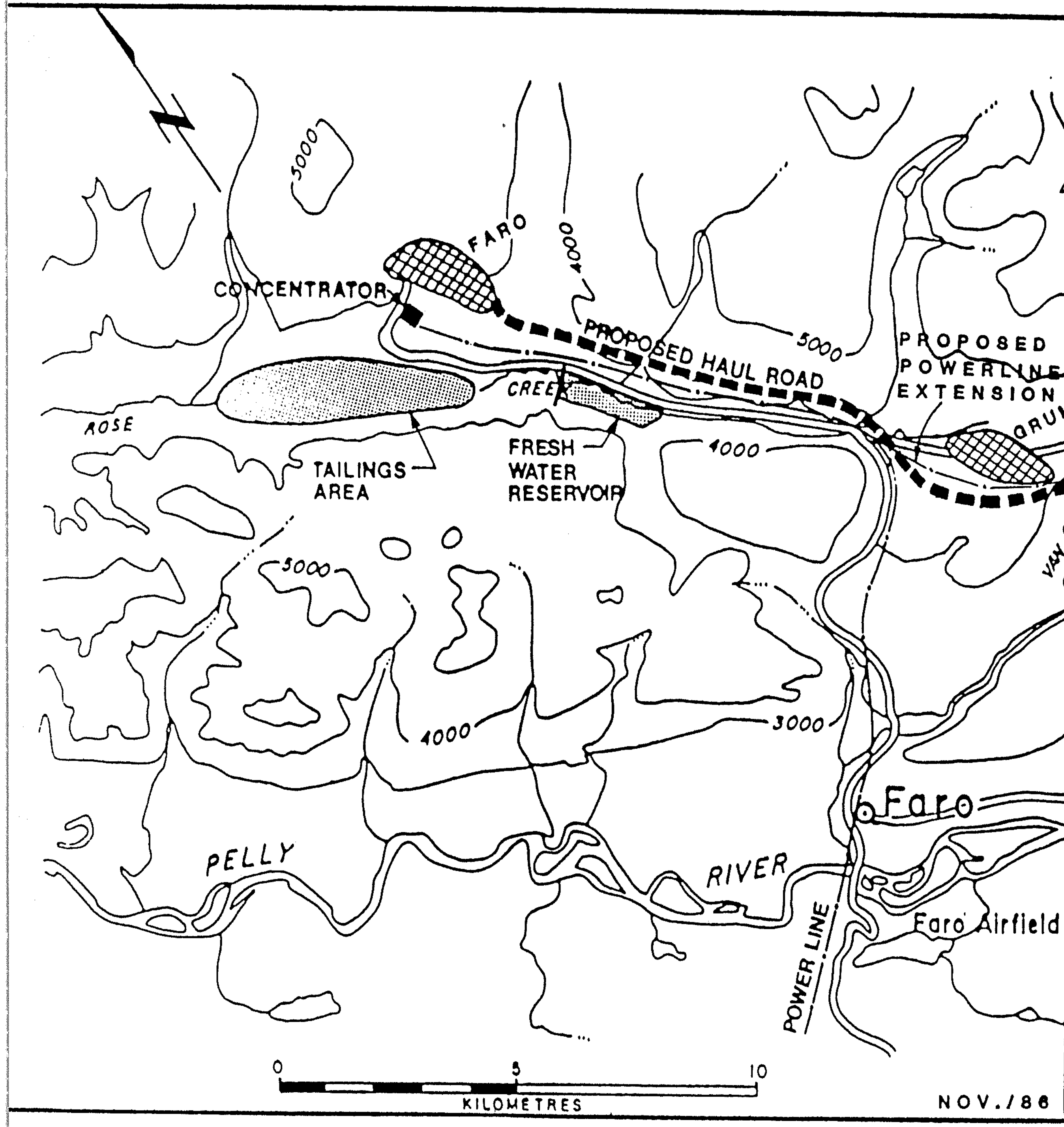
Construction of the haul road has already begun. Beginning in October 1986 waste rock from the pit is being used to construct the first stage of the haul road; the crossing of the North Fork of Rose Creek. Design of the rock drain and creek crossing at this point was done by Golder Associates in September of 1986 (ref 6).

Alignment of the haul road is essentially that developed by Stanley Associates for CAMC in December 1980 (ref. 5). The first 3000 metres of the Stanley design has been realigned in order to reduce the adverse grade on that portion of the haul road. Also, the last 3000 metres will be realigned to bring the haul road south of the Grum pit.

Completion of the haul road is expected to take two construction seasons; engineering and design work should be completed early in 1987 in order for the haul road to be complete by late 1988.

The crossing of the north Fork of Rose Creek will take approximately 16 million tonnes of fill.

Figure 6.2-1 shows the approximate location of the proposed haul road.



CURRAGH RESOURCES

VANGORDA PLATEAU
HAUL ROAD

N.T.S. 105 K TAY RIVER

FIG. 6.2-1

6.3 Vangorda Creek Diversion

In order to mine the Vangorda pit, Vangorda Creek is to be diverted along the west valley wall. This will require construction of a diversion dam and a diversion channel. Water flow will then be returned to the creek bed via an existing drainage path. Figure 6.3.1 shows the location of the diversion structures.

An engineering report on the creek diversion was prepared for CAMC in 1979 by Golder Associates (ref. 1) and much of this report forms the basis for this preliminary design.

6.3.1 Diversion Dam

The diversion dam is to be located near the present mine road, at an elevation of approximately 1180 metres. The crest of the dam is to be at an elevation of 1185 metres. An emergency overflow weir section (draining into the pit) is to be incorporated at the east end of the dam. Since the dam is designed to accomodate a fifty year return flood, and the life of the pit is less than three years, the risk and consequence of overflow into the pit is slight.

The glacial till to be excavated as a result of the mining activities will provide a suitable impervious core for the dam. Also, excavated bedrock is to be used as rip-rap to armour the upstream face and the overflow weir. Approximately 30,000 cubic meters of fill will be required for the dam construction.

6.3.2 Diversion Channel

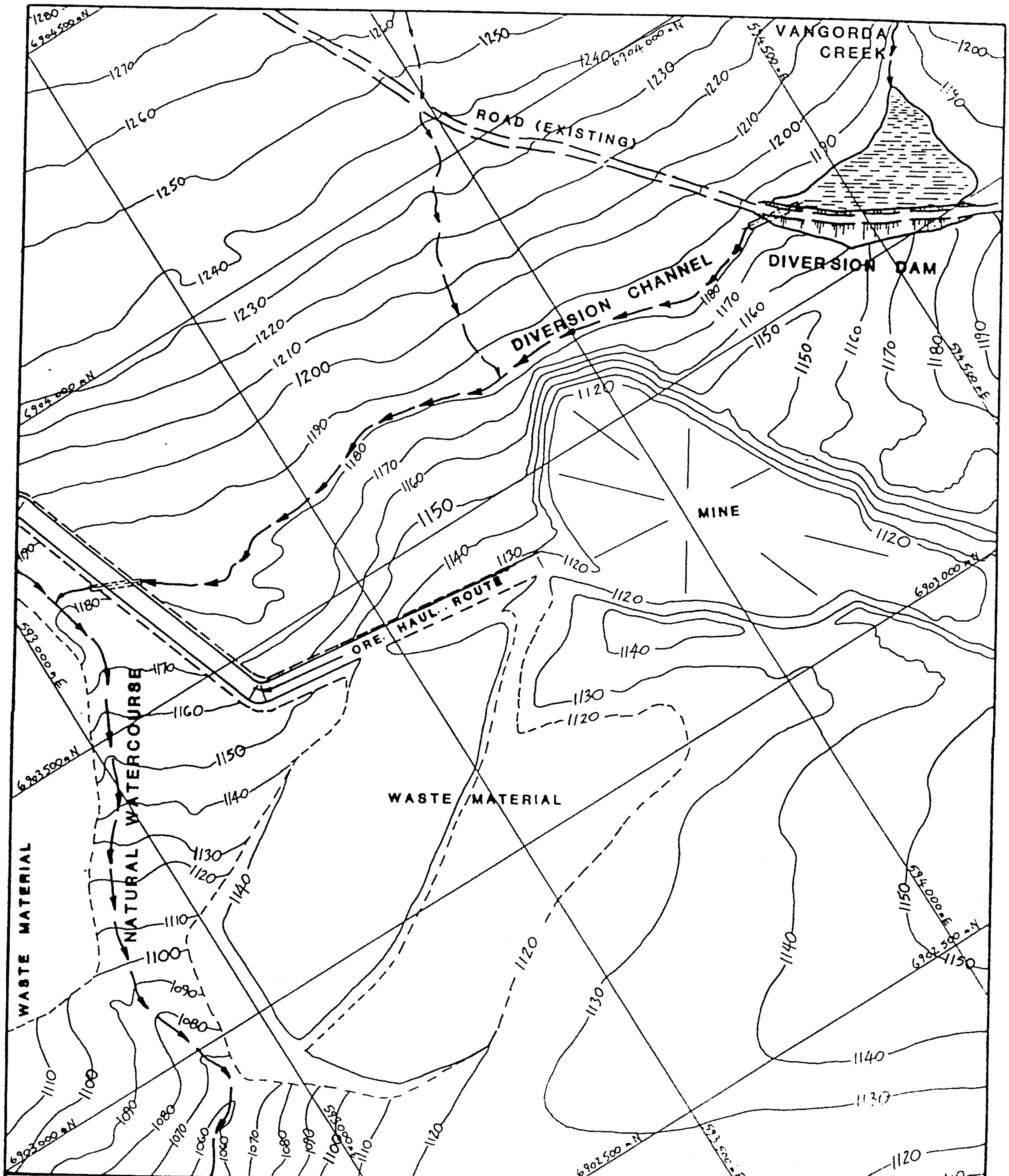
The diversion channel is to be routed on the west valley wall along the 1182 m contour to an existing drainage path, returning the water flow to the original creek bed. The channel will be approximately 1400 meters long. The proposed alignment of the ditch was selected so that the water flow will avoid both the mine and the waste dump. The channel is designed to accomodate a 50 year return flow of 20.5 m^3/sec.

Test pits along the channel alignment indicate that the bedrock material is rippable, and that permafrost is not to be expected. The excavated bedrock will provide a suitable armouring layer; the mining operation will provide a secondary source of rock for the armouring.

6.3.4 Construction

In order to reduce costs and optimize equipment utilization, construction of the diversion structures will start at the same time as mining commences. This will allow Curragh Resources personell and equipment to do the bulk of the construction, with only minor contracting out or leasing of equipment required. Also, mine waste will provide much of the construction materials.

Because mining activities will be attendant on completion of the diversion, careful planning and control of the construction will be required.



CURRAGH RESOURCES	DR.BY: CGY
PROPOSED	NOV./86
VANGORDA CREEK DIVERSION	6.3 - 1

6.3.5 Abandonment

Upon abandonment, it is recommended that the diversion structures remain in place. This alternative is preferable to allowing the water to flow through the waste dump.

6.4 Buildings

Because of the proximity of the facilities at Faro (shops, office, dry, etc.), only a minimum number of buildings are suggested.

