

016990

THE VANGORDA PLATEAU
PRELIMINARY MINING STUDY

FARO AREA, YUKON

CASE VP 1-1

NOVEMBER 1986



Curragh
Resources



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1.0 EXECUTIVE SUMMARY

1.1 Production Schedule

The production schedule for this mine plan is presented in Table 1.1-1. This schedule is exclusive of Faro Pit operations which may be running concurrently.

:

Production Summary

	1988	1989	1990	1991	1992	1993	1994	1995	1996	1997	1998	1999	Total
Mined:													
Waste	7418146	22940242	20100031	20652096	21138900	28603010	33089496	29990029	28168891	31868767	8179673	0	252149281
Ore	600216	1294491	4887262	4335191	3848385	74225	758530	3858001	5679132	1979256	4137466	0	31452155
Mill:													
Feed	0	1012000	4015000	3717701	2035000	2189000	1187680	4015000	4015000	4013776	4024671	1227326	31452154
Pb Concentrate	0	52519	242119	142737	67707	75671	50619	169263	187060	138473	171700	23416	1321284
Zn Concentrate	0	64871	283342	256061	143675	157724	90062	323267	377469	257072	335045	36516	2325105
Total Concentrate	0	117390	525461	398798	211382	233395	140681	492530	564529	395545	506746	59932	3646389
Strip Ratio													
Per Tonne Feed		22.7	5.0	5.6	10.4	13.1	27.9	7.5	7.0	7.9	2.0	0.0	8.0

TABLE 1.1 - 1
PRODUCTION SUMMARY

1.2 Costs

Capital and operating cost estimates are presented in Table 1.2-1. These costs are all FOB Skagway. This cost schedule is exclusive of any Faro Pit operations which may be running concurrently. All values are in constant (fourth quarter 1986) Canadian dollars.

Cost Summary

	1987	1988	1989	1990	1991	1992	1993	1994	1995	1996	1997	1998	1999	Total
Capital Cost	5,995,000	10,470,535	4,507,718	9,232,109	8,732,108	18,072,107	16,193,472	14,400,089	12,420,090	14,800,089	4,920,089	1,790,397	0	121,333,802
Operating Cost		8,465,746	40,957,768	91,244,345	86,481,603	81,008,360	65,983,763	64,712,151	112,106,586	116,592,503	107,820,612	81,249,660	13,790,592	850,413,688
Total Cost	5,995,000	18,936,281	45,465,485	100,476,454	95,213,711	79,080,467	82,177,235	79,112,240	124,526,675	131,392,592	112,740,701	83,040,057	13,790,592	971,947,490
Cost per Tonne Conc.			387.30	191.22	238.75	374.11	352.10	562.35	252.83	232.75	285.03	163.87	230.10	266.55

TABLE 1.2 - 1
COST ESTIMATE

1.3 Recommendations for Future Work

This study, while endeavoring to be complete, suffers from limitations in time and available information. It is therefore recommended that the following work be done in order to provide a more realistic production plan:

- carry out geotechnical studies on overburden slope and dump stability
- examine alternate means of overburden removal
- examine the usefulness of overburden dewatering
- accurately define the orientation of foliation surfaces in the rock
- redesign the Grum Pit and its stages in order to levelize the waste stripping
- finalize the Vangorda Haul Road alignment
- undertake more metallurgical test work of the Grum ore in order to define (a) the metallurgical response of the individual ore types, and (b) the effect of oxidation on stockpiles ore
- define the degree and extent of oxidation in the southeast end of the Vangorda orebody
- assess the Champ Zone of the Grum orebody for possible inclusion into the mine plan
- complete exploration drilling in the area of the waste dumps to ensure that no reserves are sterilized
- fill-in drilling as required on both orebodies
- remodel both orebodies

2.0 INTRODUCTION AND TERMS OF REFERENCE

Introduction and Terms of Reference

Curragh Resources owns several lead zinc silver deposits in the Anvil Range in addition to the Faro mine. These deposits offer the opportunity to significantly extend the utility of investment in reactivating the Faro operation and to extend the life of both Curragh and the Faro area.

Considerable work and study, as well as previous expenditure, has been directed at the development of additional reserves by the former owners, Cyprus Anvil Mining Corporation and Kerr Addison Mines. This work has focused largely on the potentially open pit mineable Grum and Vangorda deposits on the Vangorda Plateau 14 km southeast of the Faro concentrator.

Included were:

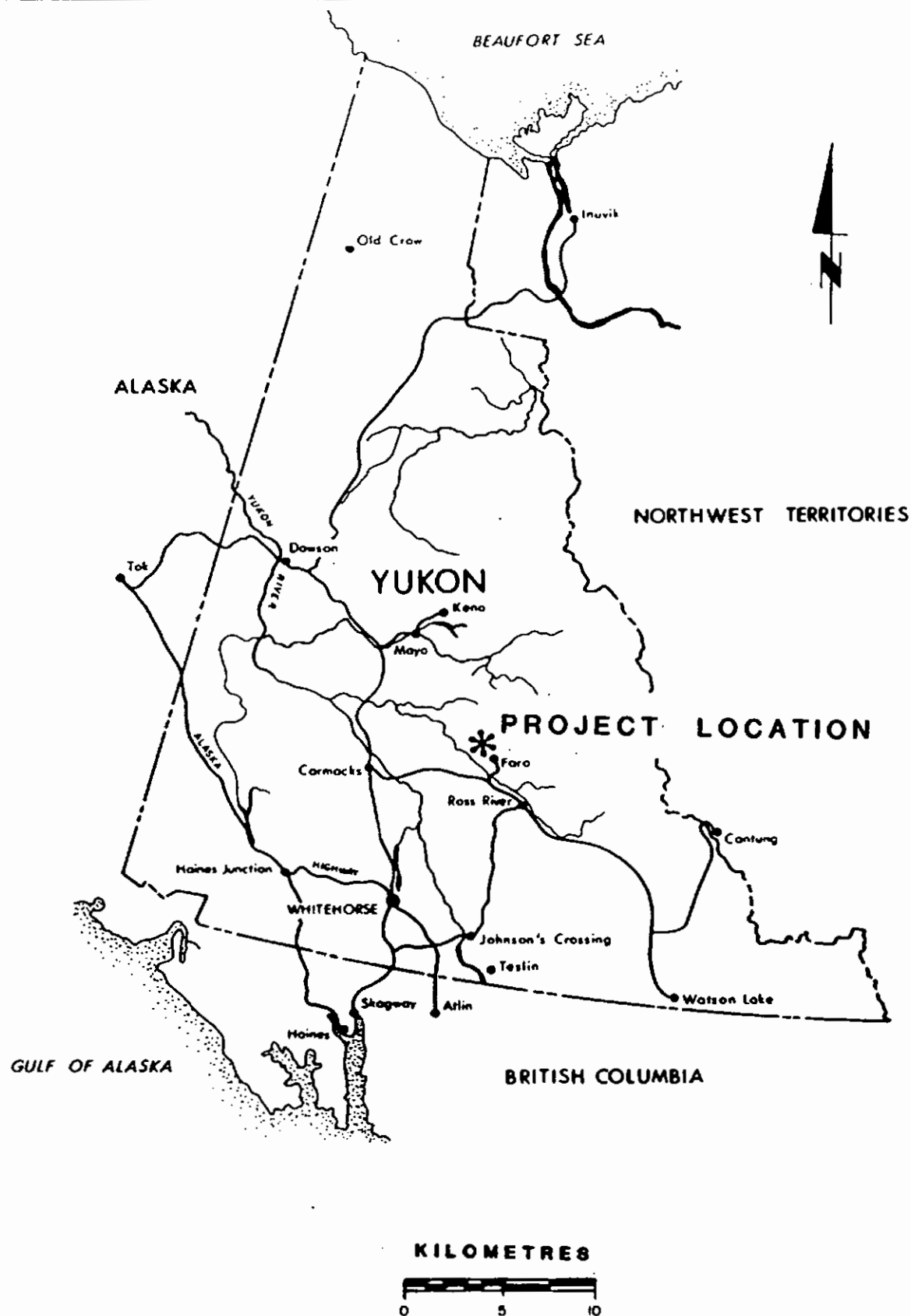
- 1.) Major renovations to the Faro concentrator to handle finer grind.
- 2.) Study of transportation alternatives between the Plateau and the Faro concentrator.
- 3.) Metallurgical testing of Vangorda Plateau ores.
- 4.) Extensive fillin drilling and geological interpretation directed at revised reserve calculations.
- 5.) Preliminary mine plans.
- 6.) Environmental baseline studies relevant to mining development on the Plateau.

In this study the above work was reviewed and incorporated in a mine plan for the Grum and Vangorda integrating the deposits with declining production from the Faro deposit as it nears exhaustion. New reserves amenable to computer aided mine design were calculated. Specific instructions recieved from Mr. C.H. Frame at a planning meeting in Whitehorse on 22 October, 1986 were:

- 1.) to use current production statistics and costs for the Faro operation as outlined in the 1987 budget with adjustments to reflect Vangorda Plateau ore versus Faro.
- 2.) to use, as much as possible, surplus equipment from the Faro operation as it becomes available.
- 3.) to develop a plan requiring the minimum pre-production capital investments possible and to levelize stripping.
- 4.) to produce a minimum of 500,000 DMT per annum of concentrate for at least 10 years
- 5.) to consider underground mining to fill in ore shortages as required.

