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C U R R A G H   R E S O U R C E S

MINE PLAN VP 1-5

VOLUME 1 - REPORT

Curragh Resources

MINE PLAN VP 1-5

VOLUME I - REPORT

FEBRUARY 1987

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1.0 EXECUTIVE SUMMARY

Under the following mine plan, known as case "VP 1-5", Faro mine operations, supplemented by production from the Vangorda Plateau, will continue well into 1999.

During that time, over 300 million tonnes of waste and 52 million tonnes of ore will be mined to produce 5.8 million tonnes of concentrate at a cost of $250.46 per tonne.

The cost and production summary is presented in Table 1-1.

Case VP 1-5

Cost and Production Summary

	87-04-01 to 87-12-31	1988	1989	1990	1991	1992	1993	1994	1995	1996	1997	1998	1999	2000	Total
Mined															
Total Waste	18,111,327	22,729,466	28,335,255	27,819,102	40,703,489	36,801,883	33,482,098	35,193,404	28,168,891	36,776,571	3,271,869				311,393,355
Total Ore	4,153,825	7,398,604	4,336,515	5,044,338	6,914,107	7,774,911	621,260	3,858,001	5,679,132	4,461,736	1,654,986				51,897,415
Strip Ratio	4.36	3.07	6.53	5.51	5.89	4.73	53.89	9.12	4.96	8.24	1.98				6.00
Mill Feed															
Tonnes	3,539,950	4,759,869	4,697,500	4,034,269	4,314,751	4,314,751	4,479,270	4,015,000	4,015,000	4,015,000	4,011,486	4,957,529	1,282,771		52,437,147
%Pb + Zn	8.23	7.99	7.25	8.58	7.49	7.41	7.59	9.16	9.47	8.86	7.10	4.05	5.04		7.62
Concentrate															
Lead Con Tonnes	153,169	194,584	181,956	240,010	165,137	146,497	157,150	188,772	187,138	179,202	137,121	90,548	31,261		2,052,543
Zinc Con Tonnes	271,522	358,612	308,270	282,454	300,798	310,261	344,115	353,848	376,395	340,155	266,509	168,732	55,169		3,736,840
Total Tonnes	424,691	553,196	490,226	522,464	465,935	456,758	501,264	542,620	563,532	519,357	403,629	259,280	86,430	0	5,789,383
Cost															
Capital	12,928,250	17,130,000	7,620,000	16,635,000	18,935,000	11,765,000	12,220,000	21,505,000	12,195,000	3,005,000	3,000,000	11,000,000	10,000,000	10,000,000	167,938,250
Operating	81,176,250	111,735,500	107,773,278	107,705,437	120,454,677	117,592,568	108,933,356	117,215,438	117,425,126	122,775,723	82,260,138	70,425,559	16,589,152		1,282,062,200
Total Cost	94,104,500	128,865,500	115,393,278	124,340,437	139,389,677	129,357,568	121,153,356	138,720,438	129,620,126	125,780,723	85,260,138	81,425,559	26,589,152	10,000,000	1,450,000,450
Per Tonne Concentrate	221.58	232.95	235.39	237.99	299.16	283.21	241.70	255.65	230.01	242.19	211.23	314.04	307.64		250.46

Table 1 - 1

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:

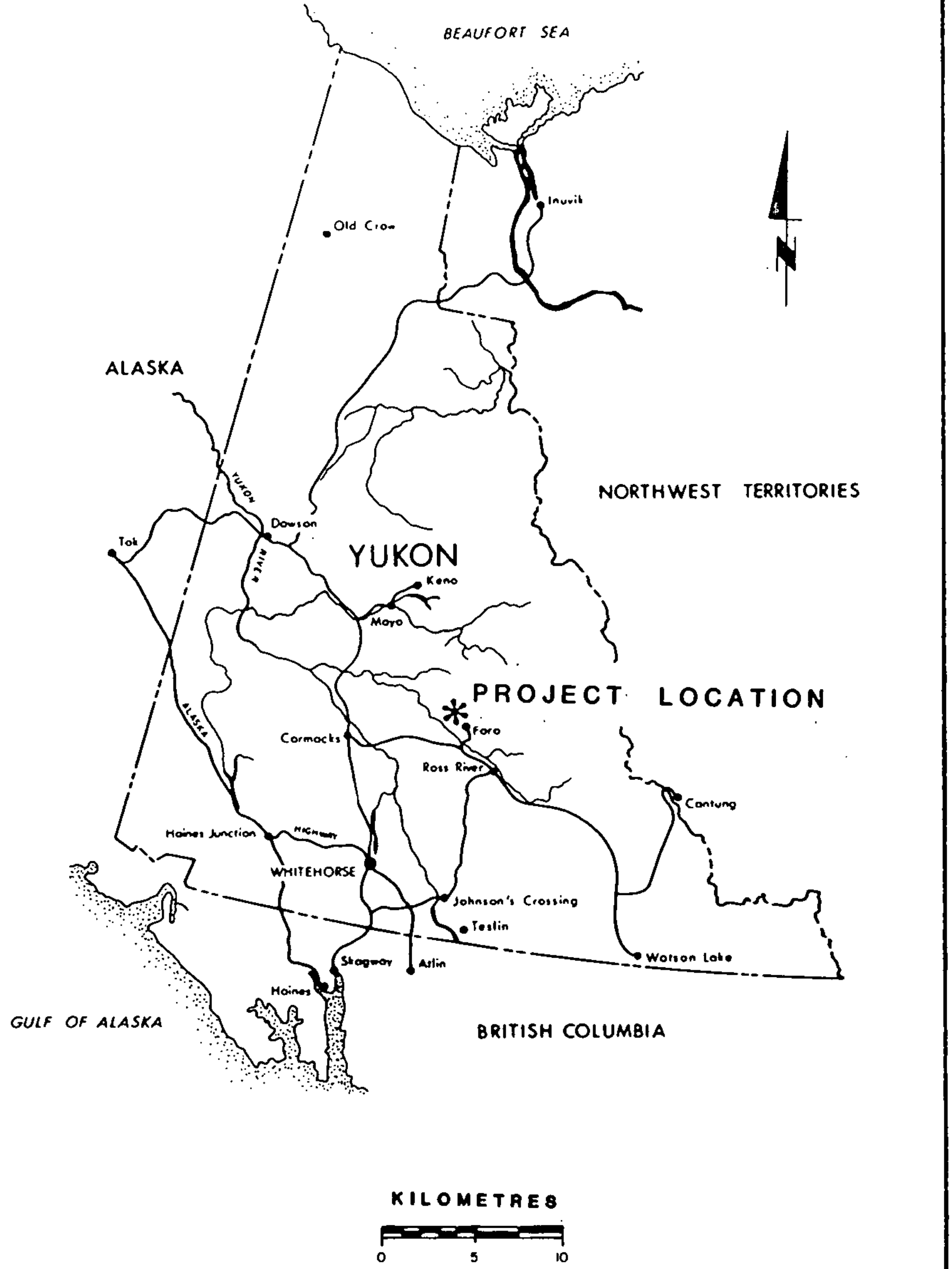
1. Deposit geology
2. Metallurgical response of the ores
3. Open pit mineable reserves, pit design, and pit planning
4. Detailed quantifying and sequencing of mining blocks
5. Detailed sequencing of mill feed and concentrate production
6. Infrastructure requirements
7. Operating equipment requirements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

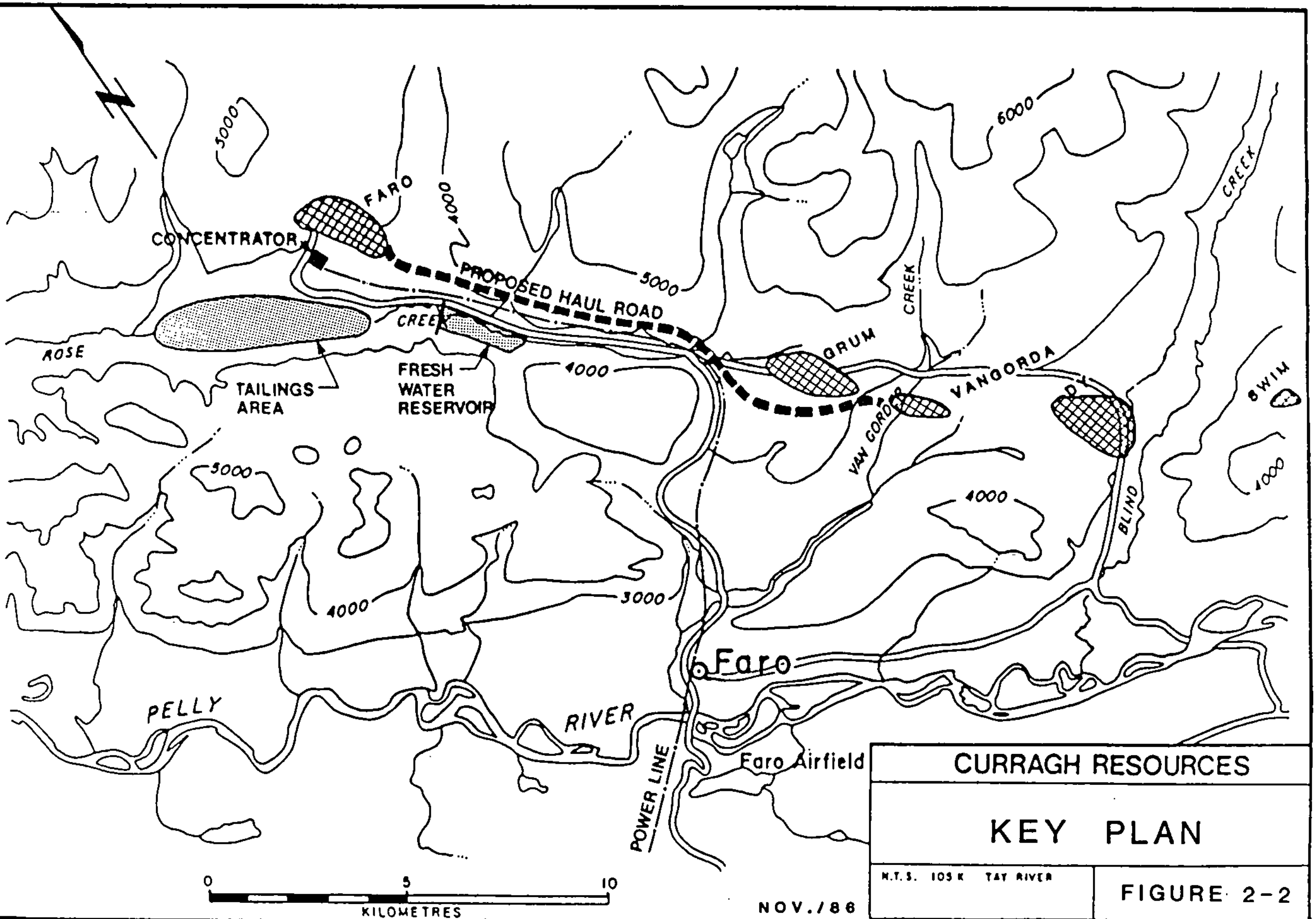
1. Alternate means of overburden stripping
2. Alternate haulage methods (waste and ore)
3. Assessment of the Champ Zone of the Grum deposit
4. Assessment of the 1986 Faro pit drilling program.

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



NOV./86

CURRAGH RESOURCES

KEY PLAN

N.T.S. 105 K TAY RIVER

FIGURE 2-2

3.1 District Geology

3.1.1 Introduction

The Anvil Range Pb-Zn-Ag District is located in the central Yukon Territory near the town of Faro (figure 2.1). The district contains one of the world's largest reserves of lead and zinc in several deposits (figure 3.1) including the recently re-opened Faro mine.

3.1.2 Regional Geology

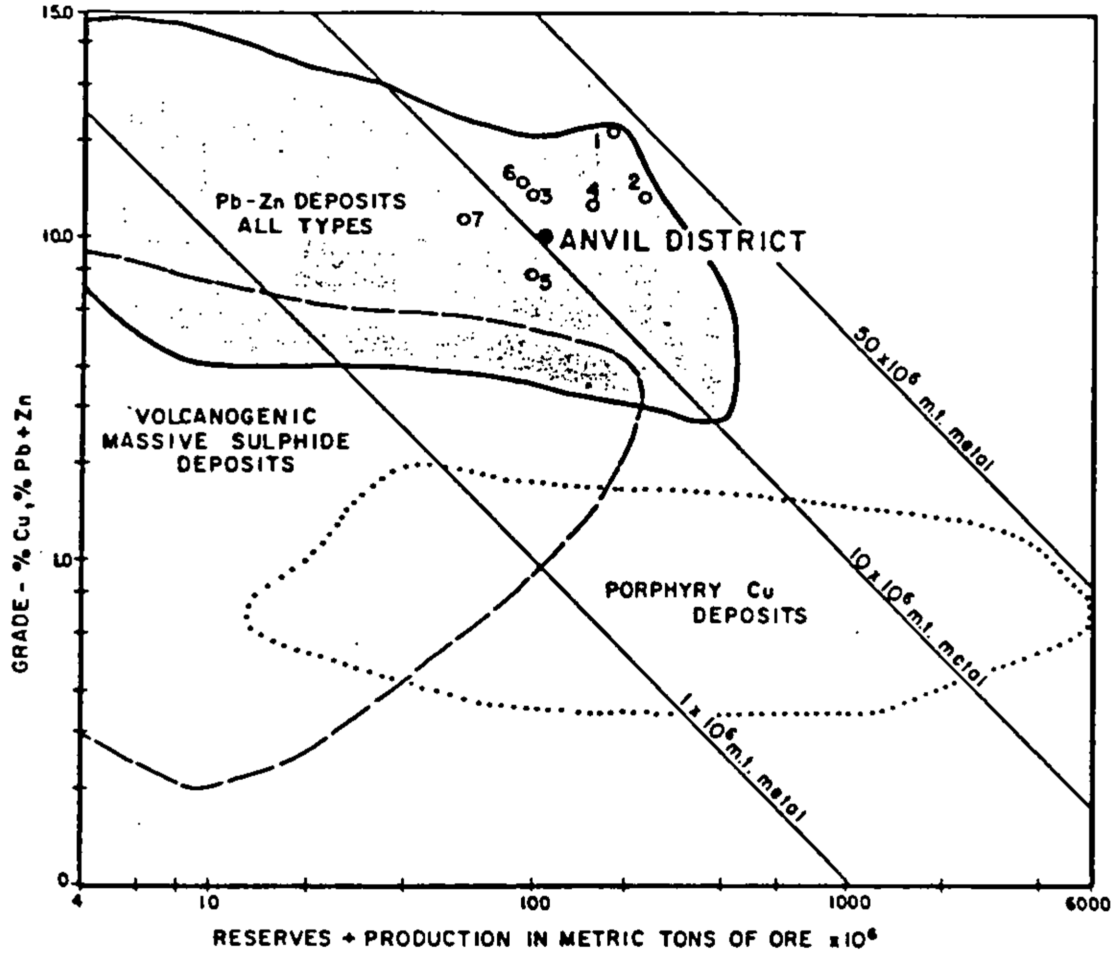
The Anvil District is part of the Selwyn Basin (figure 3.2), a large area of central Yukon where deep water shales accumulated along the ancient North American continental margin during the Paleozoic. The shales of the Selwyn Basin host most of Canada's large stratiform lead-zinc deposits, making it a metallogenic province of world wide significance. Unlike the remainder of the Selwyn Basin, the rocks and ores of the Anvil District are metamorphosed thus the shales are converted to phyllites and schists. The central part of the District in underlain by a large granitic body that cores an enlogate dome exposing the metamorphic sequence (Figure 3.3). The District contains several stratiform, lead-zinc-silver bearing, pyritic, massive sulphide deposits hosted by Cambrian metasediments on the southwest flank of the dome. The Tintina Fault, one of the major right lateral Cordilleran strike slip faults, passes just south of the district (Figures 3.2 and 3.3) but is not directly related to the ores.

3.1.3 District Stratigraphy

3.1.3.1 Introduction

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

The lower division ranges in age from late Precambrian to perhaps Early Silurian. It is approximately 5 km thick and divisible into three major mappable units (fig.3.4). From the base these are non-calcareous metapelite of Mt. Mye formation, calcareous metapelite of Vangorda formation and basalt and black phyllite of Menzie Creek formation. Established formal stratigraphic nomenclature does not apply directly to this area but the rocks are very similar to those of Kechika Group (Gordey, 1981) south of the district in Pelly Mountains. The lead zinc deposits occur within a restricted portion of the lower division.



1 BROKEN HILL, AUSTRALIA

2 MACARTHUR RIVER, AUSTRALIA

3 MT.ISA, AUSTRALIA

4 SULLIVAN, CANADA

5 HOWARDS PASS, CANADA

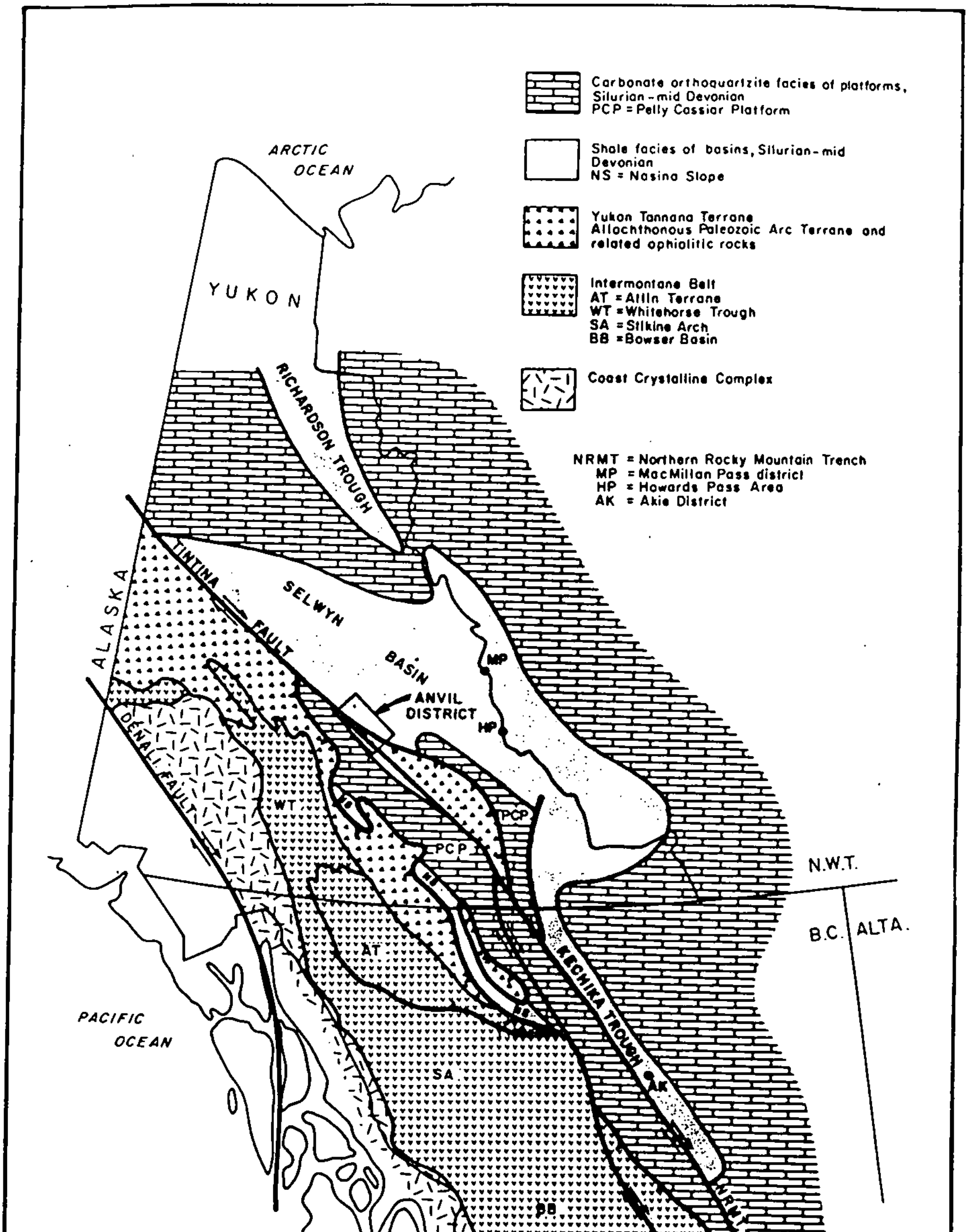
6 RED DOG, ALASKA

7 MEGGEN, WEST GERMANY

FIGURE 3.1

COMPARISON OF SIZE GRADE CHARACTERISTICS OF SOME MAJOR LEAD-ZINC DEPOSITS

JUNE 1986



Carbonate orthoquartzite facies of platforms, Silurian-mid Devonian
PCP = Pelly Cassiar Platform
Shale facies of basins, Silurian-mid Devonian
NS = Nasina Slope
Yukon Tannana Terrane
Allochthonous Paleozoic Arc Terrane and related ophiolitic rocks
Intermontane Belt
AT = Atlin Terrane
WT = Whitehorse Trough
SA = Stikine Arch
BB = Bowser Basin
Coast Crystalline Complex
NRMT = Northern Rocky Mountain Trench
MP = MacMillan Pass district
HP = Howards Pass Area
AK = Akie District
ARCTIC OCEAN
YUKON
RICHARDSON TROUGH
TINTINA FAULT
SELWYN BASIN
ALASKA
ANVIL DISTRICT
DENALI FAULT
MP
HP
WT
PCP
AT
N.W.T.
B.C.
ALTA.
KECHIKA TROUGH
PACIFIC OCEAN
SA
AK
BB
NRMT

Figure 3.2

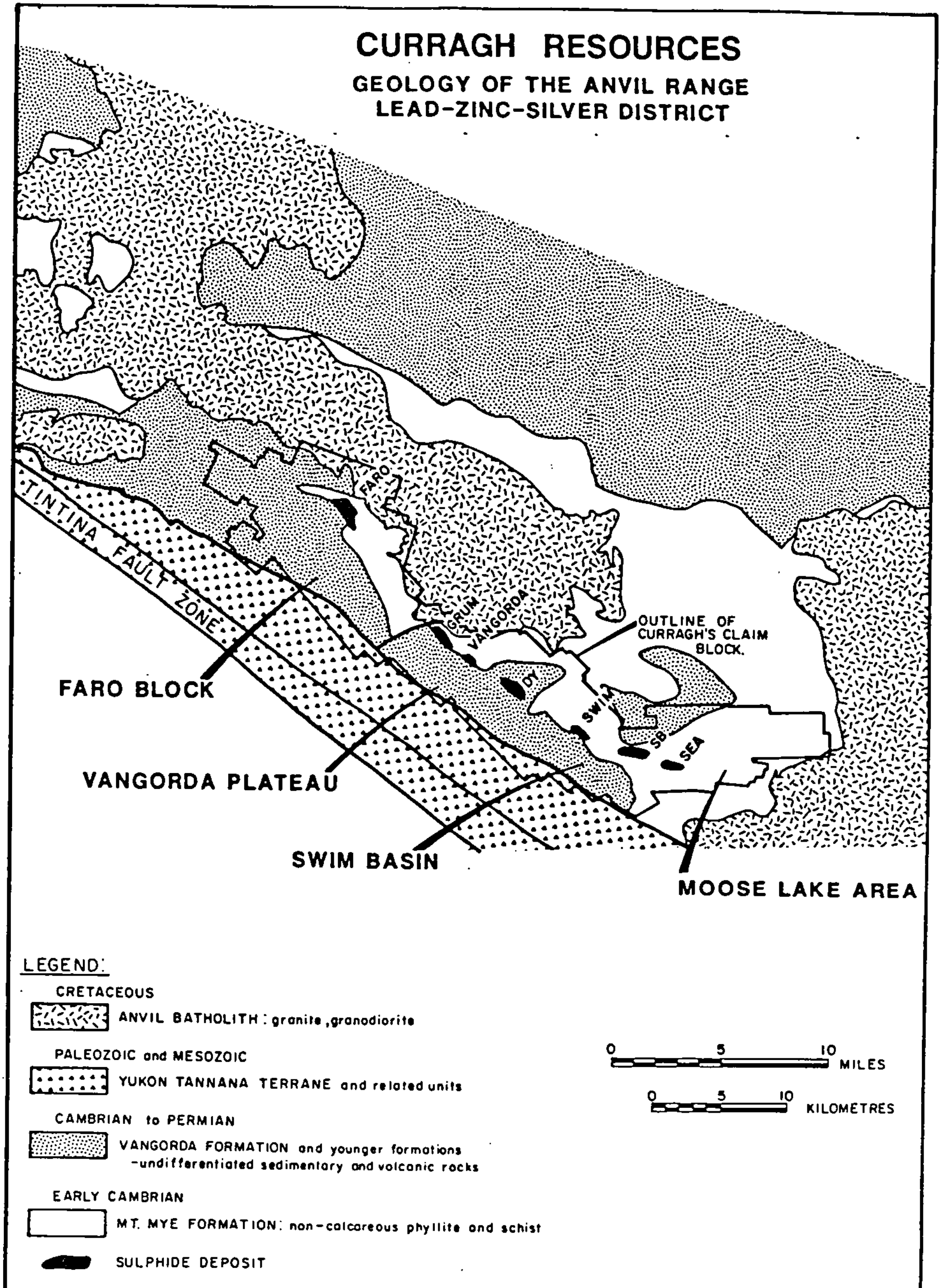

CURRAGH RESOURCES
GEOLOGY OF THE ANVIL RANGE
LEAD-ZINC-SILVER DISTRICT
TINTINA FAULT ZONE
FARO
GRUM
VANGORDA
DY
SWIM
SB
SEA
OUTLINE OF CURRAGH'S CLAIM BLOCK.
FARO BLOCK
VANGORDA PLATEAU
SWIM BASIN
MOOSE LAKE AREA
LEGEND:
CRETACEOUS
ANVIL BATHOLITH: granite, granodiorite
PALEOZOIC and MESOZOIC
YUKON TANNANA 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 3.3

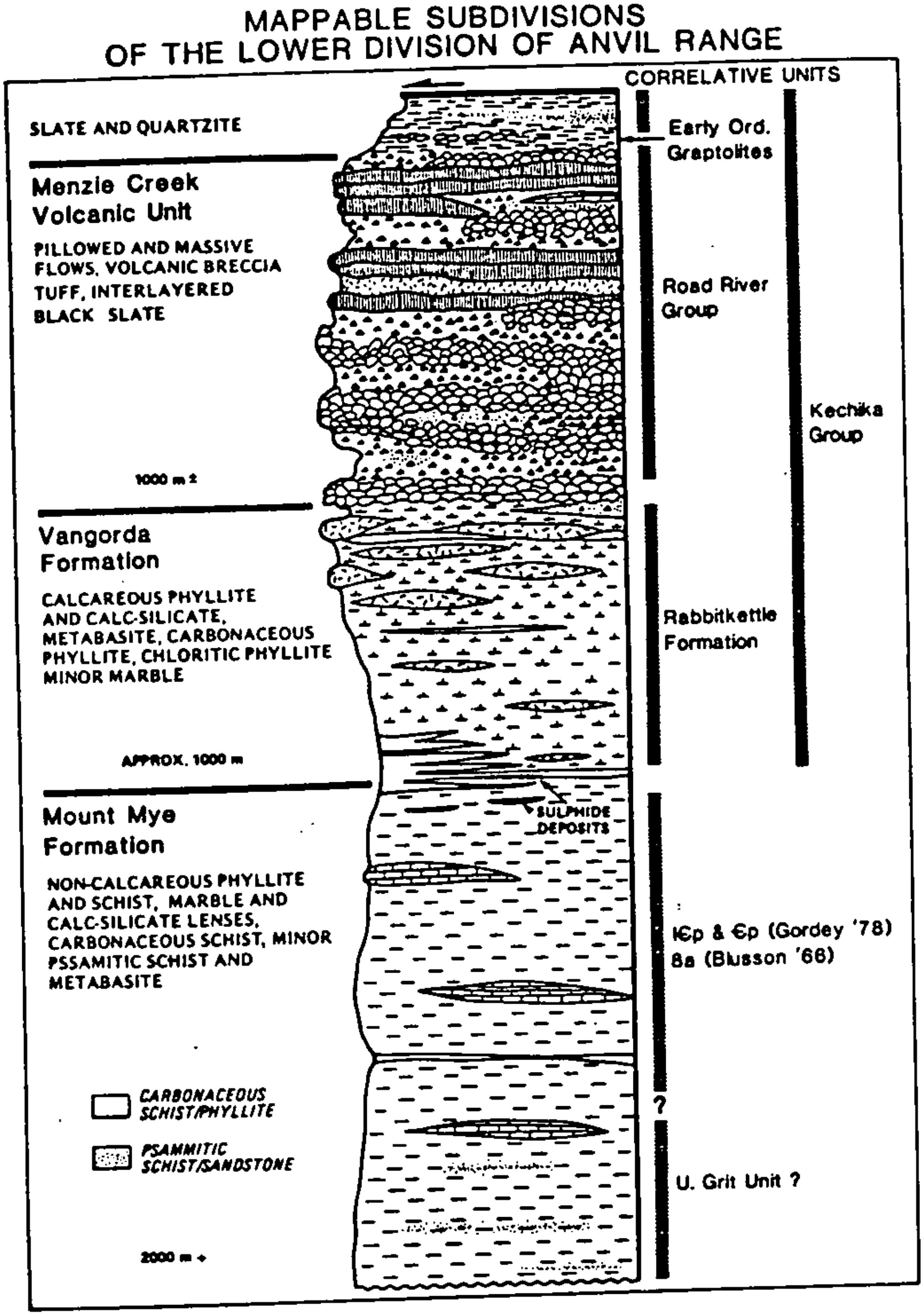


Figure 3.4 Diagramatic stratigraphic section of the lower Paleozoic of Anvil Range showing the ore deposits in relation to stratigraphy. Note that the bulk of the metavolcanics or metabasites are younger than the ore deposits but that the deposits are approximately coincident with the first appearance of substantial mafic igneous material in the section. Note also the anomalous thickness of carbonaceous rocks near the ore deposit trend.

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

3.1.3.2 The Lower Division

3.1.3.2.1 Mt. Mye formation

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

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

3.1.3.2.2 Vangorda formation

The Vangorda formation is characterized by light to medium-gray, calcareous, phyllitic rocks made up of very thin (0.1-2 cm) interlayers of a) medium grey, non-calcareous, weakly carbonceous, muscovite-chlorite pelite and b) light grey, generally calcareous quartz ± calcite ± dolomite siltstone. In areas of more intense metamorphism, such as near the Faro deposit, the calcareous phyllite is altered to a harder, banded, green, purplish brown and creme coloured calc-silicate. Other rock types interbedded with the calcareous phyllite include metabasite and meta-tuffs, graphitic phyllite, and phyllitic limestone.

Most metabasite bodies are medium-grained and equigranular, thus they may have been sills; however, locally amygdaloidal margins and a common association with thin bedded, tuffaceous rocks suggests at least some were flows. Whole rock compositional data shows that the metabasites are all of basaltic composition. The bodies range from 1 to 100 meters in thickness and are up to several kilometers in length.

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

The Vangorda formation is lithologically similar to, though more argillaceous than, Rabbitkettle Formation seen to the east (Gordey, 1978, Gabrielse et al., 1973). Based on this correlation the Vangorda formation may range in age from middle or upper Cambrian through lower Ordovician.

3.1.3.2.3 <u>Menzie Creek formation</u>

The Menzie Creek formation is a unit of basaltic metavolcanic rocks consisting of pillowed and massive flows with comparable amounts of massive, coarse, monolithic breccias and lesser, thin-bedded tuff and/or volcanic sandstone and siltstone. Carbonaceous phyllite and brown siltstone interbeds northeast of the Anvil Batholith contain graptolites of middle Ordovician or lower Silurian age (Tempelman-Kluit, 1972) suggesting correlation with the widespread Road River Formation black shale and chert to the northeast. The Menzie Creek formation varies from zero to about 1.5 kilometers in thickness in and near the district. It has been traced for 100 kilometers along strike and 30 kilometers across strike, showing that it is one of the largest of several basaltic units of its age in and around the Selwyn Basin.

3.1.3.3 <u>Relation of Stratigraphy to ore deposits</u>

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

3.1.4 <u>Deformation, Metamorphism and Plutonism</u>

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

The first deformation (D_1) produced a regional metamorphic foliation (S_1) axial planar to tight to isoclinal mesoscopic folds (F_1) in bedding (S_0). Mesoscopic early folds are rarely preserved in the district. Northeasterly inclined to upright, northeasterly verging megascopic folds with shallow northwesterly or southwesterly plunging axes appear to have formed at that time.

During the second event (D_2), S_1 was strongly crenulated and ubiquitous close to tight mesoscopic folds in S_1 were produced (figure 3.5). Some of the largest megascopic folds known to have been formed during D_2 are those at the Grum Deposit (Figure 3.8) and comparable folds in the Swim Deposit (Figure 3.12). Parallel to the axial planes of these D_2 folds is a crenulation cleavage (S_2) which imparts a well developed lithon structure and pronounced fissility to most rocks of the district, especially the strongly banded phyllites of the Vangorda formation. F_2 axial planes and S_2 dip shallowly, with axes subparallel F_1 axes.

Three later, less intense periods of folding and associated faulting followed. The later events (D_3 through D_5) generally produced open folds and weak crenulations in S_2 related to broad, regional structures. An important exception to this general rule is found in the vicinity of the Faro deposit where the fourth event (D_4) is quite intense. At Faro tight mesoscopic folds are developed in nearly pervasive S_2 with appreciable mica growth along S_4 (see Figure 3.7 for examples of fourth phase affecting outline of the Faro deposit).

During the later stages of the fold deformation history a large granitic body (Anvil Batholith) was intruded into the metamorphic sequence. Anvil Batholith ranges in composition from granodiorite to quartz monzonite and textures include equigranular massive, megacrystic massive and various strongly to weakly foliated variants. Several K/Ar ages on the granitic rocks yield ages of 85-100 ma (Tempelman-Kluit, 1972). Intrusion of the Anvil Batholith further deformed the metamorphic sequence so that the overall structure of the district is an elongate dome cored by the Batholith (Figure 3.2). In the later stages of batholith emplacement large extensional fault displacements occured along its margins. These faults determine the present day limits of several of the deposits (Figures 3.9, 3.11 and 3.12).

Metamorphism was concurrent with deformation and was most intense during the early deformations, especially D_2. Metamorphic facies developed range from middle amphibolite facies to lower greenschist facies in a low pressure Buchan facies series. Metamorphic isograds are roughly concentric about the Anvil Batholith. Faro, close to the Batholith (figure 3.3) is strongly metamorphosed, while deposits such as Vangorda are less intensely metamorphosed. This difference in metamorphism is reflected in decreased grain size, increased degree of mineral intergrowth, and lesser iron content of sphalerite in the less metamorphosed deposits. This has a significant impact on metallurgical performance of Anvil district ores.

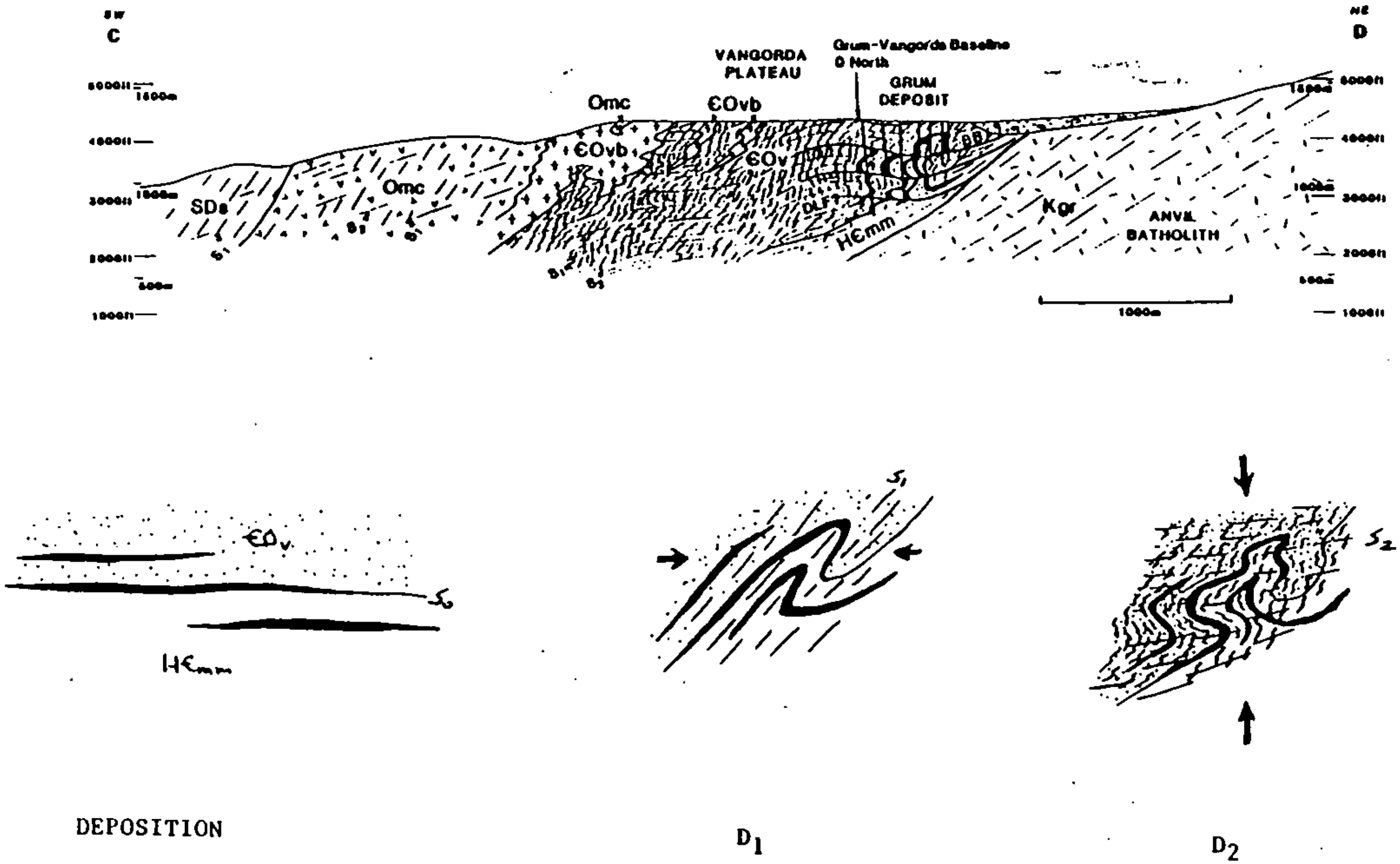


Figure 3.5 Cross section through Vangorda Plateau and Grum deposit (86 W). The Grum deposit provides the best example of the D_1/D_2 interference pattern in the district. The deposit is involved in a large Z (or N) shaped D_1 fold refolded by S shaped D_2 folds. The steeply dipping S_1 crenulated by shallowly dipping S_2 is typical of the structural relations on the Vangorda Plateau where greenschist facies rocks dominate. Post D_2 folds gently warp the S_2 foliation. The inserts show the sequential development of Grum from a sequence of stacked en echelon ore layers parallel to S_0 through D_1 and D_2 to produce the geometry observed today

3.1.5 Ore Deposits

3.1.5.1 General Description

The lead, zinc, silver deposits of Anvil Range are of the sediment hosted, stratiform, massive pyritic sulphide type (Gustafson & Williams, 1981; Large, 1980) or sedex type (Carne and Cathro, 1982). They occur as a single thick sulphide lens with little or no interbanded metasedimentary rocks (e.g. Faro) or as multilayered deposits with several thinner lenses stacked approximately one above the other with substantial metasedimentary or metavolcanic interlayers (e.g. Grum and Dy). An individual mineralized layer was deposited parallel to the bedding of the host sediments. It consisted of an upper, often centrally positioned, lead-zinc rich, massive sulphide facies and a lower and peripheral, lower grade, quartzose, disseminated sulphide facies.

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

Present day deposit lengths are generally two to three times widths; unfolded deposit dimensions range up to 4000 m across their ameboid shapes. Individual sulphide horizons commonly are 10 to 40 m in thickness. The upper and lower contacts of sulphide horizons are invariably sharp while laterally the sulphides grade into the enclosing host rocks.

All deposits are composed of a small number of different sulphide rock types. As noted above the suplhide rock types are broadly divisable into massive sulphides and quartzose, disseminated sulphides. There are pyritic, baritic, pyrrhotitic and carbonate bearing variants of massive sulphide types and carbonaceous and non-carbonaceous variants of the quartzose sulphide rock types. The typical spatial distribution of these different types is shown in figure 3.6 with great vertical exaggeration.

The simplified arrangement of the sulphide rock types in the horizons is important since lead-zinc grade and metallurgical performance varies by ore type. The baritic massive sulphides are always high grade, easily grindable and yeild good grade concentrates with good recoveries. On the other hand the lower and distal graphitic quartzites are commonly low grade, hard and produce lower grade concentrates or low recoveries. Other ore types exhibit intermediate characteristics and performance.

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

There are presently five known lead zinc bearing mineral deposits along a prominant curvilinear trend on the south flank of Anvil Arch (Figure 3.3). From northwest to southeast they include Faro, Grum, Vangorda, Dy and Swim. Additionally two lead-zinc deficient sulphide occurances, the SB and Sea, are also known. Diagramatic sections through each of the major deposits are shown in Figures 3.7 through 3.12.

3.1.5.2 <u>Description of Sulphide Rock Types</u>

3.1.5.2.1 <u>Massive Pyritic Sulphides:</u> (Unit 2E / 2F)

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

3.1.5.2.2 <u>Baritic, Massive Pyritic Sulphides:</u>

The baritic sulphides (Unit 2G) are strongly and thinly banded massive sulphide/sulphate rock consisting of pyrite, galena, sphalerite and commonly magnetic in a gangue of off-white barite and lesser carbonates (calcite, dolomite, ankerite and probably barytocalcite). The amount of barite may be as high as 50%; non-sulfidic, massive barite does not occur in the Anvil deposits. There is a complete gradation between this and the above facies with 10% visible barite by volume being the dividing line. This facies is usually quite high grade (10-15% combined lead-zinc). Sphalerite is characteristically honey coloured to reddish brown. Pyrrhotite is not commonly seen in the baritic facies except in the Faro deposit where overall pyrrhotite is more abundant.

3.1.5.2.3 <u>Carbonate-bearing, Massive Pyritic Sulphides:</u>

The carbonate bearing sulphides (Unit 2K) are similar to massive pyritic sulphides but contain greater than 10% carbonate (calcite, dolomite, ankerite) either as interstitial gangue or as coarse patches and irregular blebs. This is a minor facies and is not known with certainty to always be an original composition variant. The most common occurance of coarse pinkish beige to tan, ankerite patches may represent recrystallized original carbonate or re-worked pre/syn-metamorphic veins. This variant is generally lead-zinc

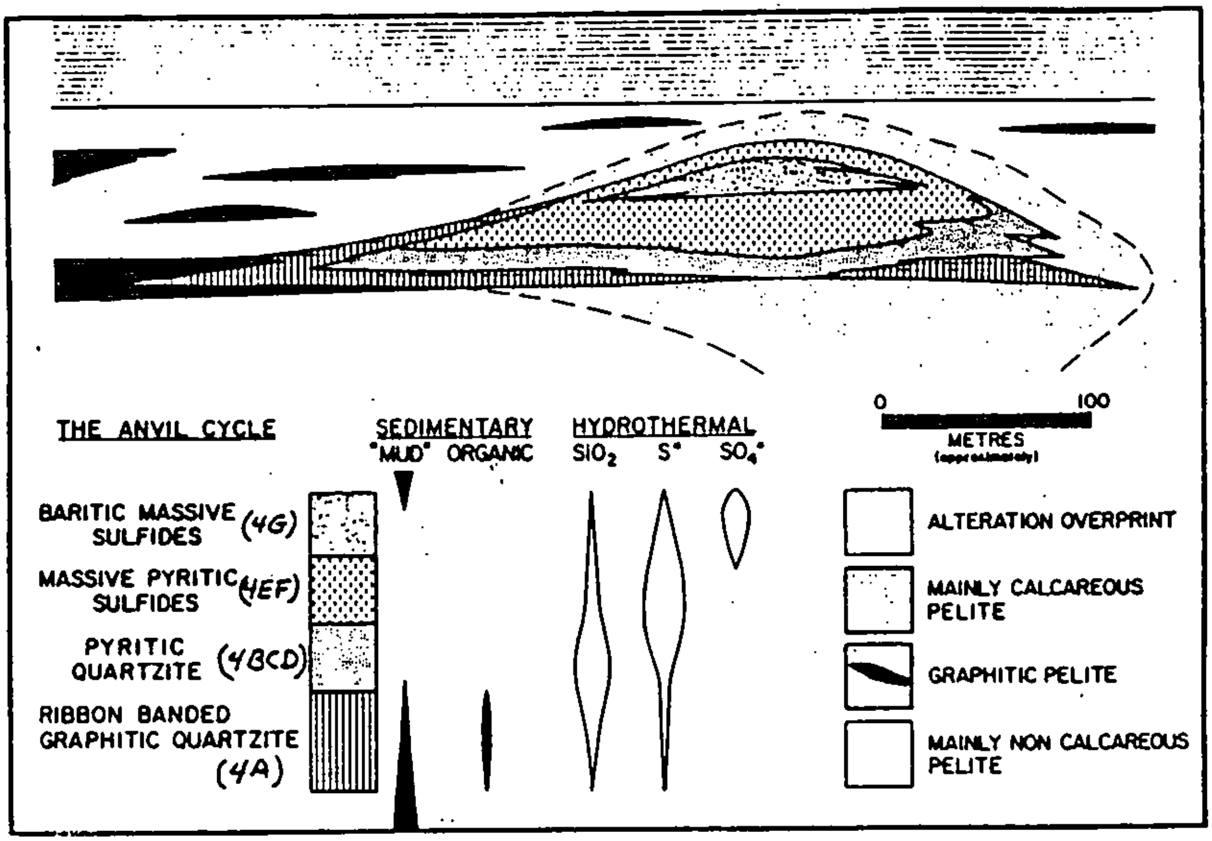


Figure 3.6 An idealized Anvil deposit based on cross sections of the Faro deposit. Such lateral and vertical zoning can be found in all deposits of the district. Massive sulphides are the central and upper lithofacies with pheripheral and lower quartzose disseminated lithofacies.