A lube and fuel station will be required. This would be located near the Grum pit exit. Equipment operating in the Vangorda pit will have to travel to the Grum pit once per shift for servicing.

A two bay maintenance facility is suggested. This would be a prefabricated building without a crane and would be used primarily for PM's. Major servicing of equipment would be done at the Faro facilities.

Appendix A

REFERENCES

1. Golder Associates; A report to Cyprus Anvil Mining Corporation on Engineering Recommendations for the Proposed Diversion of Vangorda Creed; December, 1979.

2. Cyprus Anvil Mining Corporation; The Development of the Vangorda Plateau Ore Deposits; 1981.

3. Montreal Engineering Co. Ltd; Cyprus Anvil Mining Corporation Ltd., Grum Deposit, Phase Two Geotechnical Studies, Draft Report; December 1979.

4. Cyprus Anvil Mining Corporation; Vangorda Geotechnical Study, Feasability Stage, October 1980.

5. Stanley Associates Engineering Ltd., Vangorda Plateau Haul Road Preliminary Design Study, December 1980.

6. Golder Associates; Proposed Rock Drain North Fork of Rose Creek, Faro, Yukon, September, 1986.

7. Kilborn Engineering (B.C.) Ltd; Curragh Resources Corporation Technical and Cost Review of the Cyprus Anvil Mine, Faro, Yukon, Volumes I and II, 1985.

APPENDIX B

METALLURGY

2.2 FARO ZONE III METALLURGY

2.21 Preliminary Laboratory Testwork

Interest in the effects of finer primary grinding on Anvil metallurgy was stimulated by testwork performed at Faro during the fall of 1978 on a pyrrhotitic ore type. The indications from this and subsequent work coincided with data from other sources and suggested that optimum metallurgy for most Faro ore types would be obtained by primary grinding to at least the 50 micron P_{80} range. Table 2 shows the sources of data relating to work to determine optimum grind for Faro ore types.

TABLE 2

PRIMARY GRIND LEVEL FOR OPTIMUM METALLURGY

DATA SOURCE	TYPE OF TESTS	NO. OF TESTS	OPTIMUM GRIND P_{80}	OPTIMUM GRIND %-325#
Cyprus Anvil Test Laboratory	Rougher & Cleaner Tests	100	40-45	85
Kamloops Research Laboratory	Rougher & Cleaner Tests	120	45-50	80
Sachtleben - FDR	Cleaner Testing	30	52	78
Mitsui Mining & Smelting	Rougher & Cleaner Tests	30	37-70	80-93

Notes: a) Numbers of tests are estimates only.

b) Japanese work identified pyrrhotitic ore as benefiting most from fine grinding.

c) Tests are all laboratory scale work on 1.0 or 2.0 kg samples.

Detailed laboratory testwork by ore type was then planned and carried out at the Kamloops Research Laboratory in order to ensure that these preliminary results were representative of the majority of ore types (Ref. Report KM008 - June 10th 1979). Parallel work at the Cyprus Anvil laboratory on a different suite of samples, but encompassing the same test methods yielded the similar results. All data from this latter phase of the program is summarized in the Table 3 below; For completeness, the pilot plant data, described in detail later, is also shown.

TABLE 3

METALLURGICAL IMPROVEMENTS WITH FINER GRINDING

DATA SOURCE	GRIND RANGE INVESTIGATED (P_{80} microns)	GRIND RANGE INVESTIGATED (%-325#)	METALLURGICAL IMPROVEMENTS ACROSS RANGE LEAD GRADE	LEAD RECOVERY	ZINC GRADE	ZINC RECOVERY
Rougher Tests KRAL-78	135 - 40	50 - 85	7.0	*	2.50	*
Rougher Tests CAMC 78-79	125 - 37	45 - 90	11.3	*	5.9	*
Cleaner Tests KRAL-79	125 - 45	45 - 80	2.5	7.0	2.1	9.5
Cleaner Tests CAMC-79	125 - 40	45 - 85	3.2	5.2	4.4	4.0
Plant Test Nov.13-17/78	125 - 60	45 - 70	3.0	5.8	3.0	4.0
Cleaner Tests KRAL-79	145 - 30	30 - 90	*	9.6	*	12.0
Lakefield Pilot Plant I-79	130 - 45	40 - 85	5.9	3.6	3.6	5.9
II-79	140 - 55	35 - 80	4.5	1.0	4.5	4.5

Notes: a) * signifies that results were adjusted to a constant grade or recovery for the purposes of analysis of data.

b) Pilot plant data refers to the two separate samples tested.

c) CAMC - Cyprus Anvil Mining Corporation - Faro
KRAL - Kamloops Research and Assay - Kamloops
Lakefield - Lakefield Research Laboratory - Ontario

2.22 Pilot Plant Test Program

In order to verify the results of the laboratory test program, a pilot plant investigation was planned and executed. The work was performed at the testing facility of Lakefield Research of Canada, Limited, Lakefield, Ontario, during September and October of 1979.

During the course of the investigation at Lakefield, two bulk samples were subjected to a detailed investigation. The technical aspects of the work are described in Lakefield Reports LR 2202 Volumes I-IV. Supervision and direction of the testing was the responsibility of W. Muir, Plant Metallurgist, Cyprus Anvil Mining Corporation and P.J. Brown, Consulting Metallurgist, Met Engineers Ltd. At all times during the program, the senior Lakefield metallurgist, Serge Bulatovic provided the indespensible link between client and pilot plant operating personnel.

The work was performed using the standard reagent pattern currently employed at Anvil and with a flotation circuit designed to approximate to that proposed in the modified circuit at Anvil. Grinding and regrinding effects were the principle parameters investigated in the program. The program had two major objectives which were:

(a) The determination of the optimum economic grind level.

(b) Estimation of plant metallurgy at this optimum point.

2.23 Determination of the Optimum Grind

Using the results of the pilot plant program at various grind levels and by estimating the capital expenditure needed to achieve the various grind levels, it became possible to estimate optimum economic grind level. This calculation was performed by using the metallurgical data generated by the pilot plant work, a capital expenditure estimate generated by Kilborn Engineering Ltd. for various grinding configurations and an operating unit cost increment. The results of this study are summarized in table 4 below, and confirm that the economic and the metallurgical optima are near coincident at a P_{80} of about 50 microns.

TABLE 4

OPTIMUM ECONOMIC GRIND LEVEL

CASE	PRIMARY H.P.	REGRIND H.P.	TOTAL H.P.	GRIND TARGET P_{80}	CAPITAL COST $\$x10^6$	OPERATING COST INCREASE $\$x10^6$	RELATIVE NET PRESENT VALUE $\$x10^6$
1	11,700	1,350	13,050	50	43.0	6.8	153.9
2	9,200	2,350	11,550	70	43.1	5.1	129.8
3	7,700	1,850	10,550	100	38.6	4.3	112.7
4	5,200	1,350	6,550	130	0	0	117.4

Notes: a) Reference - Memo Brown to Taggart Dec. 16, 1979 "Optimum Grind Calculations".

b) Capital cost includes power plant: Clearly case 4 is with no expansion.

c) Operating cost includes manpower, steel and reagents.

d) Data assumes milling a 50:50 blend of Grum and Anvil Zone III ore.

In reviewing the data in table 4, it is important to note that for the four alternatives considered that the difference in capital and installed power is very small indeed. The difference in predicted plant metallurgy for each case is significant.

The calculation of Relative Net Present Value was achieved by making certain assumptions about costs and prices: The values computed are appropriate for comparative purposes but may not be used as absolute values.

2.24 Predicted Metallurgy at Optimum Grind

Pilot plant data at the economic optimum grind level of 50 microns was averaged for each of the two bulk samples tested and the results used as a basis to predict the plant metallurgy at the optimum grind.

Both ore samples treated were taken from the Faro Zone I pit and were selected by the Engineering department of Cyprus Anvil as being representative of the ore type known as 4E. Type 4E is a predominately pyritic specie which comprises the bulk of the ore in the Zone III ore body. Since both samples suffered moderate to severe en route oxidation, some of the minerals were rendered non flotable - this was especially evident in sample No. 1. Shown below in Table 5 are the sulphide and non-sulphide contents of the two samples illustrating the degree of sample oxidation.

TABLE 5

PILOT PLANT BULK SAMPLES - HEAD ASSAYS

SAMPLE	ASSAYS %			
	Pb	Pb Ox.	Zn	Zn Ox.
Bulk Sample No.1	2.35	0.40	3.85	0.25
Bulk Sample No.2	2.40	0.35	4.34	0.22

Note: "Oxide components" encompass all non-sulphides including carbonates, hydroxides, etc.

Because of the oxidized condition of the samples, corrections were made to recorded data to compensate for the anamolous oxide contents. The corrections were based on experience, and established practice which required that a certain fraction of the reported oxide be removed from the head assay in the recovery calculations. These corrections had the effect of increasing the reported pilot plant lead recovery substantially; zinc metallurgy was only marginally influenced.

TABLE 6

CALCULATED SULPHIDE HEAD ASSAYS

SAMPLE	ASSAYS % LEAD ASSAY HEAD	ASSAY OXIDE	CALCULATED EFFECTIVE HEAD	ZINC ASSAY HEAD	ASSAY OXIDE	CALCULATED EFFECTIVE HEAD
Bulk Sample No. 1	2.35	0.40	2.05	3.85	0.25	3.72
Bulk Sample No.2	2.40	0.35	2.14	4.34	0.22	4.23

In keeping with conservative methods of predicting data used elsewhere in this study, it was assumed that, of the oxidized lead and zinc, only one quarter would be recoverable; (Table 6) Thus the calculated effective lead assays were derived. The results of the pilot plant data modified in this manner are shown below in table 7 and 8.

TABLE 7

LAKEFIELD PILOT PLANT TEST RESULTS

PILOT PLANT SAMPLE NO.1

50 MICRON GRIND

TESTS 9, 10 and 24

	WEIGHT %	ASSAYS Pb	Zn	DISTRIBUTION Pb	Zn
Feed	100.0	2.03	3.66	100.0	100.0
Lead Conc.	2.58	68.4	2.6	86.9	1.8
Zinc Conc.	6.35	0.38	52.1	1.2	90.4
Tails	91.71	0.27	0.31	11.9	7.8

Notes: a) See notes under table no. 8.

TABLE 8

PILOT PLANT SAMPLE NO.2

50 MICRON GRIND

TESTS 32, 33, 34 and 35

	WEIGHT %	ASSAYS Pb	Zn	DISTRIBUTION Pb	Zn
Feed	100.0	2.12	4.26	100.0	100.0
Lead Conc.	2.62	73.6	2.6	91.0	1.6
Zinc Conc.	6.99	0.48	55.2	1.6	90.6
Tails	90.39	0.17	0.37	7.4	7.8

Notes: a) Head assays were determined from the pilot plant grinding circuit overflow with a correction made for non-recoverable oxide species.

b) Weight percentages taken from assay calculations averaged for the pilot plant runs noted. Checked also by three product formula method.

c) Concentrate and minor element determined by assaying pilot plant composites.