- 6.) to carry out sensitivity analyses at $\pm 5\%$ on critical elements such as pit equipment productivities, feed grade, mill recoveries, etc.
- 7.) to thoroughly review the basic geologic data, identify shortfalls and implement plans to rectify.
- 8.) to evaluate all potential points of weakness in the plan and propose future work to eliminate them.
- 9.) to complete the plan by November 30, 1986
- 10.) costs and quantities only to be considered; no cash flow to be done -FOB on ship in Skagway.
- 11.) Costs expressed as 1986 4th quarter dollars with no escalation.

The resulting plan is outlined in the following pages.



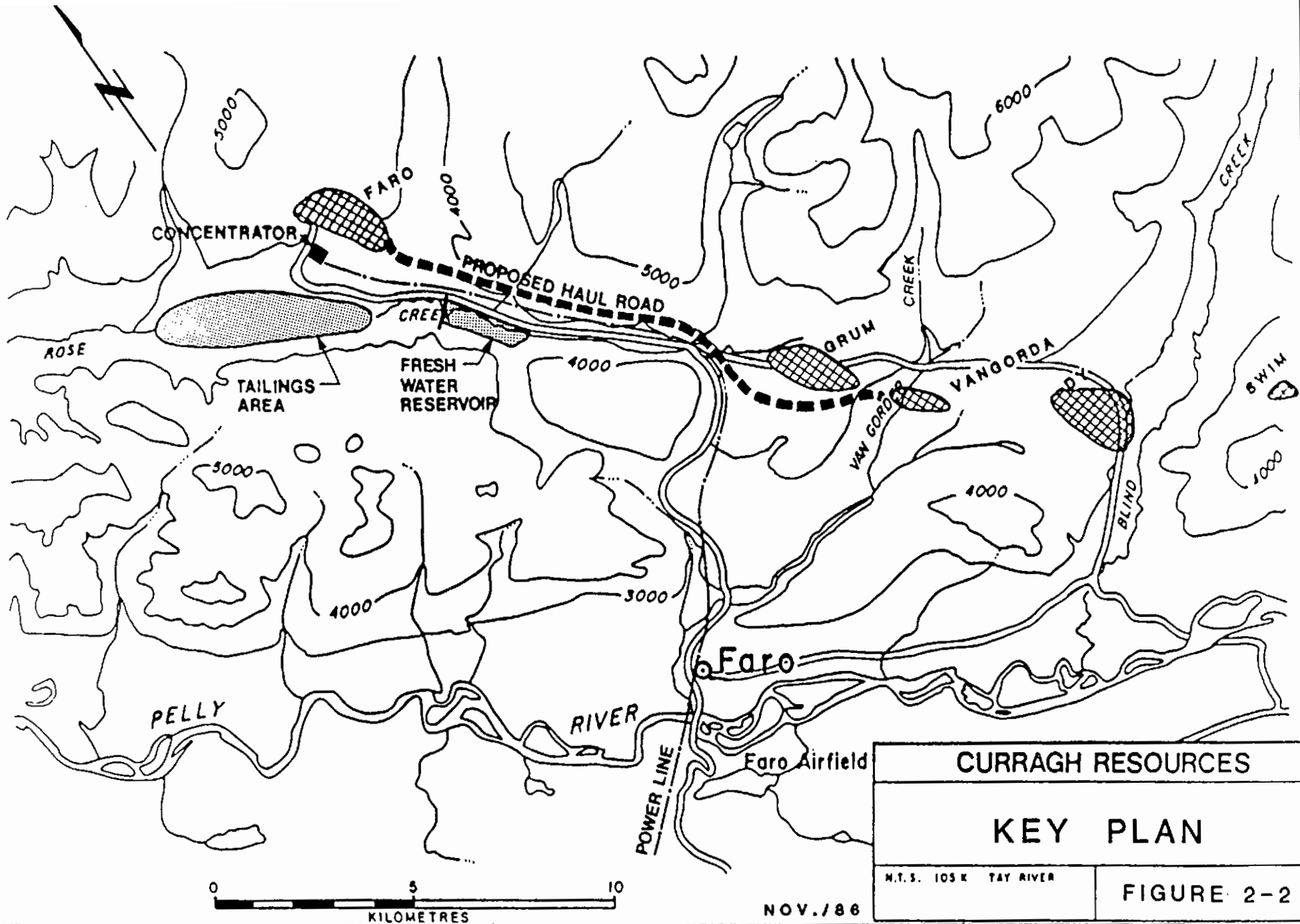
CURRAGH RESOURCES

LOCATION PLAN

PREPARED BY: CGY

NOV./86

FIGURE 2-1



3.0 GEOLOGY

3.1 DISTRICT GEOLOGY

3.1.1 Introduction

3.1.2 Regional Geology :

3.1.3 District Stratigraphy

3.3.1 General

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3.1.5.2.1.5 General Comments

3.1.5.2.2 Quartzose Disseminated Lithofacies

3.1.5.2.2.1 Ribbon Banded Graphitic Quartzit

3.1.5.2.2.2 Pyritic Quartzite

3.1.5.2.2.3 General Comments

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3.1.5.5 Genetic Model

3.1.6 Geology of the Vangorda Plateau

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. This ore can support mining activities in the Anvil District for many years following the exhaustion of the Faro deposit. Most of these deposits are within economical haulage distance to the Faro concentrator (figure 2.3 and 3.2).

Regional Geology

The Anvil District is part of the Selwyn Basin (figure 3.4), 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 metalliferous province of world wide significance.

The Anvil District differs from the remainder of the Selwyn Basin because the rocks and ore deposits are metamorphosed and significantly recrystallized. This has resulted in coarser grain size with improved metallurgical response. This geologic factor, along with the size of the Faro deposit and its location, have combined to determine that Faro is as yet the only producer of the Selwyn Basin.

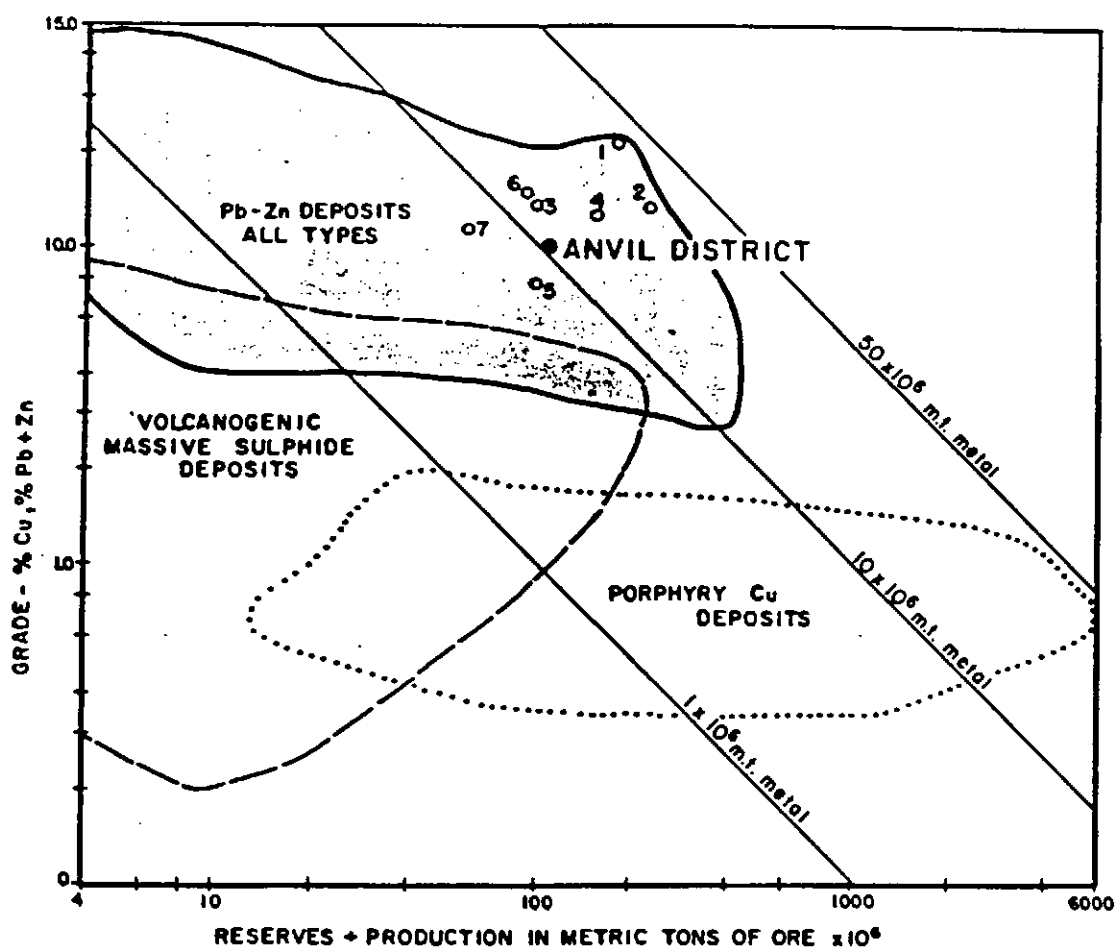
District Stratigraphy

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

The lower division ranges in age from late Precambrian to perhaps Early Silurian. It is approximately 5 km thick and divisible into three major mappable units (fig. 3.5). 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 or interval but the rocks are very similar to those of Kechika Group (Gordey, 1981) south of the district in Pelly Mountains.