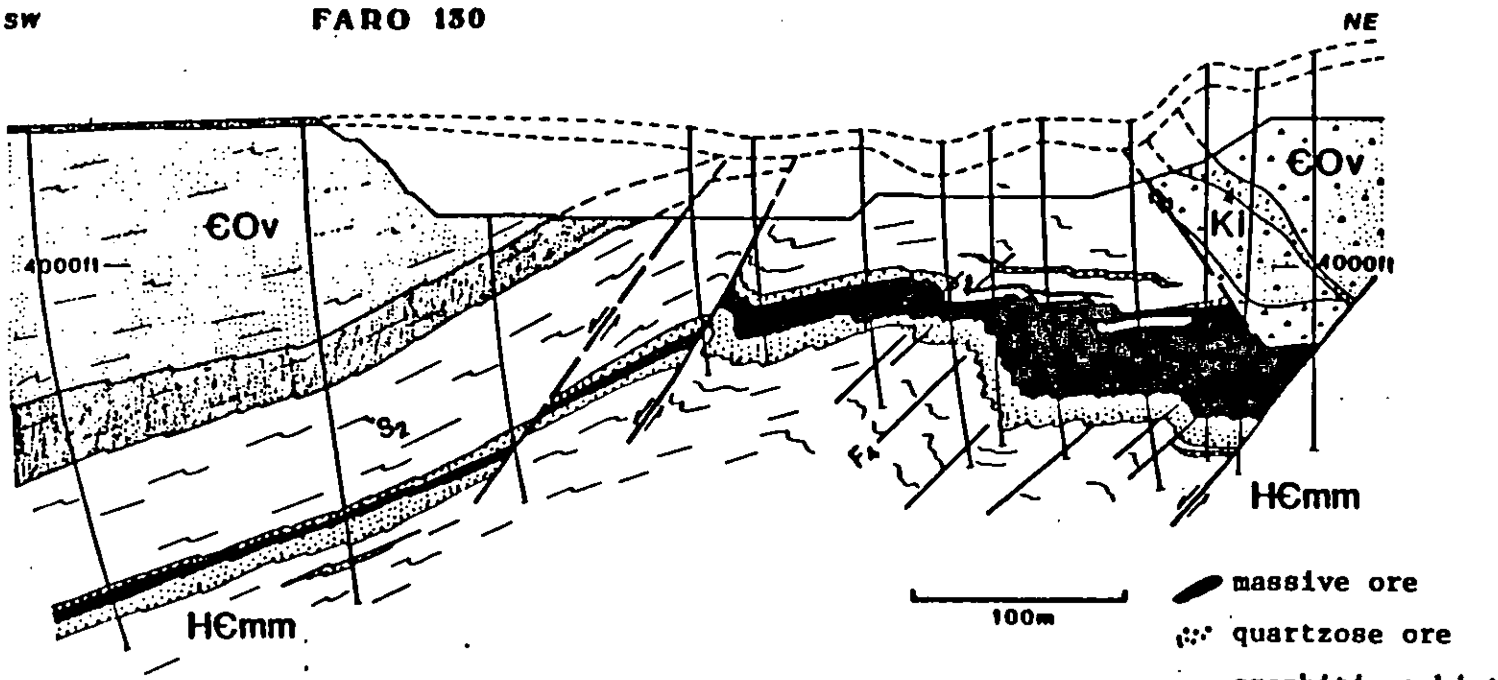


Figure 3.7 Cross section 130 through the southeast end of Faro zone 3. The pit outline shown is the present outline (as of June 1982 at suspension of mining). The faults are part of the Big Indian Fault set that separated zone 2 from zone 3, they are normal faults and cut across the section at a small angle. The triangular symbols at the northeast end of the section indicate the "breccia cap", a large body of post metamorphic breccia apparently formed by explosive activity during dyke emplacement.

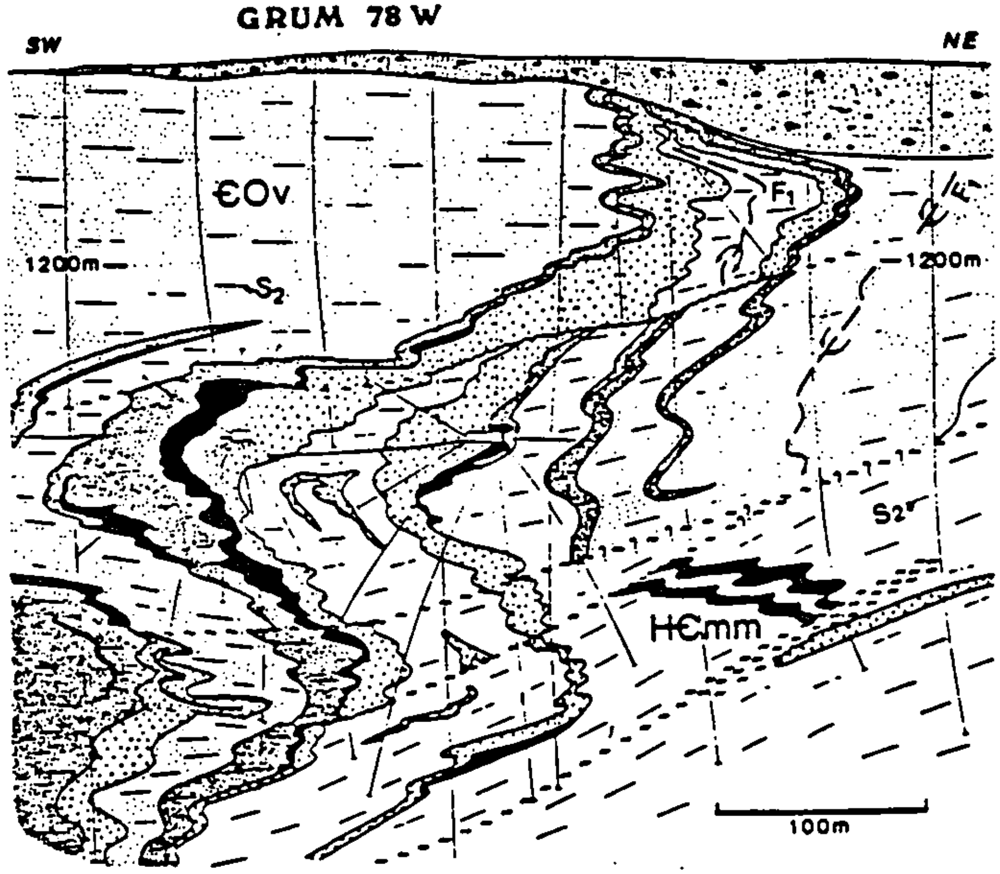


Figure 3.8. Cross section 78 W through the Grum deposit. The deposit forms a complex D_1/D_2 interference pattern which, despite the density of drilling, is not yet completely resolved. The faults appear to have slip lines directed across the plane of the cross section such that they "telescope" different deposit domains and appear not to make good sense on an individual section. The F_1 closure just beneath the overburden is confirmed on several densely drilled sections to the northwest down fold plunge.

massive ore

quartzose ore

graphitic phyllite

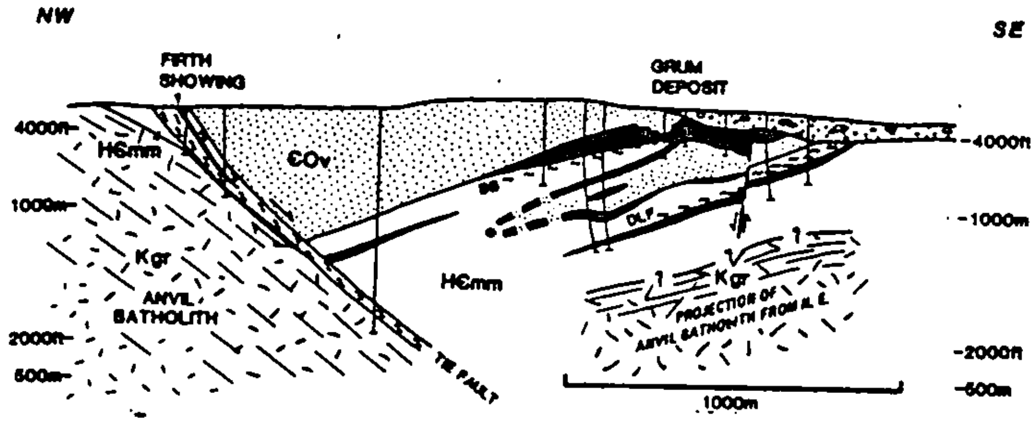
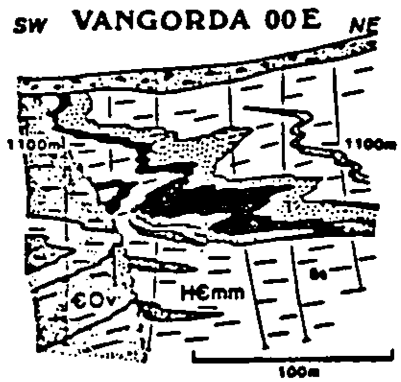


Figure 3.9 A diagramatic longitudinal showing the plunge of the Grum folded structure and the relation of the Grum deposit to the Firth showing. Firth appears to represent slivers of Grum caught in a large extensional fault, the Tie Fault, that separates footwall amphibolite facies metamorphic and granitic intrusive rocks from hangingwall greenschist facies

all sulphide lithofacies



massive ore

quartzose ore

graphitic phyll.

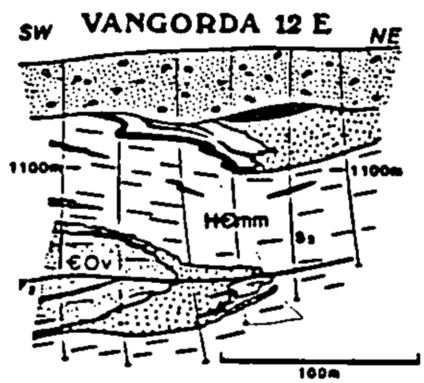


Figure 3.10 Cross sections OE and 12E through the Vangorda deposit.

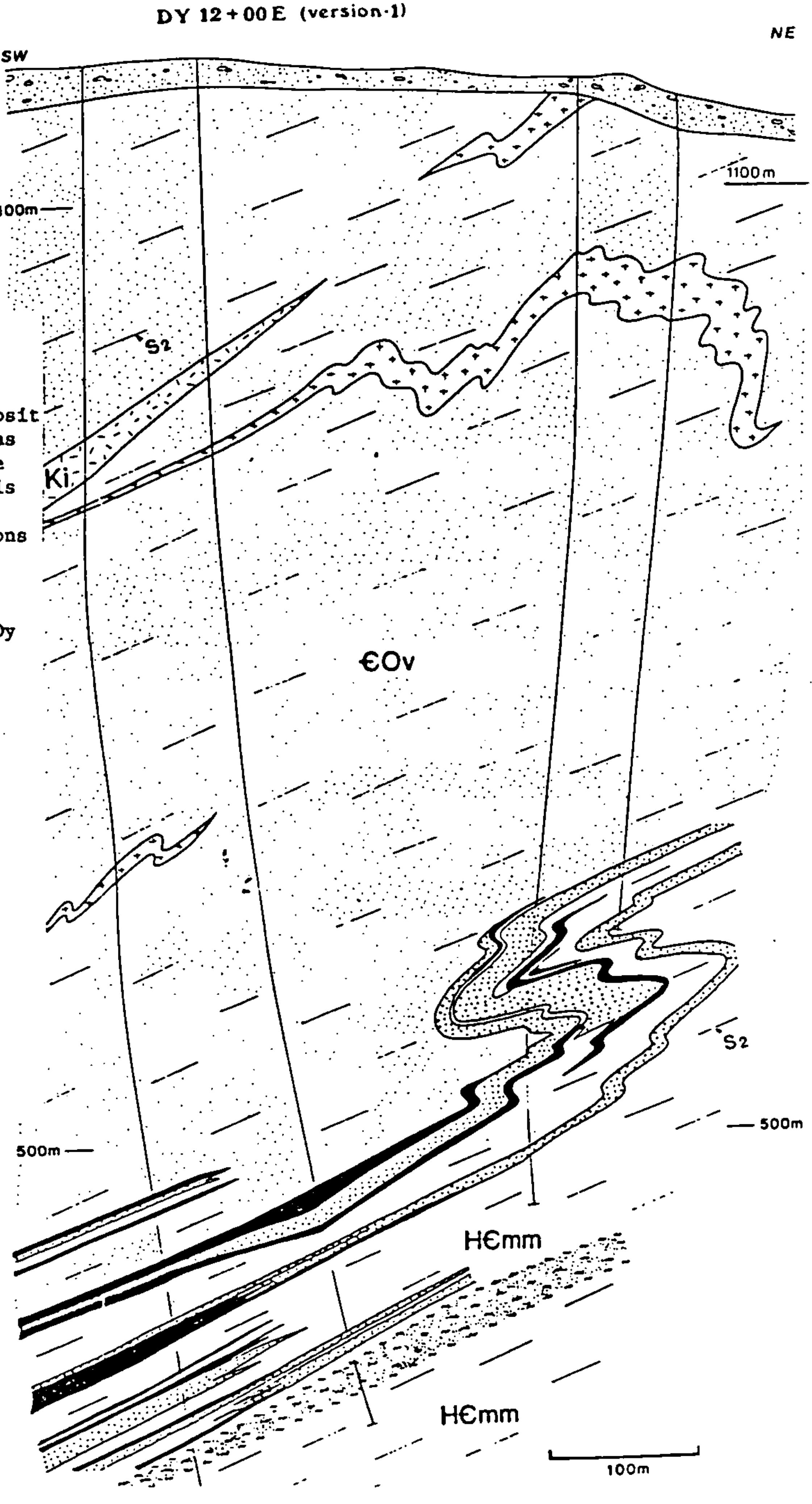


Figure 3.11

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.

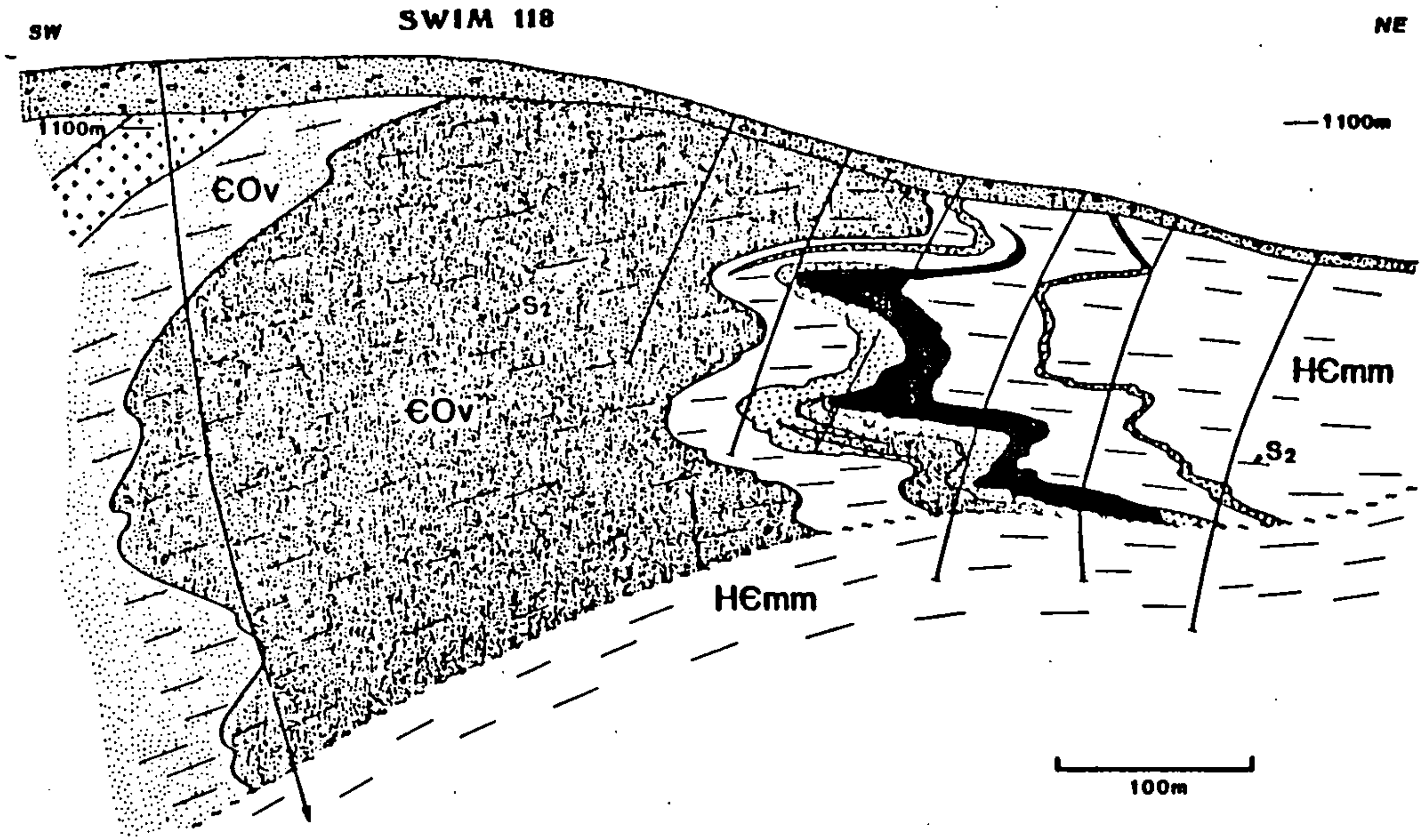


Figure 3.12 Cross section 118 through the Swim deposit. The basal massive sulphides in the main horizon are divisible into an upper baritic portion and a lower non baritic which is underlain locally by a thin quartzose unit thus the appearance of a reversed cycle is not real. The upper quartzose mineralization presumably represents the onset of a second incomplete (on this section at least) cycle. Displacement on the fault at the base of the deposit does not appear to be in the plane of the section, the resolution of this problem is a matter for furthur exploration.

poor. The variants with white interstitial gangue can be high grade and locally they texturally resemble the baritic sulphides

3.1.5.2.4 Pyrrhotitic Massive Sulphides:

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

3.1.5.2.5 Ribbon banded, "graphitic", pyritic quartzite:

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

3.1.5.2.6 Pyritic quartzite:

The pyritic quartzites (Units 2B, C, D)are light to medium grey, generally poorly banded, moderately to weakly foliated, micaceous quartzites with highly variable lead-zinc and pyrite contents. Pyrite contents are generally 10% to 40% ranging between 2 and 60%. Although there is a complete gradation from massive to quartzose ores there is usually little problem in separating this facies from the massive pyritic sulphides as the vast majority of examples have less than 40% total sulphides. A minor variant of this facies (unit 2B) shows low pyrite (< 5%) content with lead-zinc sulphides predominant. Barite in major amounts is uncommon in the quartose this facies; carbonate species are not typical but locally are abundant. Chalcopyrite, pyrrhotite and magnetite-bearing varieties are common. Sphalerite in the high grade examples is characteristically a vibrant reddish brown. At Faro the more sulphide rich variants of this facies are well developed along the northeast edge of zone 3. They are spectacularly barren but contain elevated copper contents and are rich in magnetite. A similar facies is developed at Vangorda and locally at Grum where the rocks are also quite gold rich and more clearly in the deposit footwall.

3.1.5.3 Alteration

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

3.1.5.4 Lithologic Terminology

A consistent alphanumeric code for lithology for all Anvil District deposits was introduced a number of years ago to facilitate storage of lithology data in a computerized database. Since occasional reference to these codes is made in the following sections a brief note of explanation is in order. The system works on the basis of a number followed by a letter and then a series of numbers and or symbols. The first number refers to metamorphic grade and hence structural level: 2 means amphibolite facies (Faro) and 4 means greenschist facies (all other deposits). The letter refers to the major lithology as shown in Table 3.1. The remaining letters and symbols are modifiers as outlined in Table 3.2. Thus 4A4 is a lead-zinc rich carbonaceous pyritic quartzite; 2A479 would be the same from Faro with pyrrhotite and chalcopyrite.

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

The mine models generally require an integer lithologic code and it has generally not been possible to use the same codes for all models; these codes are explained below for each model.

TABLE 3.1 THE ALPHA PART OF THE LITHOLOGIC CODE FOR SULPHIDE ROCK TYPES

A	thinly banded carbonaceous pyritic quartzite
B	weakly to non-pyritic quartzite
C	lead-zinc poor pyritic quartzite
D	lead-zinc rich pyritic quartzite
E	pyritic massive sulphide
F	buckshot textured pyritic massive sulphide
G	pyritic massive sulphide with >10% barite gangue
H	pyrrhotitic massive sulphide
J	non-pyritic or pyrrhotitic massive sulphide or massive magnetite
K	pyritic massive sulphide with >10% carbonate gangue
Q	foliated vein type quartz with sulphides

TABLE 3.2 LITHOLOGIC MODIFIERS FOR SULPHIDE ROCK TYPES

0	normal
1	siliceous
2	coarse porphyroblastic pyrite bearing
3	fine pyrite rich
4	lead-zinc rich
5	carbonaceous
6	barite bearing
7	pyrrhotitic
8	magnetite bearing
9	chalcopyrite bearing
*	undifferentiated carbonate bearing
#	calcite bearing
@	ankerite bearing
$	dolomite bearing

3.1.6 References

Blusson, S.L., 1966, Frances Lake, Yukon Territory and District of Mackenzie; Geol. Surv. Canada, Map 8-1967.

Carne, R.C., and Cathro, R.J., 1982, Sedimentary exhalative (sedex) zinc-lead-silver deposits, northern Canadian Cordillera: Can. Inst. Min. Metall. Bull., v.75, p. 66-78.

Gabrielse, H., Blusson, S.L. and Roddick, J.A. 1973, Geology of Flat River, Glacier Lake and Wrigley Lake map areas; Geol. Surv. Canada, Memoir 366, 153 p.

Gordey, S.P., 1978, Stratigraphy and structure of the Summit Lake area, Yukon and Northwest Territories; in Current Research, Part A, Geol. Surv. Canada, Paper 78-1A, p.43-48.

Gordey, S.P., 1981 Stratigraphy, structure and tectonic evolution of southern Pelly Mountains in the Indigo Lake area, Yukon Territory; Geol. Surv. Canada, Bulletin 318, 44p.

Gustafson, L.B., and Williams, N. 1981, Sediment-hosted stratiform deposits of copper, lead, and zinc: Econ. Geol., 75th Anniversary Volume, p. 139-178.

Kuo, S.L., 1976, Geology and geochemistry of stratabound ore deposits in south central Yukon Territory and southwestern District of Mackenzie, Northwest Territories; Ph.D. dissert., Univ. of Alberta, Edmonton.

Large, D., 1980, Geological parameters associated with sediment-hosted submarine exhalative Pb-Zn deposits: Geologisch. J., Reike D, Heft 40, p.49-129.

Tempelman-Kluit, D.J., 1972, Geology and origin of the Faro, Vangorda and Swim concordant zinc-lead deposits, central Yukon Territory; Geol. Surv. Canada, Bulletin 208, 73p.

3.1.6 Geology of the Vangorda Plateau

The Grum and Vangorda deposits are located in a part of the Anvil District referred to as the Vangorda Plateau. The Plateau is defined as the area between the headwaters of Rose Creek on the northwest and Blind Creek on the southeast. It is essentially the drainage basin of Vangorda Creek.

The Plateau is low rolling country between the rugged topography of the Mt. Mye massif and Sheep Mountain. The bedrock exposure is very poor, there are several areas where glacial overburden is many tens of metres thick. The area is heavily tree covered below 1220 m (4000 feet) elevation with thick brush above that.

The geology of the northwest part of the Plateau is outlined on three 1:5000 scale maps that cover the portion of the area to be affected by development of the Grum and Vangorda deposits. Three additional maps show the distribution of drill holes in the same area. The maps and the sections noted below are available in supporting documents filed at Curragh's Whitehorse and Toronto offices.

The stratigraphy is as outlined previously. Most of the Plateau is at greenschist facies with the high grade metamorphic rocks in the core of the Anvil Arch and the granitic rocks of the Anvil Batholith being separated from the low grade phyllites by a complex system of extensional faults. The geological boundary between the Vangorda Plateau and the Faro Block is the Tie fault, one of these extensional faults with about 1 km of throw.

The structure of the metamorphic sequence underlying the Vangorda Plateau is indicated on a set of cross sections also at a scale of 1:5000.

The stratigraphic position of the ores is indicated on those sections. The available drilling and mapping indicates the presence of a large isoclinal, S shaped, second phase fold which overturns the stratigraphic sequence in the vicinity of the trend of ore deposits. Because of this the depth to the favourable horizon increases rapidly to the southwest of the line of deposits and open pit potential in that area is nil. All available deep drillholes (at least those deep enough) southwest of the line of ore deposits have intersected a thick sequence of siliceous graphitic phyllite with minor disseminated pyrite and traces of sphalerite but none of the mineralized facies typical of the deposits. All indications are that the deposits are associated with the linear zone of the thickening of this graphitic phyllite and that the phyllite itself southwest of the line of thickening has limited potential. Northwest of the line of deposits stratigraphic levels exposed are deeper than the ore horizons thus the potential in that area is also limited.

Along the line of deposits there are several areas requiring further drill testing. Several holes will be required northeast of the Vangorda deposit where a blind second phase fold hinge is predicted which could repeat the ore horizon and offer potential for additional reserves. Between Vangorda and Grum is an uplifted fault block that exposes the hinge of the large overturned fold. There are several drill holes in this area but few of them have fully tested the favourable stratigraphy. This area being an upthrown block, is underlain by the more southwesterly portions of the favourable horizon as traced through the S shaped second phase folds. At both Grum and Vangorda the thickness and grade of mineralization in the deeper more southwesterly part of the favourable horizon decreases substantially from that in the main deposit area while the thickness of graphitic phyllite increases. The limited sulphide intersections in the upthrown block between Grum and Vangorda are consistent with this observation in both the adjoining downthrown blocks since thick graphitic phyllite is found at the favourable stratigraphic horizon. Nonetheless at Dy, southeast of this area, it appears that a second mineralized center is developed southwest of the main deposit line with thick sections of good grade mineralization. This observation along with the fact that Anvil type deposits are characterized by rapid, highly unpredictable facies changes shows that there is a possibility of a completely isolated separate mineralized centre within this fault block. It is highly unlikely that such a mineralized center could be found in an open pit environment however. Only a few holes would be required to evaluate the area between Grum and Vangorda prior to construction of waste dumps. It is essential that this be done because this area is some of the most attractive exploration ground in the Anvil Range.

3.1.7 References

Blusson, S.L., 1966, Frances Lake, Yukon Territory and District of Mackenzie; Geol. Surv. Canada, Map 8-1967.

Carne, R.C., and Cathro, R.J., 1982, Sedimentary exhalative (sedex) zinc-lead-silver deposits, northern Canadian Cordillera: Can. Inst. Min. Metall. Bull., v.75, p. 66-78.

Gabrielse, H., Blusson, S.L. and Roddick, J.A. 1973, Geology of Flat River, Glacier Lake and Wrigley Lake map areas; Geol. Surv. Canada, Memoir 366, 153 p.

Gordey, S.P., 1978, Stratigraphy and structure of the Summit Lake area, Yukon and Northwest Territories; in Current Research, Part A, Geol. Surv. Canada, Paper 78-1A, p.43-48.

Gordey, S.P., 1981 Stratigraphy, structure and tectonic evolution of southern Pelly Mountains in the Indigo Lake area, Yukon Territory; Geol. Surv. Canada, Bulletin 318, 44p.

Gustafson, L.B., and Williams, N. 1981, Sediment-hosted stratiform deposits of copper, lead, and zinc: Econ. Geol., 75th Anniversary Volume, p. 139-178.

Kuo, S.L., 1976, Geology and geochemistry of stratabound ore deposits in south central Yukon Territory and southwestern District of Mackenzie, Northwest Territories; Ph.D. dissert., Univ. of Alberta, Edmonton.

Large, D., 1980, Geological parameters associated with sediment-hosted submarine exhalative Pb-Zn deposits: Geologisch. J., Reike D, Heft 40, p.49-129.

Tempelman-Kluit, D.J., 1972, Geology and origin of the Faro, Vangorda and Swim concordant zinc-lead deposits, central Yukon Territory; Geol. Surv. Canada, Bulletin 208, 73p.

3.2 Faro Geology and Reserves

3.2.1 History

The Faro deposit was discovered in 1964 while drill testing airborne electro-magnetic anomalies supported by other indications. Mining at Faro began in late 1969 and continued until 1982 when high costs and falling prices forced temporary closure of the mine.

In November 1985, Curragh Resources bought the Faro mine and other deposits in the Anvil Range from Cyprus Anvil Mining Corporation. Waste removal from the Faro pit resumed in early 1986. The Faro concentrator resumed production in June 1986. During 1986 1,842,000 tonnes or high and low grade ore was mined.

3.2.2 General Geology

3.2.2.1 Stratigraphy and lithology

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

The immediate host rock of the orebody is biotite-muscovite-andalusite schist (unit 1D) that grades downwards into a coarse, gneissic botite-muscovite schist (unit 1C). A discontinuous graphitic phyllite unit about 6 m thick is interlayered with the schists about 25 m above the ore deposit. There are also several thin interbands of strongly foliated chlorite actinolite schist, or bleached and carbonated equivalents of this mafic schist, above the orebody.

The Vangorda formation at Faro is represented by hard, dense, banded calc-silicates (unit 3D) rather than the calcareous phyllite that characterizes the Vangorda Plateau. This fact is of considerable importance in blasthole drilling at Faro because of the rocks hardness. Amphibolite up to 10 m thick is interbanbed with the calc-silicates and there are several thin graphitic phyllite layers. The basal unit of the Vangorda formation in the Faro deposit area (unit 3A) consists of graphitic phyllite, amphibolite and calc-silicates mixed in subequal amounts.

Post metamorphic igneous intrusive rocks are more widely developed at Faro than elsewhere in the district. There are two clans of importance: a) a equigranular to subporphyritic hornblende diorite to quartz diorite clan (unit 10E) and b) a quartz-feldspar porphyry clan (unit 10F). The former occurs as a large dyke truncating the deposit at its northwest end, a smaller dyke along the fault between zones 1 and 3, an inferred sill beneath the breccia cap (see below) and several smaller dykes. The latter forms highly irregular and unpredictable intrusive bodies in the north part of zone 3.

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

3.2.2.2 Structure

Faro is deeper in the structural sequence than other parts of the Anvil District. Consequently the structural picture is rather different. The second deformation (D_2) effect is very strong at Faro. Virtually all sign of the first deformation has been completely overprinted by D_2.

The D_2 axial planar schistosity (S_2) is strongly developed and is the plane of greatest fissility in all metamorphic rocks of the Faro area. S_2 dips 10° to 20° towards the southwest or west. Second phase folds are generally isoclinal with shallowly northwest or southeast plunging axes. D_2 is so strongly developed that the structural sequence can be essentially viewed as a stratigraphic sequence with bedding parallel to S_2. The ore deposit is a tabular body parallel to compositional layering and S_2; internal layering in the ores is also parallel to S_2.

Three generations of later folds deform S_2, compositional layering and the ore deposit into close to tight northeasterly verging folds with axial planes dipping 45° to 60° towords the southwest and generally west or northwest plunging axes. The late folds commonly have amplitudes of approximately one m and folds of several tens of m are inferred in the base of the deposit (Figure 3.7). The size of these folds and the extent to which the deposit margin and internal banding geometry is defined by late folds as opposed to faults is one of the major uncertainties in ore reserve estimation at Faro.

Faults postdating the fold deformation (and concurrent metamorphism) are widely developed at Faro. Two sets are particularly important: 1) a N20°W striking and steeply west dipping set and 2) a East or N60°E striking and generally steeply to moderately south dipping set. These two sets define a graben structure. Zone 3, which contains the remaining reserves, is the central downtwrown block and Zones 1 and 2, now largely mined out, are the upthrown blocks. Many other fault sets are more locally developed, particularly in the area of the JB phase. Seemingly random faults with small but cumulatively significant displacement pose one of the most serious limitations on accurate local bench reserve estimation from exploration drillhole information.

S_2, along with joints and small gouge zones parallel to S_2, is the primary element for consideration in slope stability. The shallow to moderate southwest dip means that the northeast wall of the pit is relatively unstable. Failures to date appear to be surficial and involve platy rock fragments bounded by S_2

sliding slowly towords the pit. The possibility of larger scale failures involving slip on S_2 backed by some of the larger faults dipping towords the pit cannot be dismissed. The relatively more massive and stronger rock mass of the northeast edge of the sulphide deposit is expected to butress the northeast pit wall as the pit deepens.

3.2.2.3 Deposit Geology

Before mining, the Faro deposit was 2000 m along strike, 800 m across strike and from a few meters to 90 m thick. The deposit is a flat-lying, elongate, asymmetric lens with a thick northeast side and a thin tapering southwest side.

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

Ore type zoning is particularly strong at Faro. It follows the scheme outlined above with a massive variably baritic upper portion and a quartzose variably carbonaceous lower part (figures 3.6 and 3.7). In addition there is a prominent very low grade semi-massive zone along the northeast edge of zone 3 and unusually abundant (compared to other Anvil district deposits), but erratically distributed, pyrrhotitic mineralization in the southwest part of the deposit. Grade zoning follows ore type zoning so that the base and northeast edge of the deposit contains the lower grade mineralization whereas the upper and southwest portion contains the higher grade mineralization. Zoning was also obvious in plan view at Faro. Zone 1 was rich in baritic ores thus high grade, zone 2 at the other end of the deposit was rich in carbonaceous quartzose ore types thus low grade and metallurgically undesireable. Zone 3 has intermediate characteristics.

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

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

3.2.3 Drilling Density

The Faro deposit was drilled off on 43 m (141 feet) spaced sections with holes spaced nominally at 43 m along the sections. Most holes were vertical. In some parts of the deposit fill in drilling to 43 m has not been completed. In 1986 additional fill in (to about 25 m) was added to the northeast side of the AY phase because of difficulties in making accurate projections with available data. The results of this new drilling are not yet in a form suitable for mine planning.

3.2.4 Reserve Calculation

3.2.4.1 Method and Procedure - "FI" Model

The reserves used for Faro mine planning are derived from the "FI" mine model, generated from October to December 1985 by Cyprus Anvil Mining Corporation. The FI model is one of several computer block models of the Faro deposit; reference to some others will be made herein. The F3 and T3 date from 1981 and prior to completion of the FI model were the most recent complete models available. The F4 is more recent but was never completed and thus is not useable for mine planning. The FI model is an interim model that combined parts of the F3 and F4 and incorporated extensive drilling results postdating completion of the F3 and T3 models. The 1985 Kilborn analysis of the Faro mine project used the T3 model results.

3.2.4.1.1 Block geology and drillhole information

The FI Model is a computer based block model with block size 15.25m X 15.25m X 6.1m high (50 ft. X 50 ft. X 20 ft. high). The blocks are oriented North-South and East-West, at 45° to the elongation of the ore deposit and the geological sections. The Mintec Medsystem release 10 software package was used to generate the model and derived reserves. The model has since been imported to Curraghs software package, PC Mine, but all model building calculations have were done by Mintec's software. Reserves for Curragh's newly defined mining phases are calculated by PC Mine.

The geological interpretive base was derived from two sources. In the southeast part of Zone 3 (Sections 124 to 133) the geological interpretation is the most up to date possible (1983) and is the same as that used for the F4 model. In the remainder of Zone 3 (Sections 117 to 123) this new geological interpretation was not yet available thus the interpretation used was that developed in 1981 for the F3 and T3 models. This did not take advantage of 1984 drilling. The interpretations differ in the relative importance of folds and faults which results in significant differences in bench to bench geology, but for an overall section thru the deposit the cross-section area, hence the volume, is not very different. There are some artificial discontinuities evident in the resulting model where the two interpretations join (this is largely in the CY phase ore).

Block geological code assignment was based on 6 m (20 foot) spaced bench plans of the geology. A block whose area is underlain by more than 50% sulphide rock type was coded as an ore type otherwise it was coded as waste. The actual block geological code was defined by the unit occupying the maximum plan view area within the block. One rock code was assigned to each block and that code was assumed to apply uniformly to the entire block. The rock codes used are listed on table 3.3.

TABLE 3.3 LITHOLOGIC CODES USED FOR THE FI MODEL

Rock Code	Description	Density Used *
0	air	0
1	undifferentiated sulphides	0.085
2	2A ribbon banded graphitic quartzite	0.083
3	2BCD pyritic quartzite	0.090
4	2CE semi-massive sulphides	0.099
5	2EF pyritic massive sulphides	0.107
6	2GE baritic massive sulphides	0.112
7	2H pyrrhotitic massive sulphides	0.104
10	WASTE all types except calc-sil or sulph.	0.076
11	WASTE calc-silicates 3D	0.076
12	WASTE in blocks partially above topography	0.076

* Density is in tonnes per cubic foot

The drill hole database used includes all holes in the deposit to the time of model construction; thus all holes prior to Curragh's acquisition of Faro but none of Curragh's 1986 holes are included.. In the northwest half of the deposit the geology has not been adjusted to reflect holes put down since 1981 but the assays were used for interpolation.

All drillholes were relogged to a common standard between 1982 and 1984 and extensive checking of the assay and survey data for consistency was carried out by Cyprus Anvil's staff. Not all holes have been surveyed for downhole deviation. In these cases an average deviation based on nearby surveyed holes was used. As discussed below there are now known to be errors in the collar locations of some holes on the order of 50 feet laterally. This is apparently due to a small angular error in restablishing survey control at some time and cannot be corrected now. The error is largest in the more southeasterly holes thus is significant in the JB phase and especially zone 2 but is thought to be less important in the AY phase. Apparently not all holes in the JB are affected. Holes postdating the mid 70's are thought to be consistent with the current survey control.

3.3.4.1.2 Composite Calculation

Drill hole assays were composited on a 6 m (20 ft.) bench basis. Assays, were weighted by length within the bench and specific gravity of the constituent samples. High assay valves were rolled back to the 95th percentile level before compositing. Internal waste (3 m or 10 feet thick or less) was included in the composites at zero grade. Composites were again clipped to the 95th percentile before interpolation. External waste and waste bands greater than 1/2 bench height (3 m or 10 feet) were not included in the composite intervals resulting in a composite shorter than 6 m (20 feet) long. This was done on the premise that waste or ore thicker than half a bench height could be separated during mining. This assumption is of questionable validity and the method of composite calculation will lead to grades that require a higher dilution in order to quote mill feed than a calculation that averages an entire bench regardless of material type. Part of the rationale in this method of compositing is that a given composite will be used on more than one bench thus a composite from the margin of the deposit will be used to estimate the grade of the interior of the deposit and in that case it would not be appropriate to have averaged in a large amount of unmineralized material. All previous Anvil District models have followed exactly the same compositing scheme. Thus this is not an explanation of differences between model reserves.

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

3.2.4.1.3 Interpolation

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

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

Where more than one composite was available to estimate a given block they were weighted isotropically by the inverse square of distance to the point being estimated as well as by the length of the composite. The length weighting of composites was done to avoid biasing large volumes of ore with assays representative of only a small amount of material. In retrospect this proceedure appears to have deweighted the assays from the margins of the deposit relative to those in the core of the deposit. Because of the grade zoning in the deposit this probably has led to an overestimation of grade in marginal situations but has little or no effect in the core of the deposit. This proceedure is different from all previous Faro deposit models except the little known F4 and probably is the reason that the FI model differs from other calculations in marginal benches and has tended to overestimate grade in the JB phase (see below).

The implications of the geological matching requirement during interpolation have not been tested due to lack of time however some statements can be made in light of the rock type - grade correlations and grade zoning described above. Since massive ores tend to be higher grade than disseminated ores and massive ores are more central and higher in the deposit than disseminated ores, there will be a tendency to average grade both by rock type and in space if there is no matching required. The use of matching will tend to make massive ores higher grade and disseminated ores lower grade than would be the case without matching. Because of the geometry of the deposit and its zoning the higher grade ore will be more central and higher in elevation in the case of matching than it would be without matching. When a cutoff is applied to the block values computed through geological code matching the tonnage above cutoff will be lower and the average grade higher than a reserve computed without a matching requirement furthermore the limits of ore will be higher in elevation and closer to the centre of the deposit. Because of these rock type and zoning characteristics of the Faro deposit it is considered essential to use matching to produce an accurate picture of grade distribution. It is however inescapable that a reserve computed through geology matching will require a higher dilution factor than one computed without matching. It is however considered more realistic for dilution calculations to be made after the model is constructed rather than during interpolation when it will occur in uncontrollable and unpredictable fashion.

Specific gravity was treated as an assay and interpolated into blocks. This is due to the variability of SG by rock type and by grade as well as regional variations of SG within one rock type. Sulphide blocks outside of interpolation range were assigned an average specific gravity based on rock type. The SG values used for interpolation were the pulp SG not the whole rock SG thus the value did not reflect porosity in the insitu intact material. A number of comparative SG tests have been done on Anvil District ores to determine the degree of overstatement of SG. In light of the results of these tests block SG values were reduced by 5% for quartzose ore types or 10% for massive ore types to correct for porosity in the insitu whole rock. Previous Faro models have not had this correction made if pulp SG was used (such as the F3 and T3 - these models actually used an average SG derived from the pulp SG data not an interpolated SG). Cyprus Anvil's practice in quoting model results as mill feed predictions was generally to reduce the grade by 5% but not to adjust the tonnage; since the tonnage was already overstated by use of the pulp SG value this was nearly the same as adding 5% dilution. Curragh's approach is to attempt to estimate the insitu tonnage and apply an appropriate dilution factor later rather than attempt to make two sets of corrections at once.

3.2.4.1.4 Reserve Reporting

Reserves were computed for 6 ore types: 2A, 2BCD, 2CE, 2GE, and 2HE. Geological reserve computation was by a weighted average of all blocks in the model below topography that exceed a ceretain arbitrary lead and zinc content. Pit reserves are reported by computing the weighted average of all blocks above cutoff lying between two surfaces gridded on the same block grid as the block model. The two surfaces are an upper surface representing topography or the previous phase bottom and a lower surface representing the current phase bottom. Blocks partly above or below the surface are multiplied by the fraction of the block that is between the surface elevations for that block when computing the weighted average.

3.2.4.2 Methods and Procedure - F8608 Model

In August and September 1986 a new computer model of the AY and

BY phases, the F8608 model, was constructed. This was largely in reaction to the poor performance of the FI model in the JB phase and concerns over the base geologic interpretation in the AY and BY phase areas as well as some of the computational methodology used for the FI model. This model only covered the northwest portion of Zone 3 thus has not been used for long range planning, it is only included here for comparison purposes in order to help quantify the uncertainty of estimation of the FI model on which the planning is based.

3.2.4.2.1 <u>Block Geology and Drillhole Data</u>

The F8608 model is a 3D block model made using PC Mine software. Block size was 10.7 m (35 feet) along the deposit, 7.6 m (25 feet) across the deposit and 6 m (20 feet) high. The coordinate grid was rotated 45° so that rows of blocks are parallel to the geological cross sections of the deposit rather than the mine survey grid.

The geological control used in the model is based on new cross and longitudinal sections for the northwest part of zone 3 completed by R.S. Tolbert in early 1986. The 43 m (141 foot) spaced sections were simplified and intermediate cross sections created at 21.5 m (70.5 foot) intervals (or 10.7 m (35 foot) where required). These sections provided geological control for block geology rather than geological bench plans as have been used for all previous Faro deposit models. This was done not only because it is a quicker process to make cross sections but also because a flat lying stratiform deposit is more logically viewed in section perpendicular to its direction of predictability. The sectional model approach also allows the use of different bench heights without changing the geological bench plans so that bench height optimization could be studied. A drawback to this approach is that <u>if</u> section to section geology has not been closely coordinated and a control section is not provided for each row of blocks then the bench plans of grade distribution and ore type have a patchy appearance with obviously angular contacts parallel to the sections. This is the case with the F8608 model but this problem is largely due to having rushed the model to completion without taking time to refine the base geologic interpretation. The effect on reserves is not thought to be great but the lack of "smoothly flowing" bench plans could cause problems in the mine planning stage.

As in all other models a block is considered homogenous and of one material type. Block coding was based on digitized geological sections with the geology at the centre of the block assigned to the entire block. Assignments were made entirely by machine and checked manually. In most cases 2 rows of blocks were assigned according to the geology of one section since the sections were 21.5 m (70 feet) apart in most cases.

Rock types used were the same as those used for the FI model with the exception of the waste lithologies where several additional units were used (Table 3.4).

TABLE 3.4 LITHOLOGIC CODES USED FOR THE F8608 MODEL

Rock Code	Description	Density Used *
0	air	0
1	undifferentiated sulphides	0.085
2	2A ribbon banded graphitic quartzite	0.083
3	2BCD pyritic quartzite	0.090
4	2CE semi-massive sulphides	0.099
5	2EF pyritic massive sulphides	0.107
6	2GE baritic massive sulphides	0.112
7	2H pyrrhotitic massive sulphides	0.104
8	WASTE schist and phyllite 1D, 1CD and 1C	0.076
9	WASTE calc-silicate 3D	0.076
10	WASTE calc-silicate breccia 3Dbx	0.076
11	WASTE intrusive rocks 10E and 10F	0.076
12	WASTE bleached schist envelope 2L and 1D4	0.076
13	WASTE unconsolidated overburden	0.060
14	WASTE graphitic phyllite 1E and Unit 3A	0.076
20	WASTE in blocks partially above topography	0.076

* Density is in tonnes per cubic foot

Drillhole data used was that already imported into the PC Mine for the FI model and is described above.

3.2.4.2.2 Composites

The composites used were those generated for the FI model since they had geological codes assigned and considerable time was saved by cutting this corner. The length of the composite was not used in PC Mine. A significant drawback to this approach was that the geologic codes of the composites do not correspond exactly with the sectional geology. In general the massive and quartzose ore type distinction is close but subdivisions within massive or quartzose are not necessarily. This necessitated changes to the geological matching scheme during interpolation. Note that problems with non full bench height compositing of external and internal waste still would be present with this composite data set.

3.2.4.2.3 Interpolation

Variogram analysis of Faro composites was not generally sucessful. A tendency of a larger along deposit than across deposit range was indicated. The search volume (and hence anisotropy) used was tailored to be a close approximation to that used for the FI Model with the exception that the search ellipsoid was tilted to follow the layering of the deposit in three domains. First pass interpolation used a seach ellipsoid looking 69 m (225 ft.) along the deposit trend 46 m (150 ft.) across it and 7.6 m (25 ft.) vertically. This was enlarged to 91 m(300 ft.) along the deposit, 61 m (200 ft.) across and 11.3 m (37 ft.) vertically in 3 passes.

Because to FI model composite data set was used geological matching had to be relaxed from that used in the FI model considerably in order to avoid large numbers of uninterpolated blocks. Generally any massive ore type could accept the assay value of any other massive ore type. The distinction between carbonaceous and non-carbonaceous quartzose ore types was dropped. Massive sulphide assays were not allowed to influence quartzose sulphide blocks and vice versa. Semi-massive (2CE) blocks were allowed to accept quartzose and pyritic massive ore type assays.

A minimum of 2 composites was required to interpolate a block; the maximum number of composites allowed was 6. There was no limit possible on the number of composites from a single drillhole but the flat search ellipsoid used precludes more than two.

Composite values were weighted by the inverse square of the distance to the point being estimated. There was no weighting by length of composite and no minimum length of composite stipulated.

A large number of test interpolations were run using different interpolation parameters to check the grade distribution achieved versus the number of blocks that could not be interpolated. The most faithful reproduction of known grade distribution was acomplished by using a flat search ellipsoid so that only composites along the layer being estimated could be used. More spherical search volumes create a false impression of homogenous grade distribution by assigning high grades across strata to areas known to be barren. To have a flat search ellipsoid in PC Mine requires the use of a high degree of vertical anisotropy. The logic of the software treats anisotropy by adjusting distance in different directions. An apparent distance equal to the actual distance divided by the anisotropy factor is calculated and used for search and weighting criteria. This results in every point on the edge of the search ellipsoid being treated as if it were the maximum radius of the ellipsoid away from the point being estimated. Points a small distance off the principal plane of the search ellipsoid are considered further away from those on the plane in cases of a high vertical factor. Test interpolations and trace blocks run during the tests did not reveal any problems arising from this treatment of distance but this is one of the major differences between the F8608 and FI model methodologies.