In studying the overall results of the two test series two factors were considered significant:

i) The sample feed grades were quite low compared to predicted mill feed for Zone III: It would be quite reasonable to expect better results with the higher feed grade.

TABLE 9

COMPARISON OF HEAD ASSAYS

SAMPLE	HEAD ASSAYS Pb	Zn
Bulk Sample No. 1	2.35	3.85
Bulk Sample No. 2	2.40	3.85
Zone III Average Estimated	2.90	4.50

Note: a) Zone III data is from latest mine model and does not include phase 6 ore.

ii) The results from sample No.2 were considered to be quite exceptional. There were several indications to support this view, especially the high zinc concentrate grades which suggested a somewhat atypical zinc mineral structure, the rapid rate of flotation of the minerals, and the results from the Faro plant when treating similar material. Also the sample exhibited most unusual precious metal distributions.

The pilot plant data was analyzed and a conservative metallurgical balance constructed for actual plant operation. For comparison are shown some locked cycle test results generated on samples of Zone III drill core. (Table 10)

TABLE 10

PLANT METALLURGICAL BALANCE

DATA SOURCE	CONCENTRATE	ASSAYS Pb	ASSAYS Zn	DISTRIBUTION Pb	DISTRIBUTION Zn
Pilot Plant Work Sample No.1	Lead	68.4		86.9	
	Zinc		52.1		90.4
Pilot Plant Work Sample No.2	Lead	73.6		91.0	
	Zinc		55.2		90.6
Locked Cycle Tests	Lead	77.2		88.8	
	Zinc		53.8		83.5
Predicted Plant Results Faro Zone III	Lead	67.0		87.5	
	Zinc		53.5		88.5

Note: a) All data refers to metallurgy at a P_{80} of 50 microns.

b) Locked cycle tests performed at Cyprus Anvil during 1980 on samples of Zone III drill core. Data reported in January 1981 in a report by A. McIntyre.

2.25 Gold, Silver and Minor Elements

Based on assays of diamond drill composites, the Zone III ore was estimated to contain appreciable quantities of silver and minor amounts of gold. Averaged data indicated that the mean silver content would be 35 g/tonne Ag while gold would be about 0.065 g/tonne Au. The estimate of the predicted gold and silver metallurgy was conducted as follows:

i) Silver Metallurgy

Two pilot plant runs using sample No. 1 were assayed for silver; the results are shown below.

TABLE 11

SILVER RECOVERY AND GRADE AT 50 microns GRIND

TEST NO.	PRODUCT	WEIGHT %	ASSAYS %, g/tonne Pb	Zn	Ag	% DISTRIBUTION Pb	Zn	Ag
PP-9	Pb Cl. Conc. Zn Cl. Conc. Zn Comb. Tail.	2.58 6.56 90.86	69.1 0.43 0.59	2.4 52.5 0.41	501.13 38.32 6.62	75.8 1.2 23.0	1.6 88.8 9.6	60.3 11.7 28.0
	Cyclone O'Flow	100.00	2.35	3.88	21.46	100.00	100.00	100.00
PP-10	Pb Cl. Conc. Zn Cl. Conc. Zn Comb. Tail.	2.61 6.24 91.15	68.2 0.34 0.57	2.82 52.0 0.52	486.36 34.86 7.62	76.7 0.9 22.4	1.9 85.6 12.5	58.2 10.0 31.8
	Kason U'Size	100.00	2.32	3.79	21.82	100.00	100.00	100.00

Assuming that the silver recovery increases linearly with lead recovery as with most Faro area ore types, then increasing the lead recovery to 87.5% should result in some increase in silver recovery from the reported 58-60% level. Based on experience and the long term silver recovery data at Anvil, a silver recovery figure of 65% was selected.

Unfortunately the bulk samples were quite low in silver content - about 21.5 g/tonne Ag: correction for this was achieved by calculating the quantity of silver recovered into the lead concentrate at fixed recovery (e.g. 600 g/tonne silver in the lead concentrate) from a predicted mill feed of 34 g/tonne Ag.

ii) Gold Metallurgy

Some support for both the gold and silver predicted recovery levels were obtained by reassaying a locked cycle test on samples from Faro Zone III obtained in the 1978 drilling program. The data in Table 12 shows that a gold recovery of 35-40% may be anticipated in the lead concentrate. Using the predicted mine model feed grade and a gold recovery of 33% the gold concentration in the lead concentrate was calculated (Table 14).

TABLE 12

GOLD AND SILVER METALLURGY - LABORATORY TEST DATA

	ASSAYS (g/tonne)		DISTRIBUTION	
	Au	Ag	Au	Ag
Feed	0.17	44	100.0	100.0
Lead Concentrate	1.20	530	38.8	66.3
Zinc Concentrate	0.70	39	37.0	8.0

Notes: a) Composite obtained from a very limited drill program.

b) Head assays higher than mine model predicts.

c) Data from cycle test No.8 Ref.LR2082 using Faro Zone III ore.

iii) Mercury and Arsenic

Lacking specific assay data on Zone III lead and zinc concentrate contaminants, minor element assays for the year 1979 were averaged and used to predict mercury and arsenic concentrations.The averaged data for 1979 concentrations of minor elements is shown below in Table 13.

TABLE 13

MINOR ELEMENTS - FARO ZONE III

CONCENTRATE	ASSAYS g/tonne	
	Hg	As
Lead Concentrate	40	300
Zinc Concentrate	300	100

2.26 Predicted Plant Metallurgy

TABLE 14

PREDICTED PLANT PERFORMANCE - ZONE III ORE

	WEIGHT	ASSAY				DISTRIBUTION			
		Pb	Zn	Ag	Au	Pb	Zn	Ag	Au
Feed	100.0	2.9	4.6	35	*	100.0	100.0	100.0	100.0
Lead Conc.	3.79	67.0	3.0	600	0.75	87.5	2.5	65.0	33.0
Zinc Conc.	7.61	0.5	53.5	40	*	1.3	88.5	8.7	*
Tails	88.60	0.37	0.47	10	*	11.2	9.0	26.3	*

Note: a) The average gold content of the lead concentrates from Zone III will be below payable limits. However, due to the irregular occurrance of gold in the ore it is possible that payable gold will be encountered in some shipments.

TO: T. CLOUTIER

FROM: W. SCHEDING

DATE: JANUARY 29, 1987

SUBJECT: HEADGRADE/RECOVERY RELATIONSHIPS FOR LEAD AND ZINC

Confirmation of the original estimates of these relationships has been requested for the Faro Zone 3 feasibility study. These trends were derived from experience, since no valid long-term historical database was available in September, 1986. Since then, four months of daily data has become available, which is less disturbed by start-up irregularities. As a result, statistical confirmation of these trends is now possible.

In addition, 1981 CAMC monthly data has been used to confirm the trends on a historical basis - earlier data is invalidated by the coarser grind, lower tonnage and simpler ore types prior to the last expansion or by the upset period after the expansion itself. 1982 data is unsuitable by virtue of the large proportion of oxide stockpile being treated.

It should be noted that, while knowledge of these trends could affect the economics of certain sections of the orebody, they will have very little effect on the overall picture, since lower or higher than expected recoveries at low grades will be matched by higher or lower than expected performance at high grades - especially for a nearly linear relationship. What is critical is the mean performance to be expected, i.e. the base recovery at the average head grade for a given ore type. This is something that can only be derived by experience with a relatively predictable orebody - and we are short in experience on this orebody and the orebody is short on predictability. If asked to predict lead recoveries from 2E ore on January 13, 1987, 63% would have been a reasonable estimate. Two days later, that estimate would realistically be 78%, as the correct treatment route is discovered. Constant review of recoveries to be expected from various ore types is required.

Despite the above, four different methods for obtaining the headgrade/recovery relationships are discussed below. They are all based on varying recovery to a constant concentrate grade, which is more realistic given marketing constraints.

<u>(i) 25% science / 25% common sense / 50% experience</u>

Using experience gained at other operations, and knowing that flotation is a first order rate process, recovery is expected to increase with headgrade when producing a concentrate of constant grade. As the headgrade increases, the concentration ratio decreases, rapidly at first, then more slowly. The ease of removal of valuable mineral (i.e. recovery) will be inversely proportional to the concentration ratio. This leads to a trend which is well-described by multiplication of a base recovery by the log of the headgrade:

Recovery = Base recovery * (a * ln (headgrade) + k)
(at mean headgrade)

The second term of the R.H.S. of the equation must equal 1.0 at the mean headgrade. By taking logs of the headgrade and matching them to tailings grades expected by experience, factors `a' and `k' can then be selected to give the desired curvature and are fine-tuned as historical data is collected.

<u>(ii) 50% `constant tail' / 50% `constant recovery'</u>

Operating plants invariably fall within these two extremes. The headgrade/recovery relationship for a constant tail is defined by the two-product formula. Once `c' (concentrate grade) and `t' (tails grade) are fixed, curves can be drawn for lead and zinc. The `constant recovery' extreme is a straight line on the headgrade/recovery graph. If the mean of the two expressions is chosen as the most likely curve for future Zone 3 ore, curves can be drawn as shown in the graphs. The results are in reasonable agreement with the curves from method (i), if compared at the same base level recovery.

<u>(iii) 1981 CAMC Monthly Data</u>

Linear regression of these few data points give the following relationships:

Lead: Recovery = 56.29 + (6.83 * headgrade (%))

Zinc: Recovery = 50.69 + (5.02 * headgrade (%))

Correlation coefficients were 0.82 and 0.66 for lead and zinc respectively, giving confidence levels in the greater than 95% range. These lines are also plotted on the graphs. As can be expected from a linear approximation of a curve, the gradients are less than those of the curves at high headgrades. This leads, of course, to unrealistic numbers at very high grades e.g. 100% lead recovery at a 6.4% lead head and 100% zinc recovery at a 9.8% zinc head.

This makes this kind of relationship undesirable for calculating recovery to be expected from smaller zones of high or low grade ore far from the mean headgrades for the orebody. Simple regression also cannot correct for the effect of variations in concentrate grade on the recoveries used in the database. These are subject to the grade/recovery relationship for the ore (stronger than the headgrade/recovery relationship). This factor, and others which are quantifiable, can be purged from the database by multi-linear variable regression analysis - which comes later.

<u>(iv) 1986/1987 Curragh Resources Daily Data (Oct - Jan)</u>

This data was treated as with the 1981 data, except that there were many more data points (over 100 in all). Oxide ore was removed from the analysis because of its atypical response and because it should not feature in future Zone 3 ore supply. Because of operational problems and frequent ore changes, the relationships obtained did not have very high correlation coefficients. In terms of degree of confidence in the existence of a trend, however, the large number of data points ensures that the 't' test is positive at a higher confidence level (greater than 99%).