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 lead zinc deposits occur within a restricted portion of the lower division. The upper division is host to stratiform barite deposits and to a number of interesting geologic problems beyond

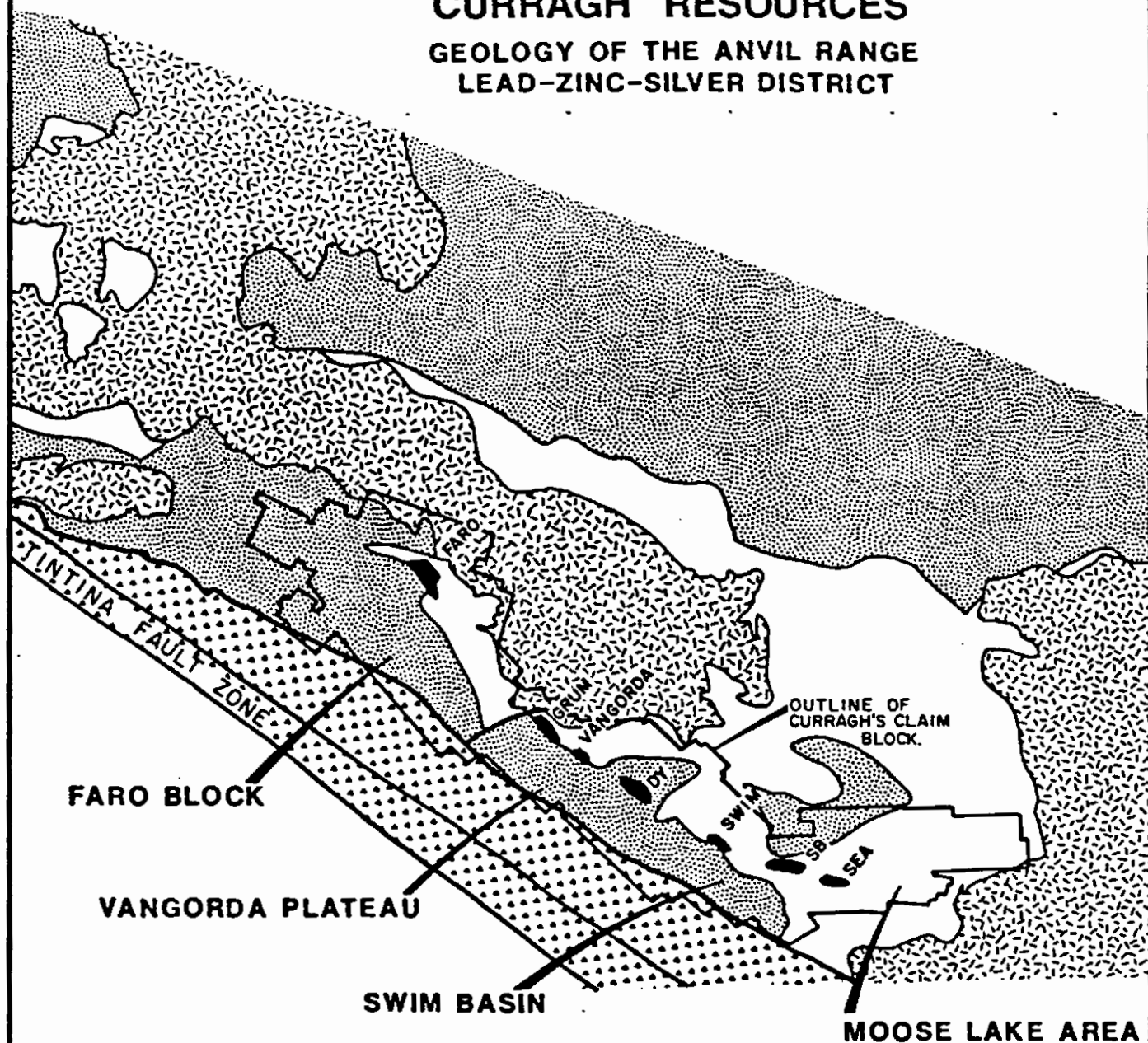


- 1 BROKEN HILL, AUSTRALIA
- 2 MACARTHUR RIVER, AUSTRALIA
- 3 MT. ISA, AUSTRALIA
- 4 SULLIVAN, CANADA
- 5 HOWARDS PASS, CANADA
- 6 RED DOG, ALASKA
- 7 MEGGEN^E, WEST GERMANY

FIGURE 2
COMPARISON OF SIZE GRADE CHARACTERISTICS
OF SOME MAJOR LEAD-ZINC DEPOSITS

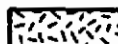
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GEOLOGY OF THE ANVIL RANGE LEAD-ZINC-SILVER DISTRICT

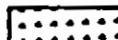


LEGEND:

CRETACEOUS

 ANVIL BATHOLITH: granite, granodiorite

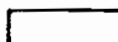
PALEOZOIC and MESOZOIC

 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 3.2

Carbonate orthoquartzite facies of platforms,
Silurian-mid Devonian
PCP = Pelly Cassiar Platform

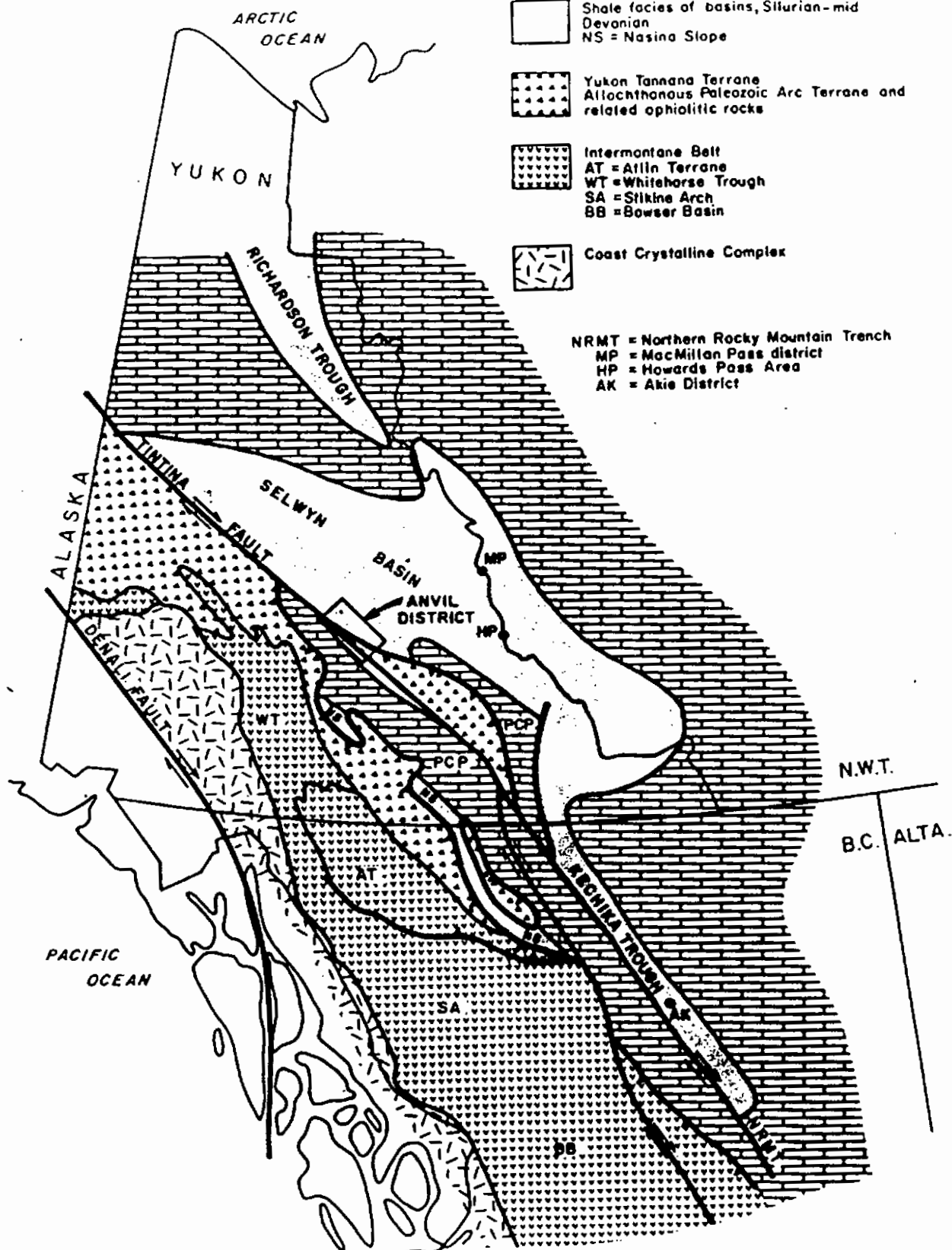
Shale facies of basins, Silurian-mid
Devonian
NS = Nasina Slope