Specific gravity is treated as an assay and interpolated into blocks. This is due to the variability of SG by rock type and by grade as well as regional variations of SG within one rock type. Prior to interpolation the composite pulp SG's were reduced by 5% for quartzose ore types or 10% for massive ore types to correct for porosity in the insitu whole rock. Uninterpolated blocks were assigned the density 0.096 tonnes/cu. ft. (see Table 3.4)

3.2.4.2.4 <u>Reserve reporting</u>

Geological reserves were not computed since the model only covers the part of the deposit between sections 117 and 125. Pit reserves are reported by computing the weighted average of all blocks lying between two surfaces gridded on the same block grid as the block model. The two surfaces are an upper surface representing topography or the previous phase bottom and a lower surface representing the current phase bottom. Blocks partly above or below the surface are multiplied by the fraction of the block that is between the surface elevations for that block.

3.2.4.3 <u>Results</u>

3.2.4.3.1 <u>Geological Reserves</u>

Geological reserves calculated from the FI model are given in Table 3.5 along with some previous figures of a comparable nature. The Dome hand calculation covers a larger area than the FI model thus cannot be compared directly. Since nearest neighbor sectional calculations such as the Dome one tend to report higher grades than inverse distance squared interpolated models the difference in grade may not be significant.

The 1985 Kilborn Report gives a rather large geological reserve with an unstated cutoff grade or source. The source is presumably the Cyprus Anvil F3 model, but no printouts could be found to confirm it. It is not clear how this number can be consistent with the Dome figure let alone the FI model. The authors predjudice is to favor the Dome sectional calculation reserve as the geological reserve for the Faro deposit since it covers a larger area than the current model and is well documented.

TABLE 3.5 GEOLOGIC RESERVES FOR THE FARO DEPOSIT-ZONE THREE
all values are undiluted and unadjusted

grade category	ore (tonnes)	Pb (%)	Zn (%)	Pb+Zn (%)	Ag (g/t)
FI COMPUTER MODEL					
+5%Pb+Zn	22,793,000	3.28	5.14	8.41	41.5
4-5%Pb+Zn	3,770,000	1.72	2.78	4.50	27.3
+4%Pb+Zn	26,563,000	3.06	4.81	7.86	39.5
total metal at +4% Pb + Zn cutoff= 2,088,000 tonnes					
DOME SECTIONAL HAND CALCULATION					
+4%Pb+Zn	29,251,000	3.13	5.03	8.16	40.8
total metal at +4% Pb + Zn cutoff= 2,387,000 tonnes					
CYPRUS ANVIL (F3 COMPUTER MODEL ?)					
+4%(?)Pb+Zn	33,000,000	3.0	4.6	7.6	35.7
total metal at +4%(?) Pb + Zn cutoff= 2,508,000 tonnes					

3.2.4.3.2 Model To Model Comparisons

Table 3.6 compares the FI model (actually the reserves computed before the SG reduction so that the numbers are comparable) with three computer calculated valves for phase A (for old phase A not the current AY) all based on the same geology but varying in computational methodology and in the case of FI for the assay database. Also shown is a hand calculated reserve for phase A based on a geologic interpretation done by the author in September 1985 incorporating all drilling data available and using a fault dominant as opposed to fold dominant geological interpretation. In this phase the FI model reports a lower tonnage at a higher grade that previous models. This is likely due to more restrictive application of geology matching during interpolation due to the greater availability of composites and the more rigorous coding but may be partly due to length weighting of composites during interpolation.

The comparison of hand calculated reserves using new geology to FI reserves is the most critical as it deals with estimates derived from very different approaches. As shown on Table 3.6, the FI model reports 9% higher tonnage than the hand model at 5% lower grade. Much of the grade reduction may be due to the comparison of a nearest neighbor to an inverse squared distance interpolation but at the worst this comparison suggests the reserves compare within 10% and within 4% on total metal.

The other phases do not compare as favorably as the A phase. The comparison of the FI, T3 and F3 models for the old A through D phases of the Faro pit is shown on table 3.7. In every phase the FI model reserve contains fewer tonnes but higher grade. The grade increase in the B to D phases is not however large enough to compensate for the drop in tonnage and there is a drop in total contained metal ranging from 7% to 10% averaging 6% for the entire pit. The reasons for this drop in total metal and the comparable drop between the F3 and T3 models is not totally clear; the most likely explanation is the restrictions on interpolation caused by the requirement for matching geology codes. The most direct test of this inference would be to reinterpolate the FI model without geology matching however this has not been done. A clue to what is happpening is found in the relative proportions of ore types above cutoff. One would expect that the grades being assigned to disseminated ore types without good geologic control would be on the average too high, consequently too much would be considered ore at a given cutoff; the converse would be expected for the massive ore types. This is the trend shown in Table 3.8.

TABLE 3.6 COMPARISON OF SEVERAL RESERVE ESTIMATES FOR CYPRUS ANVILS A PHASE

DATE	MODEL	TONNES +6% Pb+Zn	Pb (%)	Zn (%)	Ag (g/t)	Pb+Zn (%)	TOTAL METAL (tonnes)
1985	HAND	3,290,317	3.95	5.98	49	9.93	326,728
1985	F1	3,595,315	3.83	5.62	48	9.45	339,757
1982	T3	3,668,154	3.63	5.37	42	9.00	330,281
1981	F3	4,051,087	3.61	5.33	43	8.93	361,884

TABLE 3.7 COMPARISON OF THREE COMPUTER BASED MINE MODEL RESERVES FOR CYPRUS ANVIL'S A-D PIT DESIGN, FARO ZONE 3
(all values unadjusted except where noted otherwise)

**************************PRELIMINARY************************************

		TONNES +6% ORE	Pb (%)	Zn (%)	Ag (g/t)	Pb+Zn (%)	TOTAL METAL (tonnes)	UNINTERPOLATED BLOCKS (tonnes)
PHASE A								
FI		3,595,315	3.83	5.62	48	9.45	339,757	
T3		3,668,154	3.63	5.37	42	9.00	330,281	85,951
F3		4,051,087	3.61	5.33	43	8.93	361,884	43,996
PHASE B								
FI		4,457,843	3.78	5.32	48	9.10	405,664	
T3		5,129,659	3.69	5.12	45	8.81	451,923	43,775
F3		5,201,360	3.62	5.11	44	8.73	454,079	23,928
PHASE C								
FI		3,681,852	3.60	5.33	48	8.93	328,789	
T3		4,127,797	3.56	4.98	44	8.54	352,308	162,654
F3		4,694,443	3.42	4.92	43	8.34	391,704	26,797
PHASE D (includes JB phase)								
FI		3,906,211	3.38	5.57	40	8.95	349,606	
T3		4,510,291	3.21	5.30	36	8.51	383,690	252,307
F3		5,036,489	3.13	5.17	36	8.29	417,676	31,100
TOTAL A-D PIT								
FI		15,641,221	3.65	5.45	46	9.10	1,423,816	0
T3	*	17,435,901	3.52	5.19	42	8.71	1,518,201	544,687
F3		18,983,378	3.44	5.13	41	8.56	1,625,343	125,821
TOTAL A-D PIT RESERVES EXPRESSED AS EXPECTED MILLFEED (adjustments per users customary practice, see below)								
FI	#	16,345,076	3.32	4.96	42	8.28	1,352,625	
T3	@	17,435,901	3.35	4.93	40	8.27	1,442,291	
F3	@	18,983,378	3.26	4.87	39	8.13	1,544,075	

* compared to 17,180,000 quoted by Kilborn in 1985 report
F3 and T3 numbers in this table taken directly off Mintec printouts.

\# 95% mining recovery and 10% dilution at zero grade

@ minus 5% to grades, no change to tonnage

Table 3.8 PROPORTIONS OF VARIOUS ORE TYPES ABOVE CUTOFF IN THE T3 AND FI MODELS

ore type	percent of total +6% ore FI model	T3 model
A	3.9	3.4
BCD	12.3	19.9
CE	8.2	10.2
EF	51.4	49.5
GE	9.6	8.7
HE	14.5	8.3

Table 3.9 compares the expanded JB phase (again not exactly the same as the current JB phase) as calculated by the FI model and with a calculation done by Cyprus Anvil using the same assay data and geologic interpretation. Cyprus Anvil's approach was to compute the actual area of geologic units on the benches and make the most appropriate assay assignment to these areas by manual means. This comparison thus addresses the question of how adequate the block representation of the geology is and how the machine algorithms and computations compare human reasoning and manual computations. The comparison is good; the major difference being in tonnage which is probably at least in part due to the inability of 15 m (50 ft.) X 15 m (50 ft.) blocks to show every geologic unit.

The results from the FI and F8608 models the AY and BY Phases are compared in table 3.10 A to D. The F8608 computed tonnages are very close to those of the FI model or slightly higher. The grades are fairly consistently lower. The total metal for both phases is within 2.2% of that calculated by the FI model. Despite the close comparison on a large volume basis, the bench to bench variance is quite large with many benches being within only ±30% (see Table 3.10 B and D). As might be expected the larger benches in the core of the deposit compare well.

3.2.4.3.3 Comparison To Blasthole Results

The acid test of a model is to compare to actual production data. The FI model was not designed to accurately predict small domains but was intended to acheive some degree of accuracy when dealing with at best quarterly production. The model has not fared well by comparison to blasthole results. This has been traced back to two definite problems a) very high dilution by low grade sulphides caused by high grade bands that rarely occupy a full bench height and b) incorrect DDH locations; and a third possible problem, c) the length weighting of composites during interpolation.

The actual comparison of model results to JB phase blastholes is detailed in table 3.11. The table gives both the raw model and diluted model results, the 5% mining loss is not taken for this comparison since both estimates refer to the resource in the ground. The model predicts more metal than was actually blocked out; 10% more for high grade ore (combination of some +6% and some +5%) and 4% more at a 4% cutoff. The model over predicted high grade tonnage by 6.5% and grade by 5.3%. At a 4% Pb + Zn cutoff the model underpredicted tonnage by 4.4% and overpredicted grade by 7.7%. Despite these fairly close results for two quarters production the bench by bench comparison is rather poor. In the upper benches the model grossly overpredicted tonnage and under predicted grade; in the lower benches the converse was true. The upper benches model complex fault bounded slices in the Big Indian Fault Zone; this area was expected to prove to be difficult to estimate because the geology was difficult to define using the exploration drillholes. On the 3850 bench the model performed worse than

TABLE 3.9 COMPARISON OF FI MODEL TO CYPRUS ANVIL'S HAND CALCULATION FOR THE CYPRUS ANVIL JB EXTENDED PHASE

DATE	MODEL	TONNES +6% Pb+Zn	Pb (%)	Zn (%)	Ag (g/t)	Pb+Zn (%)	TOTAL METAL (tonnes)
1985	FI	816,017	3.81	6.19	50	10.00	81,602
1984	CAMC HAND	858,887	3.80	6.10	52	9.90	85,030

TABLE 3.10 A

CURRAGH RESOURCES
Comparison of FI and F8608 models - FOR AY PHASE ONLY — September 24, 1986

NO ADJUSTMENTS FOR DILUTION OR MINING LOSS

****************************NEW MODEL (F8608)********************************

BENCH NUMBER	TOE ELEV.	TONNES +6% ORE	Pb (%)	Zn (%)	Ag (g/t)	Pb + Zn (%)	LEAD METAL (tonnes)	ZINC METAL (tonnes)	SILVER METAL (grams)	TOTAL METAL (tonnes)
17	3750	3,840	3.97	5.87	79.04	9.84	152	225	303521	378
18	3730	33,320	4.15	5.13	71.75	9.28	1382	1710	2390743	3092
19	3710	49,290	3.81	5.00	54.99	8.81	1876	2464	2710654	4340
20	3690	77,620	3.42	5.66	45.75	9.07	2651	4390	3551115	7041
21	3670	51,530	3.20	5.15	38.07	8.35	1648	2653	1961953	4302
22	3650	43,590	3.14	4.16	37.53	7.30	1367	1813	1636020	3180
23	3630	76,680	3.16	4.26	37.33	7.42	2420	3266	2862771	5686
24	3610	141,460	3.13	4.88	32.90	8.01	4426	6909	4654458	11335
25	3590	121,260	2.91	4.74	31.34	7.65	3524	5749	3799682	9273
26	3570	79,820	2.91	4.84	36.36	7.74	2322	3859	2902574	6181
27	3550	205,690	3.25	5.09	44.78	8.34	6681	10466	9211004	17146
28	3530	365,850	3.61	5.27	47.45	8.88	13211	19291	17358485	32502
29	3510	499,220	3.56	5.27	44.84	8.83	17762	26299	22386522	44061
30	3490	469,190	3.87	5.41	49.20	9.28	18158	25369	23082271	43527
31	3470	361,350	3.76	5.28	45.78	9.03	13572	19068	16541158	32641
32	3450	314,490	3.17	4.67	43.67	7.85	9982	14699	13734722	24681
33	3430	253,690	3.72	5.72	45.15	9.43	9427	14498	11454357	23926
34	3410	42,300	2.79	4.27	25.95	7.06	1181	1806	1097474	2987
35	3390	16,880	2.97	4.14	23.57	7.11	501	699	397929	1200
TOTALS/AVERAGES:		3,207,070	3.50	5.15	44.29	8.65	112244	165235	142037415	277479

****************************OLD MODEL (FI)**

BENCH NUMBER	TOE ELEV.	TONNES +6% ORE	Pb (%)	Zn (%)	Ag (g/t)	Pb + Zn (%)	LEAD METAL (tonnes)	ZINC METAL (tonnes)	SILVER METAL (grams)	TOTAL METAL (tonnes)
17	3750	3,110	6.50	5.94	105.00	12.44	202	185	326550	387
18	3730	10,680	3.86	4.73	59.64	8.58	412	505	636923	917
19	3710	62,910	3.14	4.61	46.73	7.74	1972	2900	2939784	4872
20	3690	53,460	3.31	5.11	45.41	8.42	1770	2731	2427565	4501
21	3670	43,050	3.98	5.79	48.25	9.77	1714	2493	2077033	4207
22	3650	47,850	4.22	5.02	48.87	9.24	2021	2400	2338573	4421
23	3630	71,320	4.18	5.49	49.99	9.66	2980	3912	3565501	6892
24	3610	150,670	3.24	5.68	30.47	8.92	4885	8557	4591518	13441
25	3590	84,600	2.89	5.35	27.01	8.24	2441	4528	2284792	6969
26	3570	81,450	3.71	5.79	48.87	9.50	3019	4719	3980706	7738
27	3550	260,620	3.70	5.59	53.89	9.29	9640	14574	14043509	24214
28	3530	417,040	3.89	5.72	48.30	9.61	16240	23855	20142198	40094
29	3510	566,280	3.98	5.77	47.88	9.75	22555	32674	27111788	55229
30	3490	426,680	4.27	6.00	52.05	10.27	18211	25609	22209121	43820
31	3470	375,390	3.85	5.29	44.89	9.14	14441	19854	16852008	34296
32	3450	295,120	3.38	4.91	48.57	8.29	9975	14490	14334569	24465
33	3430	212,300	3.69	5.75	45.71	9.44	7828	12207	9703808	20035
34	3410	32,740	2.79	4.35	24.31	7.14	912	1425	795942	2337
35	3390	17,630	2.85	4.23	22.71	7.08	503	745	400289	1247
		3,212,900	3.79	5.55	46.92	9.34	121,719	178,363	150,762,176	300,082

TABLE 3.10 B

Variance between the F8608 and FI models for the A1 phase

PERCENT VARIANCES ((NEW-OLD)/OLD)

TONNES +6% ORE	Pb (%)	Zn (%)	Ag (%)	Pb + Zn (%)	LEAD METAL (%)	ZINC METAL (%)	SILVER METAL (%)	TOTAL METAL (%)	BENCH NUMBER	TOE ELEV.
23.5%	-38.9%	-1.2%	-24.7%	-20.9%	-24.6%	22.0%	-7.1%	-2.3%	17	3750
212.0%	7.6%	8.5%	20.3%	8.1%	235.8%	238.6%	275.4%	237.4%	18	3730
-21.6%	21.4%	8.5%	17.7%	13.7%	-4.9%	-15.0%	-7.8%	-10.9%	19	3710
45.2%	3.1%	10.7%	0.8%	7.7%	49.8%	60.7%	46.3%	56.4%	20	3690
19.7%	-19.7%	-11.1%	-21.1%	-14.6%	-3.8%	6.4%	-5.5%	2.2%	21	3670
-8.9%	-25.8%	-17.0%	-23.2%	-21.0%	-32.4%	-24.4%	-30.0%	-28.1%	22	3650
7.5%	-24.5%	-22.4%	-25.3%	-23.3%	-18.8%	-16.5%	-19.7%	-17.5%	23	3630
-6.1%	-3.5%	-14.0%	8.0%	-10.2%	-9.4%	-19.3%	1.4%	-15.7%	24	3610
43.3%	0.7%	-11.4%	16.0%	-7.2%	44.4%	27.0%	66.3%	33.1%	25	3590
-2.0%	-21.5%	-16.6%	-25.6%	-18.5%	-23.1%	-18.2%	-27.1%	-20.1%	26	3570
-21.1%	-12.2%	-9.0%	-16.9%	-10.3%	-30.7%	-28.2%	-34.4%	-29.2%	27	3550
-12.3%	-7.3%	-7.8%	-1.8%	-7.6%	-18.7%	-19.1%	-13.8%	-18.9%	28	3530
-11.8%	-10.7%	-8.7%	-6.3%	-9.5%	-21.2%	-19.5%	-17.4%	-20.2%	29	3510
10.0%	-9.3%	-9.9%	-5.5%	-9.7%	-0.3%	-0.9%	3.9%	-0.7%	30	3490
-3.7%	-2.4%	-0.2%	2.0%	-1.1%	-6.0%	-4.0%	-1.8%	-4.8%	31	3470
6.6%	-6.1%	-4.8%	-10.1%	-5.3%	0.1%	1.4%	-4.2%	0.9%	32	3450
19.5%	0.8%	-0.6%	-1.2%	-0.1%	20.4%	18.8%	18.0%	19.4%	33	3430
29.2%	0.2%	-1.9%	6.7%	-1.1%	29.5%	26.8%	37.9%	27.8%	34	3410
-4.3%	4.0%	-2.0%	3.8%	0.4%	-0.4%	-6.2%	-0.6%	-3.8%	35	3390
-0.2%	-7.6%	-7.2%	-5.6%	-7.4%	-7.8%	-7.4%	-5.8%	-7.5%	AVERAGE VARIANCE	

ACTUAL VARIANCES ((NEW-OLD)/OLD)

TONNES +6% ORE	Pb (%)	Zn (%)	Ag (%)	Pb + Zn (%)	LEAD METAL (tonnes)	ZINC METAL (tonnes)	SILVER METAL (ounces)	TOTAL METAL (tonnes)	BENCH NUMBER	TOE ELEV.
730	-2.53	-0.07	-25.96	-2.60	-50	41	-672	-9	17	3750
22,640	0.29	0.40	12.11	0.70	971	1205	51212	2176		
(13,620)	0.67	0.39	8.26	1.06	-96	-436	-6691	-531		
24,160	0.10	0.55	0.34	0.65	881	1659	32808	2540		
8,480	-0.78	-0.64	-10.17	-1.43	-66	160	-3360	94		
(4,260)	-1.09	-0.85	-11.34	-1.94	-654	-586	-20515	-1241		
5,360	-1.02	-1.23	-12.66	-2.25	-560	-646	-20520	-1206		
(9,210)	-0.11	-0.80	2.43	-0.91	-458	-1648	1838	-2106		
36,660	0.02	-0.61	4.33	-0.59	1083	1221	44235	2304		
(1,630)	-0.80	-0.96	-12.51	-1.76	-697	-860	-31481	-1556		
(54,930)	-0.45	-0.50	-9.10	-0.96	-2960	-4108	-141109	-7068		
(51,190)	-0.28	-0.45	-8.85	-0.73	-3029	-4563	-81284	-7592		
(67,060)	-0.43	-0.50	-3.03	-0.93	-4793	-6375	-137978	-11168		
42,510	-0.40	-0.59	-2.86	-0.99	-53	-240	25496	-293		
(14,040)	-0.09	-0.01	0.88	-0.10	-869	-786	-9077	-1655		
19,370	-0.21	-0.24	-4.90	-0.44	7	209	-17516	216		
41,390	0.03	-0.04	-0.56	-0.01	1600	2291	51116	3891		
9,560	0.01	-0.08	1.63	-0.08	269	381	8805	650		
(750)	0.12	-0.08	0.07	0.03	-2	-46	-69	-48		
(5,830)	-0.29	-0.40	-2.64	-0.69	-9475	-13127	-254763	-22603		

TABLE 3.10 C

CURRAGH RESOURCES
Comparison of FI and FB608 models - FOR BY PHASE ONLY

September 30, 1986

NO ADJUSTMENTS FOR DILUTION OR MINING LOSS

********************************NEW MODEL (FB608)********************************

BENCH NUMBER	TOE ELEV.	TONNES +6% ORE	Pb (%)	Zn (%)	Ag (g/t)	Pb + Zn (%)	LEAD METAL (tonnes)	ZINC METAL (tonnes)	SILVER METAL (grams)	TOTAL METAL (tonnes)
20	3690	2,770	2.55	6.17	25.05	8.72	71	171	69383	242
21	3670	11,930	3.14	7.75	27.81	10.89	374	925	331797	1299
22	3650	0	0.00	0.00	0.00	0.00	0	0	0	0
23	3630	0	0.00	0.00	0.00	0.00	0	0	0	0
24	3610	3,650	3.78	5.42	47.62	9.19	138	198	173809	336
25	3590	51,360	3.79	5.76	51.14	9.55	1948	2958	2626756	4906
26	3570	120,300	3.74	4.83	54.03	8.57	4499	5809	6499328	10309
27	3550	176,490	3.93	4.76	56.83	8.69	6931	8403	10029927	15333
28	3530	252,270	3.52	4.62	54.13	8.14	8877	11660	13655880	20537
29	3510	319,950	4.26	5.60	63.46	9.86	13636	17908	20304987	31544
30	3490	471,560	4.03	5.55	55.40	9.59	19018	26181	26125839	45199
31	3470	745,720	4.24	5.83	56.23	10.07	31633	43461	41929598	75094
32	3450	560,290	4.06	5.54	49.38	9.59	22737	31012	27666000	53749
33	3430	529,670	3.61	5.22	40.87	8.83	19116	27638	21648143	46754
34	3410	679,850	3.13	4.84	31.59	7.97	21273	32884	21476462	54157
35	3390	428,320	3.02	4.65	29.36	7.68	12948	19934	12576332	32882
36	3370	305,530	2.96	4.89	26.96	7.85	9053	14943	8236172	23996
37	3350	109,550	2.78	5.26	29.86	8.03	3040	5761	3270725	8801
38	3330	14,410	2.62	5.20	43.80	7.82	377	750	631086	1127
TOTALS/AVERAGES:		4,783,620	3.67	5.24	45.42	8.91	175668	250596	217252222	426264

********************************OLD MODEL (FI)********************************

BENCH NUMBER	TOE ELEV.	TONNES +6% ORE	Pb (%)	Zn (%)	Ag (g/t)	Pb + Zn (%)	LEAD METAL (tonnes)	ZINC METAL (tonnes)	SILVER METAL (grams)	TOTAL METAL (tonnes)
20	3690	0	0.00	0.00	0.00	0.00	0	0	0	0
21	3670	0	0.00	0.00	0.00	0.00	0	0	0	0
22	3650	2,360	6.18	7.81	61.00	13.99	146	184	143960	330
23	3630	0	0.00	0.00	0.00	0.00	0	0	0	0
24	3610	0	0.00	0.00	0.00	0.00	0	0	0	0
25	3590	1,420	3.59	4.44	50.00	8.03	51	63	71000	114
26	3570	105,810	4.32	5.41	54.45	9.73	4574	5725	5761566	10300
27	3550	181,470	4.31	4.73	63.08	9.04	7816	8584	11446220	16399
28	3530	267,200	4.40	5.18	69.03	9.58	11757	13833	18444549	25590
29	3510	272,490	4.18	5.47	66.52	9.65	11390	14905	18126307	26295
30	3490	471,420	4.32	5.77	61.81	10.09	20342	27220	29137999	47562
31	3470	751,690	4.13	5.70	56.51	9.83	31045	42846	42474243	73891
32	3450	489,640	4.13	5.71	55.03	9.84	20207	27973	26945379	48181
33	3430	503,770	3.68	5.34	39.35	9.02	18534	26896	19824861	45430
34	3410	706,020	3.30	5.06	34.35	8.36	23299	35746	24252493	59044
35	3390	450,190	3.07	4.38	36.45	7.45	13816	19736	16408075	33553
36	3370	243,000	2.96	4.79	31.70	7.76	7203	11642	7702128	18845
37	3350	148,630	2.88	5.28	34.04	8.17	4286	7849	5059811	12136
38	3330	28,830	2.61	4.92	39.32	7.52	751	1417	1133624	2168
		4,623,940	3.79	5.29	49.08	9.08	175,217	244,620	226,932,216	419,837

TABLE 3.10 D

Variance between the FB608 and FI models for the BY phase

PERCENT VARIANCES ((NEW-OLD)/OLD)

TONNES +6% ORE	Pb (%)	Zn (%)	Ag (%)	Pb + Zn (%)	LEAD METAL (%)	ZINC METAL (%)	SILVER METAL (%)	TOTAL METAL (%)	BENCH NUMBER	TOE ELEV.
ERR	ERR	ERR	ERR	ERR	ERR	ERR	ERR	ERR	20	3690
ERR	ERR	ERR	ERR	ERR	ERR	ERR	ERR	ERR	21	3670
-100.0%	-100.0%	-100.0%	-100.0%	-100.0%	-100.0%	-100.0%	-100.0%	-100.0%	22	3650
ERR	ERR	ERR	ERR	ERR	ERR	ERR	ERR	ERR	23	3630
ERR	ERR	ERR	ERR	ERR	ERR	ERR	ERR	ERR	24	3610
3516.9%	5.6%	29.7%	2.3%	19.0%	3720.4%	4592.2%	3599.7%	4202.4%	25	3590
13.7%	-13.5%	-10.8%	-0.8%	-12.0%	-1.6%	1.5%	12.8%	0.1%	26	3570
-2.7%	-8.8%	0.7%	-9.9%	-3.9%	-11.3%	-2.1%	-12.4%	-6.5%	27	3550
-5.6%	-20.0%	-10.7%	-21.6%	-15.0%	-24.5%	-15.7%	-26.0%	-19.7%	28	3530
17.4%	2.0%	2.3%	-4.6%	2.2%	19.7%	20.1%	12.0%	20.0%	29	3510
0.0%	-6.5%	-3.8%	-10.4%	-5.0%	-6.5%	-3.8%	-10.3%	-5.0%	30	3490
-0.8%	2.7%	2.2%	-0.5%	2.4%	1.9%	1.4%	-1.3%	1.6%	31	3470
14.4%	-1.7%	-3.1%	-10.3%	-2.5%	12.5%	10.9%	2.7%	11.6%	32	3450
5.1%	-1.9%	-2.3%	3.9%	-2.1%	3.1%	2.8%	9.2%	2.9%	33	3430
-3.7%	-5.2%	-4.5%	-8.0%	-4.7%	-8.7%	-8.0%	-11.4%	-8.3%	34	3410
-4.9%	-1.5%	6.7%	-19.4%	3.0%	-6.3%	1.0%	-23.4%	-2.0%	35	3390
25.7%	0.0%	2.1%	-15.0%	1.3%	25.7%	28.4%	6.9%	27.3%	36	3370
-26.3%	-3.8%	-0.4%	-12.3%	-1.6%	-29.1%	-26.6%	-35.4%	-27.5%	37	3350
-50.0%	0.4%	5.9%	11.4%	4.0%	-49.8%	-47.1%	-44.3%	-48.0%	38	3330
3.5%	-3.1%	-1.0%	-7.5%	-1.9%	0.3%	2.4%	-4.3%	1.5%	AVERAGE VARIANCE	

ACTUAL VARIANCES ((NEW-OLD)/OLD)

TONNES +6% ORE	Pb (%)	Zn (%)	Ag (%)	Pb + Zn (%)	LEAD METAL (tonnes)	ZINC METAL (tonnes)	SILVER METAL (grams)	TOTAL METAL (tonnes)	BENCH NUMBER	TOE ELEV.
2,770	2.55	6.17	25.05	8.72	71	171	2026	242	20	3690
11,930	3.14	7.75	27.81	10.89	374	925	9688	1299	21	3670
(2,360)	-6.18	-7.81	-61.00	-13.99	-146	-184	-4204	-330	22	3650
0	0.00	0.00	0.00	0.00	0	0	0	0	23	3630
3,650	3.78	5.42	47.62	9.19	138	198	5075	336	24	3610
49,940	0.20	1.32	1.14	1.52	1897	2895	74628	4792	25	3590
14,490	-0.58	-0.58	-0.43	-1.16	-75	84	21543	9	26	3570
(4,980)	-0.38	0.03	-6.25	-0.35	-885	-181	-41356	-1066	27	3550
(14,930)	-0.88	-0.55	-14.90	-1.44	-2879	-2173	-139829	-5052	28	3530
47,460	0.08	0.13	-3.06	0.21	2246	3002	63617	5249	29	3510
140	-0.28	-0.22	-6.41	-0.50	-1324	-1039	-87955	-2363	30	3490
(5,970)	0.11	0.13	-0.28	0.24	589	614	-15904	1203	31	3470
70,650	-0.07	-0.18	-5.65	-0.25	2529	3039	21042	5568	32	3450
25,900	-0.07	-0.12	1.52	-0.19	582	742	53240	1324	33	3430
(26,170)	-0.17	-0.23	-2.76	-0.40	-2026	-2861	-81060	-4888	34	3410
(21,870)	-0.05	0.27	-7.09	0.22	-868	198	-111887	-671	35	3390
62,530	0.00	0.10	-4.74	0.10	1850	3301	15594	5152	36	3370
(39,080)	-0.11	-0.02	-4.19	-0.13	-1246	-2088	-52241	-3334	37	3350
(14,420)	0.01	0.29	4.47	0.30	-374	-667	-14674	-1041	38	3330
159,600	-0.12	-0.05	-3.66	-0.17	432	5975	-282656	6427	AVERAGE VARIANCE	

TABLE 3.11 COMPARISON OF FI MODEL PREDICTED BENCH RESERVES WITH 1986 MINE PRODUCTION IN JB PHASE AS BLOCKED OUT BY BLASTHOLES

bench	high grade tonnes	Pb (%)	Zn (%)	Ag (g/t)	Pb+Zn (%)	tonnes total metal	low grade tonnes	Pb (%)	Zn (%)	Ag (g/t)	Pb+Zn (%)	tonnes total metal	all grades (+4%) tonnes	Pb (%)	Zn (%)	Ag (g/t)	Pb+Zn (%)	tonnes total metal
BLASTHOLE RESULTS																		
* 3890	44,180	3.10	4.70		7.80	3,446	0				0.00	0	44,180	3.10	4.70	0	7.80	3,446
* 3870	20,378	3.30	5.30		8.60	1,753	9,333	1.61	3.05		4.66	388	28,711	2.81	4.65	0	7.46	2,141
* 3850	29,549	2.41	3.83		6.24	1,844	7,140	1.42	2.94		4.36	311	36,689	2.22	3.66	0	5.87	2,155
* 3830	81,920	2.79	4.51		7.30	5,980	19,627	2.12	2.45		4.57	897	101,547	2.66	4.11	0	6.77	6,877
* 3810	99,840	3.19	4.41		7.60	7,568	20,480	1.89	2.51		4.40	901	120,320	2.97	4.09	0	7.06	8,489
3790	184,320	3.44	4.94		8.38	15,446	5,760	1.94	2.66		4.60	265	190.080	3.39	4.87	0	8.27	15.711
3770	217,600	2.73	4.43	35	7.16	15,580	34,560	1.69	2.69	24	4.38	1,514	252,160	2.59	4.19	33	6.78	17,094
3750	208,249	3.11	4.60	43	7.71	15,656	65,813	1.78	2.98	27	4.76	3,133	274,062	2.79	4.21	39	7.00	19,189
3730	261,111	2.99	4.66	42	7.65	19,975	82,667	2.00	3.34	34	5.34	4,414	343,778	2.75	4.34	40	7.09	24,389
3710	115,662	2.95	4.60	38	7.55	8,737	49,378	1.73	3.07	32	4.80	2,370	165,040	2.58	4.14	36	6.73	11,103
TOTAL	1,262,809	3.03	4.61		7.64	96,479	293,758	1.84	2.99		4.83	14,189	1,556,567	2.81	4.30		7.11	110,667
MODEL PREDICTIONS (NO DILUTION)																		
3890	28560	3.71	5.01	54	8.72	2,490	0					0	28,560	3.71	5.01	54	8.72	2,490
3870	10050	3.17	4.78	46	7.95	799	4500	2.37	3.18	37	5.55	250	14,550	2.92	4.29	43	7.21	1,049
3850	45500	4.33	7.64	52	11.97	5,446	8950	1.85	3.18	31	5.03	450	54,450	3.92	6.91	49	10.83	5,897
3830	76000	3.98	6.55	51	10.53	8,003	8250	2.06	2.87	32	4.93	407	84,250	3.79	6.19	49	9.98	8,410
3810	83390	3.92	6.32	52	10.24	8,539	22970	1.68	2.94	35	4.62	1,061	106,360	3.44	5.59	48	9.03	9,600
3790	115150	4.37	7.28	54	11.65	13,415	13950	1.36	3.2	28	4.56	636	129,100	4.04	6.84	51	10.88	14,051
3770	184900	3.42	5.61	49	9.03	16,696	16550	1.06	3.31	22	4.37	723	201,450	3.23	5.42	47	8.65	17,420
3750	222400	3.14	4.98	44	8.12	18,059	25200	1.55	3.06	36	4.61	1,162	247,600	2.98	4.78	43	7.76	19,221
3730	233320	3.05	5.07	37	8.12	18,946	12290	1.52	3.14	30	4.66	573	245,610	2.97	4.97	37	7.95	19,518
3710	228710	2.71	4.57	35	7.28	16,650	14710	2.23	2.51	36	4.74	697	243,420	2.68	4.45	35	7.13	17,347
TOTAL	1,227,980	3.36	5.52	44	8.88	109,045	127,370	1.65	3.03	32	4.68	5,961	1,355,350	3.20	5.29	43	8.49	115,006

MODEL PREDICTIONS DILUTED

100 PERCENT MINING RECOVERY
10 PERCENT DILUTION
0 LEAD GRADE (%) OF DILUTANT
0 ZINC GRADE (%) OF DILUTANT
0 LEAD + ZINC GRADE (%) OF DILUTANT
0 SILVER GRADE (g/t) OF DILUTANT

bench	high grade tonnes	Pb (%)	Zn (%)	Ag (g/t)	Pb+Zn (%)	tonnes total metal	low grade tonnes	Pb (%)	Zn (%)	Ag (g/t)	Pb+Zn (%)	tonnes total metal	all grades (+4%) tonnes	Pb (%)	Zn (%)	Ag (g/t)	Pb+Zn (%)	tonnes total metal
3890	31,416	3.37	4.55	49	7.93	2,490						0	31,416	3.37	4.55	49	7.93	2,490
3870	11,055	2.88	4.35	42	7.23	799	4,950	2.15	2.89	34	5.05	250	16,005	2.66	3.90	39	6.55	1,049
3850	50,050	3.94	6.95	47	10.88	5,446	9,845	1.68	2.89	28	4.57	450	59,895	3.57	6.28	44	9.84	5,897
3830	83,600	3.62	5.95	46	9.57	8,003	9,075	1.87	2.61	29	4.48	407	92,675	3.45	5.63	45	9.07	8,410
3810	91,729	3.56	5.75	47	9.31	8,539	25,267	1.53	2.67	32	4.20	1,061	116,996	3.12	5.08	44	8.21	9,600
3790	126,665	3.97	6.62	49	10.59	13,415	15,345	1.24	2.91	25	4.15	636	142,010	3.68	6.22	47	9.89	14,051
3770	203,390	3.11	5.10	45	8.21	16,696	18,205	0.96	3.01	20	3.97	723	221,595	2.93	4.93	43	7.86	17,420
3750	244,640	2.85	4.53	40	7.38	18,059	27,720	1.41	2.78	33	4.19	1,162	272,360	2.71	4.35	39	7.06	19,221
3730	256,652	2.77	4.61	34	7.38	18,946	13,519	1.38	2.85	27	4.24	573	270,171	2.70	4.52	33	7.22	19,518
3710	251,581	2.46	4.15	32	6.62	16,650	16,181	2.03	2.28	33	4.31	697	267,762	2.44	4.04	32	6.48	17,347
TOTAL	1,350,778	3.05	5.02	40	8.07	109,045	140,107	1.50	2.75	29	4.25	5,961	1,490,885	2.91	4.81	39	7.71	115,006

* High grade is +6% Pb+Zn and low grade is 4-6%; on all other benches high grade is +5% and low grade is 4-5%

usual; this has been traced back to a drillhole which is now known to be in the wrong place by comparison of pit geology to the drillhole results. This errant drillhole caused the width of a fault bounded panel to be twice what it actually was with a corresponding overestimate of tonnage. The lower benches model larger parts of the deposit in areas less affected by faulting and the model produces better results there. The last bench is starting to show the effect of another drillhole in the wrong place causing the elevation of the base of the ore zone to be estimated lower than it actually is with a resulting shortfall of tonnage. The cause of the generally greater overprediction for zinc compared to lead is not understood.

Table 3.12 shows a similar comparison for the FI model and the blasthole results in the AY phase. The tendency to underpredict the tonnage and overpredict the grade is clear. At a dilution of 50% by 4% Pb+Zn material the model results would fit the blastholes very closely; under 1% variance on total metal and within 3% on all parameters at all cutoffs (Table 3.13). It is probably not coincidence that in this area of the deposit there is a great deal of low grade sulphides close to 4% Pb+Zn with a few thin high grade bands. Table 3.14 shows the same comparison for high grade ore and the F8608 model. For the few benches mined this model seems to give a fair approximation of the pit reserves; it is within 1% on total metal and ore tonnage but 11% high on lead and 6% low on zinc.

3.2.4.3.4 <u>Conclusion</u>

The conclusion of these comparisons is that the FI model compared reasonably well to other calculations on the basis of total metal and for a large enough volume of material was within 10% of actual production statistics. Despite this, it was unuseable for bench by bench predictions whithout an appropriate dilution factor. The grade predictions of the FI model are too high probably as a result of inappropriate methodology but mainly too low a dilution factor. Most importantly, dilution is not average throughout the deposit and the choice of factor must take account local deposit structure in order to provide reasonable predictions of millfeed. Furthermore without accurate drillhole data, accurate modeling is impossible regardless of calculation sophistication. On the average the FI model will probably give a reasonable approximation of the long term mill feed at a dilution of 10% however local dilution factors should be tried in order to attempt to better reflect the short term and sensitivity to higher dilutions should be carried out at least on the marginal benches. The newer modeling techniques should be extended to the remainder of the deposit a soon as practical.

TABLE 3.12 COMPARISON OF FI MODEL PREDICTED BENCH RESERVES WITH 1986 MINE PRODUCTION IN AY PHASE AS BLOCKED OUT BY BLASTHOLES

	bench	high grade tonnes *	Pb (%)	Zn (%)	Ag (g/t)	Pb+Zn (%)	tonnes total metal	low grade tonnes *	Pb (%)	Zn (%)	Ag (g/t)	Pb+Zn (%)	tonnes total metal	all grades (+4%) tonnes	Pb (%)	Zn (%)	Ag (g/t)	Pb+Zn (%)	tonnes total metal
BLASTHOLE RESULTS																			
	3750	0	0.00	0.00	0	0.00	0	0	0.00	0.00	0	0.00	0	0	ERR	ERR	ERR	ERR	ERR
#	3730	11,520	2.04	3.89	32	5.93	683	8,333	1.63	2.69	34	4.32	360	19,853	1.87	3.39	33	5.25	1,043
	3710	123,056	2.87	4.49	40	7.36	9,057	7,140	1.46	3.14	24	4.60	328	130,196	2.79	4.42	39	7.21	9,385
	3690	116,252	2.52	4.65	30	7.17	8,335	19,627	1.75	2.97	25	4.72	926	135,879	2.41	4.41	29	6.82	9,262
	TOTAL	250,828	2.67	4.54	35	7.21	18,085	35,100	1.62	3.04	25	4.66	1,636	285,928	2.54	4.36	34	6.90	19,720
MODEL PREDICTIONS (NO DILUTION)																			
	3750	3110	6.5	5.94	105	12.44	387	0					0	3,110	6.50	5.94	105	12.44	387
	3730	10680	3.85	4.72	59	8.57	915	4500	2.37	3.18	37	5.55	250	15,180	3.41	4.26	52	7.67	1,165
	3710	80410	3.02	4.27	46	7.29	5,862	8950	1.85	3.18	31	5.03	450	89,360	2.90	4.16	44	7.06	6,312
	3690	73260	3.08	4.63	42	7.71	5,648	8250	2.06	2.87	32	4.93	407	81,510	2.98	4.45	41	7.43	6,055
	TOTAL	167,460	3.36	5.52	44	8.88	14,870	21,700	1.65	3.03	32	4.68	1,016	189,160	3.16	5.23	43	8.40	15,886

MODEL PREDICTIONS DILUTED

100 PERCENT MINING RECOVERY
10 PERCENT DILUTION
0 LEAD GRADE (%) OF DILUTANT
0 ZINC GRADE (%) OF DILUTANT
0.00 LEAD + ZINC GRADE (%) OF DILUTANT
0 SILVER GRADE (g/t) OF DILUTANT

bench	high grade tonnes *	Pb (%)	Zn (%)	Ag (g/t)	Pb+Zn (%)	tonnes total metal	low grade tonnes *	Pb (%)	Zn (%)	Ag (g/t)	Pb+Zn (%)	tonnes total metal	all grades (+4%) tonnes	Pb (%)	Zn (%)	Ag (g/t)	Pb+Zn (%)	tonnes total metal
3750	3,421	5.91	5.40	95	11.31	387						0	3,421	5.91	5.40	95	11.31	387
3730	11,748	3.50	4.29	54	7.79	915	4,950	2.15	2.89	34	5.05	250	16,698	3.10	3.88	48	6.98	1,165
3710	88,451	2.75	3.88	42	6.63	5,862	9,845	1.68	2.89	28	4.57	450	98,296	2.64	3.78	40	6.42	6,312
3690	80,586	2.80	4.21	38	7.01	5,648	9,075	1.87	2.61	29	4.48	407	89,661	2.71	4.05	37	6.75	6,055
TOTAL	184,206	3.05	5.02	40	8.07	14,870	23,870	1.50	2.75	29	4.25	1,016	208,076	2.88	4.76	39	7.63	15,886

* High grade is +5% Pb+Zn and low grade is 4-5%

\# includes ore mined from 3750 since 3730 was mined on a 40 foot lift

TABLE 3.13 COMPARISON OF FI MODEL PREDICTED BENCH RESERVES WITH 1986 MINE PRODUCTION IN AY PHASE AS BLOCKED OUT BY BLASTHOLES

bench	high grade tonnes *	Pb (%)	Zn (%)	Ag (g/t)	Pb+Zn (%)	tonnes total metal	low grade tonnes *	Pb (%)	Zn (%)	Ag (g/t)	Pb+Zn (%)	tonnes total metal	all grades (+4%) tonnes	Pb (%)	Zn (%)	Ag (g/t)	Pb+Zn (%)	tonnes total metal
BLASTHOLE RESULTS																		
3750	0	0.00	0.00	0	0.00	0	0	0.00	0.00	0	0.00	0	0	ERR	ERR	ERR	ERR	ERR
# 3730	11,520	2.04	3.99	32	5.93	683	8,333	1.63	2.69	34	4.32	360	19,853	1.87	3.39	33	5.25	1,043
3710	123,056	2.87	4.49	40	7.36	9,057	7,140	1.46	3.14	24	4.60	328	130,196	2.79	4.42	39	7.21	9,385
3690	116,252	2.52	4.65	30	7.17	8,335	19,627	1.75	2.97	25	4.72	926	135,879	2.41	4.41	29	6.82	9,262
TOTAL	250,828	2.67	4.54	35	7.21	18,085	35,100	1.62	3.04	25	4.66	1,636	285,928	2.54	4.36	34	6.90	19,720
MODEL PREDICTIONS (NO DILUTION)																		
3750	3110	6.5	5.94	105	12.44	387	0					0	3,110	6.50	5.94	105	12.44	387
3730	10680	3.85	4.72	59	8.57	915	4500	2.37	3.18	37	5.55	250	15,180	3.41	4.26	52	7.67	1,165
3710	80410	3.02	4.27	46	7.29	5,862	8950	1.85	3.18	31	5.03	450	89,360	2.90	4.16	44	7.06	6,312
3690	73260	3.08	4.63	42	7.71	5,648	8250	2.06	2.87	32	4.93	407	81,510	2.98	4.45	41	7.43	6,055
TOTAL	167,460	3.36	5.52	44	8.68	14,870	21,700	1.65	3.03	32	4.68	1,016	189,160	3.16	5.23	43	8.40	15,886

MODEL PREDICTIONS DILUTED

100 PERCENT MINING RECOVERY
50 PERCENT DILUTION
1.52 LEAD GRADE (%) OF DILUTANT
2.48 ZINC GRADE (%) OF DILUTANT
4.00 LEAD + ZINC GRADE (%) OF DILUTANT
25 SILVER GRADE (g/t) OF DILUTANT

bench	high grade tonnes *	Pb (%)	Zn (%)	Ag (g/t)	Pb+Zn (%)	tonnes total metal	low grade tonnes *	Pb (%)	Zn (%)	Ag (g/t)	Pb+Zn (%)	tonnes total metal	all grades (+4%) tonnes	Pb (%)	Zn (%)	Ag (g/t)	Pb+Zn (%)	tonnes total metal
3750	4,665	4.84	4.79	78	9.63	449						0	4,665	4.84	4.79	78	9.63	449
3730	16,020	3.07	3.97	48	7.05	1,129	6,750	2.09	2.95	33	5.03	340	22,770	2.78	3.67	43	6.45	1,469
3710	120,615	2.52	3.67	39	6.19	7,470	13,425	1.74	2.95	29	4.69	629	134,040	2.44	3.60	38	6.04	8,099
3690	109,890	2.56	3.91	36	6.47	7,114	12,375	1.88	2.74	30	4.62	572	122,265	2.49	3.79	36	6.29	7,685
TOTAL	251,190	2.75	4.51	38	7.25	18,220	32,550	1.61	2.85	30	4.45	1,450	283,740	2.62	4.32	37	6.93	19,669

* High grade is +5% Pb+Zn and low grade is 4-5%
\# includes ore mined from 3750 since 3730 was mined on a 40 foot lift

TABLE 3.14 COMPARISON OF F8608 MODEL PREDICTIONS FOR AY PHASE TO
1986 MINE PRODUCTION AS BLOCKED OUT BY BLASTHOLES

bench	tonnes +5%	Pb (%)	Zn (%)	Ag (g/t)	Pb + Zn (%)	tonnes total metal
BLASTHOLES						
3750	0					
3730	11,520	2.04	3.89	32	5.93	683
3710	123,056	2.87	4.49	40	7.36	9,057
3690	116,252	2.52	4.65	30	7.17	8,335
TOTAL	250,828	2.67	4.54	35	7.21	18,065
F8608 MODEL (UNDILUTED)						
3750	3,480	3.97	5.87	79	9.84	342
3730	34,880	4.08	5.01	70	9.09	3,171
3710	65,250	3.58	4.39	53	7.97	5,200
3690	122,570	2.93	4.79	40	7.72	9,462
TOTAL	226,180	3.31	4.73	49	8.04	18,185
F8608 MODEL WITH 10% DILUTION AT ZERO GRADE						
3750	3,828	3.61	5.34	72	8.95	342
3730	38,368	3.71	4.55	64	8.26	3,171
3710	71,775	3.25	3.99	48	7.25	5,200
3690	134,827	2.66	4.35	36	7.02	9,462
TOTAL	248,798	3.01	4.30	45	7.31	18,185

3.2.5 Additional Work Required

In order to provide reliable estimates of reserves on a short term basis it will be necessary to drill additional fill in holes. To fill in the current pattern to 43 m (141 ft.) minimum on the main 43 m (141 ft.) spaced sections will require 32 additional holes totalling 4270 m (14000 feet). Of this total 20 holes totaling 2300 m (7500 feet) are in the AY and early BY phases and have considerable urgency.