Sorting of the data between ore types was not necessary for the zinc trend, but the raw lead data gave no trend whatsoever when treated in bulk (correlation coefficient less than 0.05). After the data was batched into successive ore type campaigns of 6 - 15 days' duration, individual trends were visible for most ore/time periods, but were totally destroyed when combined with periods with very small, negative correlation coefficients. These periods were removed from the analysis and the individual trends composited to obtain overall trends as follows:

Lead: Recovery = 56.29 + (6.63 * headgrade (%))

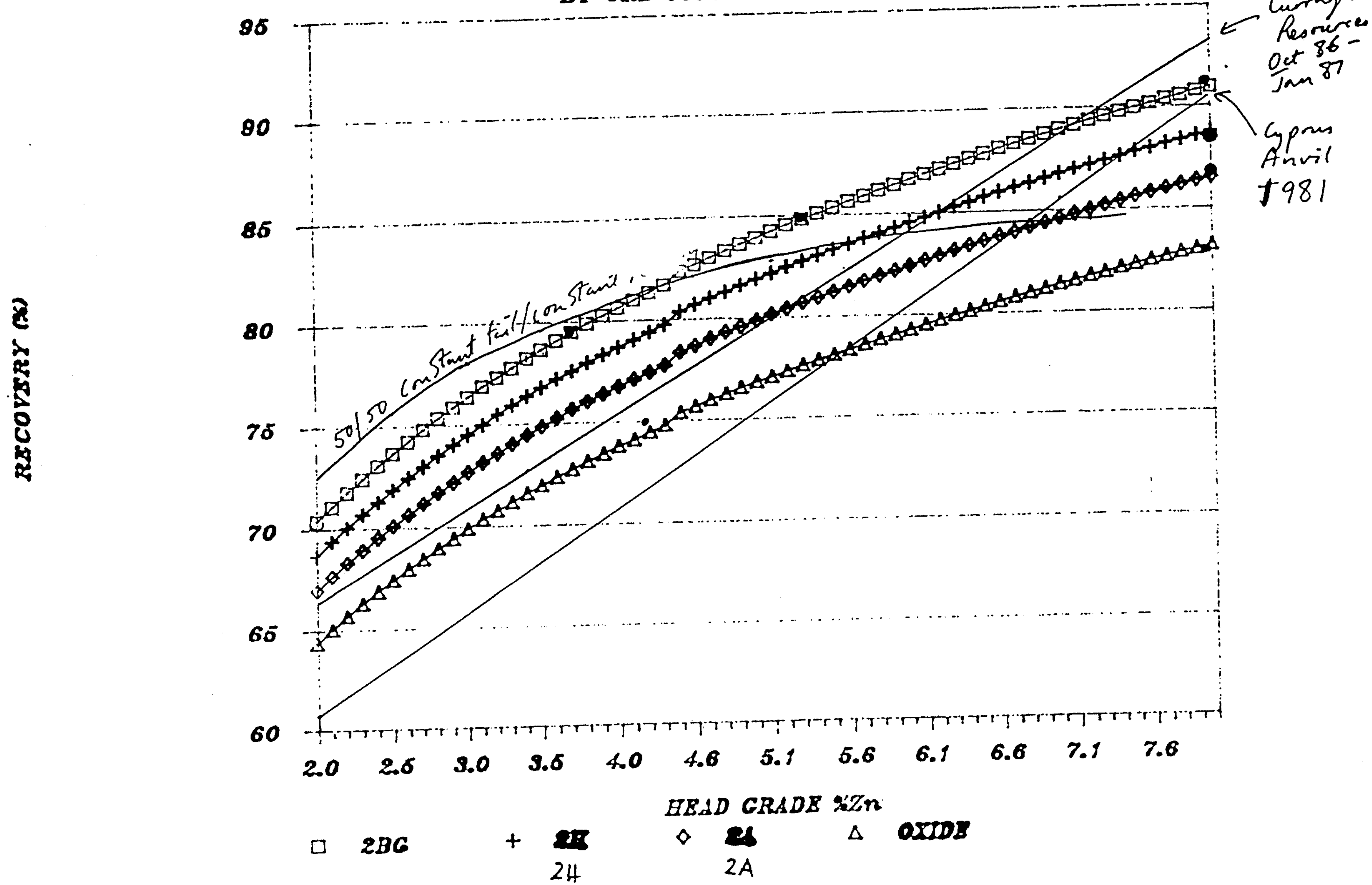
Zinc: Recovery = 37.80 + (4.35 * headgrade (%))

These trends are also plotted on the graphs. Their gradients agree well with CAMC trends and are good approximations to the curves developed in sections (i) and (ii), but only in the low to medium grade ranges. This is understandable in that they were developed with data which fell only in those ranges - lead grades below 3.8% and zinc grades below 5.5%.

In conclusion, the logarithmic trends originally developed appear to fit the statistical data quite well, and offer greater accuracy at high and low grade extremes. They are also in fair agreement with the '50% constant tail / 50% constant recovery' trend which works well at two other base metal concentrators that I am familiar with. However, their accuracy is of only minor importance compared with the estimates for the base recovery for each ore type at the mean headgrade. I recommend that some sensitivity analysis be performed to assess the impact of +3% changes in base recovery for each ore type. If the overall outcome of the feasibility study is significantly affected, more work will have to be done to estimate future performance more accurately e.g. comparison of laboratory versus plant data and generating average laboratory response curves by ore type which can then be corrected for closed circuit operation. This should be supported by a testwork program on the diamond drill core currently being assayed and typed for ore reserve estimation.

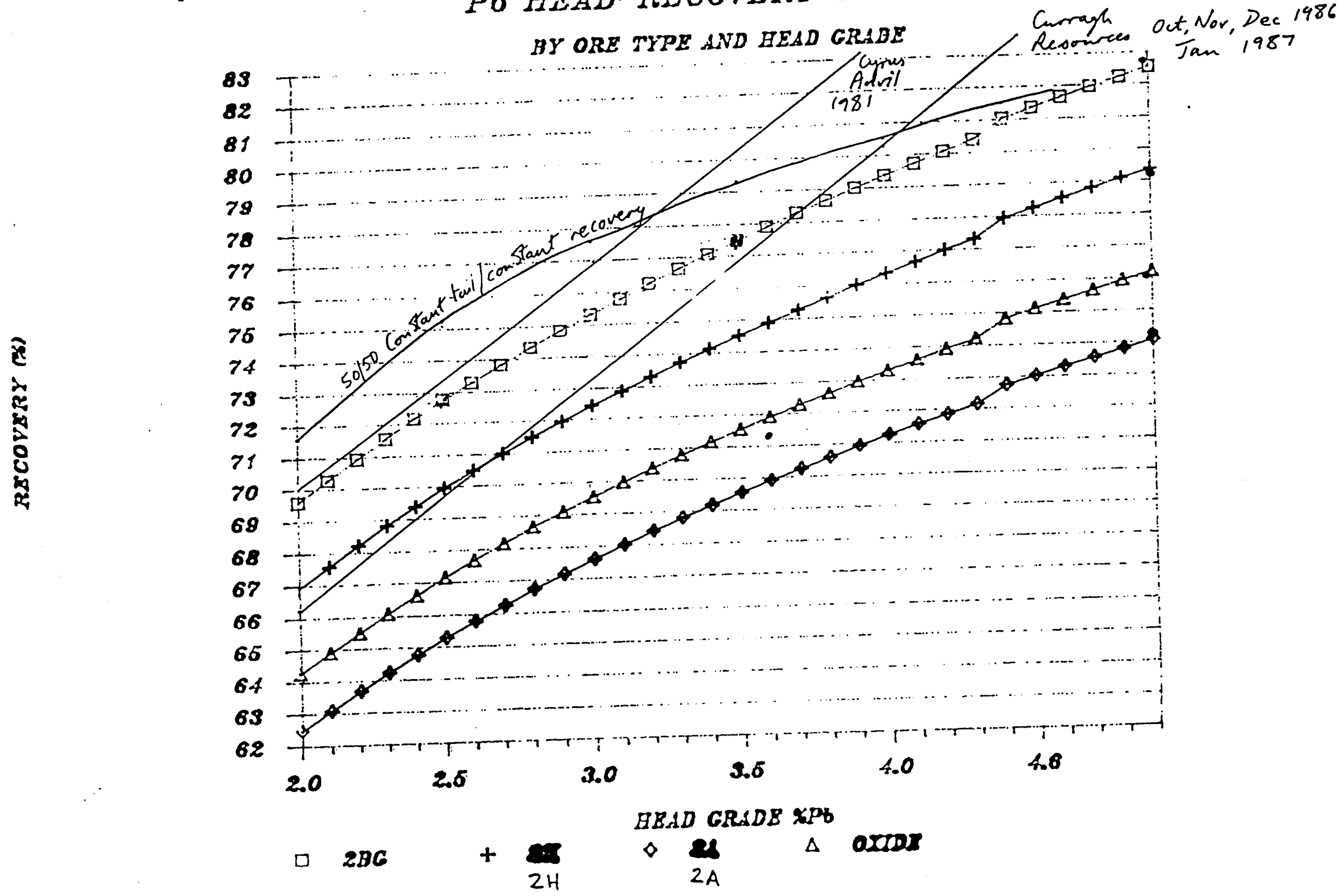
Zn HEAD–RECOVERY CURVES

BY ORE TYPE AND HEAD GRADE



Pb HEAD-RECOVERY CURVES

BY ORE TYPE AND HEAD GRADE



2.3 GRUM METALLURGY

2.3 GRUM METALLURGY

2.31 Preliminary Laboratory Testwork

Metallurgical investigations of the Grum ore body by Kerr Addison Ltd. and Noranda Mines were preceded by extensive mineralogical studies in which the entire array of ore types, believed to number twelve, were individually characterized. Having identified the various ore types, metallurgical testwork commenced at Lakefield on individual samples of each ore type, and also on composites of ore types. This compositing of ore types was a relatively late step when it was realized that many of the so called ore types were in fact metallurgically indistinguishable. The concept of compositing samples for testing was very acceptable since the costs of individually testing each ore type were prodigious.

The testwork, which was performed exclusively at Lakefield Research Ltd., Lakefield, Ontario, commenced in 1976 and culminated in late 1977 with long series of pilot plant tests. The work was directed principally by Mr. K. Konigsmann, Chief Metallurgist, Noranda Mines, and by others in the Noranda Milling Committee. A very brief synopsis of the excellent work performed at Lakefield is given below.

2.32 Critical Results from Laboratory Testwork

The earliest work performed at Lakefield demonstrated the need for a fine primary grind on Grum ores in the range 50 microns P_{80}. This observation was strongly supported by the results of the meticulous microscopic studies of Dr. Carson of Noranda Mines, which indicated an unusually fine intergrowth of mineral crystals in most ore types.

Many laboratory test series were performed to illustrate the effect of variations of primary grind on metallurgy. Typical of the tests are the data shown below in tables 15 and 16; they refer to testwork on a composite made up of seven of the twelve ore types.

TABLE 15

EFFECT OF FINENESS OF PRIMARY GRIND

TEST NO.	TIME Min.	%-200 Mesh	PRODUCTS	WEIGHT %	ASSAY% Pb	Zn	% DISTRIBUTION Pb	Zn
153	30	87.1	Pb Cleaner Concentrate Pb Rougher Concentrate	3.64 17.93	65.9 15.3	3.69 7.54	78.2 89.8	2.2 22.2
179	25	79.2	Pb Cleaner Concentrate Pb Rougher Concentrate	3.30 19.21	63.3 14.0	4.04 7.65	70.0 89.9	2.2 24.1
180	20	68.1	Pb Cleaner Concentrate Pb Rougher Concentrate	3.01 18.81	65.9 14.0	3.89 7.66	67.1 88.8	2.0 24.2

Notes: a) Data Source L.R. 1991 Vol. 7.

b) Comparative data for the effect of grind on the zinc circuit not reported for the tests.