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 = Axie District



MAPPABLE SUBDIVISIONS OF THE LOWER DIVISION OF ANVIL RANGE

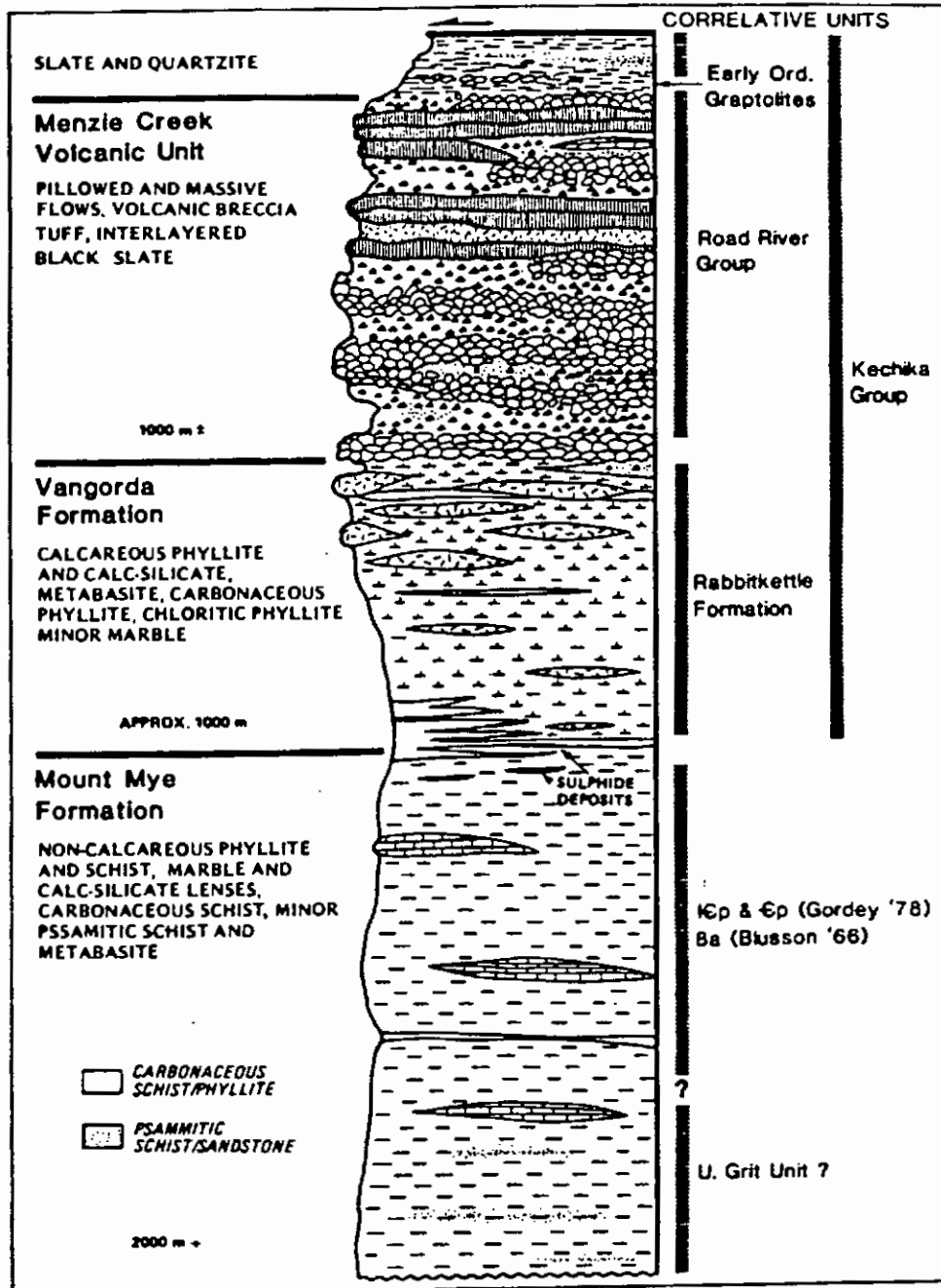


Figure 35. Diagrammatic stratigraphic section of the lower Paleozoic of Anvil Range showing the ore deposits in relation to stratigraphy. Note that the bulk of the metavolcanics or metabasites are younger than the ore deposits but that the deposits are approximately coincident with the first appearance of substantial mafic igneous material in the section. Note also the anomalous thickness of carbonaceous rocks near the ore deposit trend.

the scope of this summary.

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, interlayered, black graphitic phyllite, marble, calc-silicate phyllite or schist, metabasite and psammitic schist. The unit is at least 2 kilometers thick but 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.

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 carbonaceous, muscovite-chlorite pelite and b) light grey, generally calcareous quartz + calcite + dolomite siltstone. 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.

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. Whole rock major element and trace element data (Jennings et al., 1980) imply that the flows of the Menzie Creek volcanic unit are dominantly alkali basalt erupted in a within-plate setting

similar to metabasites of Vangorda formation. 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.

Relation of Stratigraphy to ore deposits

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.

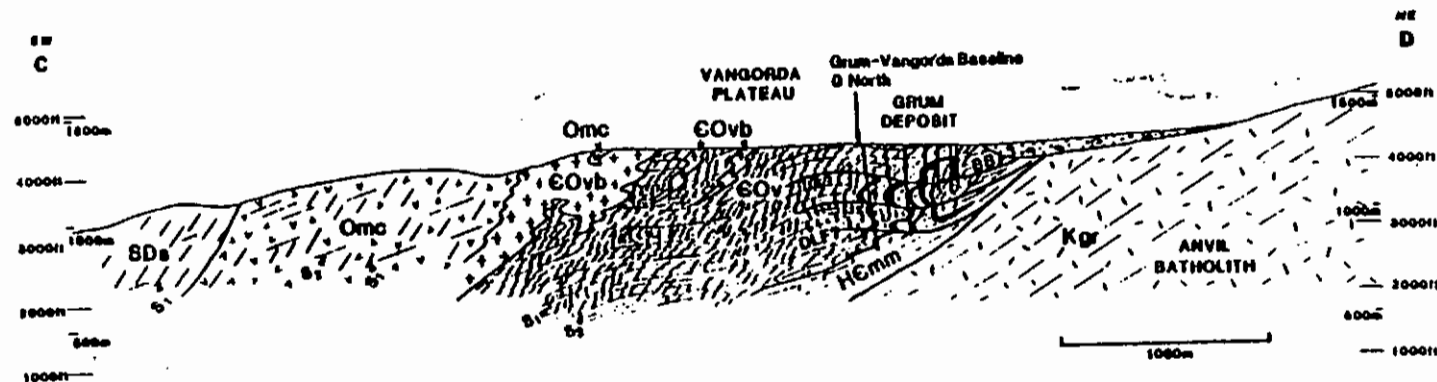
Deformation, Metamorphism and Plutonism

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 3.6). The largest megascopic folds known to have been formed during D_2 are those at the Grum Deposit (Fig. 3.9) and comparable folds in the Swim Deposit (Fig. 3.13). Parallel to the axial planes of these D_2 folds is a crenulation cleavage (S_2) which imparts a well developed lithon structure 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, with tight mesoscopic folds developed in nearly pervasive S_2 with appreciable mica growth along S_4 (see



DEPOSITION

D₁

D₂

Figure 8. Cross section through Vangorda Plateau and Grum deposit (86 W). The Grum deposit provides the best example of the D₁/D₂ interference pattern in the district. The deposit is involved in a large Z (or M) shaped D₁ fold refolded by S shaped D₂ folds. The steeply dipping S₁ crenulated by shallowly dipping S₂ is typical of the structural relations on the Vangorda Plateau where greenschist facies rocks dominate. Post D₂ folds gently warp the S₂ foliation. The inserts show the sequential development of Grum from a sequence of stacked en echelon ore layers parallel to S₀ through D₁ and D₂ to produce the geometry observed today.

Fig. 3.8 for examples of fourth phase affecting outline of the Faro deposit).

During the later stages of this 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 occurred along its margins. These faults determine the present day limits of several of the deposits (Fig. 3.10, 3.12).

Metamorphism was concurrent with deformation and was most intense during the early deformations, especially D₂. Metamorphic facies developed range from middle amphibolite facies to lower greenschist facies in a low pressure Buchan type facies series.

Metamorphic isograds are roughly concentric about the Anvil Batholith. Faro, close to the Batholith (figure 3.2) is strongly metamorphosed, while deposits such as Vangorda are less intensely metamorphosed. This difference in metamorphism is reflected in decreased grain size and increased degree of mineral intergrowth in the less metamorphosed deposits. This has a significant impact on metallurgical response of Anvil district ores.

Ore Deposits

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, massive base metal-bearing sulphide facies and a lower and peripheral, disseminated, quartzose 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 amoeboid shapes. Individual sulphide horizons commonly are 10 to 40 m in thickness. The upper and lower contacts of sulphide horizons are invariably sharp while lateral extensions grade into the enclosing host rocks.

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

The simplified arrangement of the ore 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 yield 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 prominent curvilinear trend on the south flank of Anvil Arch (Fig. 3.2). From northwest to southeast they include Faro, Grum, Vangorda, Dy and Swim. Additionally two base metal deficient sulphide occurrences, the SB and Sea, are also known. The Firth showing is best considered a faulted part of Grum while the Champ is the subcrop of one of the upper horizons of Grum. Current reserve figures for the deposits are given in Table 1.