To upgrade the drilling pattern significantly beyond the level outlined above would be prohibitive in terms of cost and the logistics of both drilling and data analysis.

While this core is still fresh it could be used for additional metallurgical testing if required.

As noted above, the remainder of the deposit modeling should be upgraded to at least F8608 model standards. A new modeling technique that treats PC Mine blocks not as homogenous single geologic entities but as the sum of two or more material types has been devised but not yet put into practice. This modeling technique could help significantly with the treatment of low grade and waste dilution and should tried soon. Work is already underway on both these objectives. The first priority however is to produce an updated AY anbd BY (or BZ) model incorporating the 1986 drilling which will help evaluate the gains from additional drilling by comparison to the F8608 model.

3.3 Grum Geology and Reserves

].3.1 History

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

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

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

3.3.2 General Geology

3.3.2.1 Stratigraphy and lithology

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

At Grum, the Vangorda formation consists of soft, highly fissile, calcareous phyllites. Metabasites in the Grum area are minor and tend to be highly foliated chlorite phyllite rather than blocky, massive greenstones that typify the Vangorda formation elsewhere. The basal carbonaceous member of the formation (unit 5A) thickens across the deposit from about 10m in the northeast to as much as 80 or 100 m southwest of the deposit. The sulphide horizons appear to be associated with the northeast pinchout of this unit. Immediately above the main ore horizon the carbonaceous rocks are soft, highly sheared and gouged but elsewhere they are moderately hard, highly fractured, black siliceous phyllites.

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

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

3.3.2.2 Structure

The ore layers at Grum are contorted into a complex, shallowly northwest plunging, polyphase fold structure. The prominent S shaped folds (figure 3.8) are second phase structures. They are superimposed on a larger Z shaped first phase fold. The dominant plane of fissility (S_2) in the phyllites at Grum is axial planar to the second phase folds and dips shallowly (10^o to 30^o) generally to the southwest. This fissility is a major factor in assessing slope stability for a Grum pit. The overall deposit elongation parallels the axial direction of the second phase folds (315^o trend 11^o plunge).

There are several important faults at Grum. The largest displacements occur on moderately (35^o - 45^o) dipping structures that truncate the deposit at both its northwest and southeast ends (figure 3.9). Neither of these structures would crop out in an open pit but smaller subparallel faults will be found in the pit. A steeply northwest dipping fault trending about 060^o, passes between sections 70W and 72W and downdrops the deposit about 60 m to the northwest. A myriad of smaller faults were mappped underground by Kerr Addison trending on the average 080^o and dipping steeply. Joints mapped underground and on surface tend to strike 060^o and dip subvertically.

3.3.2.3 Surfical Geology

The subcrop of the ore deposit is covered by up to 100 m of morainal material (tills) and better sorted galciofluvial silts, sands and gravels. These unconsolidated sediments are water saturated and may contain pockets of permafrost. The northeast wall of any pit designs at Grum must contend with thick sections of these sediments. Dewatering in advance of stripping may help increase stability substantially as well as simplify operations in the pit.

3.3.2.4 Ore Deposit Geology

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

Grum, like Vangorda and Dy, has several characteristics that distinguish it from Faro. In large part this is due to the lower metamorphic grade the deposit has reached. The most outstanding difference between Grum, and all the other Vangorda

Plateau deposits, as opposed to Faro is the form of the deposit. The Vangorda Plateau deposits consist of several distinct, highly contorted horizons separated by barren phyllite waste. Faro on the other hand is essentially one thick horizon in overall outline with lesser phyllitic waste but substantial barren sulphide waste banding. This implies that dilution by phyllite will be higher at Grum than at Faro. Faro however contains considerable internal sulphide waste thus its dilution is higher than might appear at first glance. It is none the less inescapable that Grum has more potential dilution and will have more complex mining problems than Faro. On the positive side, the dilutant at Grum will be more commonly easily identifiable phyllite rather than low grade sulphides as at Faro. Experience at Faro shows that phyllite dilution is much easier to control than low grade sulphides. Grum's higher grade if diluted at 3 times the historical 5% dilution used at Faro still gives Grum a higher average grade.

The next most obvious difference is a finer grain size and more complex mineral intergrowth, necessitating finer grinding than Faro ores. Cyprus Anvil Mining Corporation had already made modifications to its mill to accomodate this fine grind prior to shutdown in 1982. When a large proportion of feed comes from Grum it will be necessary to utlize this grinding capability at the expense of tonnage throughput.

At a given Pb + Zn cutoff grade, ores at Grum are higher grade than those remaining at Faro, particularly in precious metals relative to base metals. The average gold content of Grum is several times higher than Faro. Similarily, other elements that tend to be geochemical associates of gold: mercury and arsenic, tend to be higher at Grum. The sphalerite at Grum, and likely other Vangorda Plateau deposits, is richer in zinc due to lower metamorphic grade and resulting lesser iron content. This will help counteract higher pyrite-sphalerite middlings expected with Grum ores.

A feature unique to Grum among the Vangorda Plateau deposits is the relative abundance of quartzose ore types, particularily carbonaceous pyritic quartzites (4A) which comprise about 35% of the reserves above 4% Pb + Zn. It will undoubtedly create challenges for maintaining good lead concentrate grades, and probably necessitate stockpiling and planning campaigns of 4A during which depressants are used.

3.3.3 Drill Definition & Information Base

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

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

Between sections 62W and 86W the deposit has also been explored by 15,000 m of underground drilling in fans from a pair of parallel inclines following the deposit trend. The strike length of the deposit examined from underground is 700 m, underground workings, now flooded, total 2900 m. The fans are most complete on even numbered sections (ie: spaced 61 m apart); on the odd numbered sections in between some fill in drilling has also been done from underground. The overall density of drilling is on the order of 15 m X 30 m with local areas being much in excess of that.

In the southeast part of the deposit additional fill in drilling was done by Cyprus Anvil in 1980-1982 from the surface to more closely define shallow ore for early production.

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

Without question Grum is the best drill defined deposit in the Anvil District.

There are 9000 samples in the Grum deposit assay database. Assay intervals generally average 1.5 m in length and are keyed as closely as possible to sulphide rock types. 90% of these were determined for Cyprus Anvil by Kamloops Research and Assay Labs between 1980 and 1983. For most of these samples Pb, Zn, Cu, and Ag assays are available. For 2/3 of these samples there is also insoluble Fe, soluble (in hot concentrated HCL) Fe, Au, and pulp SG. All assays were determined using a set of Anvil District ore type standards for control. Rejects and N_2 purged pulps (by now somewhat oxidized) have been retained for additional analytical work. There are no BaO or Mn assays available nor is there systematic data available for Hg, As, Cd or any other elements.

The remaining 10% of the assays are from Kerr Addison samples for which Pb, Zn and Ag only are available. Many of these samples are from the holes not used in the ore deposit model.

3.3.4 Methods and Procedure of Reserve Calculation

3.3.4.1 Introduction

For the purpose of this evaluation a new block model, the G8606 model, was constructed in June and July 1986. New reserves were calculated for the the deposit in two portions, one, from surface (1336m maximum elevation) to 1088.5 m elevation and a second from 1088.5 to 868.0 m. elevation. This was due to software and hardware limitations bought about by a low bench height (4.5m) and correspondingly larger number of benches.

The PC Mine software package was used for grade interpolation and reserve calculation. The block geology and composites had been previously calculated using Mintec's Medsystem release 10. The results of the calculation are outlined in Table 3.19 and discussed below.

3.3.4.2 Block Geology and Drillhole information

The reserves are calculated from a computer based 3D block model based on a set of cross-sections produced by Cyprus Anvil geologists in 1982. The sections are parallel to the columns in the mine model and perpendicular to the elongation of the deposit. The cross sections are 61m apart (200 feet) and provide the only geologic control for the mine model. These are the same sections used for the sectional calculation by Cyprus Anvil in 1983 (the Simpson-Adamson calculation) recalculated by Dome in 1984. All sections are available in a supporting document available at Curragh's Toronto and Whitehorse offices.

The logging and drill hole orientation data used were the most current available for the deposit. The interpretation of the geologic detail is known to need improvement in a number of areas particularly concerning the correlation of certain horizons and the treatment of faults. A new set of more closely spaced cross and longitidinal sections is being prepared to address these points of detail. For the time being however these shortcomings are of little consequence as most of the deposit is so densely drilled that there is little scope for variations in geologic interpretation to change the volume of the deposit significantly. The details of ore distribution on a given bench can however change significantly. For the purposes of annual projections of production the current geological model is considered adequate.

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

The rock types used in the Grum model are summarized in table 3.15. Since there is a large amount of low grade quartzose mineralization in the footwall of mineralized horizons at Grum it was judged desireable to carry a separate code for base metal poor and base metal rich lithologies. This is inherrent in the lithologic codes of 4C and 4D however exactly the same distinction is made by the modifiers 4A0 and 4A4 (the latter being the base metal rich variant). A similar distinction was made between 4E0 and 4E4. The purpose of this distinction is to avoid undue averaging of grade into the footwall (and vice versa) and to improve modeling of the relatively sharp grade separation within mineralized horizons.

Sectionally assigned codes were applied to 2 columns of blocks on either side of the section. Since the Grum deposit plunges to the northwest this assignment would create a stairstep appearance in long section. By coincidence the diagonal of two blocks in long section is parallel to the deposit plunge thus a "plunge correction" was made by raising the first column of blocks one level (southeast of the section) and lowering the 4th column of blocks one level (northwest of the section). The second and third columns of blocks are kept the same. This has no affect on deposit reserves. All this block coding was done outside of PC Mine either using the Mintec Medsystem release 10 software package or by manual means. The block codes were reformated to suit PC Mine then imported.

DDH data was imported directly to PC Mine from Mintec output files after reformating but not used since composites were also imported from reformated Mintec output files.

Table 3.15

86-06 Mine Model Rock Type	Lithologic Code	Description
1	4A0	low grade, ribbon banded, graphitic, pyritic quartzite
2	4A4	high grade ribbon banded, graphitic, pyritic quartzite
3	4C	low grade pyritic quartzite
4	4D	high grade pyritic quartzite
5	4E0 & 4K	low grade massive pyritic sulphides (K = carbonate bearing)
6	4E4, 4J, 4F	high grade massive pyritic sulphides
7	4G	baritic massive sulphides
8	4H	pyrrhotitic massive sulphides
9	4L	mineralized altered wallrock phyllites
10	all waste	phyllite
11	overburden	gravel, sand & silt

3.3.4.3 Composite Calculation

Composites were calculated by Medsystem on a 4.5 m bench basis for holes steeper than 45°. For holes shallower than 45°, composites were based on 4.5 m horizontal intervals from the drillhole collar. Composite intervals can range from 4.5 m to 6.5 m depending on borehole orientation. Waste intervals less than 1/2 bench height (2.25 m) were considered internal waste and included in the composite interval. Intervals greater than 1/2 bench height were external waste and not included. This procedure was intended to accurately represent the grade of ore in blocks of all settings but does not automatically include all dilution in marginal composites. Such dilution adjustments must be made separately.

Drill hole assay data were clipped to the 95th percentile levels to avoid assigning unusually high assays to large blocks. These levels are listed in table 3.16. Intervals with no measured S.G. were assigned an SG depending on rock type as listed in table 3.17. These are based on statistical analysis of the measured data.

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

Table 3.16

Maximum Permitted Assay Values and SG Values

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

Table 3.17

SG Values Assigned For Each Major Ore Type
In Case of Missing Analytical data

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

Table 3.18

Maximum Permitted Assay and SG
For DDH Composites

PB	9.00%
ZN	17.00%
AG	150.00 G/tonne
AU	2.30 G/tonne
CU	0.34%
PSG	4.80

The final composites can be as short as 0.1 m if only a small part of a mineralized band is within a composite interval. When the composites were imported to PC Mine the length data could no longer be carried. After calculation, the composites were also clipped to the 95th percentile level as outlined in Table 3.18. Every composite was manually checked against the cross-sections to insure that the codes applied to sectional units were consistent with the composite codes.

The modeling process up to this point is described in more detail in documentation of Cyprus Anvil's "G2" model produced in 1982 by P.I. Clarke and in 1984 by L.C. Pigage. Composite calculation was the last step carried out with Medsystem, the remaining calculations, reporting and analysis was done through PC Mine.

3.3.4.4 <u>Variogram analysis:</u>

Experimental variograms were calculated for the Grum composites in vertical, across deposit (model 000°) and along deposit (model 090°) directions. As was expected the variograms are generally ambiguous. Those for lead are shown in figure 3.13 to 3.15. The variogram analysis shows the range along the deposit is about 40 m and is greater than either vertically or across the deposit. The approximately 20 m across deposit range is uncertain but it can be argued that the along deposit range is about twice the across deposit. This conforms to what was expected on geologic grounds.

3.3.4.5 <u>Interpolation:</u>

The geostatisitcal analysis done was not adeqaute to use kriging as as interpolation method thus inverse square distance weighting was used following precedent set at Faro. The search volume was an ellipsoid with major axis of 150 m parallel to the deposit plunge and with a diameter in cross section of 106 m. The elipsoid centered on the block being interpolated thus the maximum distance a sample can be used to weight a block is 75 m. A horizontal and vertical anisotopy of 1.41 was used. This results in samples along trend being weighted twice as heavily as those across trend (with an anisotopy of 1.41 a sample 53 m across strike is weighted the same as one 75 m along plunge because the sample is treated as if it is 53 X 1.41 or 75 m from the block center, once this apparent distance is squared the factor of 2 (= 1.41^2) appears). A search volume radius much larger than the range was used in order to insure that the blocks in the less intensely drilled part of the deposit would get grades assigned. Multi pass interpolation used for the F8608 model was not available when the Grum model was built.

The most important test that a sample within the search volume must pass before being used to interpolate a block is the equivalance of geological codes. This is important in Anvil district deposits because of the strong ore type zoning and

TWO DIMENSIONAL VARIOGRAM

EXTRACTION DATA USED :

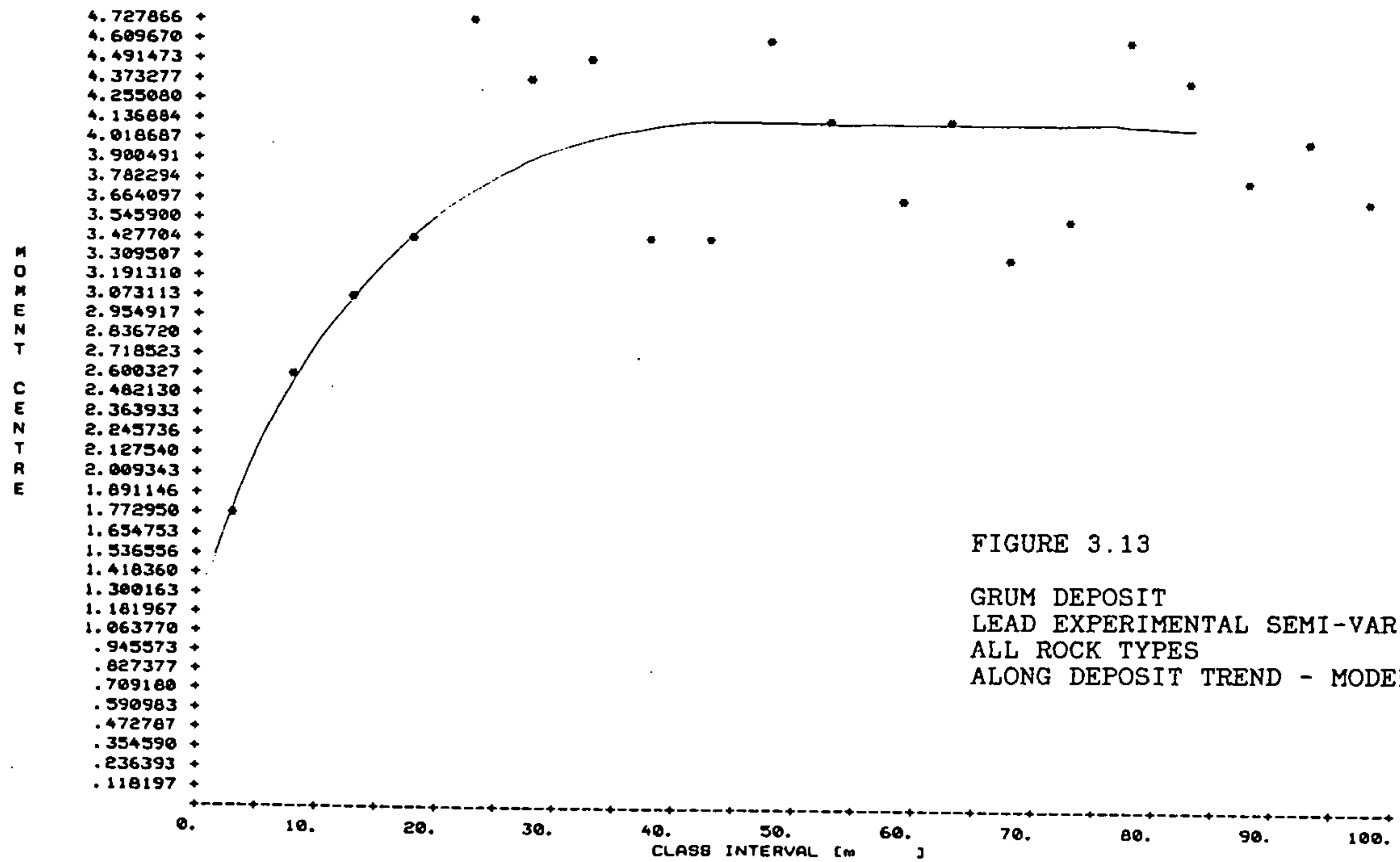


FIGURE 3.13

GRUM DEPOSIT
LEAD EXPERIMENTAL SEMI-VARIOGRAM
ALL ROCK TYPES
ALONG DEPOSIT TREND - MODEL 090

TWO DIMENSIONAL VARIOGRAM

EXTRACTION DATA USED :

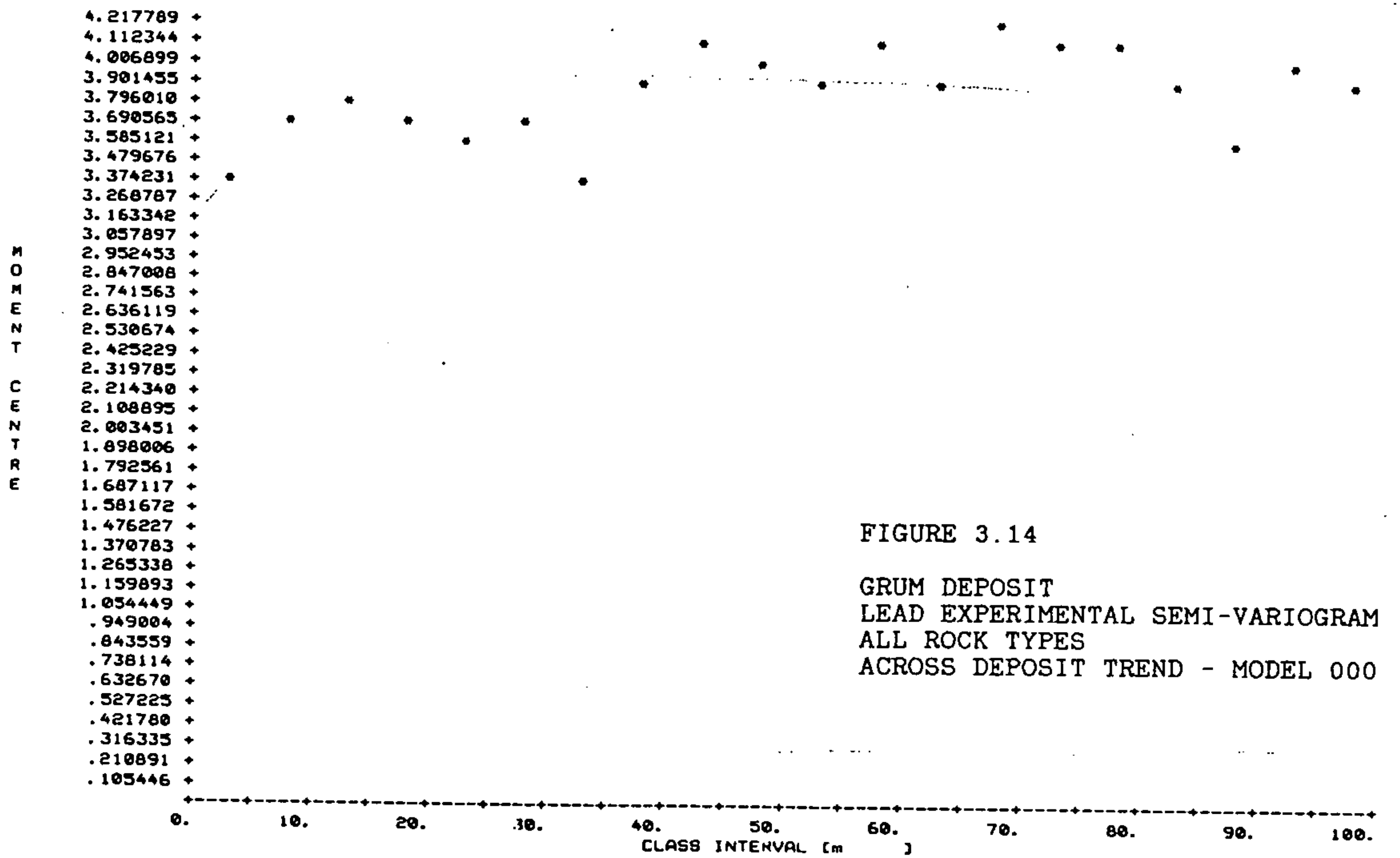


FIGURE 3.14

GRUM DEPOSIT
LEAD EXPERIMENTAL SEMI-VARIOGRAM
ALL ROCK TYPES
ACROSS DEPOSIT TREND - MODEL 000

THE SYMBOL "*" INDICATES LESS THAN 30 SAMPLES IN THAT CLASS

DOWN-THE-HOLE VARIOGRAM - SAMPLE DATA

BOREHOLE USED : AVERAGE FOR ALL SELECTED HOLES FOR LABEL NO : 2 Pb %

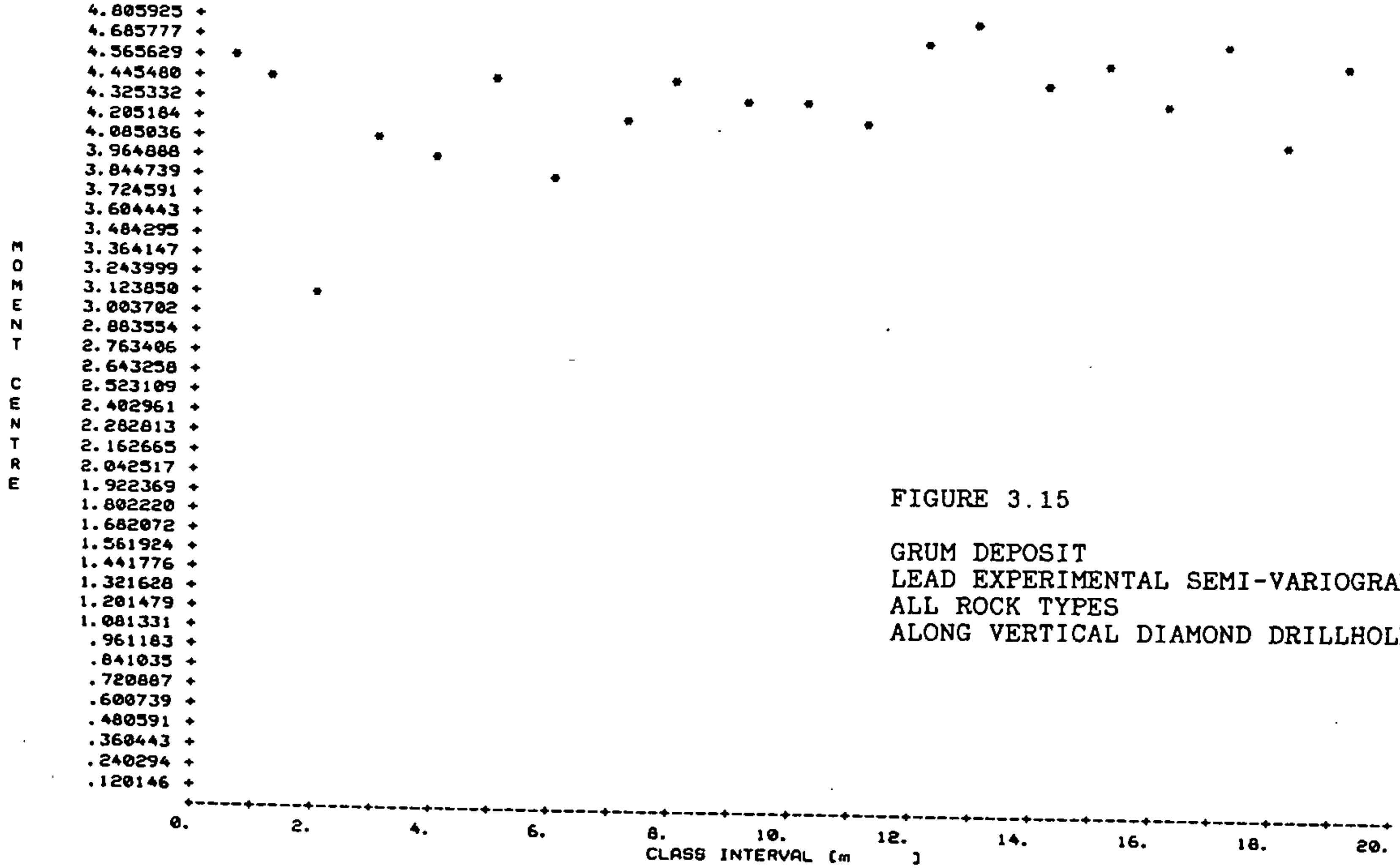


FIGURE 3.15

GRUM DEPOSIT
LEAD EXPERIMENTAL SEMI-VARIOGRAM
ALL ROCK TYPES
ALONG VERTICAL DIAMOND DRILLHOLES

THE SYMBOL "*" INDICATES LESS THAN 30 SAMPLES IN THAT CLASS coupled grade zoning. The implications of this restrictive code matching for use of the model are very significant and discussed below.

The minimum number of composites required to interpolate a block was set at 2, the maximum at 8. The minimum limit was set to avoid the possibility that one very short composite could bias an entire block, the possibility that two very short composites could be used cannot be excluded however in most cases a very short composite will be near a longer one.

Specific gravity was interpolated in the same fashion as other assays. Uninterpolated sulphide blocks were assigned an SG of 2.7. SG was reduced by 5% in the final model in order to correct the pulp SG to in-situ whole rock SG. The average SG's for the Grum deposit are given with the geological reserves in tables 3.19 and 3.20.

3.3.4.6 Geological Reserve Reporting

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

Since there are two Grum models the results of the two models were combined by a spreadsheet.

Sectional geological reserves were also computed for each cross section by reporting the reserves within a plan view polygon representing the area of influence of each section (± 30.5m from the section line). The sectional reserves were needed to compare to previous calculations which are largely sectional hand calculations.

3.3.5 Results

3.3.5.1 Geological Reserves

The geological reserves calculated for Grum are summarized in table 3.19 for the two constituent models and for the entire deposit. Sectional geological reserves are summarized in table 3.20 for the entire deposit.

Southeast of section 62W the Champ zone is estimated to contain a additional 1.7 milliion tonnes averaging 3.5% Pb, 4.3% Zn and 46 g/t Ag. This figure is based on sectional calculation and quoted at a 4% Pb+Zn cutoff with no adjustment to reflect dilution. Northwest of 86W there may be an additional 5 to 10 million tonnes of deep mineralization not yet completely drilled off.

3.3.5.2 Model to Model Comparisons

Table 3.21. and 3.22 A to D compare the newly calculated reserves (G8606 model) to previous calculations on a whole

TABLE 3.19

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

GRADE CATEGORY	VOLUME (bcm)	S.G.	ORE (tonnes)	LEAD (%)	ZINC (%)	Pb + Zn (%)	SILVER (g/t)	GOLD (g/t)
ABOVE GRUM (1336.0 m to 1088.5 m)								
+6 %	4,677,480	3.37	15,765,300	3.84	6.50	10.34	64.0	0.92
4 - 6%	1,942,920	3.12	6,065,990	1.90	3.10	4.99	33.0	0.77
+4 %	6,620,400	3.30	21,831,290	3.30	5.56	8.85	55.4	0.88
UNDER GRUM (1088.5 m to 868.0 m)								
+6 %	1,880,820	3.75	7,057,100	4.04	6.36	10.40	68.5	1.18
4 - 6%	522,720	3.37	1,760,770	2.12	2.76	4.88	35.5	0.99
+4 %	2,403,540	3.67	8,817,870	3.66	5.64	9.30	61.9	1.14
TOTAL DEPOSIT (1336.0 m to 868.0 m)								
+6 %	6,558,300	3.48	22,822,400	3.90	6.46	10.36	65.4	1.00
4 - 6%	2,465,640	3.17	7,826,760	1.95	3.02	4.97	33.6	0.82
+4 %	9,023,940	3.40	30,649,160	3.40	5.58	8.98	57.2	0.95
COMPARISON TO CYPRUS ANVIL/DOME (SIMPSON - ADAMSON) HAND CALCULATION Uncorrected - see Tables 3.14 B and D								
+4 %	8,225,911	3.96	32,611,000	3.48	5.72	9.20	59	
VARIANCE new to old	9.7%x	-14.3%x	-6.0%*	-2.2%	-2.5%	-2.4%	-3.0%	

NOTES:

* Part of this difference in tonnage is due to uninterpolated blocks. There are 707 such blocks in under grum and 316 in above grum, not all of this material will be ore however. Each block represents about 1900 tonnes of material thus there is about 2,000,000 tonnes of un accounted for sulphides.

x The large variance in volume and specific gravity between the two calculations is explained in the text and the notes for Tables 3.22 B and D.

TABLE 3.20

GEOLOGICAL RESERVES BY CROSS SECTION FROM G8606 COMPUTER MODEL, 1986
TOTAL DEPOSIT, 4% Pb + Zn CUTOFF GRADE

SECTION	VOLUME	SPECIFIC GRAVITY	TONNAGE	LEAD	ZINC	SILVER	GOLD	Pb+Zn	TOTAL METAL
	(bcm)		(tonnes)	(%)	(%)	(g/t)	(g/t)	(%)	(tonnes)
86 W	684,180	3.09	2,116,210	2.60	4.99	46.0	0.44	7.59	160,592
84 W	584,820	3.16	1,849,610	2.80	4.90	47.6	0.60	7.70	142,432
82 W	787,860	3.40	2,680,540	3.19	5.55	56.5	0.84	8.73	234,069
80 W	1,297,080	3.35	4,340,270	3.10	4.85	51.8	1.05	7.95	345,081
78 W	975,780	3.27	3,195,500	3.16	5.50	54.7	1.04	8.66	276,762
76 W	907,200	3.36	3,048,860	3.49	5.80	58.6	1.07	9.29	283,135
74 W	1,013,040	3.45	3,498,640	3.72	6.20	62.3	1.05	9.92	347,148
72 W	748,440	3.45	2,584,690	3.64	5.77	61.2	1.02	9.41	243,256
70 W	694,980	3.52	2,449,940	3.86	6.57	63.6	1.04	10.43	255,415
68 W	355,320	3.70	1,314,260	4.10	6.27	66.7	1.00	10.37	136,297
66 W	442,260	3.69	1,632,190	4.14	5.75	66.7	0.99	9.90	161,507
64 W	301,860	3.61	1,091,070	3.79	5.52	61.2	0.99	9.30	101,486
62 W	178,200	3.69	657,640	3.39	5.03	54.8	1.10	8.42	55,347
	8,971,020	3.40	30,459,420	3.41	5.60	57.3	0.95	9.00	2,742,527

* The slight difference (0.1%) in tonnage between this figure and that on table 3.19 is due to a small amount of material outside the area limits set for the cross sections when computing these reserves.

deposit and sectional basis respectively.

Unfortunately the calculations listed in table 3.21 are not exactly comparable. The first 3 (A-C) are all based on Kerr-Addison's geological interpretation and assay data. The Cyprus Anvil computer models differs in that some new assay data was used but the bulk of the data was the same as A and B.

The last 2 (D & E on table 3.21) are based on the current geological interpretation and the current assay information which nearly completely replaces Kerr-Addison's data. These two calculations reflect some ore in the northwest part of the deposit that was drilled off in 1982, this amounts to about 3,000,000 tonnes largely of low grade material on sections 78W to 86W.

The early computer models did not use geologic control on interpolation which tends to average grade between disseminated and massive mineralization resulting in a larger tonnage of lower grade material than a comparable sectional calculation with sharp grade distinctions (compare A to B and C on table 3.21).

The G8606 model has attempted to counter this averaging effect by restricting the choice of composites to the same rock type as the block being estimated. The result is a closer comparison in grade to the corresponding sectional calculation (compare D to E on table 3.21). The G8606 model however was not able to assign a grade to every block due to the stringent matching requirements and local paucity of assay information. Thus there 1023 uninterpolated blocks representing about 2,000,000 tonnes of sulphides distributed through the deposit. Only a portion of this is likely to be over 4% Pb + Zn in reality but for the purposes of the current model all of it is assigned a zero grade and included as sulphide waste. Much of this material (about 700,000 tonnes in 423 blocks) is in the lower part of section 86W partly accounting for the poor comparison to the hand calculation on that section (table 3.22D). The distribution of uninterpolated blocks by section in the two constituent models is given in table 3.23.

A further complication in comparing the current model to the hand calculated reserves obtained from the same geological interpretation is that two problems were encountered with the Cyprus Anvil/Dome calculation: a) there were some calculation errors involving volumes above cutoff and to a lesser extent tonnes on sections 78W to 86W and b) the SG used for sections 82W-86W was assumed to be 3.6 (because assay data was not complete at the time the calculation was made)rather than the measured SG that was used for the rest of the deposit. In retrospect the assumed value was too high as much of the mineralization on those sections is considerably lighter. These points have been adjusted for in Table 3.22B. Table 3.22D shows the section by section variance between calculations. Between the Cyprus Anvil/Dome calculation and the current one the grade is almost always slightly down. In large part of this is

TABLE 3.21

COMPARISON OF PREVIOUS CALCULATIONS OF THE GEOLOGIC RESERVES TO THE FB608 MODEL
4% Pb + Zn CUTOFF GRADE

	CALCULATION		ORE (tonnes)	Pb (%)	Zn (%)	Pb+Zn (%)	Ag (g/t)	TOTAL METAL (tonnes)
A	KERR ADDISON - 1977 sectional hand calculation		26,083,000	4.1	6.4	10.5	62	2,738,715
B	KERR ADDISON / NORANDA - 1978 computer block model		27,650,000	3.1	4.9	8.0	48	2,212,000
C	CYPRUS ANVIL - 1981 "61" computer block model		30,781,000	3.1	4.9	8.0	49	2,462,480
D	CYPRUS ANVIL / DOME -1983 sectional hand calculation (uncorrected, see table 3.22)	*	32,611,000	3.5	5.7	9.2	59	3,000,212
D	CYPRUS ANVIL / DOME -1983 sectional hand calculation (corrected, see table 3.22)	*	31,615,000	3.5	5.7	9.2	59	2,918,065
E	CURRAGH RESOURCES - 1986 "68608" computer block model	*	30,649,000	3.4	5.6	9.0	57	2,758,410

* Includes on the order of 2,000,000 to 3,000,000 tonnes of sulphides in the northwest part of the deposit drilled off in 1982 after the other calculations were done.

TABLE 3.22 A

RESULTS OF KERR ADDISON SECTIONAL CALCULATION, 1977
TOTAL DEPOSIT (both Vangorda Mines and AEX option) 4% Pb+Zn CUTOFF

SECTION	VOLUME (bcm)	SPECIFIC GRAVITY	TONNAGE (tonnes)	LEAD (%)	ZINC (%)	SILVER (g/t)	GOLD (g/t)	Pb+Zn (%)	TOTAL METAL (tonnes)
86 W	n/a	n/a	1,678,746	3.59	5.80	55.0	n/a	9.39	157,634
84 W	n/a	n/a	1,375,120	3.33	5.60	53.0	n/a	8.93	122,798
82 W	n/a	n/a	2,185,447	4.06	7.30	69.0	n/a	11.36	248,267
80 W	n/a	n/a	3,039,373	3.67	5.71	56.0	n/a	9.38	285,093
78 W	n/a	n/a	2,918,497	3.79	6.45	50.0	n/a	10.24	298,854
76 W	n/a	n/a	2,732,810	4.08	6.24	61.0	n/a	10.32	282,026
74 W	n/a	n/a	2,778,449	4.15	6.63	66.0	n/a	10.78	299,517
72 W	n/a	n/a	2,223,718	4.10	6.46	63.0	n/a	10.56	234,825
70 W	n/a	n/a	2,226,492	4.77	7.45	67.0	n/a	12.22	272,077
68 W	n/a	n/a	1,736,341	4.55	6.38	73.0	n/a	10.93	189,782
66 W	n/a	n/a	1,377,821	4.81	6.55	67.0	n/a	11.36	156,520
64 W	n/a	n/a	1,164,762	4.34	6.74	67.0	n/a	11.08	129,056
62 W	n/a	n/a	630,266	3.94	6.01	55.0	n/a	9.95	62,711
62W to 86W	n/a	n/a	26,067,842	4.07	6.43	61.50	n/a	10.51	2,739,161

TABLE 3.22 B

RESULTS OF CYPRUS ANVIL/DOME SECTIONAL CALCULATION (SIMPSON-ADAMSON), 1983
TOTAL DEPOSIT, 4% Pb + Zn CUTOFF GRADE

SECTION	VOLUME (bcm)	SPECIFIC GRAVITY	TONNAGE (tonnes)	LEAD (%)	ZINC (%)	SILVER (g/t)	GOLD (g/t)	Pb+Zn (%)	TOTAL METAL (tonnes)
86 W	997,045 *	3.23	3,221,532	2.88	5.21	49.0	n/a	8.09	260,622
84 W	776,317 *	3.20	2,486,855	2.86	4.83	48.0	n/a	7.69	191,239
82 W #	989,573 *	3.29	3,256,446	3.00	5.10	52.0	n/a	8.10	263,772
80 W #	1,118,740	3.38	3,783,293	3.31	5.41	56.0	n/a	8.72	329,903
78 W #	994,148	3.11	3,093,642	3.37	5.96	59.0	n/a	9.33	288,637
76 W	886,940	3.39	3,006,734	3.61	6.06	62.0	n/a	9.67	290,751
74 W	934,978	3.46	3,237,299	3.66	5.97	61.0	n/a	9.63	311,752
72 W	659,715	3.62	2,389,111	3.80	6.11	65.0	n/a	9.91	236,761
70 W	656,848	3.69	2,425,119	4.06	6.62	68.0	n/a	10.68	259,003
68 W	386,618	3.73	1,440,349	4.54	6.88	73.0	n/a	11.42	164,488
66 W	446,215	3.62	1,613,702	4.41	5.95	71.0	n/a	10.36	167,180
64 W	282,918	3.64	1,028,545	3.84	5.75	62.0	n/a	9.59	98,637
62 W	170,190	3.72	632,340	3.32	5.66	63.0	n/a	8.98	56,784
62W to 86W	9,300,243	3.40	31,614,967	3.49	5.74	59.1	n/a	9.23	2,919,529

* An assumed value for S.G. of 3.6 was used on these sections.
This is too high and has been reduced by 10% for comparison purposes.

\# Volumes above cutoff have been recalculated due to an error in the original calculation.

TABLE 3.22 C

PERCENT VARIANCE OF 68608 (TABLE 3.20) TO KERR ADDISON (TABLE 3.22 A) [(NEW-OLD)/OLD]
TOTAL DEPOSIT, 4% Pb + Zn CUTOFF GRADE

SECTION	VOLUME (bcm)	SPECIFIC GRAVITY	TONNAGE (tonnes)	LEAD (%)	ZINC (%)	SILVER (g/t)	GOLD (g/t)	Pb+Zn (%)	TOTAL METAL (tonnes)
86 W			26.1%	-27.6%	-14.0%	-16.4%		-19.2%	1.9%
84 W			34.5%	-15.9%	-12.5%	-10.2%		-13.8%	16.0%
82 W			22.7%	-21.5%	-24.0%	-18.0%		-23.1%	-5.7%
80 W			42.8%	-15.6%	-15.0%	-7.5%		-15.2%	21.0%
78 W			9.5%	-16.7%	-14.7%	9.4%		-15.4%	-7.4%
76 W			11.6%	-14.5%	-7.1%	-3.9%		-10.0%	0.4%
74 W			25.9%	-10.3%	-6.5%	-5.6%		-8.0%	15.9%
72 W			16.2%	-11.2%	-10.7%	-2.8%		-10.9%	3.6%
70 W			10.0%	-19.2%	-11.8%	-5.1%		-14.7%	-6.1%
68 W			-24.3%	-9.9%	-1.7%	-8.7%		-5.1%	-28.2%
66 W			18.5%	-13.9%	-12.2%	-0.5%		-12.9%	3.2%
64 W			-6.3%	-12.8%	-18.2%	-8.6%		-16.1%	-21.4%
62 W			4.3%	-14.1%	-16.3%	-0.3%		-15.4%	-11.7%
62W to 86W			16.8%	-16.3%	-13.0%	-6.8%		-14.3%	0.1%

TABLE 3.22 D

PERCENT VARIANCE OF 68608 (TABLE 3.20) TO SIMPSON -ADAMSON/DOME (TABLE 3.22 B) [(NEW-OLD)/OLD]
TOTAL DEPOSIT, 4% Pb + Zn CUTOFF GRADE

SECTION	VOLUME (bcm)	SPECIFIC GRAVITY	TONNAGE (tonnes)	LEAD (%)	ZINC (%)	SILVER (g/t)	GOLD (g/t)	Pb+Zn (%)	TOTAL METAL (tonnes)
86 W *	-31.4%	-4.3%	-34.3%	-9.8%	-4.2%	-6.2%		-6.2%	-38.4%
84 W *	-24.7%	-1.3%	-25.6%	-2.0%	1.4%	-0.8%		0.1%	-25.5%
82 W *	-20.4%	3.4%	-17.7%	6.2%	8.8%	8.7%		7.8%	-11.3%
80 W *	15.9%	-1.0%	14.7%	-6.4%	-10.3%	-7.5%		-8.8%	4.6%
78 W	-1.8%	5.2%	3.3%	-6.3%	-7.7%	-7.3%		-7.2%	-4.1%
76 W	2.3%	-0.9%	1.4%	-3.3%	-4.3%	-5.5%		-4.0%	-2.6%
74 W	8.3%	-0.2%	8.1%	1.7%	3.9%	2.1%		3.0%	11.4%
72 W	13.4%	-4.6%	8.2%	-4.2%	-5.5%	-5.8%		-5.0%	2.7%
70 W	5.8%	-4.5%	1.0%	-5.0%	-0.8%	-6.5%		-2.4%	-1.4%
68 W	-8.1%	-0.7%	-8.8%	-9.7%	-8.9%	-8.7%		-9.2%	-17.1%
66 W	-0.9%	2.0%	1.1%	-6.0%	-3.3%	-6.1%		-4.5%	-3.4%
64 W	6.7%	-0.6%	6.1%	-1.4%	-4.1%	-1.3%		-3.0%	2.9%
62 W	4.7%	-0.7%	4.0%	2.0%	-11.1%	-12.9%		-6.3%	-2.5%
62W to 86W	-3.5%	-0.1%	-3.7%	-2.5%	-2.5%	-3.0%		-2.5%	-6.1%

* See explanation in text-comparison of models at 0% cutoff

probably due to the clipping of the extreme high end of the assay and composite population which was not done in the Cyprus Anvil/Dome calculation. The large variance in volume on sections 80W-86W is partly due to uninterpolated blocks but must be mainly due to differences in calculation method. The computer model interpolated grade into each block whereas the hand calculation averaged a large number of intersections to arrive at the grade for large panels of ore. In light of the relative paucity of drillhole information for some panels such large variances are not surprising. The interpolation method would be expected to give a more realistic picture of grade distribution.

A check on the volume of sulphides above 0% Pb+Zn grade shows that the volumes are essentially identical for the two calculations: 11,125,546 cu. m for the Cyprus Anvil/Dome versus 11,166,660 cu. m for the present model. The comparison for tonnage above cutoff for the two calculations with adjustment for a lower S.G. on sections 82W to 86W gives 37,415,941 tonnes for Cyprus Anvil/Dome versus 37,329,090 tonnes for the current model (the original Cyprus Anvil/Dome calculation using their S.G. gave 38,536,557 tonnes above 0%). Thus it is clear that the reserves only become inconsistent when a cutoff grade above 0% is used.

<u>Table 3.15</u>

UNINTERPOLATED BLOCKS IN G8608
GRUM COMPUTER MODEL

	86 W	84 W	82 W	80 W	78 W	76 W	74 W	72 W	70 W	68 W	66 W	64 W	62 W	TOT
Above Grum	32	1	1	32	19	4	8	21	62	74	46	12	0	312
Under Grum	423	23	38	52	54	23	13	36	10	14	1	0	0	687
Total	455	24	39	84	73	27	21	57	72	88	47	12	0	999

3.3.5.3 Reliability of Reserves

The reliability of geological reserves is a difficult matter to quantify at this time. The inadequate geostatistical knowledge of the deposit has precluded determination of block estimation variance. Thus quantification of overall deposit variance was not possible.