Having established that adequate mineral liberation could be obtained at a relatively fine primary grind, attention was then focused on improving the lead concentrate grade. In almost all samples the investigators found that an extremely fine regrind of the lead rougher concentrate was mandatory in order to achieve reasonable concentrate grades at acceptable recoveries. Typical of the results obtained are those shown below in Table 16. Data in this table refers to work on the pilot plant composite which encompassed components from most ore types. The effect of regrinding on lead final concentrate grade, at almost constant recovery, is obvious.

TABLE 16

EFFECT OF LEAD CONCENTRATE REGRIND ON LEAD CLEANING

TEST NO.	PRIMARY GRIND (MIN.)	Pb REGRIND TIME (MIN)	% PASS 10 u	PRODUCT	WEIGHT %	ASSAY % Pb	Zn	% DISTRIBUTION Pb	Zn
42	30	10	31.5	Pb Cleaner Concentrate	12.12	38.6	12.9	81.0	15.6
				Pb 1st Cleaner Conc.	18.63	27.3	13.7	88.0	25.5
				Pb Combined Tailing	81.37	0.85	9.19	12.0	74.5
43	30	20	37.2	Pb Cleaner Concentrate	10.54	45.1	12.9	80.4	13.7
				Pb 1st Cleaner Conc.	18.24	28.8	14.1	89.0	26.0
				Pb Combined Tailing	81.76	0.80	8.97	11.0	74.0
45	30	30	47.4	Pb Cleaner Concentrate	8.75	51.6	11.3	76.1	9.9
				Pb 1st Cleaner Conc.	16.08	32.4	14.0	87.7	22.6
				Pb Combined Tailing	83.92	0.87	9.14	12.3	77.4
46	30	40	58.0	Pb Cleaner Concentrate	7.97	54.6	10.7	73.4	8.1
				Pb 1st Cleaner Conc.	16.54	31.3	14.1	87.3	23.0
				Pb Combined Tailing	83.46	0.89	9.2	12.7	76.0
47	30	50	62.0	Pb Cleaner Concentrate	7.96	59.7	9.53	80.0	7.6
				Pb 1st Cleaner Conc.	14.56	35.8	13.6	87.7	19.9
				Pb Combined Tailing	85.44	0.86	9.32	12.3	80.1

Notes: a) Data source L.R. 1991 Vol. 10

b) Sample: Pilot plant composite.

The route to acceptable metallurgy with Grum ore types then was established; a fine primary grind (P_{80} 50-60 microns) followed by a very fine regrind of the lead rougher concentrate. (P_{80} 15-20 microns)

In the course of these detailed studies it was found that the lead regrind mill exerted a significant influence on the zinc metallurgy; probably due to the considerable quantity of zinc minerals reporting in the lead rougher concentrate. In general however, the zinc metallurgy did not pose a major problem. The Grum zinc mineral was observed to be quite low in interstitial iron and manganese and, apart from losses in the lead cleaner circuit products, zinc recovery was good. The laboratory results suggested that zinc concentrate grades in excess of 55% Zn could be fairly easily achieved provided that both the lead and zinc regrinding circuit were optimized.

TABLE 17

EFFECT OF ZINC REGRIND ON ZINC CLEANING

TEST No.	Zn CLEANER FEED %-20μ	%-10μ	PRODUCT	ASSAY Pb	Zn	DISTRIBUTION Pb	Zn
41,47	78	50	Zinc Cleaner Conc.	2.3	56.5	5.7	80.6
			Zinc Flotation Tail	1.2	1.6	16.8	12.6
42,43,44	39	22	Zinc Cleaner Conc.	2.5	51.0	6.3	73.5
			Zinc Flotation Tail	1.3	2.3	17.3	17.6

Note: a) Data source LR2027.

b) Data refers to laboratory batch cleaner tests on a composite of the major ore types.

2.33 Pilot Plant Test Program

As the laboratory testwork neared completion at Lakefield in mid 1977, underground operations at the Grum site were directed toward obtaining a bulk sample for pilot plant testwork. The development adit had by this time penetrated the main sulphide zone and the bulk sample was taken principally from this area. Unfortunately sample make-up was somewhat biased because of the limited availability of representative quantities of each specific ore type. The sample bias was reflected in the anomalously high metal contents of the pilot plant composite.

mlwl

TABLE 18

HEAD ASSAY - PILOT PLANT COMPOSITE

	ASSAYS %					ASSAYS g/tonne		
	Cu	Pb	Zn	Fe	As	Au	Ag	Hg
Sample	0.13	6.1	10.0	20.5	0.23	1.4	98	80

Despite the problems with the sample bias there was no alternative but to commence testwork and to attempt to compensate, or correct the test results for the high grade sample when predicting probable plant metallurgy.

To assist in the prediction of plant metallurgy and as a guide in the extrapolation of results several locked cycle tests were carried out on various composites of Grum ores by Lakefield Research.

TABLE 19

LOCKED CYCLE TEST DATA - GRUM COMPOSITES

TEST NO.	COMPOSITE NO.	PRODUCT	WEIGHT %	ASSAYS %		% DISTRIBUTION	
				Pb	Zn	Pb	Zn
207	1	Pb Cleaner Conc. Zn Cleaner Conc. Zn Flot.Tailing	4.77 9.18 86.05	66.7 0.58 0.95	4.56 57.40 1.43	78.4 1.3 20.3	3.2 78.5 18.3
		Head (Calc.)	100.0	4.06	6.71	100.0	100.0
217	1	Pb Cleaner Conc. Zn Cleaner Conc. Zn Flot. Tailing	6.12 8.91 84.07	62.2 0.40 0.40	5.67 59.70 1.49	91.0 0.9 8.1	5.0 76.7 15.3
		Head (Calc.)	100.0	4.18	6.93	100.0	100.0
230	3	Pb Cleaner Conc. Zn Cleaner Conc. Zn Flot. Tailing	5.82 13.75 80.43	62.5 0.39 1.02	6.16 57.80 1.25	80.6 1.2 18.2	3.9 85.4 10.7
		Head (Calc.)	100.0	4.51	9.31	100.0	100.0
237	2	Pb Cleaner Conc. Zn Cleaner Conc. Zn Flot. Tailing	9.12 14.31 76.57	59.7 0.56 0.41	7.47 55.7 0.78	93.3 1.4 5.3	7.4 86.2 6.4
		Head (Calc.)	100.0	5.84	9.25	100.0	100.0
241	2	Pb Cleaner Conc. Zn Cleaner Conc. Zn Flot. Tailing	9.06 14.49 75.87	59.1 0.46 0.44	7.24 56.6 0.57	91.4 1.2 5.7	6.9 86.0 4.5
		Head (Calc.)	100.0	5.86	9.53	100.0	100.0

Notes: a) The composites 1, 2, and 3 are described in compositional detail in L.R. 1991 Vol. 8.

b) Tests carried out at 50-60 micron grind.

After about thirty preliminary tests, the regrinding circuits were optimized, the reagent scheme balanced and the optimum primary grind established at between 50-55 microns P_{80}. In the course of the work, and subsequent to the preliminary tests, several interesting observations were made.

a) Initially it appeared that lead regrind was not quite as critical as the laboratory work had indicated. Presumably this was because much of the laboratory work was carried out with samples averaging 9-10% combined metal; the higher grade pilot plant samples would naturally require less regrinding.

b) The collector consumption in the pilot plant was considerably lower than had been observed in the laboratory. Again this could be related to the relatively coarse mineralization associated with the high grade sample.

c) The zinc circuit first cleaner pH was found to be critical. Even very small variations ($\pm$ 0.5 units at pH 11.5) exercised enormous effects on the zinc concentrate grade.

d) High cyanide additions, in total about 250 g/tonne were found to be critical to attainment of good metallurgy.

The pilot plant program was directed by K. Konigsmann of Noranda Mines who was aided by D.M. Wyslouzil of Lakefield Research. A total of 53 tests were performed using a flotation reagent scheme very similar to that employed at Anvil (and incidentally almost identical to that used in the Anvil Pilot Plant program). The results are shown below in Table 20.

TABLE 20

GRUM PILOT PLANT TEST DATA

TEST NO.	METALLURGY LEAD GRADE	LEAD RECOVERY	ZINC GRADE	ZINC RECOVERY
1	49.6	75.9	58.9	48.9
2	53.1	74.3	56.0	73.5
3	44.7	80.7	56.0	70.9
4	44.1	80.6	50.2	66.2
5	47.6	75.6	55.8	66.1
6	59.0	81.8	54.3	78.5
7	46.6	83.1	52.6	72.3
8	49.9	79.7	49.6	78.0
9	62.4	77.8	55.1	73.8
10	44.9	79.2	36.9	74.0
11	42.2	81.6	42.6	72.7
12	47.9	74.5	38.2	76.4
13	46.4	77.8	46.0	77.6
14	52.8	77.3	47.0	78.6
15	57.2	74.8	42.3	78.7
16	56.2	76.2	49.3	75.8
17	59.2	76.2	43.3	81.4
18	57.5	74.8	50.0	79.9
19	57.9	74.2	51.5	76.7
20	59.7	70.8	48.3	74.9
21	53.6	78.1	53.9	76.1
22	62.3	71.7	54.2	78.1
23	52.3	71.0	56.3	69.1
24	58.4	63.0	52.0	81.1
25	59.3	77.4	52.1	81.1
26	57.0	76.0	56.4	76.0
27	58.4	77.7	48.3	79.8
28	51.9	78.0	46.8	75.6
29	56.4	77.6	54.5	74.2
30	54.9	75.4	50.0	78.0
31	53.7	76.3	55.0	75.4
32	65.4	78.1	54.5	79.8
33	59.2	80.4	51.8	79.0
34	64.8	77.6	53.5	77.4
35	66.6	75.7	57.4	72.0
36	54.1	77.5	52.6	78.0
37	63.1	78.3	50.2	82.3
38	60.6	77.0	53.3	79.9
39	56.9	75.3	50.0	76.6
40	57.2	75.2	56.0	72.9
41	62.7	77.2	57.4	79.6
42	60.9	75.6	48.4	79.9
43	60.7	75.9	52.1	75.5
44	54.1	77.4	52.4	65.1
45	46.8	76.5	53.8	71.7
46	55.2	76.8	50.8	73.6
47	64.6	77.9	55.5	81.5
49	62.7	76.0	56.5	78.3
50	60.8	78.1	53.6	80.6
51	65.8	74.4	52.6	82.6
52	58.3	77.5	54.0	78.0
53	60.9	75.5	54.7	78.9

Notes: a) Data source progress report No.11 Vol. I-IV L.R. 2027.

b) All results from tests using a standard flowsheet and one composite ore sample.