The Anvil deposits are distributed through a 150 m stratigraphic interval straddling the boundary of the Mt. Mye and Vangorda formations in association with a regionally developed, but laterally discontinuous "graphitic" (carbonaceous) phyllite units. Individual sulphide lenses are, or appear to be, the lateral facies equivalent of graphitic phyllite. Some lenses of some deposits (such as the upper horizons of Grum) are basal to the carbonaceous phyllite units as well as being their partial lateral equivalents. In other cases, lateral equivalence of graphitic phyllite and ore lenses has not been established.

While the bulk of basaltic meta-igneous rocks occur up-section of the Anvil deposits, the first significant pulse of basaltic activity is roughly coincident with the sulphide horizons suggesting at least a temporal relationship between ore formation and basaltic magmatism. Despite this, there is generally poor spatial association of sulphide deposits and

metabasaltic rocks. The Dy and, to a lesser extent, the Grum deposits are exceptions to this generalization. In this sense, the Anvil deposits are dominantly pelitic sediment hosted.

Detailed mapping and drilling suggest the linearly distributed deposits lie close to a northeasterly "pinch out" or "zero edge" of the associated graphitic phyllite (the basal member of Vangorda formation). To date, no sulphide deposit lithofacies have been encountered in a moderate number of drill holes through the ore-bearing horizon southwest of the deposit line. Taken together, these observations suggest some relationship between sulphide deposits, facies changes at reduced basinal margins and basaltic activity.

Description of Sulphide Lithofacies

In the following sections unit 2 refers to amphibolite facies ore types (Faro), 4 to greenschist facies (all others).

Massive Pyritic Sulphides: (Unit 2E/4E)

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, chalcopryite, 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 base metal sulphides. This texture usually is restricted to rocks with economic lead-zinc grades (Unit 4F/2F).

Baritic, Massive Pyritic Sulphides: (Unit 2G/4G) 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.

Carbonate-bearing, Massive Pyritic Sulphides: (Unit 2K/4K) similar to massive pyritic sulphides but contains 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 occurrence of coarse pinkish beige to tan, ankerite patches may represent recrystallized original carbonate or re-worked

pre/syn-metamorphic veins.

Pyrrhotitic Massive Sulphides: (Unit 2H/4H) massive, finely crystalline, usually well foliated pyrrhotite with less than 50% pyrite porphyroblasts and highly variable amounts of shalerite 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.

General Comments on Massive Sulphides

Breccia textures are more common in the massive pyritic and pyrrhotitic facies than in the barite or carbonate-bearing facies. Pyritic breccias generally involve fragments of more quartzose or less base metal rich pyritic facies in a massive pyrite plus base metal sulphide matrix. Fragments can be angular to subrounded, poorly sorted and may be in either clast or matrix support. In some cases, margins of fragments can be fit back together. In all cases, the breccias are post-metamorphic since they involve variably oriented, foliated clasts. The origin of the breccias appears to relate to ductility contrasts between the affected lithologies during sulphide flow induced by deformation and metamorphism. These are clearly not primary breccias related to feeder zones or paleoslumps prior to sulphide lithification.

Friable and porous massive sulphides are relatively common and when strongly developed often degenerate to pyrite sand. The porous massive sulphides are commonly carbonate or barite bearing and originate by post-metamorphic groundwater leaching and oxidation especially near faults.

Quartzose Disseminated Lithofacies

Ribbon banded, "graphitic", pyritic quartzite: (Unit 2A/4A) 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 with a total sulphide content usually between 10 to 30% but ranging from 2% to 60%. Pyrite is usually the dominant species but higher grade examples have sub-equal pyritic and base metal sulphides. Base-metal 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.

Pyritic quartzite: (Units 4B, C, D/2B, C, D) light to medium grey, generally poorly banded, moderately to weakly foliated, micaceous quartzites with highly variable base metal 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/4B) shows low pyrite (< 5%) content with base metal sulphides predominant. Barite in major amounts is uncommon in the quartzose 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.

General Comments on disseminated ore types

Post-metamorphic breccias are also common in the disseminated sulphide lithofacies. Pyritic quartzite breccias are often well developed in the sphalerite-rich high grade facies where again ductility contrasts between the sulphides and quartzite bands dictate ductile flow in the sulphides and brittle failure, rotation and brecciation on the quartzite. Where less intensively developed the breccias grade into examples of sulphide mobilization into D₂ or later cleavages.

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 (Unit 4L). 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.

Many mineralogical variants of the alteration facies are recognized including siliceous, carbonate-bearing, talcose, chloritic, pyritic, pyrrhotitic, chalcopyrite-bearing, magnetite-bearing and base metal-bearing species. Careful attention has been paid to the distribution of these facies in an attempt to define feeder zones for all deposits. To date, little success has been had in this regard as no unequivocal feeder zones have been recognized. Several instances of

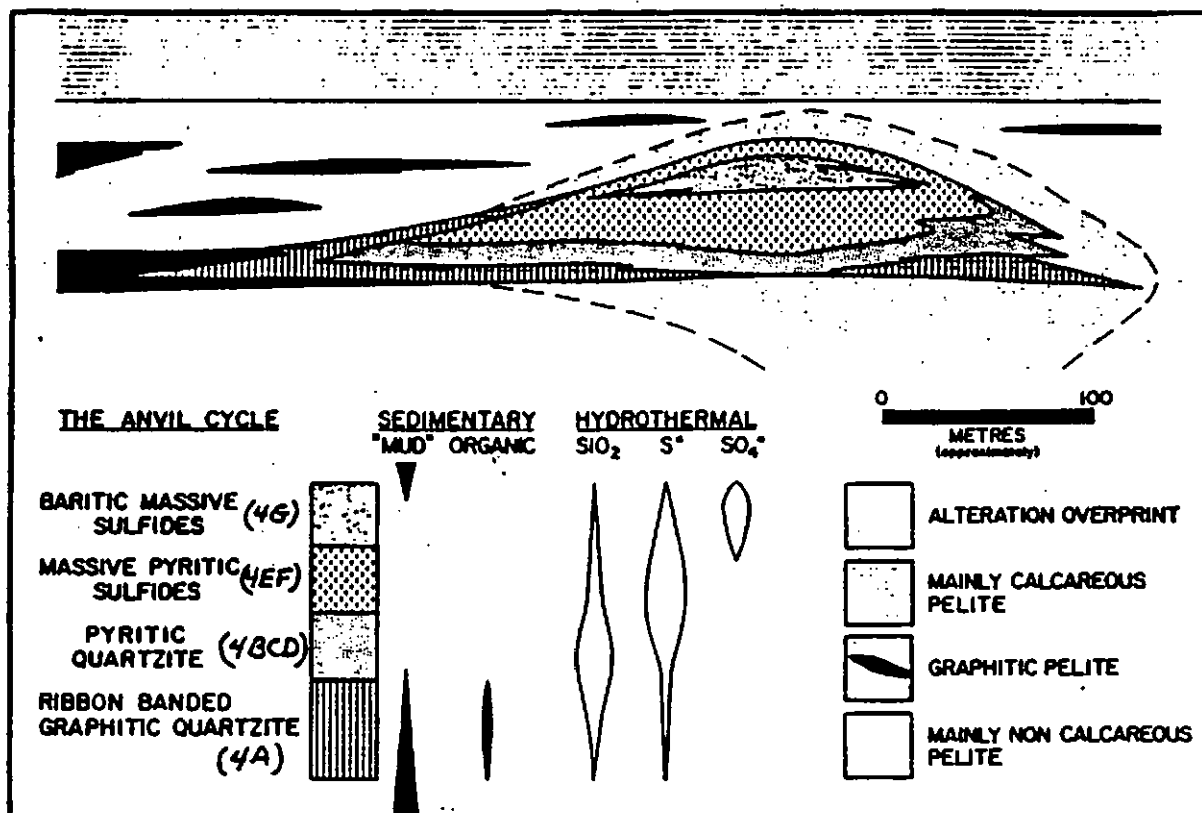
suspected pre-D₁ and certainly pre D₂ quartz-chlorite-pyrrhotite-chalcopryrite veinlets or stringers have been observed in the altered stratigraphic footwalls of several horizons (Swim deposit in particular) but not in sufficient abundance to define a stringer or feeder zone comparable to volcanogenic deposits. Recognition of a feeder zone is considerably hampered in this terrane by the polydeformational overprint.

In the multi-layered deposits at greenschist facies grade, all mineralogical variants of the alteration facies are commonly recognized, often with the best degree of development in the footwall of a mineralized horizon. The only amphibolite-grade example, the Faro deposit, shows a much less varied phase assemblage (muscovite, quartz, pyrite $\pm$ marcasite) in altered rocks with development of a substantial hanging wall as well as footwall alteration envelope. This simplified phase assemblage may be due to re-equilibration of the greenschist alteration assemblage at higher grades of metamorphism. The prominent hanging wall alteration may be due to continued post-hydrothermal activity or to sulfurization or other metasomatic reactions in the wallrocks during metamorphism perhaps caused by mobile sulfur from the inversion of pyrite to pyrrhotite in the deposit. It is interesting to note the development of massive pyrrhotitic facies is greatest in the Faro deposit which also shows the most well defined broadest and most symmetrically developed alteration envelope.