The density of drilling is sufficient to limit the possibility of major changes in deposit volume due to variance in interpretation. Volume ranges of ±10% would be possible. Variance possible due to calculation methods is also not quantified but changes of a few percent would be possible through adjustment of interpolation parameters such as anisotropy, weighting scheme and range and perhaps more by altering the geology matching scheme.

Based on the reproducability of geological reserve calculations for the Grum deposit and the central position of the current model in the range of values it seem reasonable to conclude that on a global basis the geological reserves presented herein are reasonable and probably reliable within ± 10%, as such can be considered proven. Local reserves on the other hand are not nearly as reliable. In the vicinity of the dense drill pattern around the underground workings the local reserves are reasonably accurate however towards the periphery there is considerably more possible variance. On a bench basis variances of ±50% would not be surprising towards the deposit periphery but lesser variances in the core of the deposit are likely. Small tonnage benches would of course be subject to high variances.

The earliest production comes from a long trough like pit following the subcrop of the ore horizons. This part of the deposit is the most densely dilled from the surface but only the part southeast of 72W has been drilled from underground. The drill density is at least one hole every 30.5 m (100 feet) on 61 m (200 foot) spaced sections, close to the current density of Vangorda drilling. There is room for improvement especially in light of the complex structure involving steeply dipping ore layers. In general the earliest production is from an area that is only moderately reliable but it is underlain by highly reliable deeper reserves.

How well the reported reserves will relate to mill feed depends on the type and degree of dilution allowed. It is recommended to use a dilution of no less than 10% at zero grade with alternate dilutions ranging to about 25% at 3% Pb + Zn depending on the extent of low grade sulphide dilution experienced. As a starting point 15% at zero grade would be reasonable but sensitivity to dilution should be considered. As is the case for Faro dilution will be site specific and a constant average dilution will be an oversimplification.

The need to use a higher dilution than the historic 5% used for Anvil District deposits in the past is brought on largely by the restrictive geology matching during interpolation. This matching was not done previously thus the models tended to dilute themselves in unpredictable ways during interpolation. In order to present a more realistic portrayal of grade distribution especially with respect to grade averaging into otherwise barren sulphides, particularly footwall sulphides, the modeling technique was changed. It is however necessary to change dilution practice at the same time. This was not done at Faro when using the FI model with the result that prdicted tonnages were far too low and grade far too high compared to blasthole indicated tonnage and grade.

The other major problem with the Faro FI model in the JB phase has been traced back to drillholes now known to be in the wrong place due to a survey error many years ago. This became apparent by comparing drillhole geology to pit geology. At Grum similar problems are not expected since more careful surveying has been done and survey calculations have been checked for errors; those found have been corrected. At Grum the drillhole collars are still present thus surveyed locations could be checked if desired. This was not the case at Faro.

3.3.6 Additional Work needed

The complex geology of the Grum deposit and its multi-horizon nature creates a great deal of difficulty in producing an interpretation that it is consistent from section to section. Horizons are indistiguishable from one another in drill core thus it is only the iterative process of section by section interpretation and comparison that can solve this problem, if indeed a solution is possible. The current interpretation has several inconsistencies that should be corrected. Additionally 60.5m spaced sections are too far apart and do not take account of all drillhole information available. A new interpretation is needed that will provide the basis for a better and more reliable model. 30m spaced sections should be made and longitudinal sections as well as cross sections should be interpreted in order to better portray fault problems. This interpretation has already been started and should be finished at the earliest possible date.

The Champ zone should be included in the overall deposit model since mining that area might influence the overall pit economics by lowering the main pit exit. Considering both zones together might enhance the economics of both.

The geostatistical treatment of Anvil District deposits has always been inadequate. This is largely due to the wide spacing of drill holes. At Grum the drill pattern is dense enough to produce meaningful analysis once the basic geologic data is organized by the above interpretation. Such analysis should allow better interpolation methods to be used than the simple inverse square method used to date.

One of the major inadequacies of geologic data at Grum is geotechnical data. There is reason to believe that the S_2

foliation under Doal Lake (in the northeast part of the proposed pit) may be dipping northeast rather than southwest. This could have significant implications for slope stability and the size of the area required to be pre-stripped. A program of oriented core drilling is required to evaluate this question. These holes should also sample the overburden to evaluate its clay content and hydogeology. If the overburden is amendable to dewatering and can be kept dry then steeper slopes will be possible than in the current water saturated state.

There is little known geotechnically of the proposed waste dump sites; the foundation conditions in these areas should be studied.

More metallurgical work is required particularly on the 4A carbonaceous ore type. Its abundance at Grum could have a serious impact on mine reserves if an economic treatment scheme cannot be worked out. Separate consideration of the optimum grind for 4A should be given since in hand specimen 4A is not noticably finer grained than 2A (unlike the massive ores) and fine grinding exaggerates the problems of treating 4A. The northwest half of the ore in the first years pit will be almost entirely 4A.

The frequency distribution of specific gravity for Grum ores is strongly bimodal with modes corresponding to the quartzose and massive ores. Consideration should be given to the possibility of mechanical separation of the bulk of the ore types using this density contrast rather than relying on separation during mining.

It may be necessary to carry out fill in drilling in the area of early production to more comfortably outline early reserves. This work could be coordinated with the collection of metallurgical samples.

3.4 Vangorda Geology and Reserves

3.4.1 History

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

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

3.4.2 General Geology

3.4.2.1 <u>Stratigraphy and lithology</u>

The Vangorda deposit consists of one major sulphide horizon about 50 to 120 m beneath the basal carbonaceous member of the Vangorda formation. The host rocks for the deposit are dominantly non-calcareous phyllites, probably part of the Mt. Mye formation, however formational assignments near this deposit are ambiguous. The reason for the ambiguity is largely due to the strong wall rock alteration developed around the deposit. Most phyllites, especially in the deposit footwall, are bleached, locally silicified and/or chloritic and sulphide bearing.

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

3.4.2.2 <u>Structure</u>

The Vangorda deposit occurs in the hinge of a large second phase fold. Overall the deposit has the shape of a reclining M or a 3 in cross section, however there is considerable uncertainty in the details of fold morphology. The deposit is elongate in the northwest-southeast direction parallel to F_2 fold axes. It has been traced over a 1300 m X 200 m area.

The northwest half of the deposit plunges about 10^o towards the northwest but the southeast half has a sub-horizontal plunge.

The S_2 foliation dips shallowly toward the southwest as at Grum but is locally quite variable.

The deposit is truncated by a steep normal fault at its northwest end. Many gouge zones were observed in drill core but the orientation of the structures responsible for them is not known. A number of faults parallel to S_2 are predicted. These are "required" to make the structure and stratigraphy fit. These low angle structures are best thought of as sheared out fold limbs, they are not generally gouge zones and will pose no more serious a problem for slope stability than the S_2 foliation itself and the myriad of small gouge zones that parallel it. Several analogous structures are thought to be present at Grum.

3.4.2.3 Deposit Geology

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

The deposit consists of the same sulphide rock types as the other deposits but two types are particularly prominant. In the footwall (also the interpreted stratigraphic footwall) of the deposit is a sulphide rich quartzite (4C and 4EC). This quartzite grades downwards into siliceous phyllite and ultimately altered phyllite. Parallel to this downward decrease in silica is a downward decrease in the abundance of sulphides from quartz rich semi-massive sulphide (4EC) at the top to weakly pyritic altered phyllite at the base (4L). Most of the sulphides in the quartzite are pyrite, however pyrrhotite is generally present and locally abundant or dominant. Magnetite is unusually well developed in the quartzite. The quartzite contains only minor lead and zinc but is relatively rich in copper and unusually high in gold (see Table 3.24). The quartzite is similar to the semi-massive zone along the northeast edge of Zone 3 at Faro and one of the lower ore panels at Grum. From a reserve estimation point of view, The significance of the barren pyritic quartzite is that it is beneath and sharply delinated from high grade massive sulphides. For this reason it is important to restrict the selection of assay composites during grade interpolation to equivalent, geology otherwise excessive averaging of grade into the deposit footwall and lowering of the overall deposit grade will occur. Previous models of Vangorda have been subject to this averaging effect, resulting in a larger tonnage of lower grade than more selective calculations and creating the impression of ore deeper than it actually extends.

The massive sulphides that overlie the pyritic quartzite are commonly baritic and rich in lead and zinc. The unit is actually a mixture of about 50% 4E and 50% 4G ore types but separate treatment of pure types at Vangorda is not realistic from either the point of view of mining or the level of detail carried in this model. Metallurgical predictions must take into account the mixed nature of this dominant ore type compared to the relatively pure samples on which test work was done. Of the mineralization exceeding 6% Pb & Zn, 90% is barite bearing massive sulphides (4EG). Most pyritic quartzite (4C and 4EC) is sulphide waste on the basis of lead zinc content (see Table 3.24).

Of the other sulphide rock types only 4A is of any importance. As is usual for these deposits it tends to be low grade and peripheral to the deposit. Much of the 4A is actually part of the upper horizon associated with the carbonaceous phyllite.

The shallow depth of burial of the deposit may create metallurgical difficulties because of oxidation. Early metallurgical work seemed to show this however later work done by Cyprus Anvil on fresh core achieved better results. The limited core observed by the writer was not visibly oxidized and oxidation is not extensively described in most drill logs below the first few meters of bedrock. Recoveries in massive sulphides are generally good except locally near Vangorda creek and in the southeast end of the deposit (at Faro oxidized massive sulphides yield poor core recoveries). In much of the southeast end of the deposit it is not possible to give recoveries based on recent drilling but old holes did not core well and it seems prudent to assume that the portion of the deposit southeast of 12E, where till cover is thin, will be oxidized. This could affect as much as 1,900,000 tonnes of baritic sulphide or 37% of the baritic sulphides in the geological reserves.

3.4.3 Drilling Density and Information Base

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

In the remainder of the deposit there are 8 recent NQ or HQ drill holes but the bulk of the drilling (45 holes) is by rotary methods. The pattern is also 60 m X 30 m. There appears to have been a sampling problem with some of the rotary holes that has resulted in unreasonable assays. These holes have not been used in the current reserve calculation but they cast doubt on the remainder of the rotary holes. For this reason, where a Prospector Airways hole was also available <u>and</u> good recovery was obtained, the older hole has been used in preference to a rotary hole.

For most assay intervals there are Cu, Pb, Zn, Ag, Au., soluble (in hot HCl) Fe, insoluble Fe and BaO assays. Old diamond drill holes generally only have Pb, Zn and Ag. The newer assays are the same as those described for Grum.

The lead assays from the older holes are suspect because a correction for barium interference was not made.

3.4.4 Methods and procedure of reserve calculation

3.4.4.1 Introduction

The Vangorda deposit reserves were generated using a computer based 3D block model, the V8607 model, made during June and July 1986. The PC MINE software package was used for all stages of model construction.

3.4.4.2 Block geology and drillhole information

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

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

2.) Stratigraphic facings implied by Anvil cycles were followed wherever possible. Of particular importance was the barren pyritic quartzite (units 4C and 4EC) which shows strong indications of being a footwall facies.

3.) Symmetric sulphide intersections were taken to imply folds.

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

Assumption 1 and 2 together require highly attenuated folds in order to produce a consistant interpretation. The resulting fold shapes are consistent with the deformation style observed in nearby outcrops on a small scale and with the inferred shapes of more closely controlled folds at Grum. All sections are included in supporting documents at Curragh's Toronto and Whitehorse offices.

At the density of drilling at Vangorda it is not realistic to produce interpretations that differ widely in deposit volume (changes in volume of ±10% to 15% may be possible due to interpretation) thus the current interpretation is adeqaute for long term planning even if there is disagreement on the treatment of structural detail.

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

The cross sections were digitized and block assignments made by machine from digitized outlines. The logic used for block assignment is based on the geology at the center of a block. The block is assigned one geology code and the block is thereafter considered to be entirely that rock type.

Block size used was 4.5 m high, 4.5 m across strike and 10.0 m along strike. This is essentially the smallest block size practical that allows maximum resolution of geologic detail. Rows of blocks are parallel to the geologic sections. The same geological codes were applied to 3 rows of blocks on either side of a section so that the blocks are essentially 60 m long but each 10 m segment is treated separately for purposes of interpolation. There are 45 levels in the model extending from 1209.5 m to 1007.0 m elevation (1979 CAMC Datum).

Drill hole data was imported as ASCII files into the mine modelling system from the Hewlett Packard HP3000 based database for Vangorda drillhole information. A number of reformatting steps were needed.

3.4.4.3 Composite calculation

The assays were composited on the basis of geology with an attempt to conform to a 4 to 6 m length. This composite coding ensures that an interpreted geologic unit will only be assigned an assay from a length of drill core that was actually used to define the unit. In some cases these defined units will actually contain two or more different geological types but this simplification was necessary in order to produce units that would be reasonable from a mining point of view with the minimum number of necessary ore types. In general these mixed types are related for example: mixed 4E and 4G or mixed 4C and 4A. The minimum composite length was about 1 m. Internal waste was included in the composites at a 0% assay valve. Unlike most previous models, except the F4, bench composites were not used.

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

3.4.4.4 Interpolation

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

elongation: model 090° (ie: along deposit trend)
plunge: - 10° (ie: 10° northwest plunge)
horizontal anisotopy: 1.41
vertical anisotopy: 1.41
maximum range: 90 m (1.5 X section spacing)
minimum # composites used for a block :2
maximum # composites used for a block :8

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

These interpolation parameters allowed most of the blocks to be assigned a value. Two rock types proved difficult to interpolate, 2EH and 2E. Since there is relatively little of this material it was generally not possible to find 2 composites within a search volume. For these rock types the lower limit was reduced to one composite. Any remaining uninterpolated blocks (total of 97) were assigned an SG of 3.0 and grades were left at 0.0% (ie: the blocks do not get counted in reserves).

Specific gravity was treated as an assay and interpolated into blocks based on measured pulp SG of composite intervals. These SG's were determined by air pycnometer on assay pulps consequently do not account for void space. For tight quartzose ore types these SG's are high by about 5% and for vuggy, porous, massive ore types they are high by as much as 10% based on comparison of whole rock and pulp SG's done on Grum, Faro and Dy ores. In order to recast the SG data in terms of whole rock values the SG was reduced by 5% in the final model.

Geological reserves were computed by weighted average of block values between certain Pb plus Zn grades for all levels below topography.

3.4.5 Results

3.4.5.1 Geological Reserves

Geological reserves at a 4% and 6% Pb+Zn cutoff grade are summarized in Table 3.24. The results quoted are for all mineralization regardless of potential pit outlines. There has been no allowance made for dilution or mining recovery in the geological reserves.

3.4.5.2 Comparison of Geological Reserves

Two hand calculations have been done for the entire Vangorda deposit. Both are based on the old drilling and assays for the deposit. Table 3.25 compares these results to the present model.

The oldest one by Prospector Airways is based on the triangular

TABLE 3.24 GEOLOGICAL RESERVES FOR VANGORDA DEPOSIT FROM V8607 MODEL - NO DILUTION

rock type		% of ore type	tonnes (x1000)	density (tn/bcm)	Pb (%)	Zn (%)	Pb + Zn (%)	Ag (g/t)	Au (g/t)
PLUS 6 % Pb + Zn									
4A	1	7.1	382.33	2.81	3.17	4.66	7.83	39.74	0.68
4C	2	0.0	1.42	3.50	6.40	1.01	7.41	57.51	1.77
4EC	3	0.4	21.41	3.65	2.97	3.79	6.75	46.65	0.80
4E	4	0.1	4.16	3.43	2.57	4.12	6.69	46.50	0.63
4EG	5	90.8	4,917.88	3.96	4.51	5.83	10.34	64.03	0.75
4EH	6	1.7	90.23	3.65	6.82	5.05	11.87	83.15	0.54
TOTAL		100.0	5,417.43	3.89	4.45	5.72	10.17	62.55	0.74
4 % TO 6 %									
4A	1	58.6	1,194.40	2.83	1.86	3.02	4.88	26.48	0.46
4C	2	9.5	194.12	3.27	2.34	2.08	4.42	25.82	0.79
4EC	3	14.1	286.99	3.68	2.28	2.40	4.68	33.18	0.85
4E	4	10.2	207.58	3.76	1.90	2.87	4.77	37.94	0.14
4EG	5	6.7	137.28	3.75	2.47	2.76	5.23	36.96	0.76
4EH	6	0.9	19.13	3.78	2.43	2.95	5.38	34.31	0.41
TOTAL		100.0	2,039.51	3.16	2.01	2.81	4.83	29.30	0.53
MINUS 4 %									
4A	1	36.2	3,721.08	2.92	0.90	1.44	2.34	12.48	0.33
4C	2	42.8	4,393.21	3.27	0.79	1.00	1.79	14.50	0.65
4EC	3	17.9	1,842.51	3.67	1.33	1.34	2.67	21.57	0.94
4E	4	2.9	299.17	3.65	1.50	1.40	2.90	17.48	0.28
4EG	5	0.1	5.88	3.23	0.67	0.86	1.53	11.66	0.65
4EH	6	0.1	9.09	2.99	0.39	0.49	0.88	5.95	0.08
TOTAL		100.0	10,270.94	3.23	0.95	1.23	2.18	15.12	0.57
TOTAL DEPOSIT (all grades)									
4A	1	29.9	5,297.81	2.89	1.28	2.03	3.31	17.60	0.38
4C	2	25.9	4,588.75	3.27	0.86	1.04	1.90	15.00	0.66
4EC	3	12.1	2,150.90	3.67	1.47	1.51	2.98	23.37	0.93
4E	4	2.9	510.92	3.69	1.67	2.02	3.69	26.03	0.23
4EG	5	28.5	5,061.05	3.96	4.45	5.74	10.19	63.24	0.75
4EH	6	0.7	118.46	3.62	5.62	4.36	9.98	69.33	0.48
TOTAL		100.0	17,727.88	3.42	2.14	2.78	4.92	31.24	0.62
PLUS 4 %									
4A	1	21.1	1,576.72	2.83	2.18	3.42	5.60	29.69	0.51
4C	2	2.6	195.54	3.27	2.37	2.07	4.44	26.05	0.80
4EC	3	4.1	308.40	3.68	2.32	2.50	4.82	34.11	0.84
4E	4	2.8	211.75	3.75	1.91	2.90	4.81	38.11	0.15
4EG	5	67.8	5,055.17	3.96	4.45	5.74	10.20	63.30	0.75
4EH	6	1.5	109.36	3.67	6.05	4.68	10.74	74.60	0.52
TOTAL		100.0	7,456.94	3.68	3.78	4.92	8.71	53.46	0.69

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

The Kerr Addison reserves are based on a sectional calculation by J. Paxton using sections between 4W and 30E by C.L. Smith in 1966.

These sections show unconnected enechelon pods of ore thus the tonnage should be lower than the current model where the assumptions outlined above imply the pods should be connected into fold patterns.

Previous computer models of Vangorda have only covered the northwest end of the deposit (2W-13E). For purposes of comparison to these models a separate reserve calculation was made for the area covered by these models. The results are given in Table 3.26 and 3.27.

As expected the current model reports higher grades but somewhat unexpectedly the tonnage is higher or comparable. This may be due to the fold interpretation used since zones that might be thought of as disconnected pods are connected into continuous horizons. The tonnage of the V8607 is higher than the 80-01 model (Table 3.27) because the volume of 4% material is about 10% higher, this tends to confirm the above inference and is within the realm of expected variance for the density of drilling.

3.4.5.3 <u>Reliability of Geological Reserves</u>

As a check on model computations, the areas of geological units output during digitizing of unit outlines was used to compute the volume of the overall deposit at a 0% combined cutoff grade. The average SG's of the ore types was used to calculate tonnes of mineralization. This calculation gave a total deposit tonnage of 18,921,000 tonnes compared to 18,660,930 from the block model (all these tonnages are prior to SG reduction), or 18,732,000 if the 97 uninterpolated blocks are included. No check calculation of the grades was done.

There was not sufficient time available to compute reserves with different sets of interpolation parameters. During test interpolation, increasing the degree of anisotopy tended to increase the spread in extreme block values. The effect on the average is not known but presumably above a given cutoff the average grade would increase.

On a small gold deposit modeled with PC Mine (P. Clarke, personal communication, 1986) changing only the anisotopy parameters from isotropic to about the same anisotopy used for

TABLE 3.25 EARLIER GEOLOGICAL RESERVE ESTIMATES FOR THE ENTIRE DEPOSIT COMPARED TO THE V8707

	tonnes (x1000)	density (tn/bcm)	Pb (%)	Zn (%)	Pb+Zn (%)	Ag (g/t)	Au (g/t)	total metal (tonnes)
PROSPECTORS AIRWAYS / CHISHOLM ET AL. (extent unknown - may be 4W to 40E)								
high grade (+4%?)	8,528	4.0 ?	3.16	4.96	8.12	60.33	0.69	692,474
low grade	11,431	----------	----------	----------	not determined	----------	----------	
total deposit	19,959							
KERR-ADDISON / PAXTON (recalculated to 3W to 29E)								
high grade (+4%?)	6,942	4.0 ?	---nd---	---nd---	8.67	---nd---	---nd---	601,871
low grade	9,139	3.5 ?	----------	----------	not determined	----------	----------	
total deposit	16,081							
THIS CALCULATION (3W to 29E)								
high grade (+4%)	7,457	3.68	3.78	4.92	8.71	53.46	0.69	649,224
low grade (-4%)	10,271	3.23	0.95	1.23	2.18	15.12	0.57	
total deposit	17,728	3.42	2.14	2.78	4.92	31.24	0.62	

TABLE 3.26 EARLIER GEOLOGICAL RESERVE ESTIMATES FOR THE NORTHWEST PART OF THE DEPOSIT COMPARED TO THE V8607 MODEL

	tonnes (x1000)	density (tn/bcm)	Pb (%)	Zn (%)	Pb+Zn (%)	Ag (g/t)	Au (g/t)	total metal (tonnes)	% variance (new-old)/old
79-09 MODEL (i.e. the Vangorda Plateau AFE model)									
high grade (+4%)	6,751	4.20	3.5	4.6	8.1	50.7		546,831	-11.4
low grade (-4%)									
total deposit									
80-10 MODEL (i.e. the V1 model)									
high grade (+4%)	5,209	3.83	3.3	5.2	8.5	54.5	0.65	441,608	9.8
low grade (-4%)	4,165								
total deposit	9,375	3.70							
KERR ADDISON / PAXTON (recalculated to 3W to 13E)									
high grade	4,906	4.00			8.6			420,796	15.2
low grade	5,831								
total deposit	10,738								
THIS CALCULATION (3W TO 13E)									
high grade (+4%)	5,471	3.59	3.9	5.0	8.9	55.6	0.72	484,718	
low grade (-4%)	5,695								
total deposit	11,166	3.39	2.4	3.1	5.5	36.7	0.74		

TABLE 3.27 COMPARISON OF COMPUTER MODELS FOR THE NORTHWEST PART OF THE VANGORDA DEPOSIT - V8607 TO 81-10 (or V1)

The 81-07 model was a computer based 3D block model based on more detailed geologic interpretation resulting in geologic bench plans rather than just sections. Block size was 12m X 12m X 6m. Measured S.G. was used.

	81-10 MODEL	86-07 MODEL	VARIANCE (NEW-OLD)/OLD
4.0 % CUTOFF			
TONNES	5209.5	5470.9	5.0
S.G.	3.77	3.59	-4.8
Pb (%)	3.31	3.88	17.1
Zn (%)	4.34	4.98	14.8
Ag (g/t)	47.8	55.6	16.3
Au (g/t)	0.74	0.72	-2.7
METAL (tonnes)	398.5	484.5	21.6
VOLUME (bcm)	1381.5	1524.6	10.4

NOTE: in both 3.27 there is some doubt as to the limit of the volume modeled to the NW. The 86-07 model is definitely quoted for 3W to 13E, the older model may only cover 2W to 13E or possibly 2W to 12E. In the former case the 86-07 tonnage would be about 35,000 tonnes too high, in the latter they would be about 160,000 tonnes too high. In the latter case the deposit volume and tonnage would be within 1%.

Vangorda and Grum increased the tonnage above cutoff by about 5% and the average grade above cutoff by 1 to 2%. This may have no relevance to Vangorda but it does give some idea of the effect of changing only computational parameters.

The above considerations along with concerns about the adequacy of drill spacing suggest that it would not be wise to consider the Vangorda reserves to be better than ±10% to ±20%. This is especially true in the southeast part of the deposit where ±50% might be a more realistic estimate of possible variance to be expected because of the rotary drilling.

When reporting reserves as mill feed it will be necessary to use a high dilution as at Grum.

3.4.6 Additional Work Required

The major shortfall in the work at Vangorda is the drill density. While the density may appear adequate in plan view the picture in section is another matter. On some sections holes have deviated away from one another creating a relatively large gap between the ore intersections. Some of these problems unfortunately occur where large proportions of the tonnage are inferred to exist.

The 200' X 100' drill pattern is probably adequate to define large deposits on a global basis however Vangorda is a small high grade target. Everything rests on the accuracy of the high grade reserves inferred to exist. The current drill density is not adequate to reliably predict the reserves of such a small body. Fill-in sections and fill-in holes on the current sections must be put down at the earliest possible time before committment to this project is made. A staggered triangular pattern should be drilled rather than the current retangular pattern.

The question of reliability of the rotary drilled part of the deposit must be answered. Effort should be made to core this part of the deposit using large diameter core and heavy use of modern artificial mud ingredients.

The other major risk for the Vangorda deposit development is the state of oxidation of the Vangorda ore, this uncertainty must be resolved. Considerable metallurgical work on the fill-in drill core should be carried out in order to answer the question of the extent of oxidation of Vangorda massive sulphides, especially in the southeast part of the deposit.

Metallurgical work should be done on the mixed massive sulphide ore types used in the reserve definition rather than the pure logging ore types that previous work has been keyed to.

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

TABLE 5.1-2
MINING RESERVES

CURRAGH RESOURCES

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

Material	Grade	Volume BCY	Tonnes	HEAD GRADE %Pb+Zn	HEAD GRADE %Pb	HEAD GRADE %Zn	HEAD GRADE Ag g/t	HEAD GRADE Au g/t	CONCENTRATE Pb DMT	CONCENTRATE %Pb	CONCENTRATE Ag g/t	CONCENTRATE Au g/t	CONCENTRATE Zn DMT	CONCENTRATE %Zn	% RECOVERY Pb	% RECOVERY Zn	% RECOVERY Ag	% RECOVERY Au
Waste		21204410.	43051336.															
Calc-Silic		1842640.	3761097.															
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

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° strike, dipping 28° 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

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

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

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

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.

PC-MINE VERSION 1.10 | Curragh Resources | SOFTWARE BY GEMCOM SERVICES INC
SERIAL NO : 20320 | Vangorda 5507 Geological Model | MODULE 1.04
19/11/1986 | | PAGE 1

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	4D 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 : .97
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	1700.00	.0187	.0624	.0366
STOCKPILE	1120.00	800.00	1000.00	.0187	.0624	.0366
WASTE	1120.00	800.00	1000.00	.0187	.0624	.0366

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 0 - S/P	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	4B 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	4EB 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	77.493
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	649.196

TABLE 5.2 - 5 (CONTINUED)

REVENUE/HEAD GRADE INFORMATION - VANGORDA 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

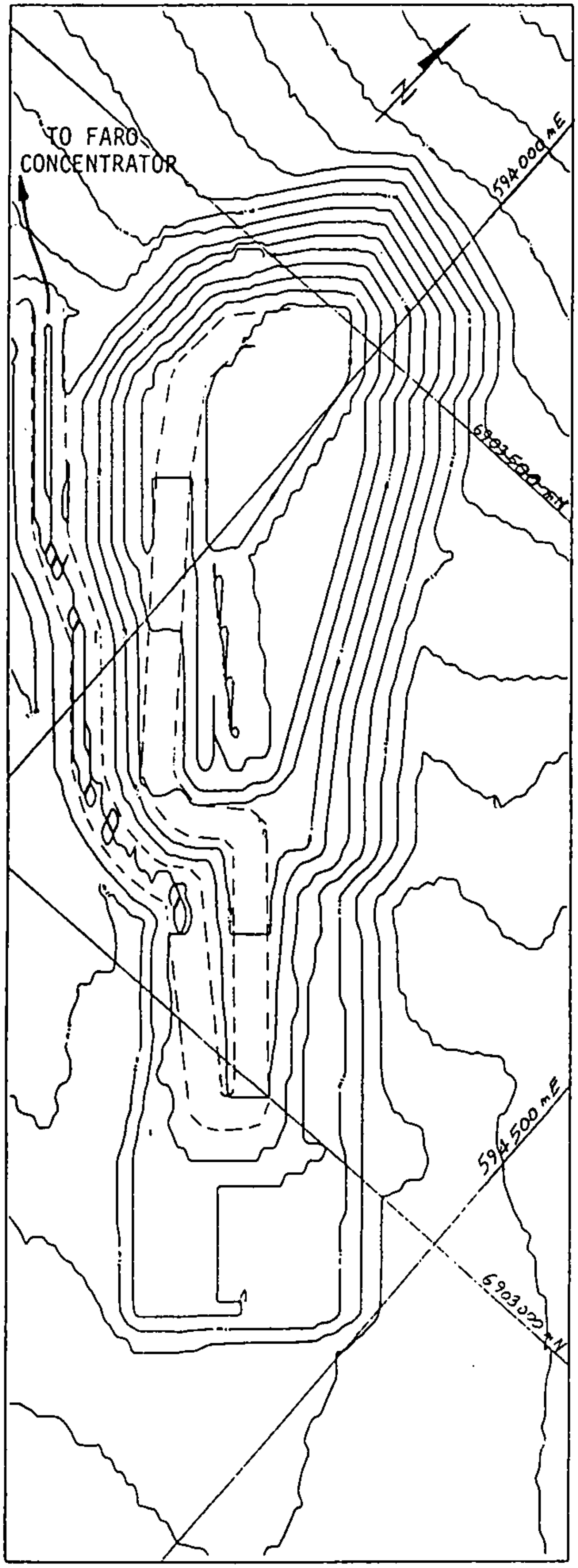
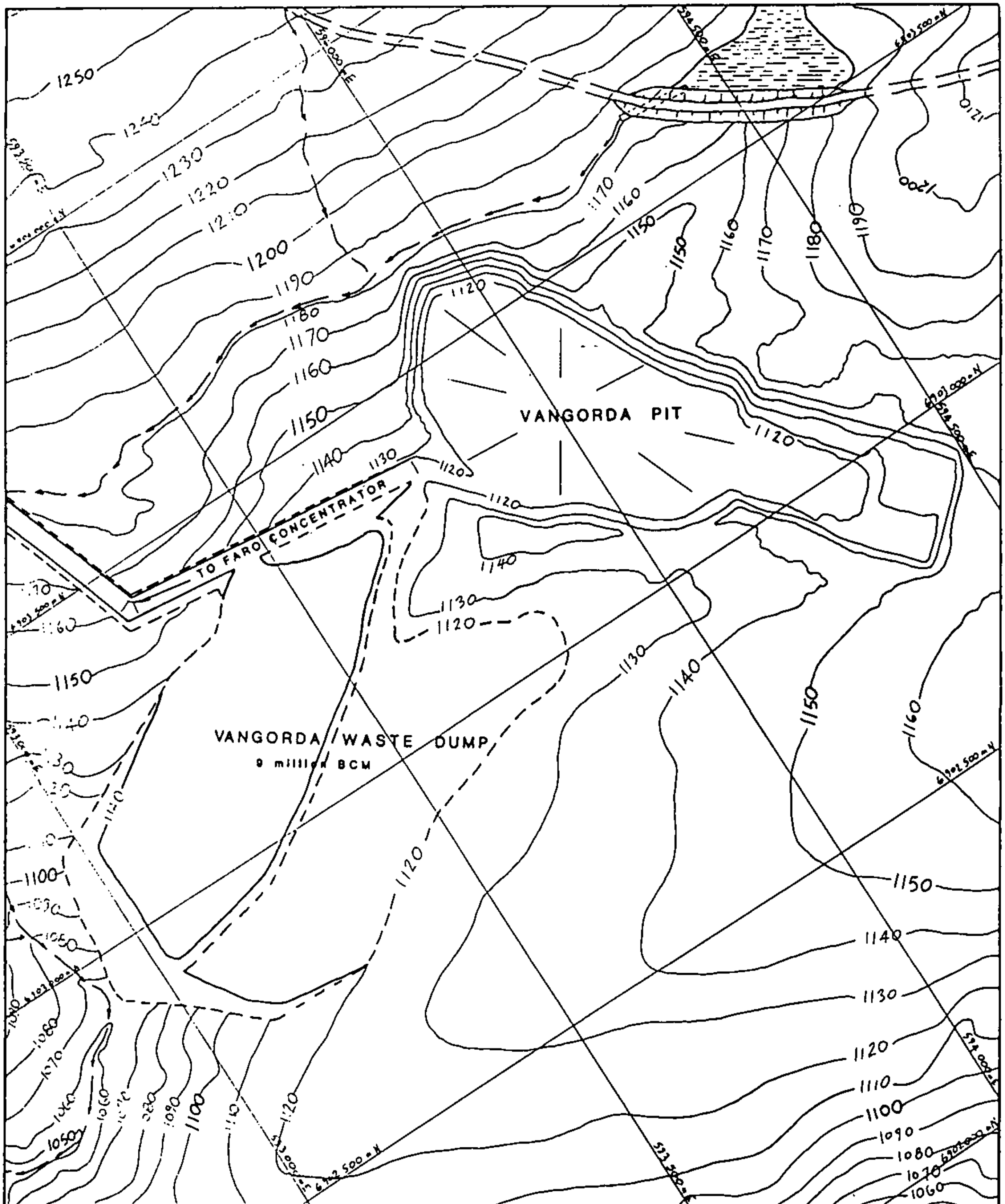


FIGURE 5.2 - 1

VANGORDA ULTIMATE PIT



0 100 200 300 400 500 METRES

CURRAGH RESOURCES	DR.BY: CGY
VANGORDA WASTE DUMP	NOV./86
GENERAL AREA	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.

<u>5.3.4 Pit Reserves</u>

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

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

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

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

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

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.

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 : .66
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	13700.00	.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

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 : 20030
13/11/1986

Curragh Resources
Grum 8606 Geological Model

SOFTWARE BY GEMCOM SERVICES INC
MODULE 1.04
PAGE 1

PRINTOUT OF ROCK-TYPE INFORMATION FOR RECORDS [illegible]

METALLURGICAL DATA

ROCK DESCRIPTION CODE	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, AA, C, E, ...	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 D Type	ore	%Pb+Zn	5.000	4.000	.0	80.0	83.0	63.0	33.0
5 E Type	ore	%Pb+Zn	5.000	4.000	.0	80.0	83.0	64.0	33.0
6 EA 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	649.196

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					
Waste Rock	39207270					
Overburden	15067690					
Sulphide Waste	1254449					
All Waste	55529409					
		%(Pb+Zn)	% Pb	% Zn	Ag g/t	Au g/t
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					
Waste Rock	47445630					
Overburden	2776652					
Sulphide Waste	99787					
All Waste	50322069					
		%(Pb+Zn)	% Pb	% Zn	Ag g/t	Au g/t
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					
Waste Rock	13486850					
Overburden	9447329					
Sulphide Waste	29778					
All Waste	22963957					
		%(Pb+Zn)	% Pb	% Zn	Ag g/t	Au g/t
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
Waste Rock	23499650
Overburden	34733
Sulphide Waste	1704585
All Waste	25238968

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

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

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

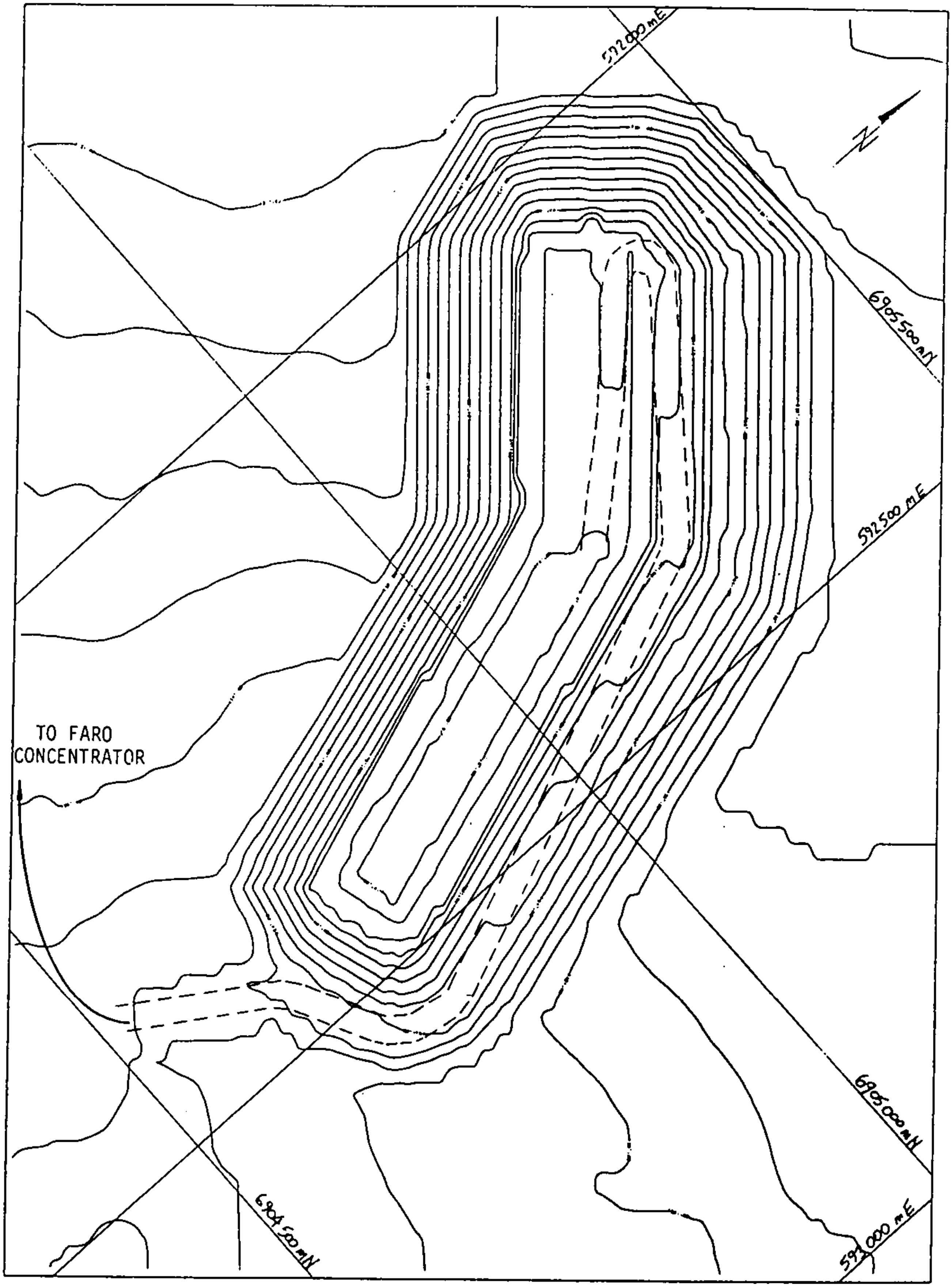


FIGURE 5.3 - 1

GRUM STAGE ONE PIT

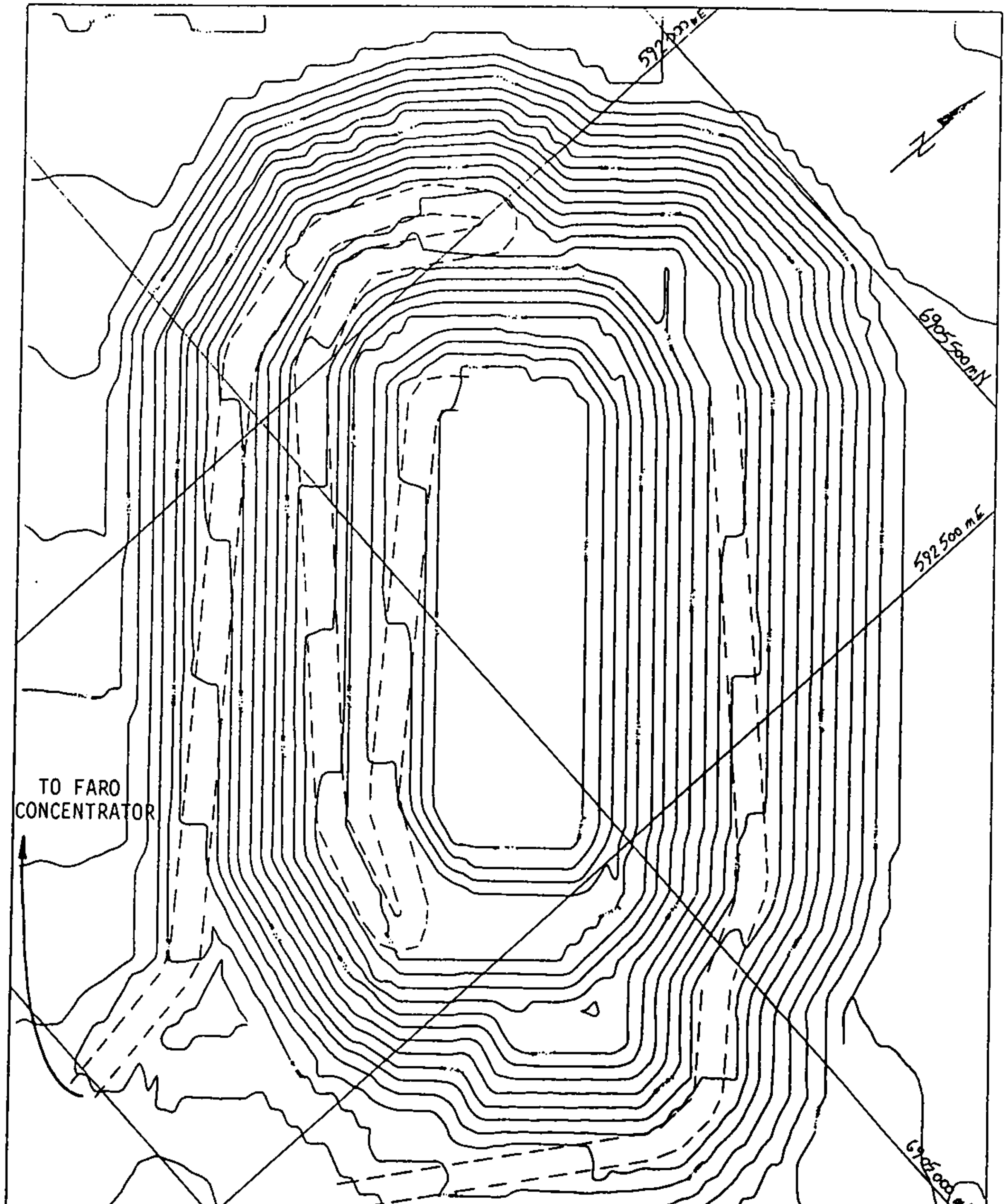
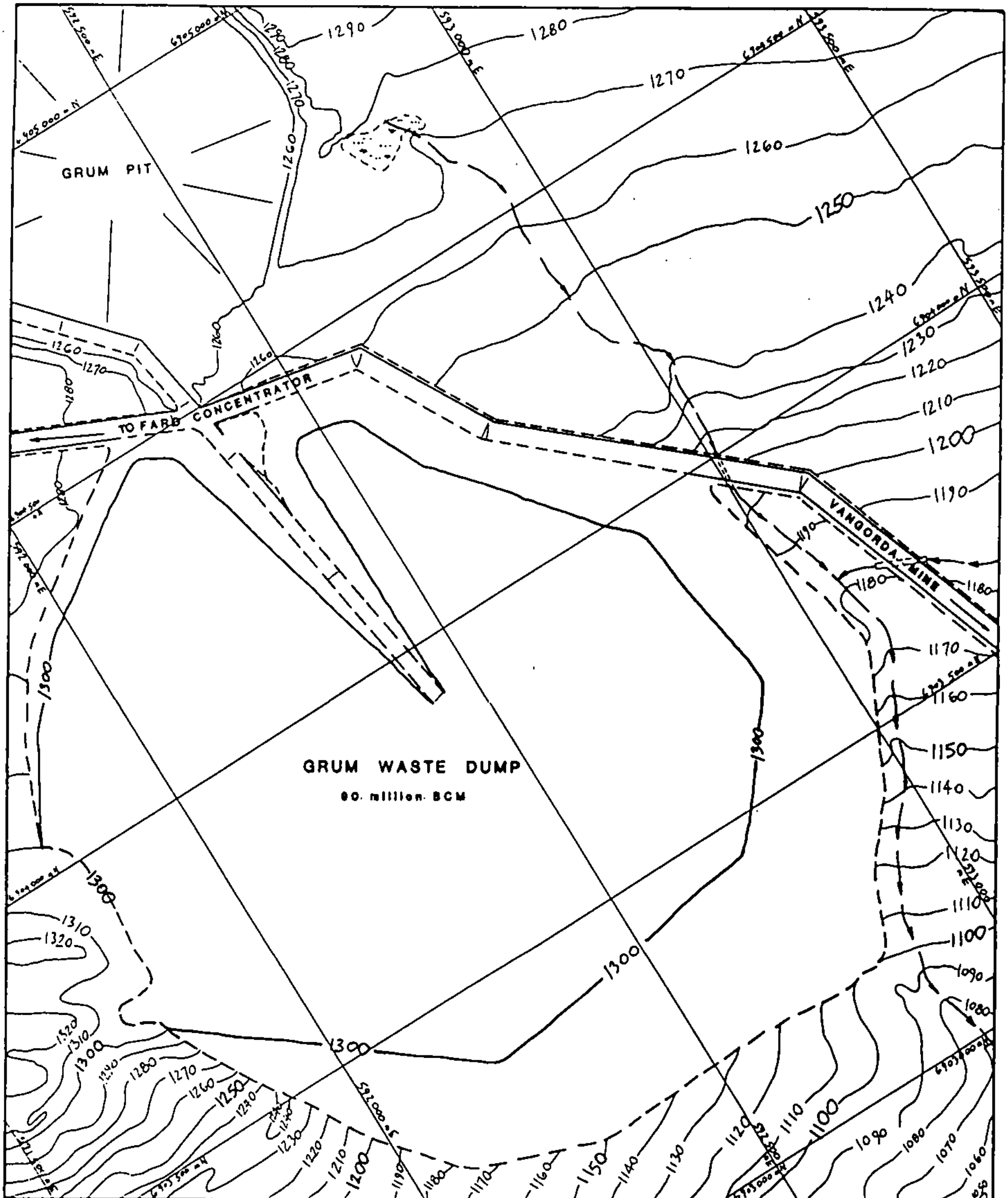


FIGURE 5. 3- 2

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.