2.34 Analysis of Pilot Plant Results

The data from the tests in the latter part of the program was examined in minute detail by the Noranda Milling Committee. Noting that the sample tested comprised principally of massive sulphide ore types, the committee elected to produce two metallurgical balances: One representing the massive sulphide ore metallurgy while the other indicated their estimate of the overall average Grum metallurgy. This latter estimate was based on the results of laboratory cleaner and the locked cycle tests on various samples.

The committee were of the opinion that the true average metallurgy of all Grum ores would be somewhat better than for the sulphide zone ores. Accordingly they modified their best pilot plant using the results of the locked cycle tests to reflect this belief and increased both lead and zinc recoveries with the same concentrate grades. The two balances are shown below in Table 21.

TABLE 21

PREDICTED METALLURGY FROM PILOT PLANT RESULTS

ORE TYPE	CONCENTRATE	ASSAYS				DISTRIBUTION			
		Pb	Zn	Au	Ag	Pb	Zn	Au	Ag
Massive Sulphide	Lead	62	10	4.8	925	77	*	33	72
	Zinc	2.5	56	*	*	*	81	*	*
Average	Lead	62	8	5.1	950	80	*	33	72
	Zinc	2.0	56	*	*	*	84	*	*

Notes: a) Data source - Noranda Milling Committee Report Dec. 1977.

b) Au and Ag in g/tonne.

2.35 Gold, Silver and Minor Elements

Some assays on three pilot plant runs yielded some interesting results on precious metal concentrations and distributions. Concentrates collected during various pilot plant runs were assayed for mercury and arsenic. The results are discussed below in section 2.36.

TABLE 22

GOLD & SILVER METALLURGY - GRUM PILOT PLANT

TEST NO.	PRODUCT	WEIGHT %	ASSAYS (g/tonne) Au	Ag	% DISTRIBUTION Au	Ag
PP25	Pb Cleaner Concentrate	8.02	3.40	857	48.8	72.5
	Zn Cleaner Concentrate	16.04	0.15	73	4.9	12.4
	Zn Combined Tailing	75.94	0.30	19	46.3	15.1
	FLOTATION FEED (calc)	100.00	0.68	95	100.0	100.0
PP35	Pb Cleaner Concentrate	6.77	4.46	950	50.6	74.8
	Zn Cleaner Concentrate	12.30	0.15	69	3.4	9.8
	Zn Combined Tailing	80.93	0.30	16	46.0	15.4
	FLOTATION FEED (calc.)	100.00	0.68	86	100.0	100.0
PP37	Pb Cleaner Concentrate	7.32	3.77	950	46.6	77.2
	Zn Cleaner Concentrate	16.72	0.30	74	9.8	13.8
	Zn Combined Tailing	75.96	0.30	11	43.6	9.0
	FLOTATION FEED (calc.)	100.00	0.68	90	100.0	100.0

Notes: a) Data source - Progress Report No.11 Vol.I L.R.2027. Gold & Silver assays reported only on these tests.

b) Each data set based on an 8 hour duration pilot plant run.

2.36 Predicted Plant Metallurgy

Because of the significant effect which mill feed grade exercises on metallurgy and since the pilot plant sample was known to be biased, there are reasons to believe that the Grum average metallurgy, predicted by the Noranda Milling Committee, may be optimistic. The principal reasons are as follows:

a) Lead grade was difficult to achieve in the pilot plant, and lead recovery was quite low in those tests in which concentrate grades in excess of 60% Pb were obtained.

b) Silver recovery at 72% probably reflects the anomalously high silver content in the pilot plant feed. Silver recovery into the lead concentrate at Cyprus Anvil averages somewhat less than 60% and is expected to reach 65% only at very high lead recoveries.

Based on actual experience of the operation of a metallurgical plant in the Faro area with the constraints of climate, limited manpower availability and variable ore types, a conservative approach would indicate that a somewhat more pessemistic balance be adopted. Accordingly the lead concentrate grade was reduced from 62% to 60% Pb, but the lead recovery was kept constant. Silver recovery was reduced to 65% to reflect lower silver grade expected in the diluted ore from the Grum deposit.

The gold in the mill feed was assumed to be proportional to the lead content; hence the diluted feed grade required that the pilot plant gold head assay be reduced by about 60%. A gold recovery figure of 33% was used in the calculations - identical to that assumed for Zone III ore.

TABLE 23

PREDICTED METALLURGY - GRUM ORE

CONCENTRATE	ASSAYS Pb%	Zn%	Au	Ag	As	Hg	DISTRIBUTION Pb%	Zn%	Au%	Ag%
Average Mill Feed-Lead Conc.	60	11	3.5	750	100	90	80	*	33	65
-Zinc Conc.	2.5	55	*	*	50	650		83		

Notes: a) Au, Ag, As, and Hg assays in terms of g/tonne.

b) Silver recovery recalculated on the basis of approximately 50 g/tonne Ag in the mill feed.

c) Average mill feed including dilution, will be about 9% combined lead and zinc metal.

2.4 VANGORDA METALLURGY

2.4 VANGORDA METALLURGY

2.41 Preliminary Laboratory Work

Since its discovery in the mid fifties the Vangorda ore body remained, until quite recently a metallurgical enigma. Early metallurgical work produced erratic results and even the most optimistic metallurgist could predict only the production of a bulk lead-zinc concentrate. Probably the reason for the very poor initial metallurgical results was that the early diamond drill core was of small diameter and hence core recovery was poor, incidentally negating the chance of an accurate ore body model, and ensuring the rapid oxidation of the minerals, thus preventing the generation of reproducible laboratory test data.

Redrilling in the late sixties, and recognition of at least three ore types by the various investigators led to the generation of a few groups of reasonably encouraging metallurgical results. After about five years of sporadic testwork in many laboratories, there emerged a picture of an ore which depicted an extremely fine mineral crystal intergrowth, very markedly activated iron and zinc minerals, and a prediliction for rapid and deleterious mineral oxidation.

The Table 24 below is a synopsis of the early published work on Vangorda ore - unfortunately very few of the investigators noted the ore type being tested. The testwork covers a wide range of metallurgical conditions varying from lime to soda ash modulated circuits, various collectors and depressants and several cleaning schemes. All work exhibited one common factor however - Vangorda ores required an extremely fine primary grind of the order 30-50 μ P_{80} to permit any sort of separation to be achieved.

TABLE 24

PRELIMINARY LABORATORY TEST DATA - VANGORDA ORE

REF.	FEED % Pb	FEED % Zn	LEAD CONC. GRADE	LEAD CONC. REC.	ZINC CONC. GRADE	ZINC CONC. REC.	TAILINGS % Pb	TAILINGS % Zn	GRIND MESH #	SOURCE
1	4.0	4.9	49.9	77.0	49.4	60.4	0.15	0.66	99%-200	Dowa Mining Company Report. March 25, 1969.
2	4.6	4.7	56.6	83.8	54.4	79.7	0.63	0.54	64%-325	Galligher Company Report. July 17,1969. Series 2 tests.
3	4.1	5.2	51.7	89.7	55.4	78.1	0.38	0.65	82%-325	Brunswick Mining & Smelting Report. 1969
4	3.2	5.2	46.1	77.6	53.0	49.5	-	-	-	Brunswick Mining & Smelting attachment "Refractory Ores"
5	4.0	4.9	49.8	77.0	49.4	60.4	-	-	-	Brunswick Mining & Smelting attachment "Refractory Ores"
6	3.9	5.1	56.5	72.5	50.8	51.5	-	-	-	Brunswick Mining & Smelting attachment "Refractory Ores"
7	4.4	5.0	22.7	85.2	24.2	60.0	0.52	0.44	82%-325	Noranda Report "Vangorda & Brunswick M & S Ore Samples" December 9-13, 1969
8	5.8	6.1	29.1	85.4	11.0	60.1	0.54	1.22	80%-325	Noranda Report "Preliminary Test work on Vangorda Cores" April 2, 1970.
9	3.4	3.7	22.3	75.6	12.5	64.2	0.58	0.70	80%-325	Noranda Report "Preliminary Test work on Vangorda Cores" April 2, 1970.
10	1.6	2.7	37.4	77.9	49.2	77.4	0.34	0.36	55%-325	Noranda Report "Preliminary Locked Tests" Feb. 26, 1975
11	3.6	4.1	53.6	77.9	51.8	77.3	0.64	0.63	63%-325	Noranda Report "Preliminary Locked Tests" Feb. 26, 1975
12	3.7	6.8	48.7	78.4	54.4	80.4	0.51	0.74	77%-325	Noranda Report "Preliminary Locked Tests" Feb. 26, 1975
13	6.2	10.5	25.0	93.2	48.6	64.6	0.34	0.72	72%-325	Noranda Report "Preliminary Locked Tests" Feb. 26, 1975
14	3.1	4.4	61.7	78.3	55.3	86.6	0.19	0.14	80%-325	Dowa Mining Company "Metallurgical Test of the Vangorda Ore" May 1975.

Notes: a) All data shown in this table was abstracted from laboratory reports. Various schemes were employed but the soda ash - cyanide reagent scheme was most favoured.

b) The importance of a fine primary grind and the need for regrinding was repeatedly stressed by the investigators.

2.42 Detailed Type Testing

In the Summer of 1979 the Vangorda deposit was redrilled by Cyprus Anvil with relatively large diameter NQ drill bits and the core subjected to a meticulous geological examination and logging. The core was divided into three main geological species, type 4G, 4E and 4A, and then subdivided again according to assay range of lead and zinc.

Each group of samples was then subjected to detailed metallurgical testwork at the Kamloops Research Laboratory: The tests being designed to determine the effect of primary grind and regrinding on metallurgy when the particular ore type was treated with the Anvil reagent scheme. During the period June to November a total of 55 open circuit cleaner tests were completed on the three major ore types under study.

The results, are shown in summary form below in Table 25 as generated and also in a corrected form based on established laboratory procedures for the evaluation of metallurgical test data. The corrections involve the redistribution of the cleaner tailings to one or other of the concentrates according to experience.

Of special note are the extreme finess of grind employed in most tests ranging from 20-50 microns P_{80} in the primary grind. That these very fine grinds can be achieved is due to the extreme friability of the Vangorda ores.

Derivation of the metallurgy by ore type was fairly simple and was achieved by averaging the best corrected results obtained in the laboratory testwork. Almost invariably the best results occurred at very fine grind levels coupled with fine regrinding. Since the tests covered a wide range of test conditions for each major ore type, considerable attention was addressed to selection of the results for inclusion in each data set.

At no time during this phase of the testwork, nor in subsequent work performed in 1980, was a point detected beyond which fine grinding exercised a deleterious effect on metallurgy. These latter studies were conducted in the range 15-40 microns primary grind, with samples of various ore types (Ref. KM032 October 1980). As a point of general interest more and more complex sulphide operations are employing extremely fine primary grind levels. Meggen and Brunswick grind to 30-40 while at Huelva and Aznalcolla in Spain the primary grind is performed at less than 30 μ.