Idealized Anvil Deposit

There are sufficient similarities between the Faro deposit and other Anvil district deposits for it to serve as a model of common deposit characteristics. Figure 3.7 is a generalized, pre-deformation vertically exaggerated cross-section of Faro illustrating these features. In addition to all deposits showing a spatial relation to the Mt. Mye/Vangorda boundary and having a variably developed alteration overprint, they have a distinct arrangement of sulphide lithofacies. This arrangement in a vertical and lateral sense is so commonly seen within and between deposits, it has been termed the Anvil Cycle. The base of the cycle is marked by ribbon-banded, graphitic, pyritic quartzites succeeded upward by pyritic quartzites, massive pyritic sulphides and baritic massive pyritic sulphides (fig. 3.7). This array is also seen laterally with ribbon-banded, graphitic, pyritic quartzites forming the marginal or distal facies of a deposit inward to the baritic massive facies.

It is important to note that Anvil Cycles are developed on a wide variety of scales and to varying degrees of completeness. The most common scale is that of a cross-section through an entire deposit making recognition in individual boreholes or exposures often difficult. A series of complete and partial cycles may cumulatively form a mega-cycle on the scale of a complete deposit, e.g. Faro, or on the scale of a single sulphide horizon within a multi-layered deposit, e.g. Grum or Dy. Complete cycles are seen over a one meter stratigraphic



3.7

Figure 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 peripheral and lower quartzose disseminated lithofacies.

interval (or less), emphasizing the scale at which facies ordering can occur.

Metal zoning in some cases complements this facies distribution pattern in a crude way. The quartzose disseminated sulphide facies at the base of an ideal cycle tend to be zinc enriched. The massive, upper facies are slightly lead-silver enriched, particularly the upper most baritic facies. However, commonly there is no evidence in the assays of metal zonation in an individual horizon. As might be expected studies of metal zoning are greatly hampered by the structural complexity of the Anvil deposits. Consequently, definitive deposit wide studies are not yet available.

On the basis of scanty and preliminary data, copper and to a lesser extent gold seem to be preferentially distributed in siliceous facies of the footwall-biased alteration overprint or in the pyritic quartzite facies of the stratiform sulphides; this again is a characteristic that varies from deposit to deposit.

Facies zoning can be used in a tenuous way as top indicators in polydeformed horizons to decipher fold patterns: the more complete facies cyclicity shown, the greater degree of confidence. It is stressed that top directions defined by the unambiguous distribution of Mt. Mye and Vangorda formation lithologies always take precedence over those interpreted from sulphide facies ordering.

Genetic Model

The Anvil deposits are examples of synsedimentary, stratiform, massive sulphide deposits considered to be submarine exhalative in origin. Evidence for their synsedimentary origin includes:

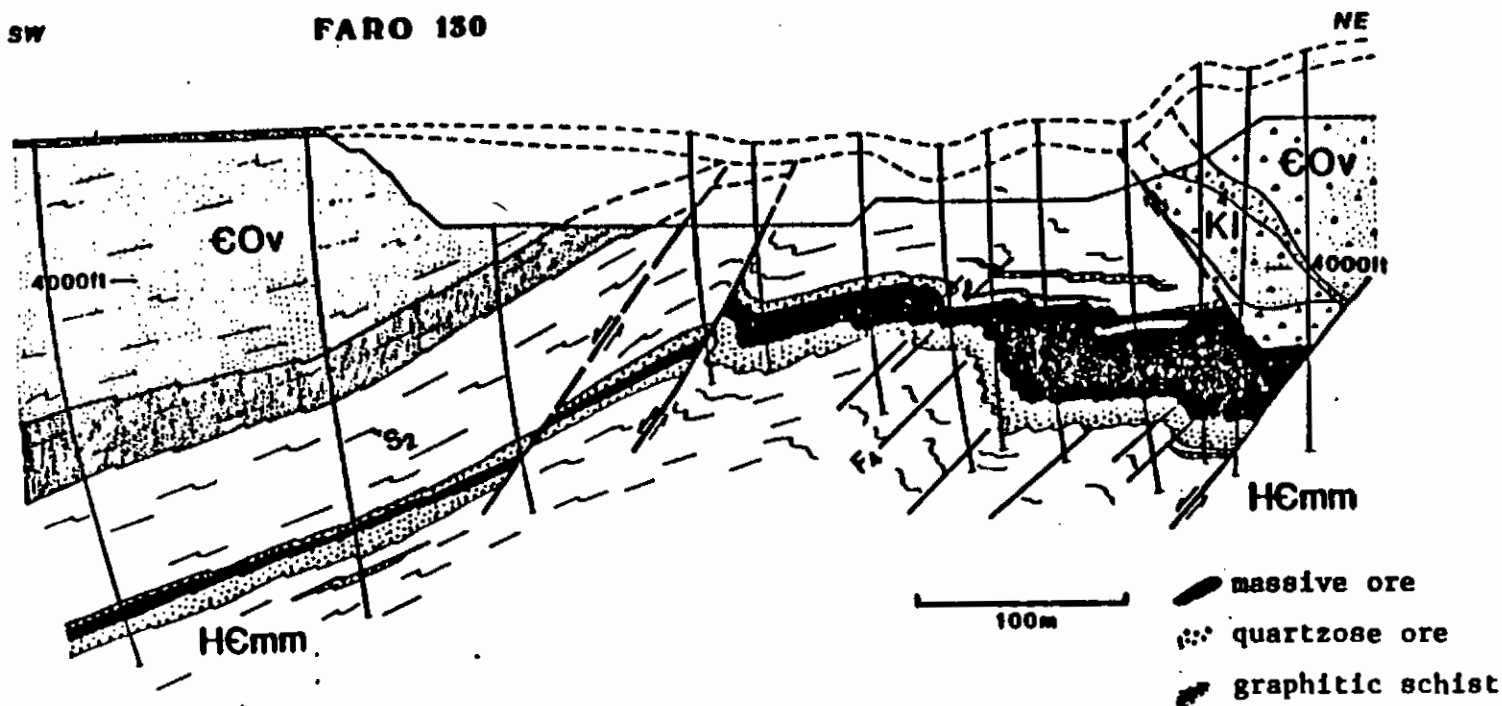
- 1.) The prevalent and well developed compositional layering or banding in many or most sulphide facies commonly with large variation in proportions of sulphide species between bands.
- 2.) The interlayering of sulphides with unmineralized metasedimentary and probable metavolcanic rocks, commonly on scale of centimeters.
- 3.) The occurrence of all deposits within a relatively restricted vertical stratigraphic interval.
- 4.) The curvilinear depositional trend crudely associated with graphitic carbonaceous pelite facies change.
- 5.) The metamorphic and deformational overprints which clearly show the ores are pre-metamorphic.

No unequivocal evidence supporting the notion of an exhalative origin is preserved in the district however and it is important

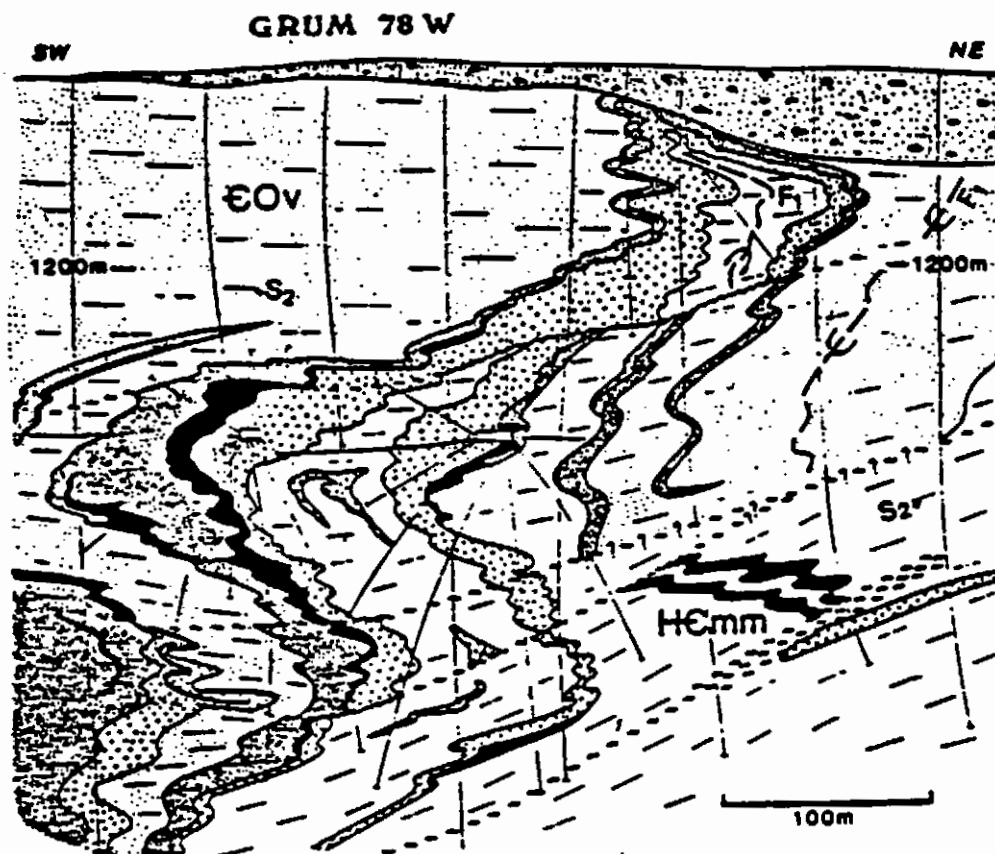
SW

FARO 130

NE



3.8
Figure 1. 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.



3.9

Figure 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

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 accomplished 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 models methodologies (as well as the Grum and Vangorda models).