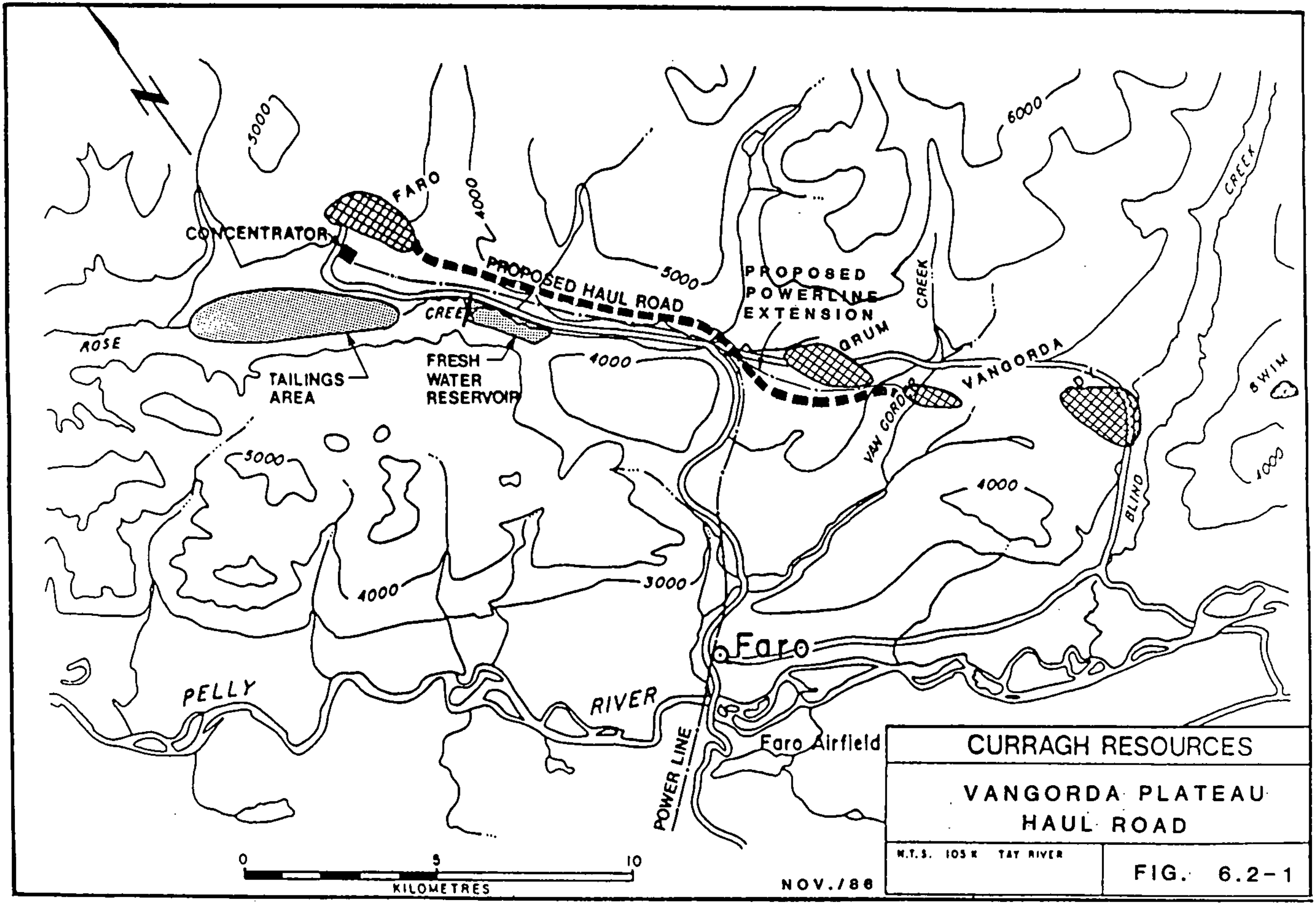
Construction of the haul road has already begun. Beginning in October 1986 waste rock from the pit is being used to construct the first stage of the haul road; the crossing of the North Fork of Rose Creek. Design of the rock drain and creek crossing at this point was done by Golder Associates in September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

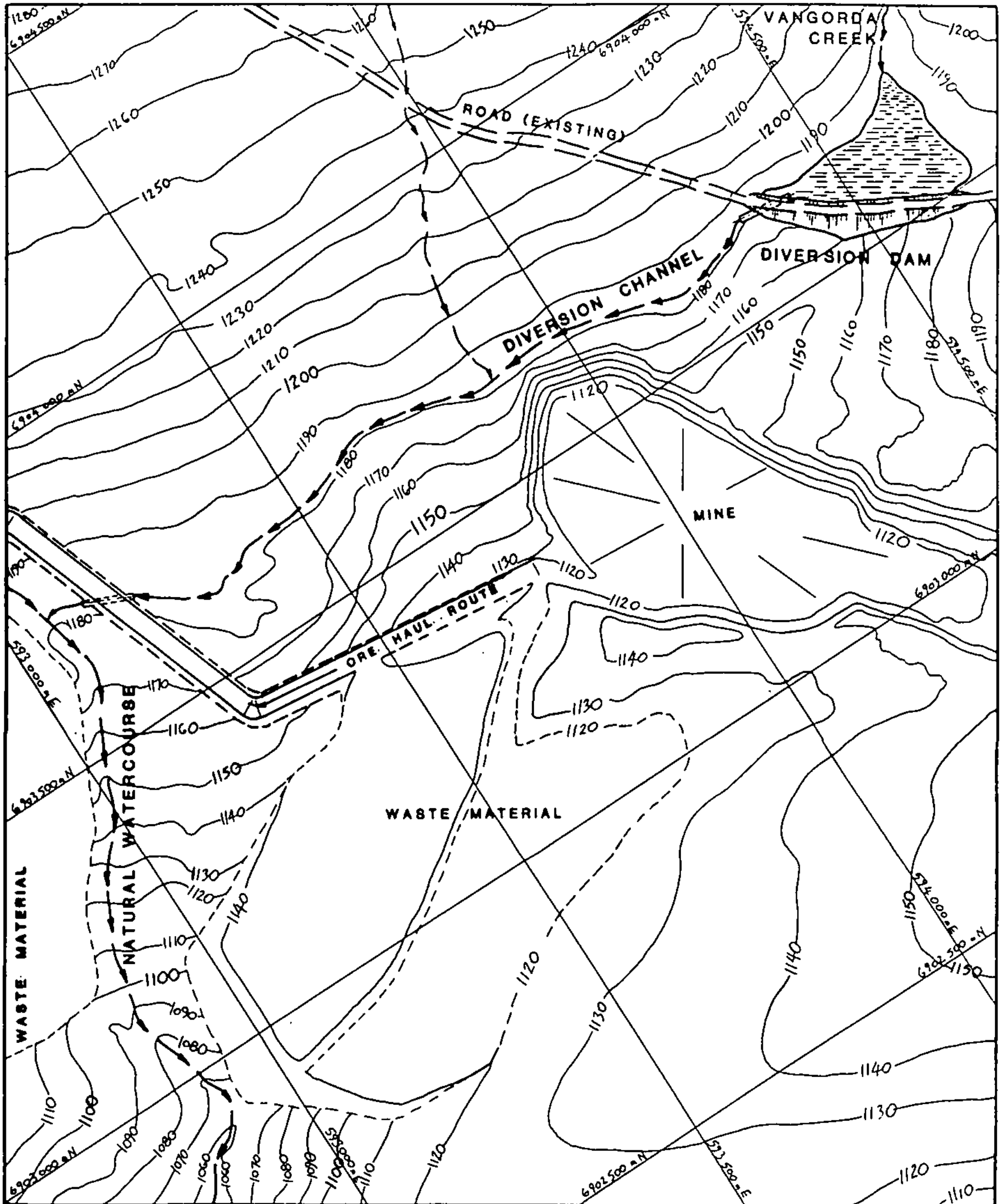
The diversion channel is to be routed on the west valley wall along the 1182 m contour to an existing drainage path, returning the water flow to the original creek bed. The channel will be approximately 1400 meters long. The proposed alignment of the ditch was selected so that the water flow will avoid 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
	NOV./86
PROPOSED 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.

7.0 PRODUCTION SCHEDULE - MINING

7.1 Shovel Productivity

Shovel productivities are forecasted based on current operating experience in the Faro pit, and are detailed in Table 6.1-1. This productivity is defined as the "base rate" or "A" and is for a shovel digging waste on a 12 metre (40 foot) bench.

Shovel productivities in the Faro pit are decreased when necessary to reflect operating conditions. These adjustments are labeled in Table 6.1-2.

Shovel productivities in the Grum and Vangorda pits are reduced to 95.6 percent to represent an average production rate for all operating conditions.

The production rate for the 23 cu. meter (30 yd) shovel scheduled to begin operating in 1991 has been taken at an industry average rate of 18.2 million tonnes per year.

TABLE 7.1-1

Shovel Productivity

	P & H	MARION
tonnes/manned hr (Avg)	1465	1465
Total minutes/12 hr shift	720	720
Delays (minutes):		
Shift change	15	15
Lunch	60	60
Misc.(moving, blasting)	30	30
Net Delay	105	105
Net Manned Minutes	615	615
Net Manned Hours	10.25	10.25
Utilization	85%	85%
Physical Availability	85%	70%
Tonnes/hour	1058	872
Tonnes/calendar day	25,528	21,023
Tonnes/365 day year	9,317,583	7,673,304

Rate Code	% of Base	Description
A	100	40 foot bench, straight waste
B	90	Tight working conditions or low face
C	80	20 foot face, ore
D	70	very low face or poor working conditions

TABLE 7.1-2
Digging Rate Criteria

7.2 Mining Plan

The mining plan has been developed under the following guidelines:

1. Provide a constant mill feed of the best ore possible at the maximum capacity of the mill,
2. Targetted mill feed rates are 13,500 t/d for Faro ore and 11,000 t/d for Vangorda Plateau ore, to a maximum concentrate production rate of 1,600 t/d.
3. As much as possible, delay waste stripping while maintaining item 1 above.
4. As much as possible, maintain the existing operating equipment fleet size.

The annual mining schedule is summarized in Table 7.2-1.

7.2.1 Faro Pit Operations

The annual mining schedule for the Faro pit is presented in Table 7.2-2.

Mining - 1987

During 1987, a maximum of four and a minimum of three shovels will be operating in the Faro pit. Ore production will come from the AY phase of the pit, from the 3570 through 3430 benches. While the AY phase is being mined, the BZ phase will be stripped. By year end the AY phase will be completely mined out and the BZ phase will be producing ore from the 3510 bench.

Case VP 1-5
Mining Summary

	87-04-01 to 87-12-31	1988	1989	1990	1991	1992	1993	1994	1995	1996	1997	1998	1999	Total
Faro Pit														
Waste	18 111 327	15 311 320	5 395 013	7 719 071	4 618 747	2 292 619	0	0	0	0	0	0	0	53 648 097
Ore	4 153 825	6 798 388	3 042 024	157 076	2 578 916	3 852 301	0	0	0	0	0	0	0	20 582 530
Strip Ratio	4.36	2.25	1.77	49.14	1.87	0.60	NA	NA	NA	NA	NA	NA	NA	2.61
Vangorda Pit														
Waste	0	6 665 590	6 029 032	8 792 771	0	0	0	0	0	0	0	0	0	21 487 393
Ore	0	600 216	1 236 775	4 621 775	0	0	0	0	0	0	0	0	0	6 458 766
Strip Ratio	NA	11.11	4.87	1.90	NA	NA	NA	NA	NA	NA	NA	NA	NA	3.33
Grum Pit														
Waste	0	752 556	16 911 210	11 307 260	35 884 742	34 509 264	33 482 098	35 193 404	28 168 891	36 776 571	3 271 869	0	0	236 257 665
Ore	0	0	57 716	265 487	4 335 191	3 922 610	758 530	3 858 001	5 679 132	4 461 736	1 654 986	0	0	24 993 389
Strip Ratio	NA	NA	293.01	42.59	8.28	8.80	44.14	9.12	4.96	8.24	1.98	NA	NA	9.45
All Pits														
Waste	18 111 327	22 729 466	28 335 255	27 819 102	40 703 489	36 801 883	33 482 098	35 193 404	28 168 891	36 776 571	3 271 869	0	0	311 393 355
Ore	4 153 825	7 398 604	4 336 515	5 044 338	6 914 107	7 774 911	758 530	3 858 001	5 679 132	4 461 736	1 654 986	0	0	52 034 685
Strip Ratio	4.36	3.07	6.53	5.51	5.89	4.73	44.14	9.12	4.96	8.24	1.98	NA	NA	5.98
Total Tonnes Mined	22 265 152	30 128 070	32 671 770	32 863 440	47 617 596	44 576 794	34 240 628	39 051 405	33 848 023	41 238 306	4 926 856	0	0	363 428 040

Table 7.2 - 1

Mining - 1988

Three shovels will be operating in the Faro pit at the start of 1988. Ore production will come from the BZ phase as the CZ phase is being stripped. The BZ phase will be completed in mid 1988 as ore production begins in the CZ phase. At year end two shovels are operating in the CZ phase.

Mining - 1989

Only one shovel operates in the Faro pit throughout 1989. Ore production will come from the CZ phase until it is completed in mid year. After that waste stripping of the DY phase will start.

Mining - 1990

Waste stripping of the DY phase continues throughout the year, with only minor amounts of ore being released.

Mining - 1991

Ore production comes from the DY phase.

Mining - 1992

Ore production comes from the DY phase; mining is complete by year end.

87-02-01

Pit Production Summary: Faro

	87-04-01 to 87-12-31	1988	1989	1990	1991	1992	1993	1994	1995	1996	1997	1998	1999	Total
Waste rock	15 338 504	11 731 706	4 999 577	7 377 049	3 367 805	236 689								43051330
Calc Siliicate	1 695 614	1 767 803	2 736	279 089	35 855									3781097
Sulphide Waste	1 077 209	1 811 811	392 700	62 933	1 415 067	2 055 930								6815670
All Waste	18 111 327	15 311 320	5 395 013	7 719 071	4 818 747	2 292 619								53648097
+ 4 % tonnes	4 153 825	6 798 388	3 042 024	157 076	2 578 916	3 852 301								20582530
%Pb+Zn	7.65	7.04	7.27	5.14	6.10	7.34								7.12
%Pb	3.19	2.86	2.92	2.24	2.28	2.59								2.81
%Zn	4.46	4.16	4.35	2.90	3.82	4.76								4.32
Ag g/t	43	38	40	31	24	23								35
Au g/t	0.10	0.08	0.11	0.28	0.12	0.19								0.12
Concentrate Equivalent														
Pb Conc DMT	167 009	241 415	110 594	4 086	70 819	128 792								722715
%Pb	60.95	60.26	60.01	61.30	59.60	57.40								59.82
Ag g/t	576	565	577	596	442	356								520
Au g/t	0.00	0.00	0.00	0.00	0.00	0.00								0.00
Zn Conc DMT	292 995	444 061	207 585	6 676	151 731	289904								1392953
%Zn	50.91	50.85	50.62	50.95	50.88	50.84								50.83
Total Waste Mined	18 111 327	15 311 320	5 395 013	7 719 071	4 818 747	2 292 619								53648097
Total Ore Mined	4 153 825	6 798 388	3 042 024	157 076	2 578 916	3 852 301								20582530
Total Tonnes Mined	22 265 152	22 109 708	8 437 037	7 876 147	7 397 663	6 144 920								74230627
Strip Ratio	4.36	2.25	1.77	49.14	1.87	0.60								2.61

Table 7.2 - 2

7.2.2 Production Schedule - Vangorda Pit

Mining activities in the Vangorda pit will commence on January 1, 1988 with the arrival of the first shovel released from the Faro pit. Mining will be complete by late 1990. A total of 6.5 million tonnes of ore and 18.6 million tonnes of waste will be mined.

Following is an annual summary of activities in the Vangorda pit.

1988

Mining will commence at the 1170 metre elevation at the north end of the pit. The Vangorda Creek diversion channel will be constructed early in the year, and some of the unconsolidated overburden excavated will be used to construct the diversion dam.

During the year, approximately 500,000 tonnes of ore will be mined and stockpiled; this ore will primarily come from the southeast end of the deposit. This ore has been characterized as oxidized, but the extent and degree of oxidation has not been quantified, and no allowance for oxidation has been made in the mill production forecast.

By year end, mining will be at the 1130 metre elevation; 6.7 million tonnes of waste will have been mined.

Pit status at the end of 1988 is shown in figure 7.2-1.

1989

During 1989, approximately 1.2 million tonnes of ore will be mined. This ore will feed the Faro Concentrator during the fourth quarter. In contrast to the previous year's ore production, most of this ore will come from the northeast part of the deposit; this ore has not been characterized as oxidized.

By year end, an additional 6.0 million tonnes of waste will have been removed and the pit will be at the 1110 metre elevation.

Pit status at the end of 1989 is shown in figure 7.2-2.

1990

During 1990, the Vangorda Pit is to supply the total feed to the Faro concentrator. In order to meet the production requuirements, a second shovel will move into the pit early in 1990. Mining will be complete late in 1990; 4.6 million tonnes of ore and 8.8 million tonnes of waste will have been mined.

Pit status at the end of 1990 is shown in figure 7.2-3.

Table 7.2-3 details the annual ore production for the Vangorda Pit.

Pit Production Summary: Vangorda

	87-04-01 to 87-12-31	1988	1989	1990	1991	1992	1993	1994	1995	1996	1997	1998	1999	Total
Waste rock		1176277	1972163	5445180										8593620
Overburden		5214587	3196395	658416										9069398
Sulphide Waste		274726	660474	2689175										3624375
All Waste		6665590	6029032	8792771										21487393
+ 5 % tonnes		501642	1093336	4229724										5824702
%Pb+Zn		7.31	8.04	8.77										8.51
%Pb		3.12	3.42	3.85										3.71
%Zn		4.19	4.62	4.92										4.80
Ag g/t		44	50	54										52
Au g/t		0.49	0.58	0.62										0.60
Concentrate Equivalent														
Pb Conc DMT		24159	58614	259969										342742
%Pb		53.06	53.16	53.31										53.27
Ag g/t		608	617	602										605
Au g/t		4.04	4.31	3.87										3.96
Zn Conc DMT		29647	72594	302267										404508
%Zn		54.9	54.71	54.7										54.72
4 - 5 % tonnes		98574	143439	392051										634064
%Pb+Zn		3.86	3.9	3.91										3.90
%Pb		1.6	1.64	1.72										1.68
%Zn		2.26	2.26	2.19										2.22
Ag g/t		27	29	24										26
Au g/t		0.35	0.45	0.6										0.53
Concentrate Equivalent														
Pb Conc DMT		1984	2876	8486										13346
%Pb		54.23	56.51	55.25										55.37
Ag g/t		616	664	519										565
Au g/t		3.44	3.05	5.95										4.95
Zn Conc DMT		2828	4151	10957										17936
%Zn		52.45	52.69	52.74										52.68
Total Waste Mined		6665590	6029032	8792771										21487393
Total Ore Mined		600216	1236775	4621775										6458766
Total Tonnes Mined		7265806	7265807	13414546										27946159
Strip Ratio		11.11	4.87	1.90										3.33

Table 7.2 - 3

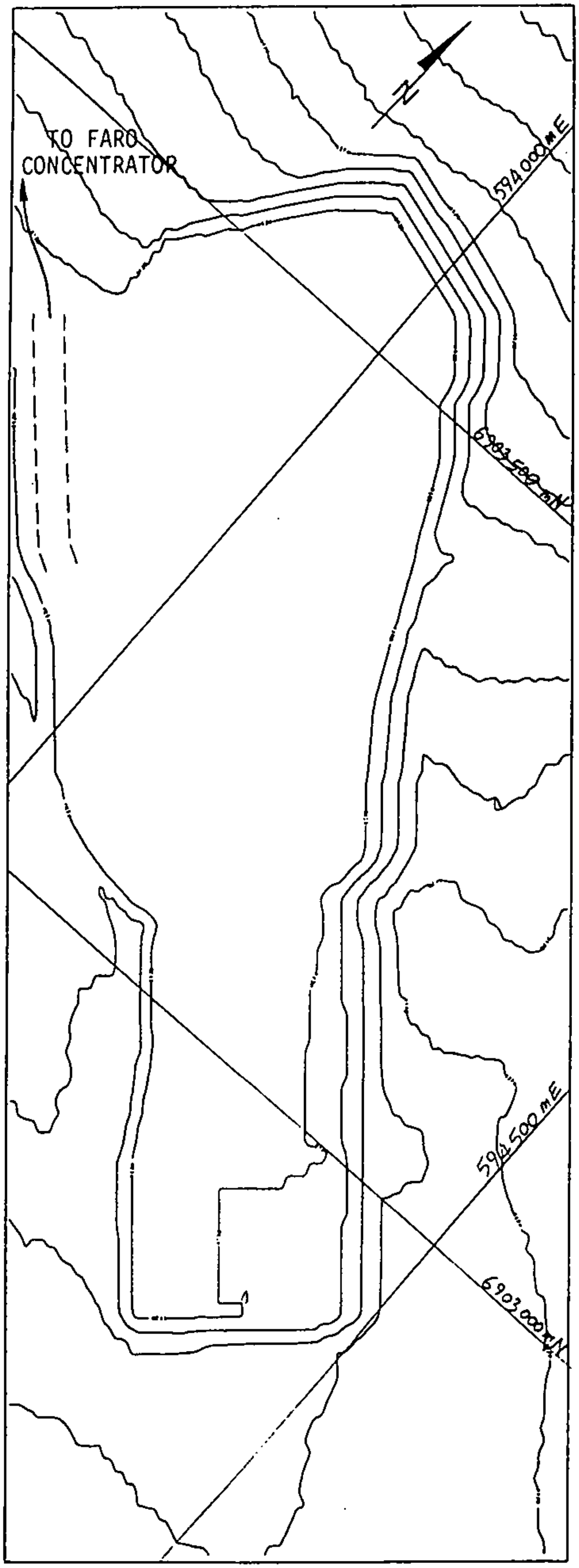


FIGURE 7. 2- 1

VANGORDA PIT END OF 1988

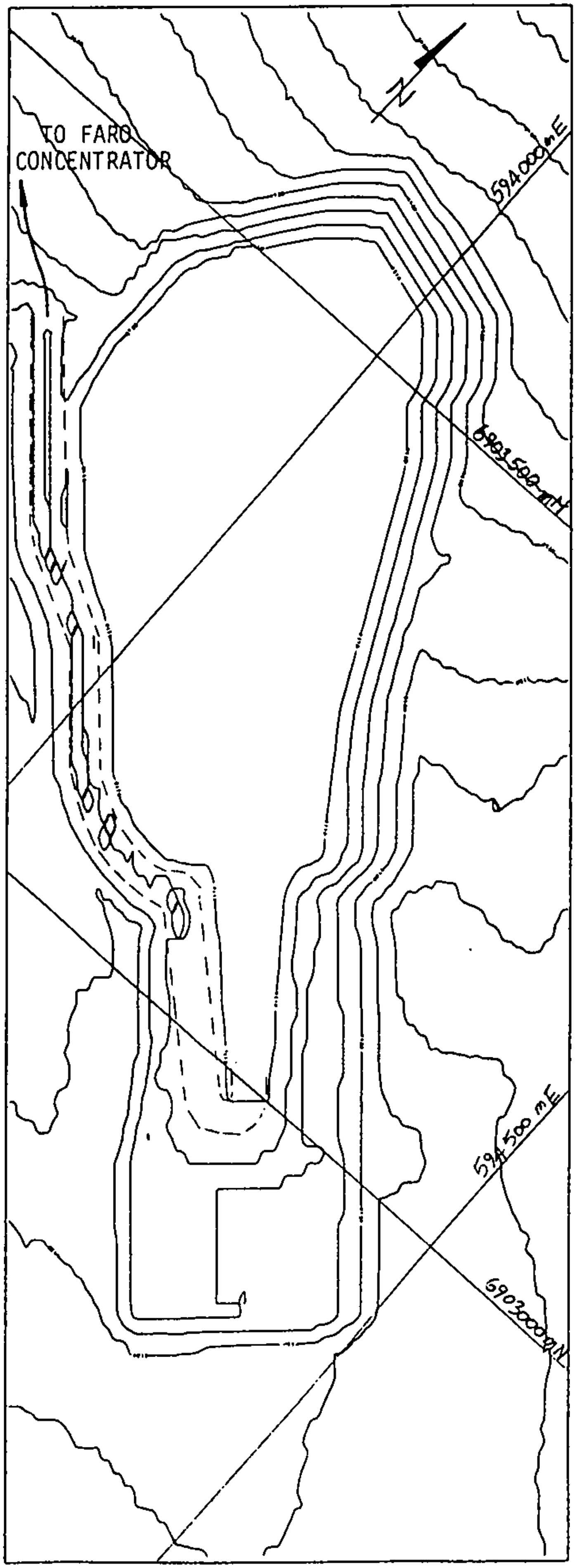


FIGURE 7.2 - 2

VANGORDA PIT END OF 1989

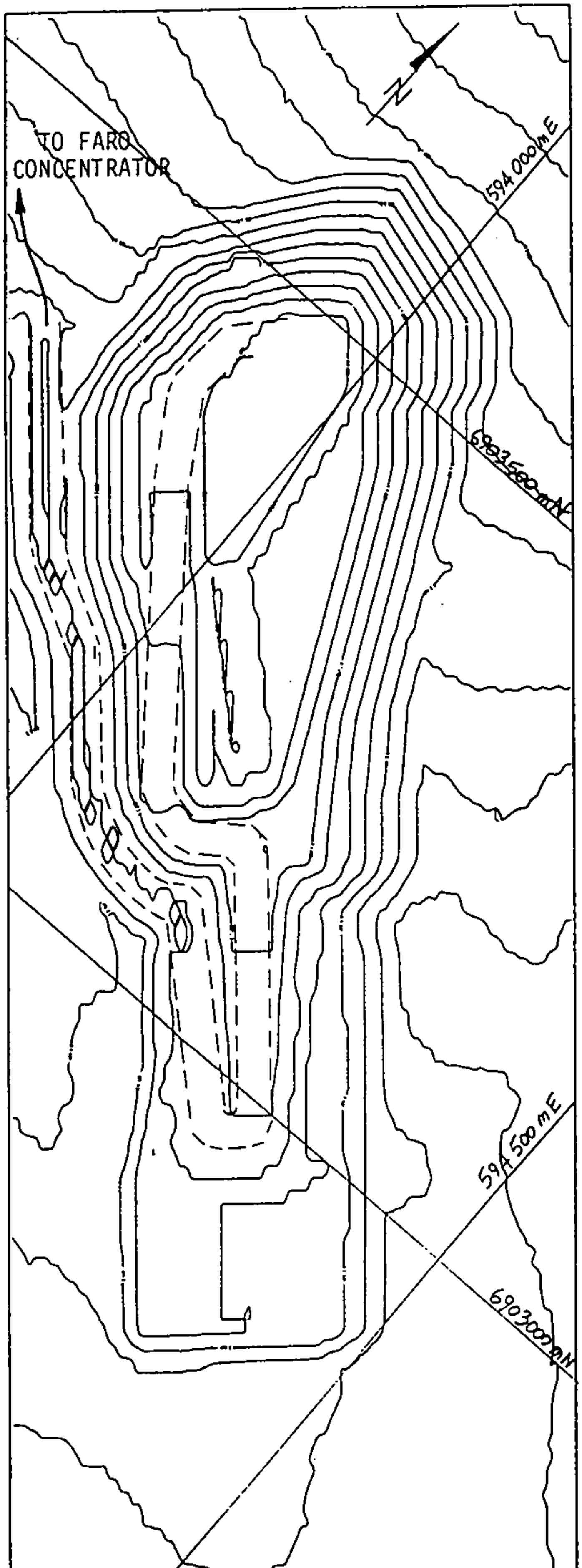


FIGURE 7.2 - 3

VANGORDA PIT END OF 1990

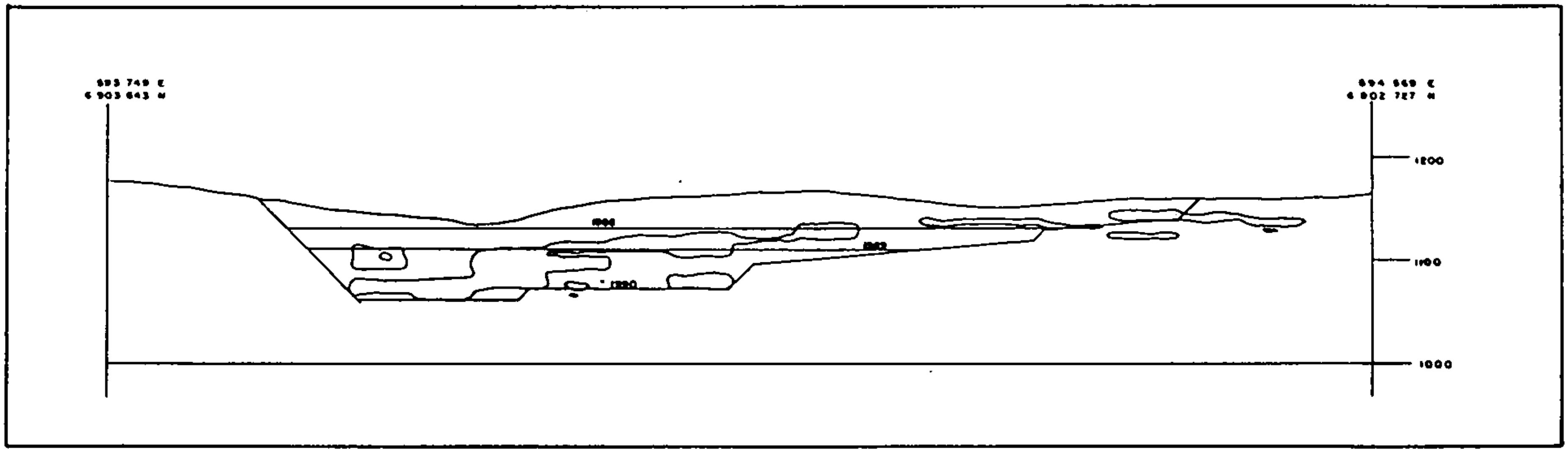


FIGURE 7.2 - 4

Longitudinal Section Through Vangorda Pit

7.2.3 Production Schedule - Grum Pit

Mining activities in the Grum Pit will commence in late 1988 and continue until 1998. A total of 26 million tonnes of ore and 234 million tonnes of waste will be mined.

Following is an annual summary of activities in the Grum Pit.

1988

Mining will begin late in 1988 with the release of the second shovel from the Faro Pit. Stripping of the Stage One pit will commence at the 1310 metre elevation. No ore will be released in 1988; approximately 750,000 tonnes of waste will be mined.

1989

Early in 1989, the third shovel to be released from the Faro pit will move to the Grum Pit, for a total of two shovels operating in this pit. These shovels will continue to strip the Stage One pit, with only minor amounts of ore (60,000 tonnes) being mined and stockpiled. About 17 million tonnes of waste will be mined.

Pit status at the end of 1989 is shown in figure 7.2-4.

1990

Only one shovel will be operating in Grum through most of 1990 as production emphasis is shifted to the Vangorda Pit. By the end of the year however, the Vangorda Pit will be complete and all three shovels will be operating in the Grum Pit. About 260,000 tonnes of ore will be mined and stockpiled; 11.3 million tonnes of waste will be mined.

Pit status at the end of 1990 is shown in figure 7.2-5.

1991

During 1991, much of the ore-bearing portion of the Stage One pit will be mined as the pit deepens from the 1230 metre elevation to the 1180 metre elevation. Most of the concentrator feed this year will be from the Grum Pit; 4.3 million tonnes of ore will be mined along with 21 milliion tonnes of waste.

Pit status at the end of 1991 is shown in figure 7.2-6.

1992

Stage One will be completed in mid 1992 as the Faro pit again starts to supply the concentrator with ore.

The remainder of the Stage One pit will release 3.5 million tonnes of ore. When Stage One has been completed, stripping will begin on the north and south sides of the Stage Two pit.

A total of 21 milliion tonnes of waste will be mined.

Pit status at the end of 1992 is shown in figure 7.2-7.

1993

A fourth shovel will begin working in the Grum pit in mid 1993, as Faro pit operations cease.

Stripping of the north and south sides of the Stage Two pit will continue through 1993, with only minor amounts of ore being released. A total of 29 million tonnes of waste and 75,000 tonnes of ore will be mined.

Pit status at the end of 1993 is shown in figure 7.2-8.

1994

Stripping on both sides of the Stage Two pit will continue until late 1994 before any appreciable amount of ore is released. A total of 800,000 tonnes of ore and 33 million tonnes of waste will be mined.

Pit status at the end of 1994 is shown in figure 7.2-9.

1995

Mining continues in the Stage Two pit in 1995, releasing 3.9 million tonnes of ore. Also, stripping of the Stage Three pit begins in this year. A total of 30 million tonnes of waste is to be mined this year.

Pit status at the end of 1995 is shown in figure 7.2-10.

1996

The year 1996 sees the completion of the Stage Two pit and continued stripping of Stage Three. A total of 5.7 million tonnes of ore and 28 million tonnes of waste are to be mined.

Pit status at the end of 1996 is shown in figure 7.2-11.

1997

Continued mining of the Stage Three pit releases 2.0 million tonnes of ore and 32 million tonnes of waste.

Pit status at the end of 1997 is shown in figure 7.2-12.

1998

The Grum pit is to be completely mined out in 1998; 4.1 million tonnes of ore and 8 million tonnes of waste are to be mined.

Pit status at the end of 1998 is shown in figure 7.2-13.

Tables 7.2-4 through 7.2-7 detail the annual production schedule for the Grum Pit.

Pit Production Summary: Grum Total

	1988	1989	1990	1991	1992	1993	1994	1995	1996	1997	1998	1999	Total
Waste rock	110054	9560761	6598990	27437797	27525833	30859340	32966253	26121767	35893272	3046568			200120635
Overburden	642502	7347807	4692635	7736435	6451748	2474821	1756006	826249	69507	0			31997710
Sulphide Waste	0	2642	15635	710510	531683	147937	471145	1220875	813791.8	225301.2			4139520
All Waste	752556	16911210	11307260	35884742	34509264	33482098	35193404	28168891	36776570.8	3271869.2			236257865
+ 5 % tonnes		57716	212216	3540616	3525872	621260	3435351	4828997	4013030.6	1505068.4			21740127
%Pb+Zn		5.89	5.55	7.26	7.47	10.87	9.15	9.48	8.70	8.99			8.55
%Pb		2.21	1.93	2.60	2.68	4.43	3.47	3.47	3.32	3.35			3.18
%Zn		3.68	3.62	4.66	4.78	6.44	5.68	6.01	5.37	5.64			5.38
Ag g/t		37	33	42	44	73	58	59	57	59			53
Au g/t		0.48	0.67	0.63	0.68	0.87	0.88	0.96	0.91	1.01			0.84
Concentrate Equivalent													
Pb Conc DMT		1564	4856	117074	121162	38549	160408	224984	177049.8	66901.2			912554
%Pb		60.00	60.00	60.00	60.00	60.00	60.00	60.00	60.00	60.00			60.00
Ag g/t		810	828	800	813	817	826	844	849	876			833
Au g/t		5.31	8.42	5.97	6.27	4.85	6.22	6.86	6.82	7.51			6.52
Zn Conc DMT		3055	10942	246695	254338	63151	303740	453999	330560	130736			1797216
%Zn		55.00	55.00	55.00	55.00	55.00	55.00	55.00	55.00	55.00			55.00
4 - 5 % tonnes			53271	794575	396738	137270	422650	850135	448705	149918			3253262
%Pb+Zn			3.78	3.89	3.97	4.06	3.88	3.87	3.86	3.88			3.89
%Pb			1.26	1.28	1.29	1.73	1.53	1.54	1.55	1.55			1.45
%Zn			2.52	2.61	2.68	2.33	2.35	2.33	2.31	2.33			2.44
Ag g/t			24	25	24	31	27	26	28	27			26
Au g/t			0.77	0.46	0.45	0.82	0.63	0.64	0.78	0.79			0.61
Concentrate Equivalent													
Pb Conc DMT			726	10963	5546	2763	7281	14838	7870.2	2634.8			52642
%Pb			60.00	60.00	60.00	60.00	60.00	60.00	60.00	60.00			60.00
Ag g/t			897	912	871	864	834	816	879	842			859
Au g/t			14.66	8.76	8.42	11.67	10.11	10.02	12.22	12.39			10.20
Zn Conc DMT			1787	27870	14341	4208	13079	25985	13590.6	4584.4			105445
%Zn			55.00	55.00	55.00	55.00	55.00	55.00	55.00	55.00			55.00
Total Waste Mined	752556	16911210	11307260	35884742	34509264	33482098	35193404	28168891	36776570.8	3271869.2			236257865
Total Ore Mined	0	57716	265487	4335191	3922610	758530	3858001	5679132	4461735.6	1654986.4			24993389
Total Tonnes Mined	752556	16968926	11572747	40219933	38431874	34240628	39051405	33848023	41238306.4	4926855.6			261251254
Strip Ratio		293.01	42.59	8.28	8.80	44.14	9.12	4.96	8.24	1.98			9.45

Table 7.2 - 4

Pit Production Summary: Grue 1

	1988	1989	1990	1991	1992	1993	1994	1995	1996	1997	1998	1999	Total
Waste rock	110054	9560761	6598990	17556870	5380593								39207268
Overburden	642502	7347807	4692635	2384716									15067660
Sulphide Waste		2642	15635	710510	525661								1254448
All Waste	752556	16911210	11307260	20652096	5906254								55529376
+ 5 % tonnes		57716	212216	3540616	3451647								7262195
%Pb+Zn		5.89	5.55	7.26	7.36								7.25
%Pb		2.21	1.93	2.6	2.63								2.59
%Zn		3.68	3.62	4.66	4.73								4.66
Ag g/t		37	33	42	43								42
Au g/t		0.48	0.67	0.63	0.68								0.65
Concentrate Equivalent													
Pb Conc DMT		1564	4856	117074	115608								239102
%Pb		60	60	60	60								60.00
Ag g/t		810	828	800	812								906
Au g/t		5.31	8.42	5.97	6.4								6.22
Zn Conc DMT		3055	10942	246695	245643								506335
%Zn		55	55	55	55								55.00
4 - 5 % tonnes			53271	794575	396738								1244584
%Pb+Zn			3.78	3.89	3.97								3.91
%Pb			1.26	1.28	1.29								1.28
%Zn			2.52	2.61	2.68								2.63
Ag g/t			24	25	24								25
Au g/t			0.77	0.46	0.45								0.47
Concentrate Equivalent													
Pb Conc DMT			726	10983	5546								17255
%Pb			60	60	60								60.00
Ag g/t			897	912	871								898
Au g/t			14.66	8.76	8.42								8.90
Zn Conc DMT			1787	27870	14341								43998
%Zn			55	55	55								55.00
Total Waste Mined	752556	16911210	11307260	20652096	5906254								55529376
Total Ore Mined	0	57716	265487	4335191	3848385								8506779
Total Tonnes Mined	752556	16968926	11572747	24987287	9754639								64036155
Strip Ratio		293.01	42.59	4.76	1.53								6.53

Table 7.2 - 5

Pit Production Summary: Grum 2

	1988	1989	1990	1991	1992	1993	1994	1995	1996	1997	1998	1999	Total
Waste rock				9880927	22145240	30859340	13363200	6184547					64433254
Overburden				5351719	6451748	421737	29135	4413					12258752
Sulphide Waste					6022	147937	471145	1209048					1834152
All Waste				15232646	28603010	31429014	13863480	9398008					98526158
+ 5 % tonnes					74225	621260	3435351	4796979					8927815
%Pb+Zn					12.49	10.87	9.15	9.46					9.46
%Pb					5.17	4.43	3.47	3.47					3.55
%Zn					7.32	6.44	5.68	5.99					5.91
Ag g/t					87	73	58	59					60
Au g/t					0.75	0.87	0.88	0.96					0.92
Concentrate Equivalent													
Pb Conc DMT					5560	38549	160408	223193					427710
%Pb					60	60	60	60					60.00
Ag g/t					840	817	826	844					835
Au g/t					3.57	4.85	6.22	6.67					6.40
Zn Conc DMT					8695	63151	303740	449539					825125
%Zn					55	55	55	55					55.00
4 - 5 % tonnes						137270	422650	848849					1408769
%Pb+Zn						4.06	3.86	3.87					3.89
%Pb						1.73	1.53	1.54					1.56
%Zn						2.33	2.35	2.33					2.34
Ag g/t						31	27	26					27
Au g/t						0.82	0.63	0.64					0.65
Concentrate Equivalent													
Pb Conc DMT						2763	7281	14812					24856
%Pb						60	60	60					60.00
Ag g/t						864	834	816					827
Au g/t						11.67	10.11	10.03					10.24
Zn Conc DMT						4208	13079	25948					43235
%Zn						55	55	55					55.00
Total Waste Mined				15232646	28603010	31429014	13863480	9398008					98526158
Total Ore Mined				0	74225	758530	3858001	5645828					10336584
Total Tonnes Mined				15232646	28677235	32187544	17721481	15043836					108862742
Strip Ratio					385.36	41.43	3.59	1.66					9.53

Table 7.2 - 6

Pit Production Summary: Grum 3

	1988	1989	1990	1991	1992	1993	1994	1995	1996	1997	1998	1999	Total
Waste rock							19603053	17937220	35893272	3046568			76480113
Overburden						2053084	1726871	821836	69507				4671298
Sulphide Waste								11827	613792	225301			1050920
All Waste						2053084	21329924	18770883	31868767	8179673			82202331
+ 5 % tonnes								32018	4013031	1505068			5550117
%Pb+Zn								12.54	8.70	8.99			8.80
%Pb								4.07	3.32	3.35			3.34
%Zn								8.47	5.37	5.64			5.46
Ag g/t								69	57	59			57
Au g/t								0.93	0.91	1.01			0.94
Concentrate Equivalent													
Pb Conc DMT								1791	177050	66901			245742
%Pb								60	60	60			60.00
Ag g/t								842	849	876			856
Au g/t								5.75	6.82	7.51			7.00
Zn Conc DMT								4460	330560	130736			465756
%Zn								55	55	55			55.00
4 - 5 % tonnes								1286	448705	149918			599909
%Pb+Zn								3.99	3.86	3.88			3.87
%Pb								1.76	1.55	1.55			1.55
%Zn								2.23	2.31	2.33			2.31
Ag g/t								26	28	27			28
Au g/t								0.42	0.78	0.79			0.78
Concentrate Equivalent													
Pb Conc DMT								26	7870	2635			10531
%Pb								60	60	60			60.00
Ag g/t								717	879	842			870
Au g/t								5.8	12.22	12.39			12.25
Zn Conc DMT								37	13591	4584			18212
%Zn								55	55	55			55.00
Total Waste Mined						2053084	21329924	18770883	31868767	8179673			82202331
Total Ore Mined						0	0	33304	4461736	1654986			6150026
Total Tonnes Mined						2053084	21329924	18804187	36330503	9834659			88352357
Strip Ratio									7.14	4.94			13.37