TABLE 25

VANGORDA METALLURGY BY ORE TYPE

TEST NO.	TEST METALLURGY				CORRECTED METALLURGY				GRIND TIME (min.)	GRIND % 325 MESH	ORE TYPE
	LEAD		ZINC		LEAD		ZINC				
	GRADE	RECOVERY	GRADE	RECOVERY	GRADE	RECOVERY	GRADE	RECOVERY			
1	45.5	43.9	54.1	69.6	45.5	58.0	54.1	74.7	15	94	1A4G
2	36.4	88.4	55.7	66.0	36.4	90.3	55.7	68.6	15	97	1A4G
3	55.7	85.6	56.7	76.4	55.7	87.1	56.7	79.0	15	96	1A4G
4	54.6	87.2	54.9	76.2	54.6	89.2	54.9	78.7	15	98	1A4G
5	57.8	83.2	56.6	75.2	57.8	85.7	56.6	77.7	15	97	1A4G
6	40.3	84.2	55.1	63.3	40.3	86.7	55.1	67.4	10	86	1A4G
7	48.2	85.0	57.5	62.3	48.2	87.5	57.5	67.9	10	88	1A4G
8	60.8	77.5	54.2	61.8	60.8	81.3	54.2	69.3	10	89	1A4G
13	57.3	83.1	55.9	71.7	57.3	83.4	55.9	75.2	15	94	1A4G
14	61.4	79.7	55.9	77.9	61.4	81.4	55.9	80.6	15	94	1A4G
9	36.4	72.5	47.7	63.3	36.4	76.5	47.7	69.3	10	91	1B4G
10	45.9	76.2	51.4	72.3	45.9	79.2	51.4	75.3	15	94	1B4G
11	49.6	67.9	52.4	75.9	49.6	73.6	52.4	81.6	15	97	1B4G
12	41.8	73.6	54.1	71.5	41.8	77.2	54.1	74.5	15	95	1B4G
15	41.3	71.3	51.6	76.0	41.3	74.4	51.6	78.3	15	95	1B4G
16	50.0	77.4	55.0	78.9	50.0	80.2	55.0	81.6	25	99	1B4G
17	49.3	80.8	55.1	74.8	49.3	82.1	55.1	76.8	15	95	1B4G
18	53.6	69.6	56.8	46.5	53.6	72.6	56.8	61.5	15	93	1B4G
19	56.9	67.9	58.1	59.8	56.9	72.5	58.1	67.4	20	98	1B4G
34	43.3	78.2	53.0	57.6	43.2	81.2	53.0	64.5	20	97	2A4E
30	33.7	79.6	37.2	68.7	33.7	81.3	37.2	71.7	10	80	2B4E
31	39.9	77.7	41.2	67.5	39.9	80.7	41.2	72.5	15	91	2B4E
32	44.4	77.5	51.0	65.8	44.4	79.5	51.0	70.9	20	97	2B4E
33	29.7	74.6	40.6	59.9	29.7	77.6	40.6	65.0	5	57	2B4E
35	50.7	78.8	49.8	65.7	50.7	82.8	49.7	72.0	20	99	2B4E
20	37.9	51.9	44.9	17.7	37.9	56.3	44.9	43.2	15	93	2C4E
21	42.9	53.6	50.4	43.1	42.9	61.1	50.4	55.6	20	97	2C4E
22	45.2	74.2	52.3	69.5	45.2	76.3	52.3	74.3	20	99	2C4E
23	33.4	72.5	42.7	74.6	33.4	75.5	42.7	77.6	15	91	2C4E
24	33.7	73.3	44.6	63.2	33.4	76.3	44.6	69.2	15	92	2C4E
25	38.4	58.0	42.4	73.1	38.4	64.0	42.4	77.0	15	98	2C4E
36	34.2	59.9	45.8	57.3	34.2	65.9	45.8	67.3	10	84	2C4E
41	36.5	55.5	41.4	55.3	36.5	62.5	41.4	67.0	5	60	2C4E
37	23.3	66.9	32.1	50.4	23.3	72.9	32.1	59.4	5	49	2D4E
38	37.2	74.7	46.7	53.3	37.2	77.7	46.7	61.0	20	97	2D4E
39	37.7	65.6	45.3	57.8	37.7	68.6	45.3	63.8	15	86	2D4E
40	42.5	60.6	42.6	61.2	42.4	64.0	42.6	67.2	10	71	2D4E
42	38.9	76.3	46.8	69.9	38.9	76.3	46.8	75.9	5	37	3A4A
43	58.9	70.0	52.4	58.8	58.9	76.0	52.4	69.8	10	56	3A4A
44	67.2	62.6	49.9	76.2	67.2	70.6	49.9	82.2	15	76	3A4A
45	58.6	78.8	56.5	69.8	58.6	81.8	56.5	75.8	25	88	3A4A
46	48.8	80.5	51.5	77.2	48.8	83.5	51.5	81.2	25	88	3B4A
47	33.4	82.8	50.3	69.9	33.4	84.8	50.3	77.9	15	73	3B4A
48	31.9	82.8	44.8	72.2	31.9	83.8	44.8	75.2	10	54	3B4A
49	40.2	71.8	45.5	79.4	40.2	74.8	45.5	83.4	10	62	3C4A
50	36.9	75.0	46.3	76.0	36.9	79.0	46.3	80.0	15	74	3C4A
51	31.0	72.8	52.2	73.1	31.0	77.8	52.2	78.1	20	85	3C4A
56	30.6	74.2	52.1	74.2	30.6	79.2	52.1	79.2	25	89	3C4A
52	35.2	80.4	53.9	66.1	35.2	82.4	53.9	72.1	25	90	3D4A
53	27.2	72.7	45.5	69.8	27.2	74.7	45.5	74.8	10	59	3D4A
54	21.8	63.0	42.3	73.8	21.8	68.0	42.3	75.5	15	70	3D4A
55	20.9	75.4	48.6	66.6	20.9	79.3	48.6	72.5	20	87	3D4A

Notes: a) Test metallurgy is recovery at final cleaner concentrate grade.

b) Corrected metallurgy was arrived at by redistribution of the cleaner tailings.

c) Grind time refers to time in the laboratory test mill: Product P_{80} is proportional to grind time.

2.43 Gold, Silver and Minor Elements

In addition to the normal assays used for the computation of recoveries, the concentrates were assayed for silver, gold, mercury and arsenic. This data for selected tests is shown below in table 26 which shows average gold and silver concentrations and estimated recoveries. Averaged mercury and arsenic concentrations are reported in Tables 27 and 28 by type and for the Vangorda orebody.

TABLE 26

GOLD AND SILVER METALLURGY - VANGORDA ORE TYPES

ORE TYPE	TEST NO.	RECOVERY		GRADE	
		GOLD	SILVER	GOLD	SILVER
1A4G	3	54.6	70.6	4.8	692
1A4G	4	47.4	71.7	3.8	641
1A4G	5	42.2	63.3	4.1	685
AVERAGE	-	48.1	68.5	4.2	673
1B4G	17	28.9	62.7	7.5	584
1B4G	16	33.3	70.9	8.9	679
1B4G	11	23.5	56.3	7.2	588
1B4G	18	30.6	51.5	10.3	620
1B4G	19	23.1	50.3	8.9	696
AVERAGE	-	27.9	58.3	8.6	633
2A4E	34	59.7	81.5	7.5	638
2B4E	32	57.1	53.9	8.2	582
2B4E	35	60.2	64.4	8.9	714
2C4E	21	39.4	36.2	8.2	384
2C4E	22	39.0	42.5	6.5	360
2D4E	38	60.2	51.3	11.7	353
2D4E	39	52.8	42.7	12.3	353
AVERAGE	-	52.6	53.2	10.3	483
3A4A	44	7.0	46.4	2.4	822
3A4A	45	10.4	59.9	2.7	802
3B4A	46	9.6	61.0	2.2	579
3C4A	49	13.5	51.9	2.1	381
3D4A	52	17.0	67.9	2.5	415
AVERAGE	-	11.5	57.4	2.4	600

Notes:
a) Recoveries calculated from composite head assay and unadjusted test distribution data.
b) It is estimated that silver and gold recoveries are accurate to within $\pm 10\%$.
c) Au and Ag in g/tonne.

2.44 Predicted Plant Metallurgy

An approximate mine model based on preliminary cross sections generated by D. Hanson indicated the relative occurrance of the three major ore types. This data was then used to produce a weighted metallurgical balance for all elements of interest in the Vangorda ore body. See tables 27 and 28 below.

TABLE 27

VANGORDA METALLURGY BY ORE TYPE

ORE TYPE	RELATIVE OCCURRENCE IN DEPOSIT %	PREDICTED METALLURGY ASSAYS						DISTRIBUTION			
		Pb	Zn	Au	Ag	Hg	As	Pb	Zn	Au	Ag
4G Baritic	31	53	*	6.0	650	80	50	84	*	40	65
		*	55	*	*	440	20	*	80	*	*
4E Pyritic	39	49	*	10.0	480	60	200	77	*	50	50
		*	51	*	*	250	50	*	72	*	*
4A Quartzitic	30	48	*	2.4	600	40	500	81	*	15	55
		*	53	*	*	250	200	*	79	*	*

Notes: a) Data from best adjusted test data from type testing program.

b) Relative occurrence data calculated from the mine model available in 1980.

c) Au, Ag, As, and Hg, in g/tonne.

TABLE 28

PREDICTED PLANT METALLURY - VANGORDA ORE

CONCENTRATE	ASSAYS						DISTRIBUTION			
	Pb	Zn	Au	Ag	Hg	As	Pb	Zn	Au	Ag
Lead	50	*	6.5	575	60	250	80	*	35	55
Zinc	*	52.8	*	*	300	100	*	77	*	*

Notes: a) Metallurgy calculated from weighted average of type testing data.

b) Au, Ag, As, and Hg, in g/tonne.

2.5 COMPATABILITY TESTING

2.5 COMPATABILITY TESTING

2.51 Laboratory Testwork

Having established the metallurgy for each ore type and then deducing the overall metallurgy for each ore body, the next point to consider was the feasibility of milling the various ores concurrently. As discussed earlier in this section, both pilot plant studies and all recent laboratory testwork have been designed to emulate achievable conditions in the modified Cyprus Anvil mill.

Since the Vangorda material is only a small fraction of the total remaining ore in the area, interaction effects were studied only between Grum and Cyprus Anvil ore. The testwork described here was designed in cooperation with W. Muir, Plant Metallurgist at Faro, and various members of the Feasibility and Development Group. The work was performed at the Lakefield Research Laboratory in 1979.

The testwork comprised several open circuit cleaner tests utilizing the standard flowsheet, with grinding at about 50 microns P_{80}, and a reagent pattern approximating to that employed at Cyprus Anvil. The samples consisted of drill core from the Faro deposit and the screened remnants of the pilot plant sample treated at Lakefield during the 1977 Grum testwork. Sample composition was as follows:

TABLE 29

HEAD ASSAYS OF COMPOSITES

SAMPLE	ASSAYS % Cu	Pb	Zn	Fe	S
Grum	0.14	4.98	9.13	33.10	33.9
Cyprus Anvil	0.18	3.15	4.43	34.80	32.2

Notes: a) Grum sample originated from materials stored at Lakefield.

b) Cyprus Anvil sample was provided by W. Muir plant metallurgist and comprised principally Type 4E Pyritic species.