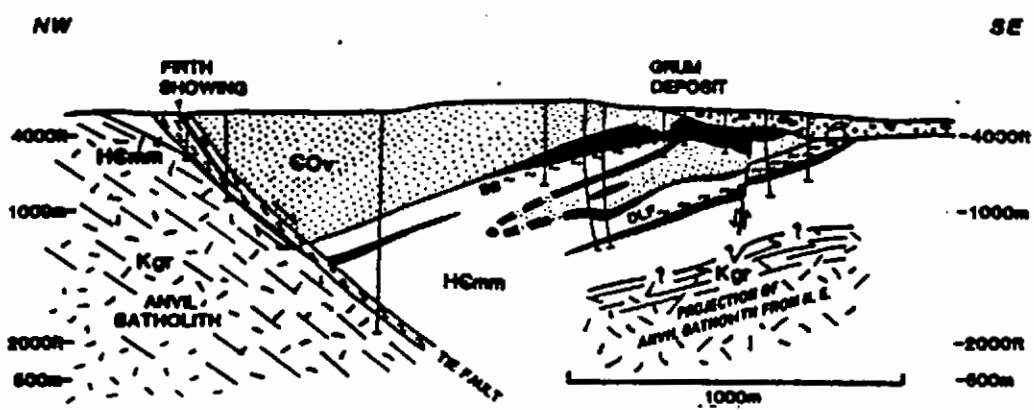
Specific gravity is treated as an assay and interpolated into blocks. This is due to the variability of SG by rock type and by grade as well as regional variations of SG within one rock type. Prior to interpolation the composite pulp SG's were reduced by 5% for quartzose ore types or 10% for massive ore types to correct for porosity in the insitu whole rock.

3.2.4.2.4 Reserve reporting

Geological reserves were not computed since the model only covers the part of the deposit between sections 117 and 125. Fit 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 Results

Table 3.2 compares the FI model with three computer calculated values 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 interpolation done by the author in September 1985 incorporating all drilling data available and using a fault dominant as opposed to fold dominant interpretation. The FI model tends to report a lower tonnage at a higher grade than



3.10

Figure 10. A diagrammatic 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

3.2 Faro

3.2.1 History

3.2.2 General Geology

3.2.3 Drilling Density

3.2.4 Reserve Calculation

3.2.4.1 Methods and Procedure - FI Model

3.2.4.1.1 Block Geology and Drillhole Data

3.2.4.1.2 Composite Calculation

3.2.4.1.3 Interpolation

3.2.4.1.3 Reserve Reporting

3.2.4.1 Methods and Procedure - F8608 Model

3.2.4.2.1 Block Geology & drillhole information

3.2.4.2.2 Composites

3.2.4.2.3 Interpolation

3.2.4.2.4 Reserve Reporting

3.2.4.3 Results

3.2 Faro

The Faro deposit is not part of this study however a section is included to emphasize its differences from the Vangorda Plateau deposits, to review Faro models and to put the model calculations into a production context to help assess their reliability

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.

3.2.2 General Geology

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 that grades downwards into a coarse, gneissic biotite-muscovite schist. The Vangorda formation at Faro is represented by hard, dense, banded calc-silicates rather than the calcareous phyllite that characterizes the Vangorda Plateau. This fact is of considerable importance in blasthole drilling at Faro because of the rocks hardness.

Post metamorphic igneous intrusive rocks are more widely developed at Faro than on the Vangorda Plateau. There are two clans of importance: a) a equigranular to subporphyritic hornblende diorite to quartz diorite clan and b) a quartz-feldspar porphyry clan.

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.

Before mining, the Faro deposit was 2000 m along strike, 800 m across strike and about 70 m thick. The deposit is a flat-lying, elongate, asymmetric lens with a thick northeast side and a thin tapering southwest side. The deposit is cut by several important faults which form a graben structure, the mined out zones 1 and 2 were the upthrown blocks, and zone 3 the

central graben. Zone 3 contains the remaining reserves.

There is essentially one thick horizon at Faro although this horizon contains numerous Anvil 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.

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 (figure 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, pyrrhotic mineralization in the southwest part of the deposit. Grade zoning follows ore type zoning so that the base and northeast edge of the deposit contains the lower grade mineralization whereas the upper and southwest portion contains the higher grade mineralization. Zoning was also obvious in plan view at Faro. Zone 1 was rich in baritic ores thus high grade, zone 2 at the other end of the deposit was rich in carbonaceous quartzose ore types thus low grade and metallurgically undesirable. Zone 3 has intermediate characteristics.

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 fillin was added to the northeast side of the AY phase because of difficulties in making accurate projections with available data.

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.

Block geology and drillhole information

The FI Model is a computer based block model with block size 50' X 50' X 20' high. (15.25m X 15.25m X 6.1m high). The Mintec Medsystem release 10 software package was used to generate the model and derived reserves. The model has since been imported to PC Mine but all calculations have were done by Mintec's software.

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). In the remainder of Zone 3 (Sections 117 to 124) this new geological interpretation was not yet available thus the interpretation used was that developed in 1981. 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.

The drill hole database used includes all holes in the deposit to the time of model construction. In the northwest half of the deposit the geology has not been adjusted to reflect this information but the assays were used for interpolation.

3.2.4.1.2 Composite Calculation

Drill hole assays were composited on a 20' bench basis. Assays, were weighted by length and specific gravity. High assay values were rolled back to the 95th percentile level before compositing (See table 1). Internal waste (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 (10 feet) were not included in the composite intervals resulting in a shorter than 20 foot composite.

A major improvement over previous models was made in geological coding of the composites. Each composite was checked manually to ensure that it was coded consistently with the sectional geology rather than machine coded by detailed logged geology. Since large interpreted units often encompass several smaller intervals of different geology, this procedure insured that the composite would be used to interpolate only relevant units.

3.2.4.1.3 Interpolation

Interpolation search volume was 225' along strike, 150' along dip and 25' vertically. Composites were selected for interpolation on the basis of block geology being equivalent to composite geology coding. No composite less than 8 feet long was used in the interpolation to avoid biasing large blocks with small data points. One composite per drillhole was allowed to minimize vertical averaging across banding in the stratiform deposit. Where more than one composite was available 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 search volume was enlarged (to as much as 250' X 175' X 105' high) and geology matching restrictions were relaxed through 4 additional passes to interpolate blocks missed by the first pass.

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. Mining reserves are computed as outlined below for the F8608 model.

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.

3.2.4.2.1 Block Geology and Drillhole Data

The F8608 model is a 3D block model made using PC Mine software. Block size was 35 feet along the deposit, 25 feet across the deposit and 20 feet high. The coordinate grid was rotated 45° so that rows of blocks are parallel to the cross sections 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 141 foot spaced sections were simplified and intermediate cross sections created at 70.5 foot intervals (or 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 largely 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 geology so that bench height optimization could be studied. A drawback to this approach is that if 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 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.

Drillhole data used was that already imported into the PC Mine

(1983). In the remainder of Zone 3 (Sections 117 to 124) this new geological interpretation was not yet available thus the interpretation used was that developed in 1981. 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.

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A major improvement over previous models was made in geological coding of the composites. Each composite was checked manually to ensure that it was coded consistently with the sectional geology rather than machine coded by detailed logged geology. Since large interpreted units often encompass several smaller intervals of different geology, this procedure insured that the composite would be used to interpolate only relevant units.

3.2.4.1.3 Interpolation

Interpolation search volume was 225' along strike, 150' along dip and 25' vertically. Composites were selected for interpolation on the basis of block geology being equivalent to composite geology coding. No composite less than 8 feet long was used in the interpolation to avoid biasing large blocks with small data points. One composite per drillhole was allowed to minimize vertical averaging across banding in the stratiform deposit. Where more than one composite was available 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 search volume was enlarged (to as much as 250' X 175' X 105' high) and geology matching restrictions were relaxed through 4 additional passes to interpolate blocks missed by the first pass.

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. Mining reserves are computed as outlined below for the FB608 model.

3.2.4.2 Methods and Procedure - FB608 Model

In August and September 1986 a new computer model of the AY and BY phases, the FB608 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.

3.2.4.2.1 Block Geology and Drillhole Data

The FB608 model is a 3D block model made using PC Mine software. Block size was 35 feet along the deposit, 25 feet across the deposit and 20 feet high. The coordinate grid was rotated 45° so that rows of blocks are parallel to the cross sections 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 141 foot spaced sections were simplified and intermediate cross sections created at 70.5 foot intervals (or 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 largely 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 geology so that bench height optimization could be studied. A drawback to this approach is that if 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 FB608 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 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.

Drillhole data used was that already imported into the PC Mine

for the FI model.

Being a sectional model the FB608 model is similar to those created for Grum and Vangorda with the exception of computational details outlined below.

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 successful. 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 search ellipsoid looking 225' along the deposit trend 150' across it and 25' vertically. This was enlarged to 300' along the deposit, 200' across and 37' vertically in 3 passes (comparable multi-pass interpolation was not available when the Grum and Vangorda models were made).

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.

previous models. This is likely due to more restrictive application of geology matching during interpolation due to the greater availability of composites for 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. The FI model reports 8.5% higher tonnes 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%.