Table 7.2 - 7

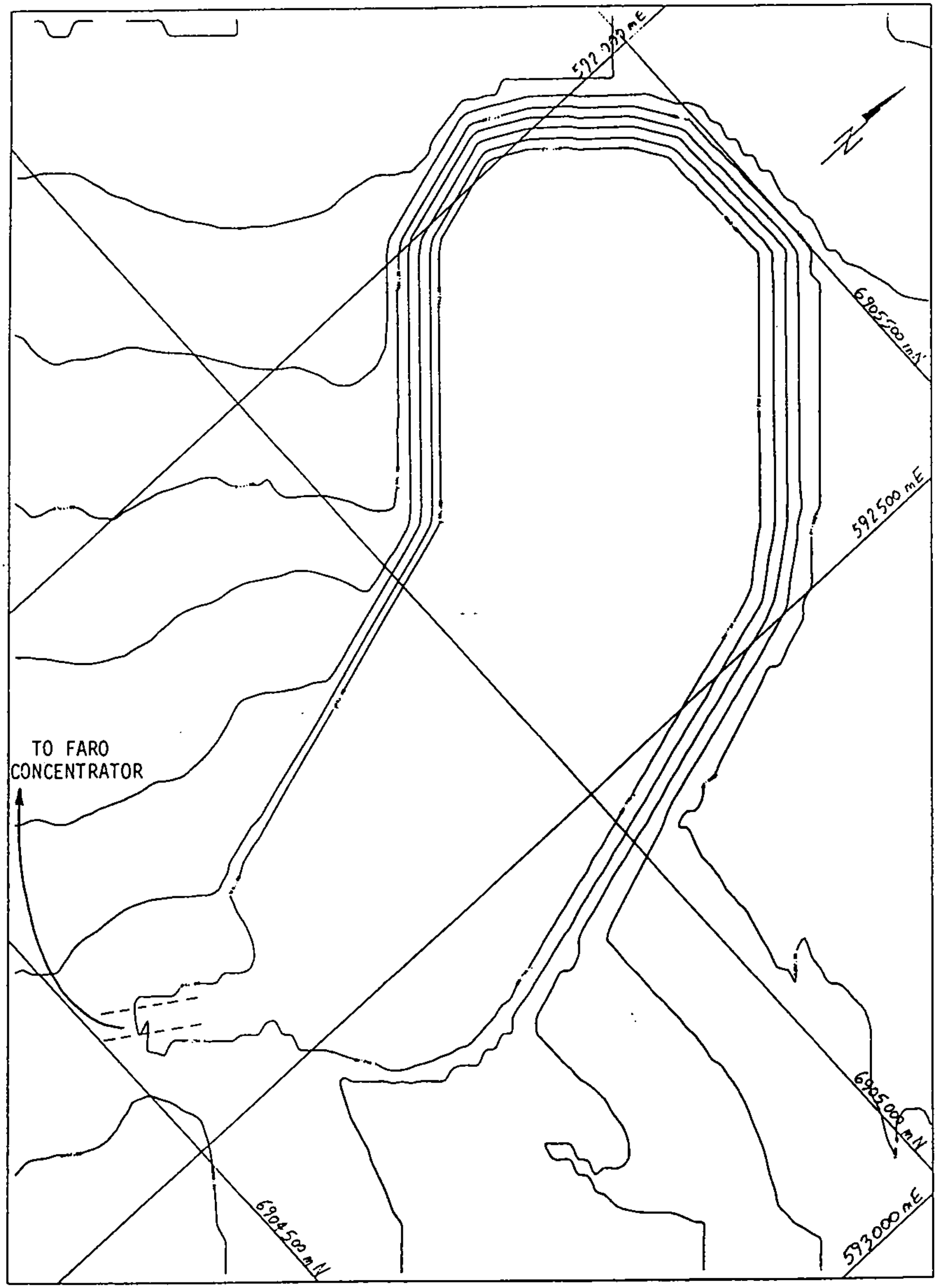


FIGURE 7.2 - 5

GRUM PIT END OF 1989

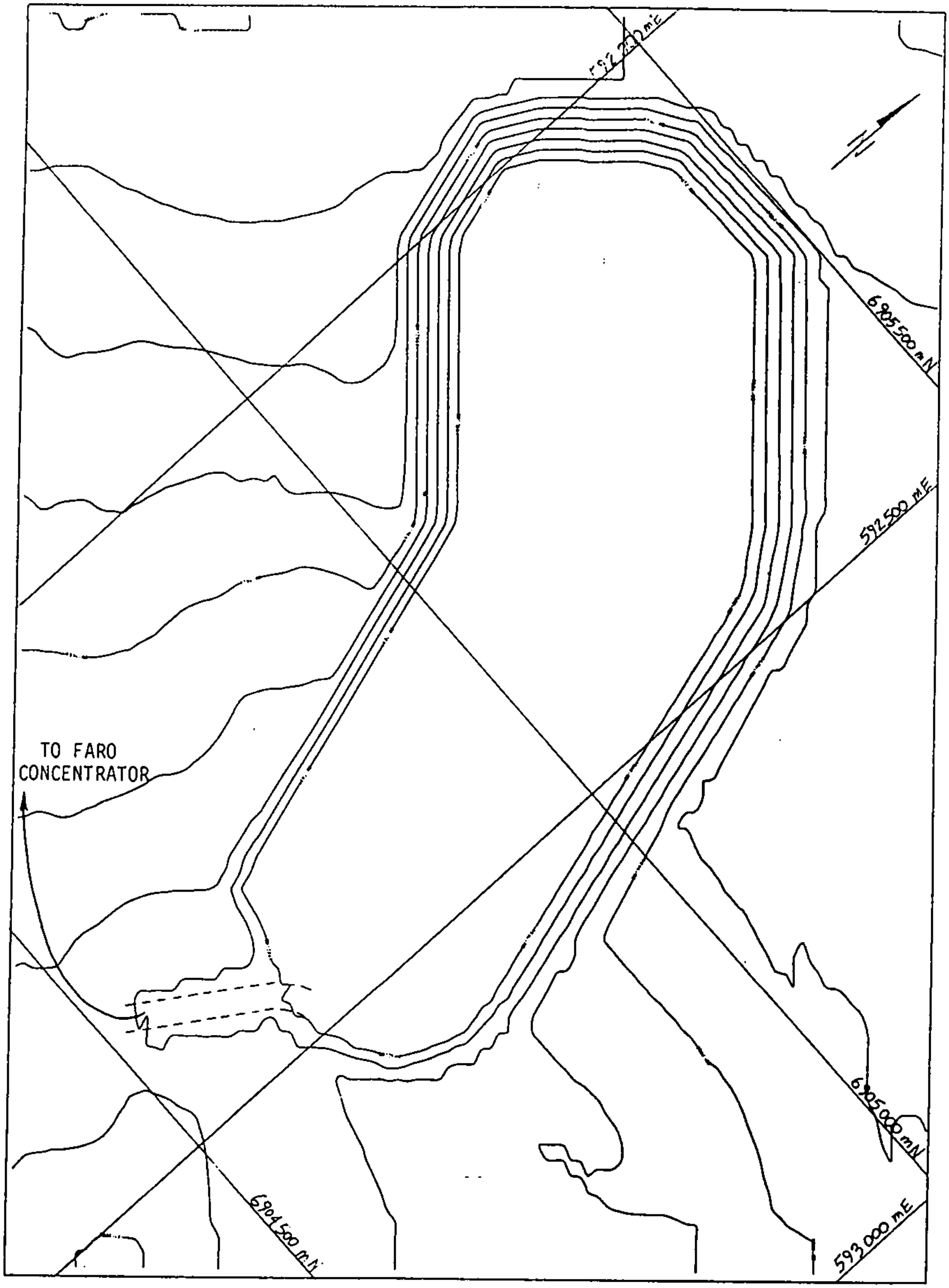


FIGURE 7.1.2 - 6

GRUM PIT END OF 1990

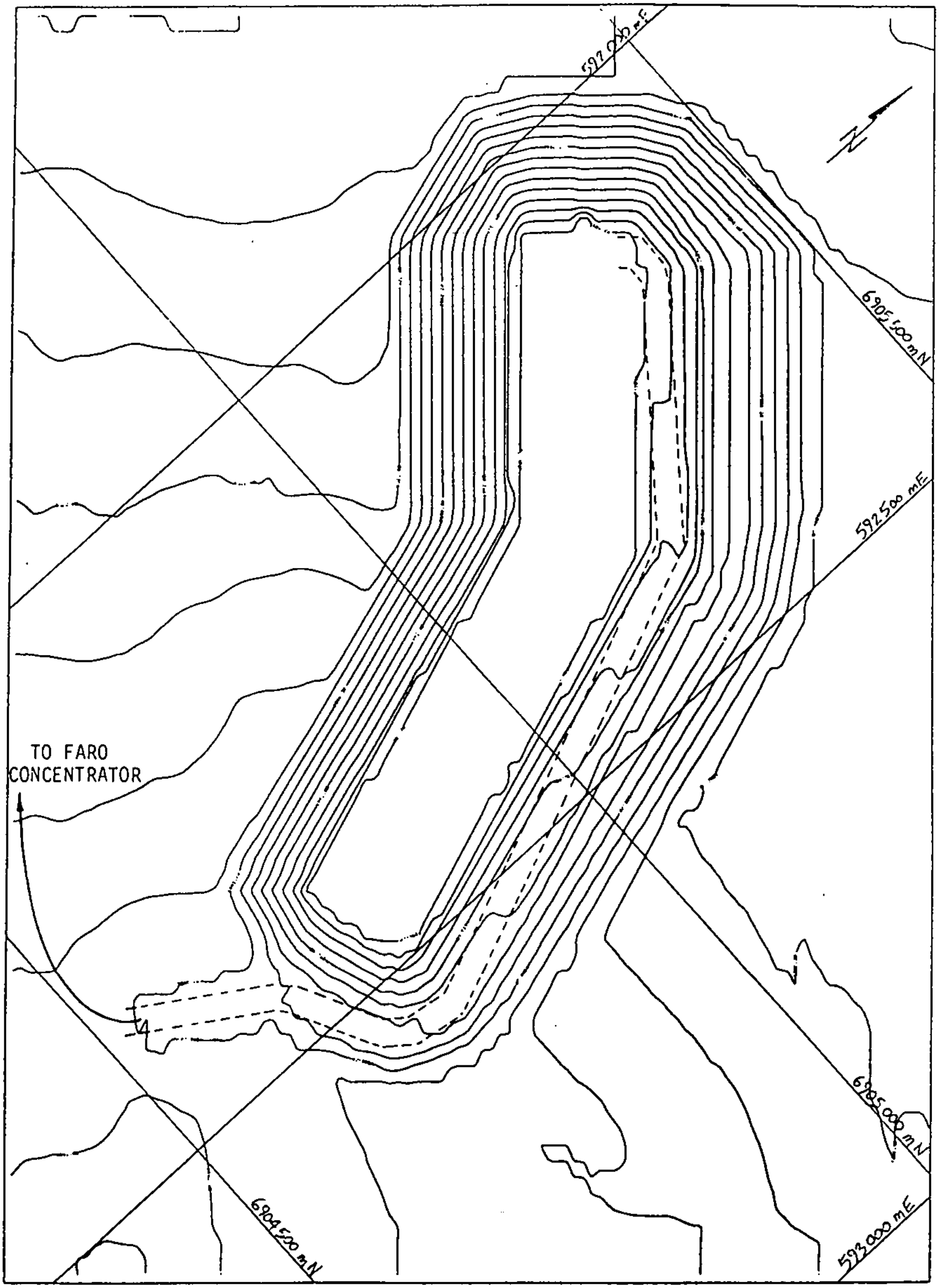


FIGURE 7.2 - 7

GRUM PIT END OF 1991

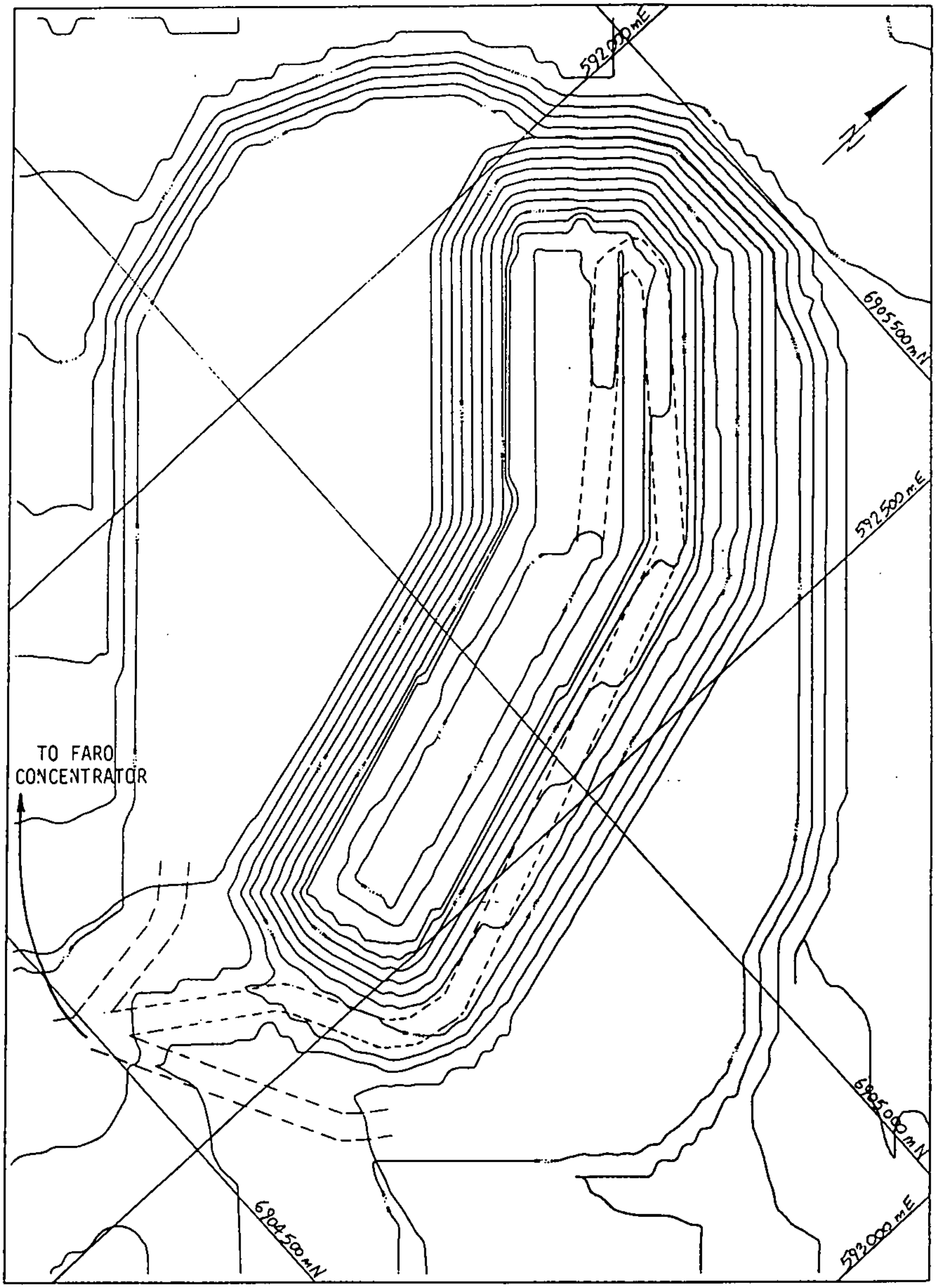


FIGURE 7.2 - 8

GRUM PIT END OF 1992

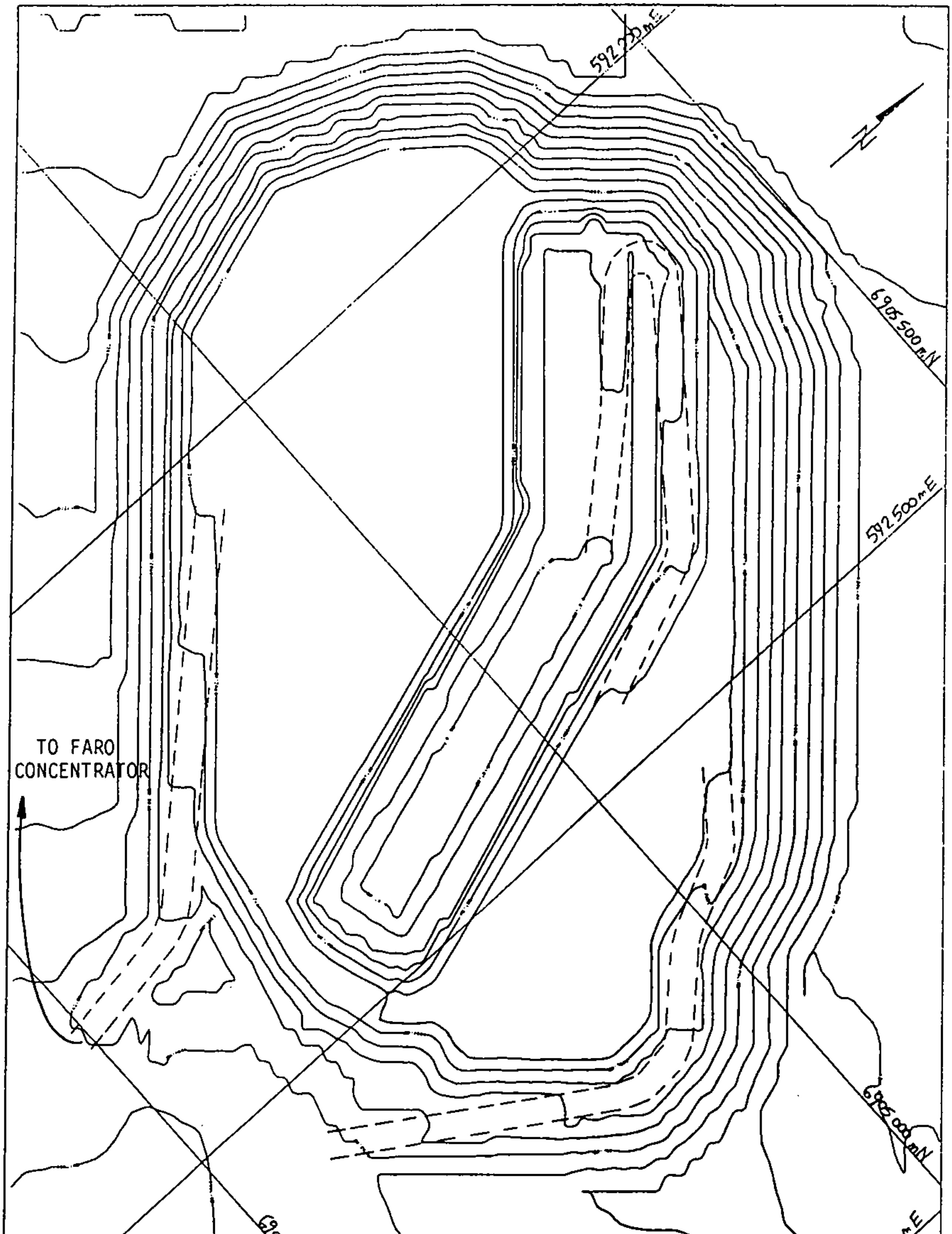


FIGURE 7. 2 - 9

GRUM PIT END OF 1993

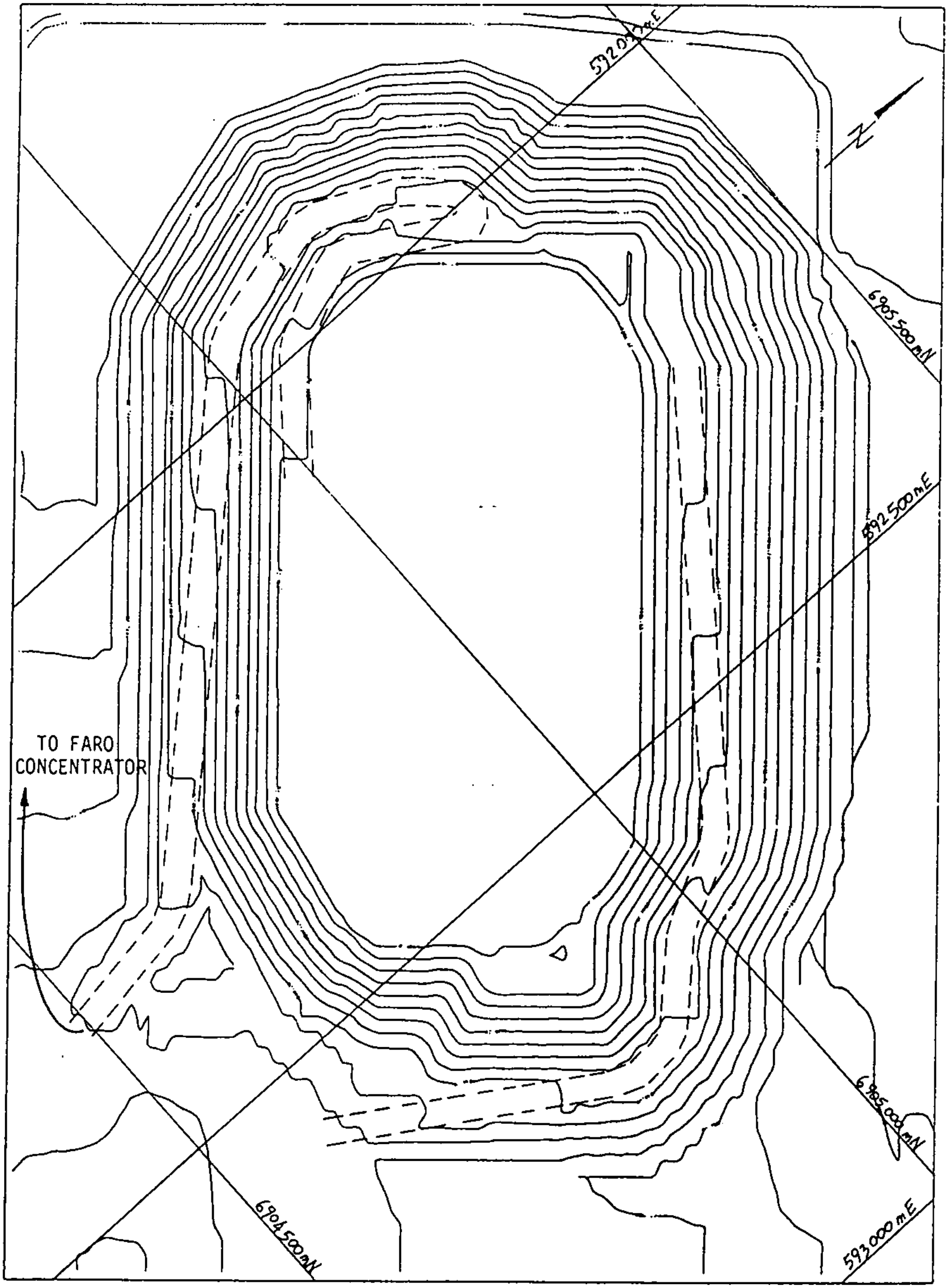


FIGURE 7.2 - 10

GRUM PIT END OF 1994

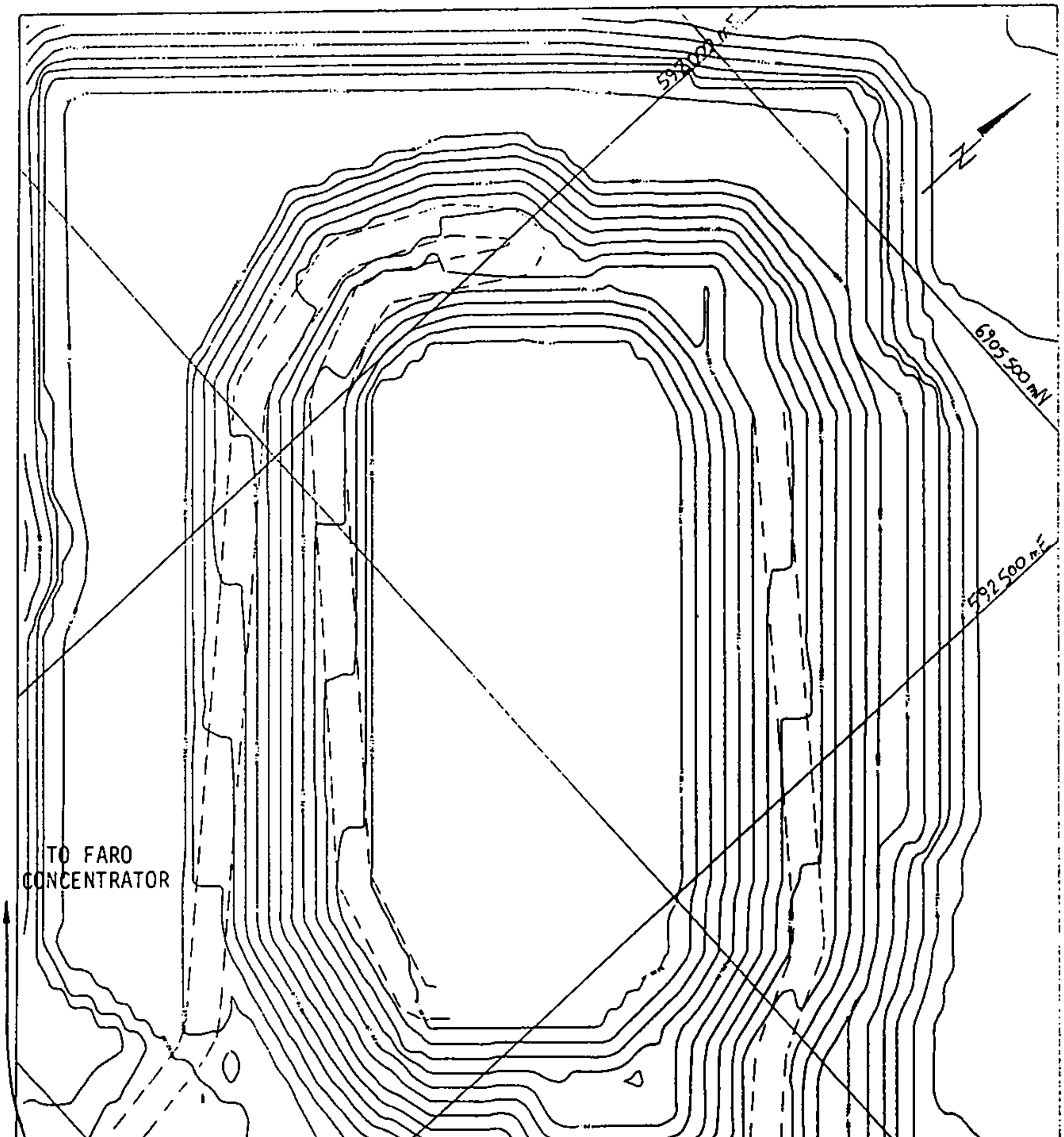


FIGURE 7.2 - 11

GRUM PIT END OF 1995

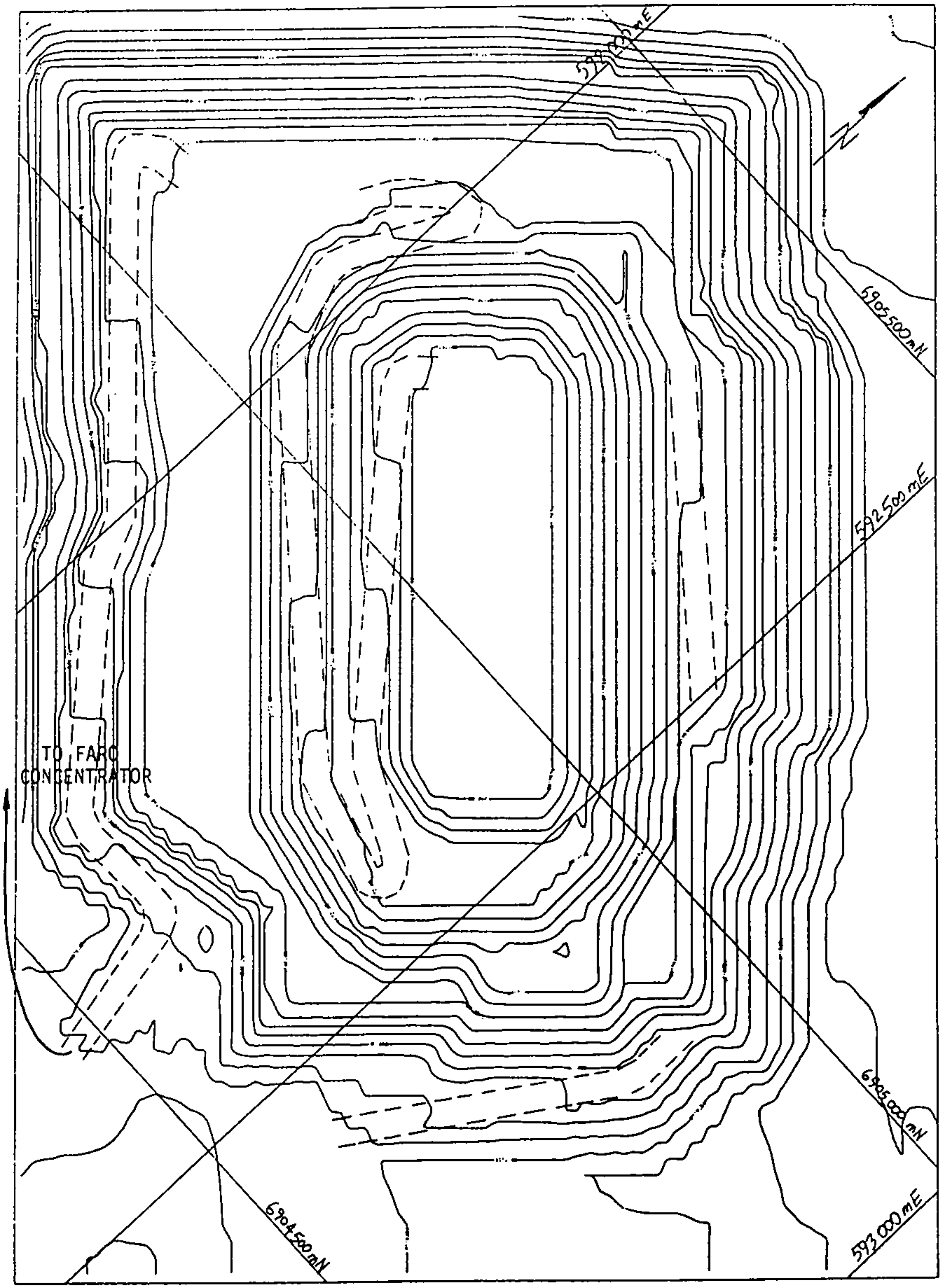


FIGURE 7. 2 - 11

GRUM PIT END OF 1996

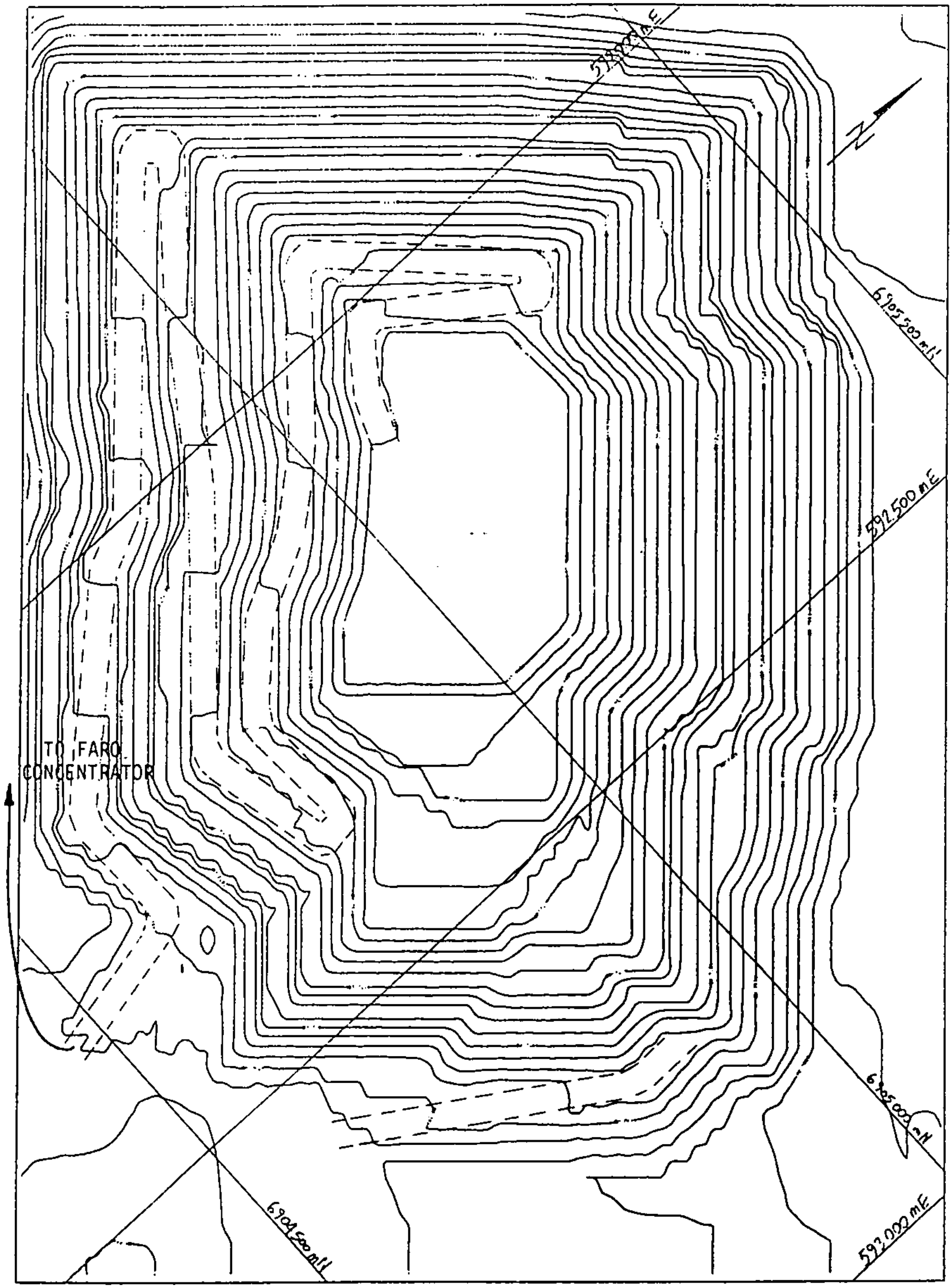


FIGURE 7.2 - 12

GRUM PIT END OF 1997

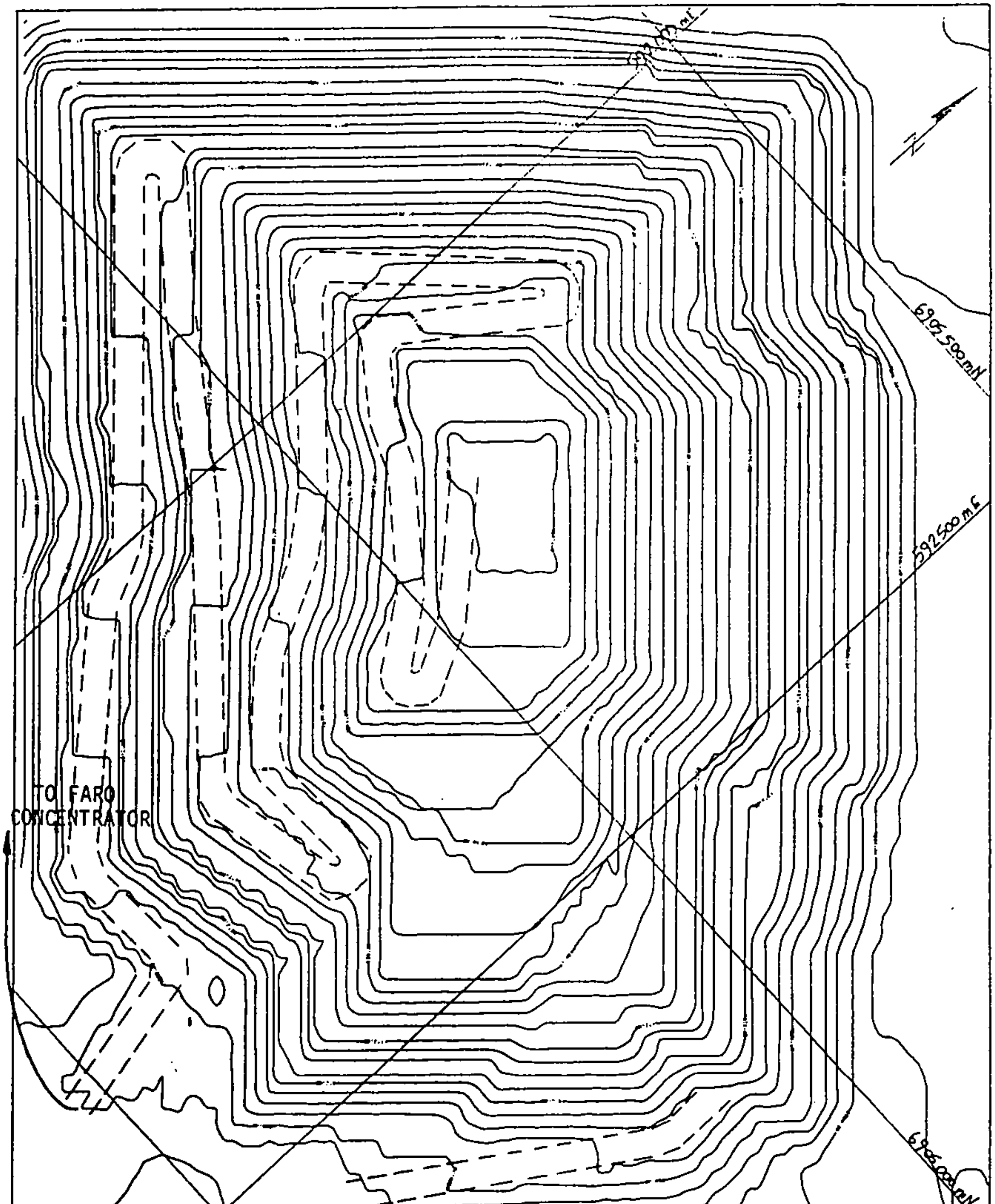


FIGURE 7.2 - 13

13

GRUM PIT END OF 1998

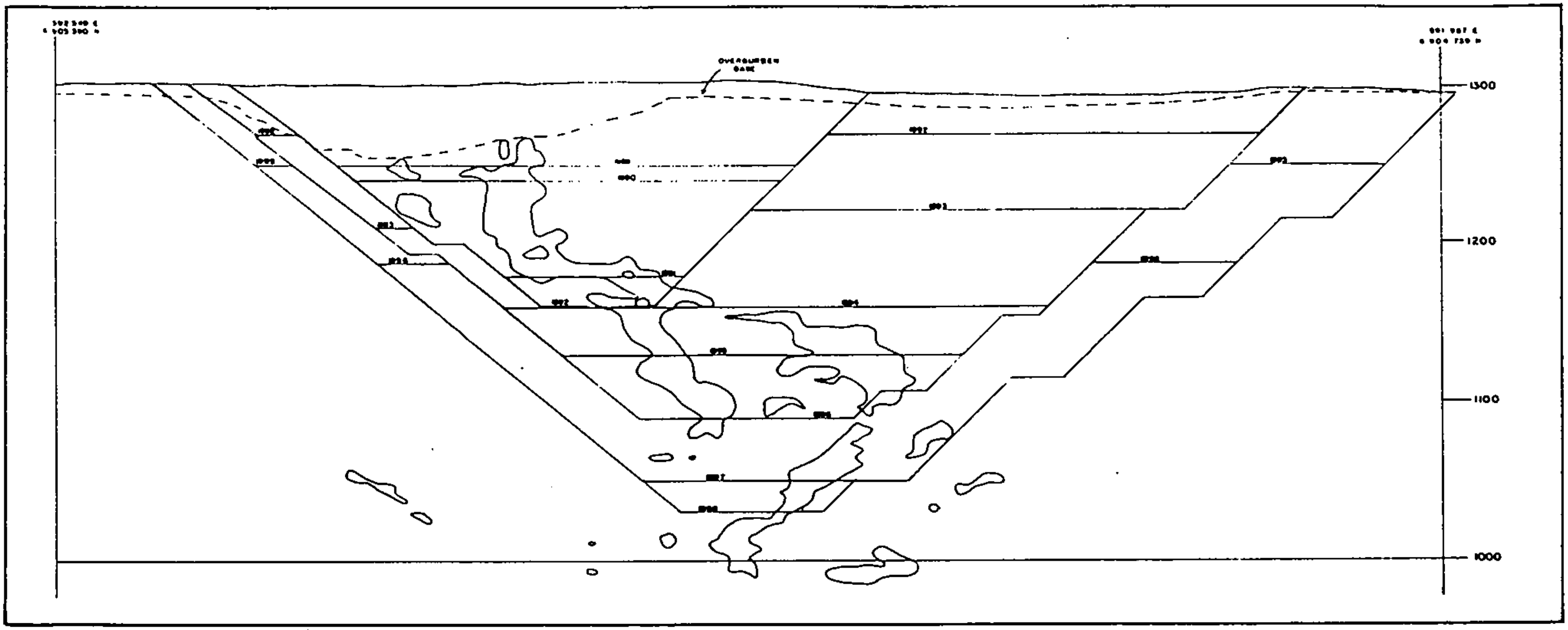


Figure 7. 2 - 14
Cross Section Through Grum Pit

7.3 Operating Equipment

Operating equipment in the pits has been estimated based on the following:

Shovels:	As required by mine plan
Trucks:	As required by mine plan
Tracked dozers:	One per operating shovel
Rubber-tired dozers:	One per two operating shovels
Graders:	One per two operating shovels, plus one for Vangorda Haul Road after 1988
Drills:	One per two operating shovels

Additionally, front end loaders have been scheduled to handle stockpiled ore, based on the following:

- Rehandle 30% of mill feed from Faro pit
- Rehandle all mill meed from Vangorda and Grum pits
- Rehandle all ore in long-term stockpile
- Productivity of 415 t/h.

In order to maintain ore production, a 23 m^3 (30 cu. yd) shovel is added to the shovel fleet in 1991.

Operating equipment is summarized in Table 7.3-1, and the truck requirement calculation is presented in Table 7.3-2.

TABLE 7.3-1

OPERATING EQUIPMENT SUMMARY

	1987	1988	1989	1990	1991	1992	1993	1994	1995	1996	1997	1998	1999
Shovel, Marion	1.0	1.0	1.0	1.0	0.6	0.3	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Shovel, P&H 2100	3.0	3.0	3.0	3.0	3.0	3.0	1.7	2.0	2.0	2.8	0.6	0.0	0.0
Shovel, 23 cu.m.	-	-	-	-	1.0	1.0	1.0	1.0	1.0	1.0	0.0	0.0	0.0
Truck, 154t	11.0	11.0	11.0	16.0	21.0	26.0	15.5	31.4	31.5	32.0	9.5	3.5	0.0
Truck, 109t	13.1	9.9	7.5	4.1	16.0	7.3	5.0	0.0	0.0	8.3	0.0	2.5	1.3
Front End Loader	2.0	0.5	0.8	1.5	1.5	1.5	1.9	1.5	1.5	1.5	1.5	1.9	0.5
Tracked Dozer	4.0	4.0	4.0	4.0	4.6	4.3	2.7	3.0	3.0	3.8	0.6	0.0	0.0
Wheeled Dozer	2.0	2.0	2.0	2.0	2.3	2.2	1.4	1.5	1.5	1.9	0.3	0.0	0.0
Grader	2.0	2.0	3.0	3.0	3.3	3.2	2.4	2.5	2.5	2.9	1.3	1.0	0.5

Curragh Resources

CIRQUE CROSS-SECTION RESERVES

Cutoff Grade (%) = 0.00

	TONNAGE tonnes	PB %	ZN %	AG g/t	PB+ZN %	ABOVE CUTOFF tonnes	PYRITIC %	PB tonnes	ZN tonnes	AG grams	(PB+ZN) tonnes	RESERVES tonnes ore	RESERVES tonnes Pb+Zn
295+50N	1,792,068	2.08	7.21	38	9.29	1,792,068	0.00%	37,275	129,208	68,098,584	166,483	5.19%	4.83%
296+40N	1,923,767	1.49	6.57	34	8.06	1,923,767	9.64%	28,741	126,341	64,516,904	155,082	5.57%	4.50%
297+50N	6,837,904	2.07	6.90	43	8.97	6,837,904	31.03%	141,401	471,786	293,931,389	613,187	19.79%	17.80%
298+50N	4,954,170	1.97	8.28	50	10.25	4,954,170	44.45%	97,812	409,992	247,309,414	507,805	14.34%	14.74%
299+20N	5,804,112	1.90	8.06	47	9.95	5,804,112	31.19%	110,079	467,609	273,554,395	577,688	16.80%	16.77%
300+00N	4,389,880	2.13	8.46	49	10.59	4,389,880	49.50%	93,706	371,372	215,895,020	465,078	12.71%	13.50%
301+30N	6,119,327	2.19	7.58	47	9.77	6,119,327	51.68%	134,111	463,819	285,084,133	597,930	17.71%	17.36%
302+50N	1,383,477	3.37	9.16	65	12.53	1,383,477	73.35%	46,570	126,792	90,197,577	173,362	4.00%	5.03%
303+00N	656,395	3.73	10.73	74	14.46	656,395	89.33%	24,491	70,421	48,256,495	94,912	1.90%	2.76%
303+50N	360,934	3.61	8.31	50	11.92	360,934	62.31%	13,028	30,006	18,123,532	43,034	1.04%	1.25%
304+00N	327,076	4.41	10.72	71	15.13	327,076	69.34%	14,415	35,056	23,279,608	49,471	0.95%	1.44%
TOTAL	34,549,110	2.15	7.82	47	9.97	34,549,110	39.68%	741,630	2,702,401	1,628,247,051	3,444,031	100.00%	100.00%

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08:49 Truck Requirements

	1987	1988	1989	1990	1991	1992	1993	1994	1995	1996	1997	1998	1999
Wabco 109 Tonne													
Number Units	16	16	16	16	16	16	16	16	16	16	16	16	16
Physical Avail.	0.70	0.70	0.70	0.70	0.70	0.70	0.70	0.70	0.70	0.70	0.70	0.70	0.70
Utilization	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80
Effective Utilization	0.56	0.56	0.56	0.56	0.56	0.56	0.56	0.56	0.56	0.56	0.56	0.56	0.56
Total Op Hours Available	58,921	78,490	78,275	78,275	78,275	78,275	78,275	78,275	78,275	78,275	78,275	78,275	78,275
Op Hours/Unit/Year	3,683	4,906	4,892	4,892	4,892	4,892	4,892	4,892	4,892	4,892	4,892	4,892	4,892
Productivity (Waste t/h)	190	190	208	219	208	208	190	190	190	190	190	190	190
Tonnes Waste Capacity	11,195	14,913	16,281	17,142	16,281	16,281	14,872	14,872	14,872	14,872	14,872	14,872	14,872
Tonnes Waste Remaining	3,367	0	0	0	0	0	0	0	0	0	0	0	0
Tonnes Hauled	3,367	0	0	0	0	0	0	0	0	0	0	0	0
Operating Hours - Waste	17,720	0	0	0	0	0	0	0	0	0	0	0	0
Productivity (Ore t/h)	170	170	180	190	190	170	170	170	170	170	170	170	170
Tonnes Ore Capacity	7,004	13,343	14,089	14,872	14,872	13,307	13,307	13,307	13,307	13,307	13,307	13,307	13,307
Tonnes Ore Remaining	4,154	4,148	0	0	0	0	0	0	0	0	0	0	0
Tonnes Ore Hauled	4,154	4,148	0	0	0	0	0	0	0	0	0	0	0
Operating Hours - Ore	24,435	24,399	0	0	0	0	0	0	0	0	0	0	0
Tonnes Long Term Stock	0	0	0	0	0	0	4079	0	0	0	0	2534	1283
Tonnes Rehandle at 30%	1,246	2,039	913	47	150	150	0	0	0	0	0	0	0
Productivity (t/h)	207	207	207	207	207	207	207	207	207	207	207	207	207
Operating Hours - Stock.	6,020	9,852	4,409	228	725	725	19,705	0	0	0	0	12,242	6,198
Operating Hours Total	48,175	34,251	4,409	228	725	725	19,705	0	0	0	0	12,242	6,198
Units Required	13.1	7.0	0.9	0.0	0.1	0.1	4.0	0.0	0.0	0.0	0.0	2.5	1.3
Units Surplus	2.9	9.0	15.1	16.0	15.9	15.9	12.0	16.0	16.0	16.0	16.0	13.5	14.7
Summary - Faro Requirement													
154 Tonne Units Required	11.0	11.0	5.0	4.2	4.3	4.1	0.0	0.0	0.0	0.0	0.0	0.0	0.0
109 Tonne Units Required	13.1	7.0	0.9	0.0	0.1	0.1	4.0	0.0	0.0	0.0	0.0	2.5	1.3
Total Trucks Required	24.1	18.0	5.9	4.3	4.5	4.2	4.0	0.0	0.0	0.0	0.0	2.5	1.3

Table 7.3 - 2

Continued

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Case VP 1-5
Truck Requirements

VANGORDA PLATEAU REQUIREMENT

Mined (1000 tonnes)

	1987	1988	1989	1990	1991	1992	1993	1994	1995	1996	1997	1998	1999
Mined		8,018	24,235	24,987	40,220	36,432	34,103	39,051	33,848	41,238	4,927		
Feed To Mill			1,012	3,930	3,815	3,815	400	4,015	4,015	4,015	4,011	2,423	
Total Moved	0	8,018	25,247	28,917	44,035	42,247	34,503	43,066	37,663	45,253	8,938	2,423	0

Euclid 154 Tonne

	1987	1988	1989	1990	1991	1992	1993	1994	1995	1996	1997	1998	1999
Number Units	0.0	0.0	6.0	11.8	16.7	21.9	32.0	32.0	32.0	32.0	32.0	32.0	32.0
Physical Avail.	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75
Utilization	0.85	0.85	0.85	0.85	0.85	0.85	0.85	0.85	0.85	0.85	0.85	0.85	0.85
Effective Utilization	0.6375	0.6375	0.6375	0.6375	0.6375	0.6375	0.6375	0.6375	0.6375	0.6375	0.6375	0.6375	0.6375
Total Op Hours Available	0	0	33,486	65,695	93,100	122,385	178,704	178,704	178,704	178,704	178,704	178,704	178,704
Op Hours/Unit/Year	0	0	5,585	5,585	5,584	5,584	5,585	5,585	5,585	5,585	5,585	5,585	5,585
Productivity (t/h)	687	687	505	619	336	330	393	273	235	235	235	235	235
Mined Tonnes Capacity	0	0	11,356	15,058	21,027	30,316	68,973	40,017	34,447	34,447	34,455	37,440	41,995
Total Tonnes Mined	0	8,018	24,235	24,987	40,220	38,432	34,103	39,051	33,848	41,238	4,927	0	0
Total Tonnes Hauled	0	0	11,356	15,058	21,027	30,316	34,103	39,051	33,848	34,447	4,927	0	0
Total Tonnes Remaining	0	8,018	12,879	9,929	19,193	8,116	0	0	0	6,791	0	0	0
Operating Hours - Mine	0	0	22,486	24,326	62,580	91,865	86,777	143,045	144,034	146,584	20,965	0	0
Productivity (to Mill t/h)	99	99	92	95	125	125	125	125	125	125	125	125	125
Tonnes to Mill Capacity	0	0	3,081	6,241	11,637	15,298	22,338	22,338	22,338	22,338	22,338	22,338	22,338
Tonnes to Mill Total	0	0	1,012	3,930	3,815	3,815	400	4,015	4,015	4,015	4,011	2,423	0
Tonnes Hauled to Mill	0	0	1,012	3,930	3,815	3,815	400	4,015	4,015	4,015	4,011	2,423	0
Tonnes to Mill Remaining	0	0	0	0	0	0	0	0	0	0	0	0	0
Operating Hours - to Mill	0	0	11,000	41,368	30,520	30,520	3,200	32,120	32,120	32,120	32,088	19,384	0
OPerating Hours Total	0	0	33,486	65,695	93,100	122,385	89,977	175,165	176,154	178,704	53,053	19,384	0
Units Required	0.0	0.0	6.0	11.8	16.7	21.9	16.1	31.4	31.5	32.0	9.5	3.5	0.0
Units Surplus	0.0	0.0	0.0	0.0	0.0	0.0	15.9	0.6	0.5	0.0	22.5	28.5	32.0

Table 7.3 - 2

Continued

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Case VP 1-5
Truck Requirements

	1987	1988	1989	1990	1991	1992	1993	1994	1995	1996	1997	1998	1999
Wabco 109 Tonne													
Number Units	2.9	9.0	15.1	16.0	15.9	15.9	12.0	16.0	16.0	16.0	16.0	13.5	14.7
Physical Avail.	0.70	0.70	0.70	0.70	0.70	0.70	0.70	0.70	0.70	0.70	0.70	0.70	0.70
Utilization	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80
Effective Utilization	0.56	0.56	0.56	0.56	0.56	0.56	0.56	0.56	0.56	0.56	0.56	0.56	0.56
Total Op Hours Available	14,315	44,238	74,069	78,261	77,763	77,763	58,730	78,490	78,490	78,490	78,490	66,214	72,275
Op Hours/Unit/Year	4,906	4,906	4,906	4,906	4,906	4,906	4,906	4,906	4,906	4,906	4,906	4,906	4,906
Productivity (t/h)	568	568	397	503	247	243	291	195	166	166	166	166	166
Mined Tonnes Capacity	8,131	25,127	29,405	39,366	19,207	18,896	17,090	15,305	13,029	13,029	13,029	10,992	11,998
Total Tonnes Remaining	0	8,018	12,879	9,929	19,193	8,116	0	0	0	6,791	0	0	0
Total Tonnes Hauled	0	8,018	12,879	9,929	19,193	8,116	0	0	0	6,791	0	0	0
Operating Hours - Mine	0	14,117	32,441	19,740	77,705	33,401	0	0	0	40,910	0	0	0
Productivity (to Mill t/h)	70	70	65	90	90	90	90	90	90	90	90	90	90
Tonnes to Mill Capacity	1,002	2,109	2,706	5,267	5	3,993	5,286	7,064	7,064	3,382	7,064	5,959	6,505
Tonnes to Mill Remaining	0	0	0	0	0	0	0	0	0	0	0	0	0
Tonnes Hauled to Mill	0	0	0	0	0	0	0	0	0	0	0	0	0
Operating Hours - to Mill	0	0	0	0	0	0	0	0	0	0	0	0	0
Operating Hours Total	0	14,117	32,441	19,740	77,705	33,401	0	0	0	40,910	0	0	0
Units Required	0.0	2.9	6.6	4.0	15.8	6.8	0.0	0.0	0.0	8.3	0.0	0.0	0.0
Units Surplus	2.9	6.1	8.5	11.9	0.0	9.0	12.0	16.0	16.0	7.7	16.0	13.5	14.7
Summary - Plateau Requirement													
154 Tonne Units Required	0.0	0.0	6.0	11.8	16.7	21.9	16.1	31.4	31.5	32.0	9.5	3.5	0.0
109 Tonne Units Required	0.0	2.9	6.6	4.0	15.8	6.8	0.0	0.0	0.0	8.3	0.0	0.0	0.0
Total Trucks Required	0.0	2.9	12.6	15.8	32.5	28.7	16.1	31.4	31.5	40.3	9.5	3.5	0.0
Summary - All Pits													
154 Tonne Units Required	11.0	11.0	11.0	16.0	21.0	26.0	16.1	31.4	31.5	32.0	9.5	3.5	0.0
109 Tonne Units Required	13.1	9.9	7.5	4.1	16.0	7.0	4.0	0.0	0.0	8.3	0.0	2.5	1.3
Total Trucks Required	24.1	20.9	18.5	20.1	37.0	33.0	20.1	31.4	31.5	40.3	9.5	6.0	1.3
154 Tonne Operating Hours	46,114	61,429	61,353	89,287	117,208	145,135	89,977	175,165	176,154	178,704	53,053	19,384	0
109 Tonne Operating Hours	48,175	48,368	36,850	19,968	78,430	34,125	19,705	0	0	40,910	0	12,242	6,198

Table 7.3 - 2

Continued

8.0 PRODUCTION SCHEDULE - MILLING

The milling schedule was produced under the following guidelines:

1. Mill feed rate of 13,500 tonnes/day for Faro ore, and 11,000 tonnes/day for Vangorda Plateau ore.
2. Mill feed to come from highest grade ore available.
3. Maximum concentrate production of 1,600 tonnes/day.
4. Metallurgical parameters as presented in Section 4.