2.52 Base Metallurgy by Ore Type

First standard open circuit cleaner tests were performed on Grum and Cyprus Anvil samples to obtain data about the base conditions. The results of two test pairs are shown below in Table 30.

TABLE 30

BASE METALLURGY FOR GRUM & CYPRUS ANVIL SAMPLES

PRODUCT	WEIGHT %	ASSAYS % Pb	ASSAYS % Zn	% DISTRIBUTION Pb	% DISTRIBUTION Zn
CYPRUS ANVIL ORE					
Pb Cleaner Concentrate	3.01	74.9	1.81	81.7	1.3
Pb 1st Cleaner Conc.	4.61	54.8	4.31	91.3	4.6
Zn Cleaner Concentrate	6.04	0.36	52.80	0.8	74.3
Zn Rougher Concentrate	30.81	0.31	12.9	3.9	92.9
Zn Flotation Tailing	64.68	0.20	0.17	4.6	2.5
CYPRUS ANVIL ORE					
Pb Cleaner Concentrate	3.06	72.8	2.03	79.8	1.4
Pb 1st Cleaner Conc.	5.68	44.9	5.21	91.4	6.8
Zn Cleaner Concentrate	5.48	0.34	53.8	0.7	67.8
Zn Rougher Concentrate	13.00	0.48	29.5	2.2	88.3
Zn Flotation Tailing	81.32	0.22	0.26	6.4	4.9
GRUM ORE					
Pb Cleaner Concentrate	6.49	60.3	7.52	81.3	5.4
Pb 1st Cleaner Conc.	20.65	21.6	12.20	92.5	27.9
Zn Cleaner Concentrate	10.27	0.50	51.70	1.1	58.5
Zn Rougher Concentrate	27.25	0.65	22.80	3.7	68.4
Zn Flotation Tailing	52.10	0.36	0.65	3.8	3.7
GRUM ORE					
Pb Cleaner Concentrate	6.85	56.2	9.12	80.6	6.9
Pb 1st Cleaner Conc.	15.12	28.8	12.4	91.2	20.7
Zn Cleaner Concentrate	10.21	0.63	52.8	1.3	59.6
Zn Rougher Concentrate	21.60	0.70	31.1	3.2	74.1
Zn Flotation Tailing	63.28	0.43	0.73	5.6	5.2

2.53 Tests with Ore Mixtures

Two more tests were then performed with a 50:50 mixture of Grum and Anvil ores. The test procedures were the same and the results are reported below in Table 31.

TABLE 31

METALLURGY OF MIXTURE 50:50 GRUM & ANVIL

PRODUCT	WEIGHT %	ASSAYS % Pb	Zn	% DISTRIBUTION Pb	Zn
Pb Cleaner Concentrate	4.72	68.6	4.57	85.1	3.2
Pb 1st Cleaner Conc.	12.27	28.8	10.70	92.8	19.6
Zn Cleaner Concentrate	8.38	0.51	52.5	1.1	65.9
Zn Rougher Concentrate	23.80	0.53	21.60	3.3	76.9
Zn Flotation Tailing	63.93	0.23	0.36	3.9	3.5
Pb Cleaner Concentrate	4.41	69.8	5.30	80.5	3.5
Pb 1st Cleaner Conc.	9.30	37.3	9.65	91.2	13.5
Zn Cleaner Concentrate	8.01	0.38	53.50	0.8	64.9
Zn Rougher Concentrate	20.87	0.56	25.8	3.1	81.7
Zn Flotation Tailing	69.83	0.31	0.45	5.8	4.8

By rearranging and averaging the data in Tables 29 and 30 it is possible to show that the results for the mixture of ores falls, as expected, between that recorded for the two ore sources when separately tested.

An interesting point about these results is that the blended mixture produced results which were slightly better than the arithmetic average of both ore types individually. This is a peculiar but not unique occurrance with blends of ores and is probably due to preferential grinding of the relatively softer Grum ore.

TABLE 32

SUMMARY OF TEST DATA

CONDITION		ASSAYS Pb	ASSAYS Zn	DISTRIBUTION Pb	DISTRIBUTION Zn
Anvil - Base Metallurgy	Lead	73.9	*	80.8	*
	Zinc	*	53.3	*	71.1
Grum - Base Metallurgy	Lead	58.3	*	81.0	*
	Zinc	*	52.3	*	59.1
50:50 Blend Metallurgy	Lead	69.2	*	82.8	*
	Zinc	*	52.8	*	65.1

Notes: a) All data from LR 2176 No. 5

b) Test data refers to batch tests which are not directly compable to the predicted plant data.

TABLE 33

COMPARISON OF

ANTICIPATED AND ACTUAL METALLURGY FOR BLENDED ORES

CONDITION		ASSAYS Pb	ASSAYS Zn	DISTRIBUTION Pb	DISTRIBUTION Zn
Anticipated Metallurgy (Arithmetic Average)	Lead	66.1	*	80.9	*
	Zinc	*	52.8	*	65.1
Actual Metallurgy	Lead	69.2	*	82.8	*
	Zinc	*	53.0	*	65.4

Notes: a) Again data refers to batch cleaner test results, unadjusted for redistribution of cleaner tails.

2.6 SPECIAL CHARACTERISTICS OF THE CONCENTRATES

2.6 SPECIAL CHARACTERISTICS OF CONCENTRATES

There are several characteristics of mineral concentrates which are of considerable significance in shipping and sales which have not yet been discussed. The three most important factors, aside from major metal contents, are concentrate flow moitures, moisture content and trace element analyses.

2.61 Flow Moistures

Flow moisture is an approximate physical test method which permits an estimate to be made of the point at which plastic flow of concentrates might occur. Clearly the onset of plastic flow will depend on mean particle size, particle shape and the range of various sizes present. The effect of plastic flow of large concentrate masses on ship stability and rolling moment are apparent.

To determine the effect of finer grinding of the concentrates on flow moisture, samples originating from the fine grind tests at Lakefield Research were subsequent to a flow moisture test. The results of these standard tests are summarized below.

TABLE 34

FLOW MOISTURE

	NORMAL	FINE GRIND
Lead	8.1	9.7
Zinc	9.8	11.2

Notes: a) Normal - Nov. 1979 test result for a six month certificate.

b) Normal P_{80} for concentrates 30-40 microns.

c) Fine grind P_{80} 10-20 microns.

d) The increase in flow moisture point with finer grinding was unexpected. Usually flow moistures tend to decrease with finer particle size.

2.62 Moisture Contained in the Concentrates

The upgraded dewatering plant will accomodate all the planned concentrate production and produce concentrates containing 4.5% moisture under ideal conditions. However, these ideal conditions assumed in the theoretical calculations of dryer capacity seldom exist in practice. Therefore, as shown in the table below the moisture contents will decrease from present levels but will probably not immediately reach the target moisture content of 4.5%. The probable initial plant performance is shown below in Table 35.

TABLE 35

MOISTURE CONTENTS OF CONCENTRATES

CONDITION	LEAD CONC. % WATER	ZINC CONC. % WATER
1978-1979 Averages	5.2	6.6
Theoretical Plant Performance	4.5	4.5
Initial Predicted Performance	4.7	5.0

2.63 Spectrographic Analyses

Some trace elements exercise considerable influence on the smelter process and as such, incur significant penalties to the seller of concentrates. The detection of these minor or trace elements is usually achieved by performing a spectrographic analysis of the concentrates.

TABLE 36

SPECTROGRAPHIC DATA - FINAL CONCENTRATES

(Results from Can-Test Vancouver)

ELEMENT		VANGORDA						GRUM		ANVIL ORE	
		TYPE 4A		TYPE 4E		TYPE 4G		PILOT PLANT COMPOSITE		PILOT PLANT COMPOSITE	
		ZINC	LEAD	ZINC	LEAD	ZINC	LEAD	ZINC	LEAD	ZINC	LEAD
Aluminum	Al	1.	1.	1.	0.1	0.2	0.2	0.1	0.05	N.D.	N.D.
Antimony	Sb	N.D.	0.1	N.D.	0.05	N.D.	0.3	N.D.	0.15	N.D.	N.D.
Arsenic	As	TRACE	*	N.D.	0.03	N.D.	0.05	0.15	N.D.	0.01	0.0002
Barium	Ba	0.1	0.3	*	*	*	*				
Beryllium	Be	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.
Bismuth	Bi	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.
Boron	B	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.
Cadmium	Cd	TRACE	N.D.	TRACE	N.D.	TRACE	N.D.	0.05	0.1	0.05	N.D.
Calcium	Ca	1.	0.5	2.	1.	1.	1.	0.10	0.07	0.05	0.07
Chromium	Cr	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.
Cobalt	Co	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.
Copper	Cu	0.3	0.1	*	*	0.1	0.3	0.2	0.5		
Gallium	Ga	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.
Iron	Fe	MAJOR	MAJOR	MAJOR	MAJOR	MAJOR	MAJOR	MAJOR	MAJOR	MAJOR	MAJOR
Lead	Pb	*	MATRIX	*	MATRIX	*	MATRIX	MAJOR	MATRIX	MAJOR	MATRIX
Magnesium	Mg	1.	0.1	2.	0.3	1.	0.5	0.02	0.02	0.1	0.01
Manganese	Mn	0.1	0.07	0.3	0.2	0.2	0.3	0.03	0.01	0.02	0.01
Molybdenum	Mo	N.D.	N.D.	TRACE	TRACE	N.D.	TRACE	N.D.	N.D.	N.D.	N.D.
Niobium	Nb	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.
Nickel	Ni	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.
Potassium	K	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.
Silicon	Si	3.	5.+	0.5	1.	0.3	1.	1.5	1.5	0.1	0.1
Sodium	Na	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.
Strontium	Sr	TRACE	0.001	0.03	0.05	0.01	0.05	N.D.	N.D.	N.D.	N.D.
Tantalum	Ta	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.
Thorium	Th	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.
Tin	Sn	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.
Titanium	Ti	0.1	0.3	0.01	0.01	0.01	0.01	N.D.	0.01	N.D.	N.D.
Tungsten	W	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.
Uranium	U	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.	N.D.
Vanadium	V	0.001	0.005	0.001	0.001	0.001	0.001	N.D.	N.D.	N.D.	N.D.
Zinc	Zn	MATRIX	*	MATRIX	*	MATRIX	MAJOR	MATRIX	*	MATRIX	*

Notes:

a) Percentages of the various elements expressed in these analyses may be considered accurate to within plus or minus 35 to 50% of the amount present.

b) Semi-quantitative spectrographic analytical results for gold and silver are normally not of a sufficient degree of precision to enable calculation of the true value of ores. Therefore, should exact values be required, it is recommended that these elements be assayed by the conventional Fire Assay Method. Quantitative and Fire Assays may be carried out on the retained pulp samples.

c) Silicon, aluminum, magnesium, calcium and iron are normal components of complex silicates.

d) MATRIX - Major constituent
MAJOR - Above normal spectrographic range
TRACE - Detected by minor amounts
N.D. - Not detected
* - Suggest assay (above 0.3%)