The mill feed schedule is presented in Tables 8-1 through 8-3.

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Case VP 1-5: Faro plus Vangorda Plateau Ore Milling Schedule

	87-04-01 to 87-12-31	1988	1989	1990	1991	1992	1993	1994	1995	1996	1997	1998	1999	Total
Mill Feed Summary														
Tonnes	3,539,950	4,759,869	4,697,500	4,034,269	4,314,751	4,314,751	4,479,270	4,015,000	4,015,000	4,015,000	4,011,486	4,957,529	1,282,771	52,437,147
% Pb + Zn	8.23	7.99	7.25	8.58	7.49	7.41	7.59	9.16	9.47	8.86	7.10	4.05	5.04	7.62
% Pb	3.41	3.25	2.97	3.75	2.82	2.66	2.75	3.49	3.47	3.35	2.66	1.56	2.01	2.96
% Zn	4.82	4.74	4.28	4.83	4.66	4.75	4.83	5.67	5.99	5.50	4.44	2.49	3.03	4.66
Ag g/t	45	41	42	53	42	41	28	58	59	57	47	26	28	44
Au g/t	0.09	0.07	0.20	0.60	0.57	0.61	0.16	0.86	0.96	0.92	0.84	0.34	0.10	0.50
Concentrate														
Pb Conc DMT	153,169	194,584	181,956	240,010	165,137	146,497	157,150	188,772	187,138	179,202	137,121	90,548	31,261	2,052,543
% Pb	61.03	60.88	58.83	53.39	57.65	59.79	58.43	60.00	60.00	60.00	60.00	56.46	55.83	58.74
Ag g/t	562	548	598	603	693	760	426	824	843	847	863	670	598	683
Au g/t	0.00	0.00	1.22	3.86	4.75	5.51	0.66	6.04	6.82	6.82	7.74	3.99	0.00	3.85
Zn Conc DMT	271,522	358,612	308,270	282,454	300,798	310,261	344,115	353,848	376,395	340,155	266,509	168,732	55,169	3,736,840
% Zn	50.86	50.87	51.57	54.64	54.42	54.49	51.28	55.00	55.00	55.00	55.00	52.36	50.04	53.37
Total Conc DMT	424,691	553,196	490,226	522,464	465,935	456,758	501,264	542,620	563,532	519,357	403,629	259,280	86,430	5,789,383
Recoveries														
Pb	77.4%	76.6%	76.8%	84.7%	78.2%	76.3%	74.4%	80.9%	80.5%	79.8%	77.2%	66.2%	67.7%	77.8%
Zn	80.9%	80.9%	79.0%	79.3%	81.3%	82.6%	81.5%	85.5%	86.1%	84.7%	82.3%	71.6%	71.0%	81.6%
Ag	54.0%	54.6%	55.5%	68.0%	63.2%	62.4%	52.7%	66.6%	66.6%	66.0%	63.3%	47.9%	52.0%	61.0%
Au	0.0%	0.0%	24.0%	38.1%	32.1%	30.9%	13.9%	33.1%	33.3%	33.0%	31.5%	21.3%	0.0%	30.4%

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Case VP 1-5: Faro Ore Milling Schedule

	87-04-01 to 87-12-31	1988	1989	1990	1991	1992	1993	1994	1995	1996	1997	1998	1999	Total
Mill Feed Summary														
Tonnes	3,539,950	4,759,869	3,685,500	104,053	499,751	499,751	4,079,270					2,534,076	1,282,771	20,984,992
% Pb + Zn	8.23	7.99	7.10	5.71	7.85	7.85	7.85					4.19	5.04	7.09
% Pb	3.41	3.25	2.87	2.43	2.67	2.67	2.67					1.60	2.01	2.79
% Zn	4.82	4.74	4.23	3.28	4.76	4.76	4.76					2.59	3.03	4.30
Ag g/t	45	41	40	33	25	25	25					25	28	35
Au g/t	0.09	0.07	0.10	0.23	0.11	0.11	0.11					0.11	0.1	0.09
Concentrate														
Pb Conc DMT	153,169	194,584	129,437	3,004	16,881	16,881	137,795					48,247	31,261	731,259
% Pb	61.03	60.88	61.14	61.04	58.21	58.21	58.21					54.63	55.83	59.70
Ag g/t	562	548	591	586	371	371	371.35					585.00	598.00	522
Au g/t														
Zn Conc DMT	271,522	358,612	243,399	5,095	37,898	37,898	309,344					92,798	55,169	1,411,735
% Zn	50.86	50.87	50.72	50.93	50.86	50.86	50.86					50.64	50.04	50.79
Total Conc DMT	424,691	553,196	372,836	8,099	54,779	54,779	447,139					141,045	86,430	2,142,994
Recoveries														
Pb	77.4%	76.6%	74.8%	72.5%			73.5%					65.0%	67.7%	74.5%
Zn	80.9%	80.9%	79.2%	76.0%			81.1%					71.6%	71.0%	79.5%
Ag	54.0%	54.6%	51.9%	51.3%			49.4%					44.6%	52.0%	52.0%
Au	0.0%	0.0%	0.0%	0.0%			0.0%					0.0%	0.0%	0.0%

Table 8 - 2

Case VP 1-5: Vangorda Plateau Ore Milling Schedule

	87-04-01 to 87-12-31	1988	1989	1990	1991	1992	1993	1994	1995	1996	1997	1998	1999	Total
Mill Feed Summary														
Tonnes			1,012,000	3,930,216	3,815,000	3,815,000	400,000	4,015,000	4,015,000	4,015,000	4,011,486	2,423,453		31,452,155
% Pb + Zn			7.81	8.65	7.49	7.40	9.18	9.16	9.47	8.86	7.10	3.90	-	7.97
% Pb			3.33	3.79	2.84	2.66	3.57	3.49	3.47	3.35	2.66	1.51		3.07
% Zn			4.48	4.87	4.65	4.75	5.62	5.67	5.99	5.50	4.44	2.38		4.90
Ag g/t			48	53	44	43	59	58	59	57	47	26		50
Au g/t			0.55	0.61	0.63	0.67	0.77	0.86	0.96	0.92	0.84	0.59		0.76
Concentrate														
Pb Conc DMT			52,519	237,006	148,255	129,616	19,355	188,772	187,138	179,202	137,121	42,301		1,321,284
% Pb			53.13	53.29	57.59	60.00	60.00	60.00	60.00	60.00	60.00	58.54		58.21
Ag g/t			614	603	730	811	815	824	843	847	863	766		772
Au g/t			4.23	3.91	5.29	6.23	5.32	6.04	6.82	6.62	7.74	8.54		5.98
Zn Conc DMT			64,871	277,359	262,901	272,364	34,770	353,848	376,395	340,155	266,509	75,934		2,325,105
% Zn			54.77	54.71	54.93	55.00	55.00	55.00	55.00	55.00	55.00	54.45		54.93
Total Conc DMT			117,390	514,365	411,156	401,979	54,125	542,620	563,532	519,357	403,629	118,234		3,646,389
Recoveries														
Pb			82.9%	84.9%	78.8%	76.7%	81.4%	80.9%	80.5%	79.8%	77.2%	67.6%		79.8%
Zn			78.3%	79.3%	81.4%	82.7%	85.1%	85.5%	86.1%	84.7%	82.3%	71.6%		82.8%
Ag			66.3%	68.3%	64.2%	63.4%	67.3%	66.6%	66.6%	66.0%	63.3%	51.3%		65.2%
Au			39.8%	38.5%	32.8%	31.5%	33.3%	33.1%	33.3%	33.0%	31.5%	25.5%		32.9%

Table 8 - 3

9.0 MANPOWER

Hourly manpower forecasts are presented in Table 9-1. This forecast is based on the current three shift cycle.

Manpower for Mine Operations is determined from the operating equipment. Mine Maintenance manpower is pro-rated from 1987 levels based on the number of units of operating equipment.

Mill Operations and Maintenance manpower remains constant through the life of the project.

Materials Management manpower decreases towards the end of the project life.

Salaried manpower is presented in Table 9-2.

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	1987	1988	1989	1990	1991	1992	1993	1994	1995	1996	1997	1998	1999
Hourly Manpower													
Mine Operations													
Shovel Operator	12.0	11.4	11.8	12.0	13.8	13.0	8.2	9.0	9.0	11.5	1.7		
Driller	6.0	5.7	5.9	6.0	6.9	6.5	4.1	4.5	4.5	5.8	0.9		
Mine Utility Blaster	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	1.0		
Mine Utility Blaster Helper	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	1.0		
Haulage Truck Driver	60.0	62.7	55.5	60.3	111.0	99.0	60.3	94.2	94.5	120.9	28.5	18.0	3.9
Equipment Operator	23.0	24.5	29.0	31.5	35.1	33.5	25.7	25.6	25.6	30.6	10.9	8.7	1.5
Mine Utility Worker	6.0	6.0	6.0	6.0	6.0	6.0	6.0	6.0	6.0	6.0	6.0	3.0	1.0
Pumpman	5.0	5.0	5.0	5.0	5.0	5.0	5.0	5.0	5.0	5.0	5.0	3.0	1.0
Trainer Operator Shovel	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0		
Trainer Operator Truck	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0		
Total	119.0	119.0	119.0	119.0	119.0	119.0	119.0	119.0	119.0	119.0	119.0	119.0	119.0
Mine Maintenance													
H D Mechanic	32.0	33.0	32.4	34.8	52.8	48.2	31.1	42.2	42.3	53.5	13.3	8.5	1.7
Welder	14.0	14.5	14.2	15.2	23.1	21.1	13.6	18.5	18.5	23.4	5.8	3.7	0.7
Electrician	12.0	12.4	12.1	13.0	19.8	18.1	11.7	15.8	15.9	20.0	5.0	3.2	0.6
Machinist	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0
Carpenter	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0
Plumber	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0
Cableman	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0
Equipment Operator	6.0	6.0	6.0	6.0	6.0	6.0	6.0	6.0	6.0	6.0	3.0	3.0	1.0
Lubeman	9.0	9.0	9.0	9.0	9.0	9.0	9.0	9.0	9.0	9.0	6.0	3.0	1.0
Utility Operator	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0
Tool Crib	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0
Labourers	6.0	6.0	6.0	6.0	6.0	6.0	6.0	6.0	6.0	6.0	3.0	3.0	1.0
Total	89.0	90.9	89.7	94.1	126.8	118.3	87.4	107.5	107.7	127.9	46.1	34.3	16.1
Mine Total	208	210	209	213	246	237	206	227	227	247	165	153	135
Mill Operations													
Flotation Operator	4	4	4	4	4	4	4	4	4	4	4	4	2
Flotation Helper	1	1	1	1	1	1	1	1	1	1	1	1	0.5
Grinding Operator	3	3	3	3	3	3	3	3	3	3	3	3	1.5
Grinding Helper	1	1	1	1	1	1	1	1	1	1	1	1	0.5
Crusher Operator	3	3	3	3	3	3	3	3	3	3	3	3	1.5
Crusher Helper	3	3	3	3	3	3	3	3	3	3	3	3	1.5
Filter Operator	3	3	3	3	3	3	3	3	3	3	3	3	1.5
Coal Crusher/Filter Op.	3	3	3	3	3	3	3	3	3	3	3	3	1.5
Trainee	4	4	4	4	4	4	4	4	4	4	4	4	2
Labourer	15	15	15	15	15	15	15	15	15	15	15	15	7.5
Heat Plant Operator	4	4	4	4	4	4	4	4	4	4	4	4	2
Heat Plant Helper	4	4	4	4	4	4	4	4	4	4	4	4	2
Reagent Operator	2	2	2	2	2	2	2	2	2	2	2	2	1
Reagent Helper	1	1	1	1	1	1	1	1	1	1	1	1	0.5
Tailings Operator	1	1	1	1	1	1	1	1	1	1	1	1	0.5
Equipment Operator	2	2	2	2	2	2	2	2	2	2	2	2	1
Steel Crew	2	2	2	2	2	2	2	2	2	2	2	2	1
Filter Sector	1	1	1	1	1	1	1	1	1	1	1	1	0.5

Note: 1987 values refer to manpower from April 1 through December 31, 1987

Table 9 - 1

File: MAN15
02/04/87
09:02

	1987	1988	1989	1990	1991	1992	1993	1994	1995	1996	1997	1998	1999
Sample Bucker	3	3	3	3	3	3	3	3	3	3	3	3	1.5
Total	60	60	60	60	60	60	60	60	60	60	60	60	30
Mill Maintenance													
Mill Mechanic	34	34	34	34	34	34	34	34	34	34	34	34	17
Electrician	9	9	9	9	9	9	9	9	9	9	9	9	4.5
Instrument Mechanic	5	5	5	5	5	5	5	5	5	5	5	5	2.5
Total	48	48	48	48	48	48	48	48	48	48	48	48	24
Mill Total	108	108	108	108	108	108	108	108	108	108	108	108	54
Materials Management													
Warehouse Person A	2	2	2	2	2	2	2	2	2	2	2	2	2
Warehouse Person B	3	3	3	3	3	3	3	3	3	3	3	3	3
Warehouse Person C	2	2	2	2	2	2	2	2	2	2	2	2	2
Warehouse/First Aid	3	3	3	3	3	3	3	3	3	3	3	3	3
Total	10	10	10	10	10	10	10	10	10	10	10	10	10
Mine Total	208	210	209	213	246	237	206	227	227	247	165	153	135
Mill Total	108	108	108	108	108	108	108	108	108	108	108	108	54
Total Hourly	326	328	327	331	364	355	324	345	345	365	283	271	199

Note: 1987 values refer to manpower from April 1 through December 31, 1987

Table 9 - 1 Continued

File: MAN15
02/04/87
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	1987	1988	1989	1990	1991	1992	1993	1994	1995	1996	1997	1998	1999
Salaried Manpower													
Mine Operations													
General Foreman	1	1	1	1	0.6								
Drill & Blast Foreman	1	1	1										
Mine Training Foreman	2	2	2	2	2	2	2	2	2	1	1		
Mine Foreman	6	6	6	6	6	6	6	6	6	3	3	3	
Mine Clerk	1	1	1	1	1	1	1	1	1	1	1		
Mine Engineering													
Chief Engineer	1	1	1	1	1	1	1	1	1	1	0.5		
Senior Mine Engineer	1	1	1	1	1	1	1	1	1	1	0.5		
Intermediate Engineer	1	1	1	1	1	1	1	1	1	1	0.5		
Mine Engineer	1	1	1	1	1	1	1	1	1	1	0.5		
Senior Geologist	1	1	1	1	1	1	1	1	1	1	0.5		
Geologist	3	3	3	3	3	3	3	3	3	3	1.5		
Surveyor	3	3	3	3	3	3	3	3	3	3	1.5		
Technician	1	1	1	1	1	1	1	1	1	1	0.5		
Clerk	1	1	1	1	1	1	1	1	1	1	0.5		
Mine Maintenance													
General Foreman	1	1	1	1	1	1	1	1	1	1	1	1	
Shop Foreman	3	3	3	3	3	3	3	3	3	3			
Field Maintenance Foreman	2	2	2	2	2	2	2	2	2	2	1	1	
Mine Electrical Foreman	1	1	1	1	1	1	1	1	1	1	1	1	
Senior Maintenance Planner	1	1	1	1	1	1	1	1	1	1	1	1	
Maintenance Engineer	1	1	1	1	1	1	1	1	1	1			
Yards & Services Foreman	1	1	1	1	1	1	1	1	1	1			
Planning Clerk	2	2	2	2	2	2	2	2	2	2	1	1	
Mine Total	36	36	36	35	34.6	34	34	34	34	30	16.5	8	0
Mill Operations													
General Forman	1	1	1	1	1	1	1	1	1	1	1	1	0.5
Training Foreman	1	1	1	1	1	1	1	1	1	1	1	1	0.5
Labour/Surface Foreman	1	1	1	1	1	1	1	1	1	1	1	1	0.5
Shift Foreman	3	3	3	3	3	3	3	3	3	3	3	3	1.5
Mill Metallurgical													
Chief Metallurgist	1	1	1	1	1	1	1	1	1	1	1	1	0.5
Senior Metallurgist	1	1	1	1	1	1	1	1	1	1	1	1	0.5
Metallurgist	1	1	1	1	1	1	1	1	1	1	1	1	0.5
Process Control Engineer	1	1	1	1	1	1	1	1	1	1	1	1	0.5
Chief Assayer	1	1	1	1	1	1	1	1	1	1	1	1	0.5
Assayer	6	6	6	6	6	6	6	6	6	6	6	6	3
Met Technician	4	4	4	4	4	4	4	4	4	4	4	4	2
Mill Maintenance													
General Foreman	1	1	1	1	1	1	1	1	1	1	1	1	0.5
Mechanical Foreman	3	3	3	3	3	3	3	3	3	3	3	3	1.5
Chief Heat Plant Engineer	1	1	1	1	1	1	1	1	1	1	1	1	0.5
Electrical Foreman	1	1	1	1	1	1	1	1	1	1	1	1	0.5
Instrument/Electrical	1	1	1	1	1	1	1	1	1	1	1	1	0.5
Maintenance Planner	1	1	1	1	1	1	1	1	1	1	1	1	0.5
Maintenance Planner	1	1	1	1	1	1	1	1	1	1	1	1	0.5
Mill Total	30	30	30	30	30	30	30	30	30	30	30	30	15

Note: 1987 Values refer to manpower from April 1 through December 31, 1987

Table 9 - 2

#14: Atherton, K. (1986) VP 1-1 Mine Plan

1988 Vangorda and Grum Mine Reserves

10.0 COSTS AND REVENUES

10.1 Operating Costs

The cost estimate for this plan is based on unit costs derived from the Curragh Resources 1987 Operating Budget. All costs are in constant (fourth quarter 1986) Canadian Dollars. The net cost presented is the total cost to load the concentrate on board ship at Skagway, Alaska.

The operating and cost estimate is presented in Table 10.1-1.

10.2 Capital Costs

The Capital cost estimate is presented in Table 10-2. Highlights of the estimate include:

- Scheduled replacement of mine operating equipment
- Scheduled purchase over six years of 30 haul trucks, the first 20 units to replace the existing Wabco fleet, the last 10 units to replace the existing Euclid fleet.
- An allowance of $6 million in each of 1988 and 1994 to raise the tailings dam.
- New structures and equipment as required for the development of the Vangorda Plateau.
- A total of $35 million for abandonment.

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Operating Cost Estimate

	Unit Cost	87-04-01 to 87-12-31 **	1988	1989	1990	1991	1992	1993	1994	1995	1996	1997	1998	1999	Total
Unit Costs, Mining															
Drill and Blast	0.20		6,025,614	6,534,354	6,572,688	9,523,519	8,915,359	6,820,672	7,810,281	6,769,605	8,247,661	985,371	0	0	
Op Hour Costs, Mining															
Shovel, Marion	227.06		1,183,437	1,183,437	1,183,437	701,615	399,626	0	0	0	0	0	0	0	
Shovel, P&H 2100	151.13		2,692,532	2,806,031	2,866,936	2,866,936	2,866,936	1,659,710	1,918,142	1,913,004	2,715,504	531,222	0	0	
Shovel, 23 cu m	230.00		0	0	0	1,541,230	1,541,230	1,541,230	1,541,230	1,541,230	1,541,230	0	0	0	
Truck, 154 tonne	126.90		7,795,404	7,785,688	11,330,544	14,873,729	18,417,568	11,418,080	22,228,494	22,353,960	22,677,538	6,732,469	2,459,830	0	
Truck, 109 tonne	107.76		5,212,135	3,970,947	2,151,749	8,451,576	3,677,328	2,123,445	0	0	4,408,466	0	1,319,149	667,904	
Front End Loader	148.21		509,971	394,864	11,148	53,570	53,570	1,813,795	0	0	0	0	905,001	458,119	
Dozer, Tracked	73.23		1,768,138	1,823,134	1,852,646	2,127,421	2,009,152	1,267,685	1,392,908	1,390,418	1,779,269	257,403	0	0	
Dozer, Wheeled	66.23		799,562	824,431	837,776	962,031	908,549	573,254	629,880	628,755	804,595	116,399	0	0	
Grader	64.18		774,813	1,205,108	1,218,040	1,338,449	1,286,623	961,705	1,016,579	1,015,488	1,185,886	518,992	406,195	0	
Subtotal, Mining			26,761,606	26,527,993	28,024,964	42,440,077	40,075,941	28,179,575	36,537,515	35,612,459	43,360,149	9,141,857	5,090,175	1,126,023	
Unit Costs, Milling															
Primary Crushing	0.133		633,063	624,768	536,558	573,862	573,862	595,743	533,995	533,995	533,995	533,528	659,351	170,609	
Secondary Crushing	0.216		1,028,132	1,014,660	871,402	931,986	931,986	967,522	867,240	867,240	867,240	866,481	1,070,826	277,079	
Grinding	1.562		7,434,915	7,337,495	6,301,528	6,739,641	6,739,641	6,996,620	6,271,430	6,271,430	6,271,430	6,265,941	7,743,660	2,003,688	
Flotation	0.258		1,228,046	1,211,955	1,040,841	1,113,206	1,113,206	1,155,652	1,035,870	1,035,870	1,035,870	1,034,963	1,279,042	330,955	
Reagent Area	1.672		7,958,501	7,854,220	6,745,298	7,214,264	7,214,264	7,489,340	6,713,080	6,713,080	6,713,080	6,707,205	8,288,989	2,144,793	
Freight backhaul	1.156		5,502,409	5,430,310	4,663,615	4,987,852	4,987,852	5,178,037	4,641,340	4,641,340	4,641,340	4,637,278	5,730,904	1,482,883	
Unit Costs, Concentrate															
Dewatering	1.513		836,986	741,712	790,488	704,960	691,075	758,413	820,985	852,624	785,787	610,691	392,291	130,769	
Drying	2.576		1,425,033	1,262,822	1,345,868	1,200,249	1,176,609	1,291,257	1,397,790	1,451,659	1,337,863	1,039,749	667,905	222,644	
Handling	50.174		27,756,056	24,596,590	26,214,120	23,377,825	22,917,377	25,150,443	27,225,439	28,274,674	26,058,214	20,251,692	13,009,112	4,336,539	
Subtotal, Mill & Conc			53,803,140	50,074,531	48,509,719	46,843,845	46,345,872	49,583,028	49,507,169	50,641,913	48,244,820	41,947,527	38,842,080	11,099,958	
Annual Costs															
Dewatering, Mine			362,600	362,600	362,600	362,600	362,600	362,600	362,600	362,600	362,600	362,600	181,300		
Technical Services			1,285,800	1,285,800	1,285,800	1,285,800	1,285,800	1,285,800	1,285,800	1,285,800	1,285,800	1,285,800	642,900		
General, Mine			7,706,500	7,706,500	7,706,500	7,706,500	7,706,500	7,706,500	7,706,500	7,706,500	7,706,500	7,706,500	3,853,250		
General, Mill			10,723,719	10,723,719	10,723,719	10,723,719	10,723,719	10,723,719	10,723,719	10,723,719	10,723,719	10,723,719	10,723,719	2,144,744	
G & A, Faro			4,711,135	4,711,135	4,711,135	4,711,135	4,711,135	4,711,135	4,711,135	4,711,135	4,711,135	4,711,135	4,711,135	942,227	
All, Whitehorse			1,381,000	1,381,000	1,381,000	1,381,000	1,381,000	1,381,000	1,381,000	1,381,000	1,381,000	1,381,000	1,381,000	276,200	
All, Toronto			5,000,000	5,000,000	5,000,000	5,000,000	5,000,000	5,000,000	5,000,000	5,000,000	5,000,000	5,000,000	5,000,000	1,000,000	
Subtotal, Annual Costs			31,170,754	31,170,754	31,170,754	31,170,754	31,170,754	31,170,754	31,170,754	31,170,754	31,170,754	31,170,754	26,493,304	4,363,171	
Total, Operating Cost		81,176,250	111,735,500	107,773,278	107,705,437	120,454,677	117,592,568	108,933,356	117,215,438	117,425,126	122,775,723	82,260,138	70,425,559	16,589,152	1,282,062,200

** Note: See 1987 Operating Budget for details of 1987 costs

Table 10.1 - 1

87-02-01

Case VP 1-5: Capital Forecast

Item	1987	1988	1989	1990	1991	1992	1993	1994	1995	1996	1997	1998	1999	20000
Structures and Equipment														
Power														
5000 metre powerline	220 000													
Substation: 138 kV	200 000													
Equipment														
Shovel: 23 cu. m.					7 000 000									
Buildings														
Lube/Fuel Station	350 000													
Two Bay Maintenance Facility		1 500 000												
Vangorda Creek Diversion	200 000													
Vangorda Haul Road	5 000 000	5 000 000												
Grum Water Diversion			1 000 000											
Shovel Moves	25 000	25 000	25 000				25 000							
Subtotal Structures and Equipment	5 995 000	6 525 000	1 025 000	0	7 000 000	0	25 000	0	0	0	0	0	0	0
Equipment Replacement - Mine														
Shovel				4 000 000										
Haul Truck				7 500 000	7 500 000	7 500 000	7 500 000	7 500 000	7 500 000					
Blasthole Drill				500 000	1 000 000									
Dozer: Cat D9 with ripper			630 000			1 260 000	1 260 000		1 260 000					
Rubber Tired Dozer: Cat 824			530 000	530 000										
Grader: Cat 16G			430 000		430 000		430 000		430 000					
Front End Loader		1 100 000		1 100 000										
Miscellaneous														
Subtotal Equipment Replacement	0	1 100 000	1 590 000	13 630 000	8 930 000	8 760 000	9 190 000	7 500 000	9 190 000	0	0	0	0	0
Mill														
Crusher Modification			1 500 000											
Raise Tailings Dam		6 000 000						6 000 000						
Capital Replacement		1 000 000	1 000 000	1 000 000	1 000 000	1 000 000	1 000 000	1 000 000	1 000 000	1 000 000	1 000 000			
Subtotal Mill	0	7 000 000	2 500 000	1 000 000	1 000 000	1 000 000	1 000 000	7 000 000	1 000 000	1 000 000	1 000 000	0	0	0
Reclamation/Abandonment														
Total								5 000 000				10 000 000	10 000 000	10 000 000
Whitehorse Office														
Exploration		500 000	500 000											
Replacement		5 000	5 000	5 000	5 000	5 000	5 000	5 000	5 000	5 000				
Subtotal Whitehorse Office	1 444 000	505 000	505 000	5 000	5 000	5 000	5 000	5 000	5 000	5 000	0	0	0	0
Contingincies														
All Items	5 489 250	2 000 000	2 000 000	2 000 000	2 000 000	2 000 000	2 000 000	2 000 000	2 000 000	2 000 000	2 000 000	1 000 000		
Total Capital Cost	12 928 250	17 130 000	7 620 000	16 635 000	18 935 000	11 765 000	12 220 000	21 505 000	12 195 000	3 005 000	3 000 000	11 000 000	10 000 000	10 000 000

Table 10.2 - 1

10.3 Revenue

Table 10.3-1 presents an estimate of pretax net operating revenue for the VP 1-5 Mine Plan, based on the following exchange rate and metal prices.

Exchange rate: CDN $1.00 = U.S. $0.725

Metal Prices

Lead	U.S. $440.00/kg,	U.S. $0.20/lb
Zinc	U.S. $880.00/kg,	U.S. $0.40/lb
Silver		U.A. $5.50/oz

Included in the net revenue calculation is the cost to retire a $50 million debt, amortized over seven years at an interest rate of 12 percent.

A simplified net present value for various discount rates is also presented. The net present value calculation refers to a start of January 1, 1987, and assumes that revenues (expenditures) occur at the end of each year.

File: REVVP15 — Case VP 1-5

02/04/87 — Revenue Estimate

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	87-04-01 to 87-12-31	1988	1989	1990	1991	1992	1993	1994
Lead Conc								
Tonnes (DMT)	153,169	194,584	181,956	240,010	165,137	146,497	157,150	188,772
% Pb	61.03%	60.88%	58.83%	53.39%	57.65%	59.79%	58.43%	60.00%
Ag g/t	562	548	598	603	693	760	426	824
Au g/t	0.00	0.00	1.22	3.86	4.75	5.51	0.66	6.04
Zinc Conc								
Tonnes (DMT)	271,522	358,612	308,270	282,454	300,798	310,261	344,115	353,848
% Zn	50.86%	50.87%	51.57%	54.64%	54.42%	54.49%	51.28%	55.00%
Total Conc								
Concentrate DMT	424,691	553,196	490,226	522,464	465,935	456,758	501,264	542,620

COST OF PRODUCTION

	87-04-01 to 87-12-31	1988	1989	1990	1991	1992	1993	1994
Cost FOB Skagway -- $CDN								
Operating	(81,176,250)	(111,735,500)	(107,773,278)	(107,705,437)	(120,454,677)	(117,592,568)	(108,933,356)	(117,215,438)
Capital	(12,928,250)	(17,130,000)	(7,620,000)	(16,635,000)	(18,935,000)	(11,765,000)	(12,220,000)	(21,505,000)
Ocean Freight								
$ / DMT Conc	$26.47	$26.47	$26.47	$26.47	$26.47	$26.47	$26.47	$26.47
Freight $	(11,241,571)	(14,643,098)	(12,976,277)	(13,829,628)	(12,333,301)	(12,090,385)	(13,268,470)	(14,363,164)
Market Commision -- $CDN								
$ / DMT Conc	$6.90	$6.90	$6.90	$6.90	$6.90	$6.90	$6.90	$6.90
Commision $	(2,930,368)	(3,817,052)	(3,382,558)	(3,605,003)	(3,214,952)	(3,151,630)	(3,458,725)	(3,744,081)
Exchange -- $CDN / $US	0.725	0.725	0.725	0.725	0.725	0.725	0.725	0.725
Smelter Charges -- $US								
$ / DMT Lead Conc	$119.30	$119.30	$119.30	$119.30	$119.30	$119.30	$119.30	$119.30
Lead Conc $	(18,273,062)	(23,213,871)	(21,707,323)	(28,633,212)	(19,700,791)	(17,477,059)	(18,747,957)	(22,520,496)
$ / DMT Zinc Conc	$157.65	$157.65	$157.65	$157.65	$157.65	$157.65	$157.65	$157.65
Zinc Conc $	(42,805,443)	(56,535,182)	(48,598,773)	(44,528,885)	(47,420,881)	(48,912,693)	(54,249,693)	(55,784,214)
Net Smelter Charge								
$US	(61,078,505)	(79,749,053)	(70,306,096)	(73,162,097)	(67,121,672)	(66,389,752)	(72,997,651)	(78,304,710)
$CDN	(84,246,214)	(109,998,694)	(96,973,925)	(100,913,237)	(92,581,617)	(91,572,072)	(100,686,415)	(108,006,497)
Total Cost -- $CDN	(192,522,652)	(257,324,344)	(228,726,038)	(242,688,305)	(247,519,546)	(236,171,655)	(238,566,966)	(264,834,179)

PAYABLE METAL

	87-04-01 to 87-12-31	1988	1989	1990	1991	1992	1993	1994
Contained Metal								
Lead DMT	93,479	118,463	107,041	128,137	95,209	87,595	91,819	113,263
Zinc DMT	138,096	182,426	158,979	154,330	163,689	169,076	176,467	194,617
Ag grams	86,080,978	106,632,032	108,763,387	144,743,749	114,521,502	111,386,005	66,943,839	155,515,877
Ag ounces	2,767,877	3,428,683	3,497,215	4,654,140	3,682,363	3,581,544	2,152,535	5,000,511
Au Grams	0	0	222,217	926,138	783,694	807,478	102,997	1,140,349
Au ounces	0	0	7,145	29,779	25,199	25,964	3,312	36,683

Table 10.3 - 1

File: REVVP15 Case VP 1-5

02/04/87 Revenue Estimate

08:59

	87-04-01 to 87-12-31	1988	1989	1990	1991	1992	1993	1994
Payable Metal								
Lead % Paid	95.10%	95.10%	95.10%	95.10%	95.10%	95.10%	95.10%	95.10%
Lead DMT Payable	88,899	112,658	101,796	121,859	90,544	83,303	87,320	107,713
Zinc % Paid	84.20%	84.20%	84.20%	84.20%	84.20%	84.20%	84.20%	84.20%
Zinc DMT Payable	116,277	153,603	133,860	129,946	137,826	142,362	148,585	163,867
Silver % Paid	92.60%	92.60%	92.60%	92.60%	92.60%	92.60%	92.60%	92.60%
Silver oz Payable	2,563,054	3,174,960	3,238,421	4,309,733	3,409,869	3,316,509	1,993,247	4,630,473
Gold % Paid	95.00%	95.00%	95.00%	95.00%	95.00%	95.00%	95.00%	95.00%
Gold oz Payable	0	0	6,788	28,290	23,939	24,666	3,146	34,849
Metal Prices -- $US								
Lead / Tonne	$440.00	$440.00	$440.00	$440.00	$440.00	$440.00	$440.00	$440.00
Zinc / Tonne	$880.00	$880.00	$880.00	$880.00	$880.00	$880.00	$880.00	$880.00
Silver / ounce	$5.50	$5.50	$5.50	$5.50	$5.50	$5.50	$5.50	$5.50
Gold / Ounce	$420.00	$420.00	$420.00	$420.00	$420.00	$420.00	$420.00	$420.00
Metal Revenue -- $US								
Lead	39,115,370	49,569,549	44,790,416	53,617,794	39,839,282	36,653,390	38,420,730	47,393,846
Zinc	102,323,678	135,170,313	117,796,830	114,352,479	121,287,106	125,278,597	130,754,988	144,203,167
Silver	14,096,798	17,462,281	17,811,316	23,703,534	18,754,277	18,240,801	10,962,861	25,467,600
Gold	0	0	2,850,953	11,881,969	10,054,463	10,359,600	1,321,413	14,636,621
Net Metal Revenue								
$US	155,535,846	202,202,143	183,249,515	203,555,766	189,935,129	190,532,389	181,459,992	231,701,234
$CDN	214,532,202	278,899,507	252,757,951	280,766,574	261,979,488	262,803,295	250,289,643	319,587,909
NET REVENUE -- $CDN								
Smelter Return	214,532,202	278,899,507	252,757,951	280,766,574	261,979,488	262,803,295	250,289,643	319,587,909
Debt Retirement	(10,955,887)	(10,955,887)	(10,955,887)	(10,955,887)	(10,955,887)	(10,955,887)	(10,955,887)	
Cost of Production	(192,522,652)	(257,324,344)	(228,726,038)	(242,688,305)	(247,519,546)	(236,171,655)	(238,566,966)	(264,834,179)
Net Revenue	11,053,662	10,619,276	13,076,026	27,122,382	3,504,055	15,675,754	766,791	54,753,730
Net Present Value at i =								
0%	11,053,662	21,672,939	34,748,965	61,871,347	65,375,402	81,051,156	81,817,947	136,571,677
8%	10,234,873	19,339,190	29,719,361	49,655,122	52,039,923	61,918,307	62,365,722	91,947,459
12%	9,869,341	18,334,963	27,642,220	44,878,985	46,867,280	54,809,104	55,155,962	77,270,075
16%	9,529,019	17,420,871	25,798,128	40,777,578	42,445,904	48,879,896	49,151,209	65,852,491
Cost to Produce Payable Zinc at Fixed Lead, Silver, Gold Prices								
$US per Tonne	$811.08	$829.88	$809.18	$728.68	$861.57	$800.17	$876.26	$637.75
$US per Pound	$0.368	$0.376	$0.367	$0.331	$0.391	$0.363	$0.397	$0.289
Cost to Produce Payable Lead at Fixed Zinc, Silver, Gold Prices								
$US per Tonne	$349.85	$371.66	$346.87	$278.63	$411.94	$303.57	$433.63	$71.46
$US per Pound	$0.159	$0.169	$0.157	$0.126	$0.187	$0.138	$0.197	$0.032

Table 10.3 - 1 Continued

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	1995	1996	1997	1998	1999	2000	Total
Lead Conc							
Tonnes (DMT)	187,138	179,202	137,121	90,548	31,261		2,052,543
% Pb	60.00%	60.00%	60.00%	56.46%	55.83%		58.74%
Ag g/t	843	847	863	670	598		683
Au g/t	6.82	6.82	7.74	3.99	0.00		3.85
Zinc Conc							
Tonnes (DMT)	376,395	340,155	266,509	168,732	55,169		3,736,840
% Zn	55.00%	55.00%	55.00%	52.36%	50.04%		53.37%
Total Conc							
Concentrate DMT	563,532	519,357	403,629	259,280	86,430		5,789,383

COST OF PRODUCTION

	1995	1996	1997	1998	1999	2000	Total
Cost FOB Skagway -- $CD							
Operating	(117,425,126)	(122,775,723)	(82,260,138)	(70,425,559)	(16,589,152)		(1,282,062,200)
Capital	(12,195,000)	(3,005,000)	(3,000,000)	(11,000,000)	(10,000,000)	(10,000,000)	(167,938,250)
Ocean Freight							
$ / DMT Conc	$26.47	$26.47	$26.47	$26.47	$26.47		
Freight $	(14,916,702)	(13,747,378)	(10,684,065)	(6,863,140)	(2,287,802)		(153,244,981)
Market Commision -- $CD							
$ / DMT Conc	$6.90	$6.90	$6.90	$6.90	$6.90		$6.90
Commision $	(3,888,373)	(3,583,563)	(2,785,042)	(1,789,032)	(596,367)		(39,946,746)
Exchange -- $CDN / $US	0.725	0.725	0.725	0.725	0.725	0.725	0.725
Smelter Charges -- $US							
$ / DMT Lead Conc	$119.30	$119.30	$119.30	$119.30	$119.30		
Lead Conc $	(22,325,530)	(21,378,818)	(16,358,491)	(10,802,352)	(3,729,437)		(244,868,399)
$ / DMT Zinc Conc	$157.65	$157.65	$157.65	$157.65	$157.65		
Zinc Conc $	(59,338,619)	(53,625,399)	(42,015,078)	(26,600,624)	(8,697,393)		
Net Smelter Charge							
$US	(81,664,149)	(75,004,216)	(58,373,569)	(37,402,976)	(12,426,830)		(833,981,277)
$CDN	(112,640,205)	(103,454,092)	(80,515,268)	(51,590,312)	(17,140,455)		(1,150,319,003)
Total Cost -- $CDN	(261,065,407)	(246,565,755)	(179,244,513)	(141,668,044)	(46,613,776)	(10,000,000)	(2,793,511,180)

PAYABLE METAL

	1995	1996	1997	1998	1999
Contained Metal					
Lead DMT	112,283	107,521	82,272	51,120	17,453
Zinc DMT	207,017	187,085	146,580	88,341	27,607
Ag grams	157,767,911	151,871,987	118,380,554	60,623,139	18,694,078
Ag ounces	5,072,923	4,883,344	3,806,449	1,949,297	601,096
Au Grams	1,276,936	1,221,856	1,061,308	361,425	0
Au ounces	41,059	39,288	34,126	11,621	0

Table 10.3 - 1 Continued

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	1995	1996	1997	1998	1999	2000	Total
Payable Metal							
Lead % Paid	95.10%	95.10%	95.10%	95.10%	95.10%		
Lead DMT Payable	106,781	102,253	78,241	48,615	16,598		1,146,579
Zinc % Paid	84.20%	84.20%	84.20%	84.20%	84.20%		
Zinc DMT Payable	174,308	157,526	123,420	74,383	23,245		1,679,208
Silver % Paid	92.60%	92.60%	92.60%	92.60%	92.60%		
Silver oz Payable	4,697,527	4,521,976	3,524,771	1,805,049	556,615		41,742,205
Gold % Paid	95.00%	95.00%	95.00%	95.00%	95.00%		
Gold oz Payable	39,006	37,324	32,419	11,040	0		241,468
Metal Prices -- $US							
Lead / Tonne	$440.00	$440.00	$440.00	$440.00	$440.00		
Zinc / Tonne	$880.00	$880.00	$880.00	$880.00	$880.00		
Silver / ounce	$5.50	$5.50	$5.50	$5.50	$5.50		
Gold / Ounce	$420.00	$420.00	$420.00	$420.00	$420.00		
Metal Revenue -- $US							
Lead	46,983,544	44,991,211	34,426,054	21,390,599	7,303,040		504,494,824
Zinc	153,391,365	138,622,591	108,609,710	65,457,183	20,455,362		1,477,703,370
Silver	25,836,398	24,870,869	19,386,243	9,927,770	3,061,381		229,582,129
Gold	16,382,557	15,675,900	13,616,143	4,636,929	0		101,416,539
Net Metal Revenue							
$US	242,593,864	224,160,571	176,038,149	101,412,481	30,819,783		2,313,196,862
$CDN	334,612,226	309,186,995	242,811,241	139,879,284	42,510,046		3,190,616,362
NET REVENUE -- $CDN							
Smelter Return	334,612,226	309,186,995	242,811,241	139,879,284	42,510,046		3,190,616,362
Debt Retirement							(76,691,208)
Cost of Production	(261,065,407)	(246,565,755)	(179,244,513)	(141,668,044)	(46,613,776)		(2,783,511,180)
Net Revenue	73,546,820	62,621,240	63,566,728	(1,788,760)	(4,103,730)	0	330,413,974
Net Present Value at i							
0%	210,118,497	272,739,736	336,306,464	334,517,705	330,413,974		
8%	128,739,179	157,744,930	185,007,610	184,297,269	182,788,336		
12%	103,791,795	123,954,159	142,228,074	141,768,944	140,828,475		
16%	85,191,846	99,387,055	111,809,067	111,507,728	110,911,757		
Cost to Produce Payable							
$US per Tonne	$574.10	$591.79	$506.59	$897.43	$1,007.99		$737.34
$US per Pound	$0.260	$0.268	$0.230	$0.407	$0.457		$0.334
Cost to Produce Payable							
$US per Tonne	($59.35)	($4.00)	($149.02)	$466.68	$619.25		$231.07
$US per Pound	($0.027)	($0.002)	($0.068)	$0.212	$0.281		$0.105

Table 10.3 - 1 Continued