Table 3.2B compares the expanded JB phase (again not exactly the same as the current JB phase) as calculated by the FI model and with 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 computations compare to manual computations. The comparison is good; the major difference being in tonnage which is probably at least in part due to the inability of 50' X 50' blocks to show every geologic unit.

The results from the FI and F8608 models the AY and BY Phases are compared in table 3.1. The F8608 composited 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 $\pm 30\%$. As might be expected the larger benches in the core of the deposit compare well.

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 achieve 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 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.

The actual comparison of model results to JB phase mining is detailed in tables 3.5A through 3.6G. These tables give the unadjusted reserves of: the FI model (3.5A), a recalculation of that model using F8608 methodology (3.5B) and the blasthole results (3.6A).

The variance and present variance to the blast hole results is also given (table 3.6B) as well as the variance of the two model calculations (table 3.5C). Two comparisons of diluted reserves to blastholes are also provided (table 3.6D-G).

In the upper benches which model fault bounded blocks of ore in

the Big Indian fault system and are significantly affected by the edges of the deposit the two model calculations compare poorly (Table 3.5C). The new (F8608 hybrid) calculation in general reports a lower grade with large erratic variances in tonnage above cutoff (probably caused by the poor grade comparison). In the lower benches the comparison is much better. These benches model the middle of the deposit away from any known faults. Overall the comparison is within 10% on total metal with the new calculation showing lower overall grade and tonnage.

The raw model results do not compare well to blasthole results (table 3.6B). The upper benches show major positive variances in tonnage and negative variances in grade suggestive of high dilution. Lower benches compare better. There is an unexplained large negative in zinc than in lead on most benches.

The model results diluted at 10% by zero grade (table 3.6D & 3.6F) compare better but the F8608 hybrid clearly requires a larger dilution (table 3.6D). Table 3.6E shows the comparison of the F8608 hybrid at 30% dilution by material averaging 3.5% Pb+Zn, an amount not unrealistic from the JB zone geometry at the deposit margin. The comparison is better with some notable exceptions. The large variance on 3850 bench was due to a drill hole being plotted at least 50' long too far southeast so that a fault bounded panel of massive sulphides appeared to be twice as wide as it actually was. The last two benches do not require as large a dilution as the higher benches; this is probably due to the simpler geometry and thicker ore zones.

There is no dilution that can make the FI model grades compare well to the blasthole grades (tables 3.6F & G). This is thought to be due to the length weighting of composites which overemphasized the higher grade core of the deposit with respect to the lower grade margins. This weighting was not done during the F8608 hybrid calculation.

The conclusion of these comparisons is that the FI model compared 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 comparisons without an appropriate dilution factor. The grade predictions of the FI model are too high probably as a result of inappropriate methodology. Most importantly, dilution is not average through 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.

Table 3.1

<u>6% Cutoff</u>		FI (New)	T3 (Old)	$\frac{FI - T3}{T3} \%$
JB	MT	169,221	360,935	-53
	Pb	4.04	3.74	+8
	Zn	6.55	5.66	+16
	Ag	52.9	50.6	+5
A	MT	3,595,315	3,668,154	-2
	Pb	3.83	3.63	+6
	Zn	5.62	5.37	+5
	Ag	47.5	41.9	+13
B	MT	4,457,843	5,129,659	-13
	Pb	3.78	3.69	+2
	Zn	5.32	5.12	+4
	Ag	47.9	44.9	+7
C	MT	3,681,852	4,127,797	-11
	Pb	3.60	3.56	+1
	Zn	5.33	4.98	+7
	Ag	47.6	44.0	+8
D	MT	3,736,990	3,393,517	-4
	Pb	3.35	3.18	+5
	Zn	5.53	5.26	+5
	Ag	39.9	35.8	+11
TOTAL				
	MT	15,641,221	17,180,062	-9
	Pb	3.65	3.53	+3
	Zn	5.45	5.18	+5
	Ag	45.9	42.1	+9
		1,423,351	1,496,384	-4.9%

December 5, 1985

Table 3.2 A
Phase A

	Tonnes	Pb %	Zn %	AG g/tonne	Pb&Zn %
F1	3,595,315	3.83	5.62	47.5	9.45
T3	4,043,405	3.63	5.37	41.9	9.00
F3	4,254,974	3.6	5.4	43.1	9.00
HAND	3,290,317	3.95	5.98	48.7	9.93
	-8.5%	+3.1%	+6.4%	+2.5%	-3.8% total metal

Table 3.2 B
Phase JBX (not JB as on Table I)

FI	816,017	3.81	6.19	49.9	10.00
HAND CALC FROM F4 BENCH GEOLOGY	858,887	3.8	6.1	52.4	9.9

Table 3.3A

CARRAGH RESOURCES

Comparison of F1 and FB608 models - FOR AY PHASE ONLY

Gregg Jilson
September 24, 1986

NO ADJUSTMENTS FOR DILUTION OR MINING LOSS

*****NEW MODEL (FB608)*****											*****OLD MODEL (F1)*****										
BENCH NUMBER	TOE ELEV.	TONNES +62 ORE	Pb (%)	Zn (%)	Ag (g/t)	Pb + Zn (%)	LEAD METAL (tonnes)	ZINC METAL (tonnes)	SILVER METAL (grams)	TOTAL METAL (tonnes)	BENCH NUMBER	TOE ELEV.	TONNES +62 ORE	Pb (%)	Zn (%)	Ag (g/t)	Pb + Zn (%)	LEAD METAL (tonnes)	ZINC METAL (tonnes)	SILVER METAL (grams)	TOTAL METAL (tonnes)
17	3750	3,040	3.97	5.87	79.04	9.84	152	225	303521	370	17	3750	3,110	4.50	5.94	105.00	12.44	202	185	326550	387
18	3730	33,320	4.15	5.13	71.75	9.28	1382	1710	2390743	3092	18	3730	10,680	3.86	4.73	59.64	8.58	412	505	636923	917
19	3710	49,290	3.81	5.00	54.99	8.81	1876	2464	2710654	4340	19	3710	62,910	3.14	4.61	46.73	7.74	1972	2900	2939784	4872
20	3690	77,620	3.42	5.66	45.75	9.07	2651	4390	3351115	7041	20	3690	33,460	3.31	5.11	45.41	8.42	1770	2731	2427565	4501
21	3670	51,530	3.20	5.15	38.07	8.35	1648	2653	1961953	4302	21	3670	43,050	3.98	5.79	48.25	9.77	1714	2493	2077033	4207
22	3650	43,590	3.14	4.16	37.53	7.30	1367	1813	1636020	3180	22	3650	47,850	4.22	5.02	48.87	9.24	2021	2400	2338573	4421
23	3630	76,600	3.16	4.26	37.33	7.42	2420	3266	2862771	5686	23	3630	71,320	4.18	5.49	49.99	9.66	2980	3912	3565501	6892
24	3610	141,460	3.13	4.88	32.90	8.01	4426	6909	4654458	11335	24	3610	150,670	3.24	5.68	30.47	8.92	4885	8557	4591518	13441
25	3590	121,260	2.91	4.74	31.34	7.65	3524	5749	3799682	9273	25	3590	84,600	2.89	5.35	27.01	8.24	2441	4528	2284792	6969
26	3570	79,820	2.91	4.84	36.36	7.74	2322	3859	2902574	6181	26	3570	81,450	3.71	5.79	48.87	9.50	3019	4719	3980706	7738
27	3550	205,690	3.25	5.09	44.78	8.34	6681	10466	9211004	17146	27	3550	260,620	3.70	5.59	53.89	9.29	9640	14574	14043509	24214
28	3530	365,850	3.61	5.27	47.45	8.88	13211	19291	17358485	32502	28	3530	417,040	3.89	5.72	48.30	9.61	16240	23855	20142198	40094
29	3510	499,220	3.56	5.27	44.84	8.83	17762	26299	22386522	44061	29	3510	566,280	3.98	5.77	47.88	9.75	22555	32674	27111788	55229
30	3490	449,190	3.87	5.41	49.20	9.28	18158	25349	23082271	43527	30	3490	426,680	4.27	6.00	52.05	10.27	18211	25609	22209121	43820
31	3470	361,350	3.76	5.28	45.78	9.03	13572	19068	16541158	32641	31	3470	375,590	3.85	5.29	44.89	9.14	14441	19854	16852008	34296
32	3450	316,890	3.17	4.67	43.67	7.85	9982	14699	13734722	24681	32	3450	295,120	3.58	4.91	48.57	8.29	9975	14490	14334569	24465
33	3430	253,690	3.72	5.72	45.15	9.43	9427	14498	11454357	23926	33	3430	212,300	3.69	5.75	45.71	9.44	7828	12207	9703808	20035
34	3410	42,308	2.79	4.27	25.95	7.06	1181	1806	1097474	2987	34	3410	32,740	2.79	4.35	24.31	7.14	912	1425	795942	2337
35	3390	16,880	2.97	4.14	23.57	7.11	501	699	397929	1200	35	3390	17,630	2.85	4.23	22.71	7.08	503	745	400789	1247
TOTALS/AVERAGES:		3,207,070	3.50	5.15	44.29	8.65	112244	165235	142037415	277479			3,212,900	3.79	5.55	46.92	9.34	121,719	178,363	150,762,176	300,082

Table 3.3B

-----PERCENT VARIANCES ((NEW-OLD)/OLD)-----

TONNES +61 ORE	Pb (%)	Zn (%)	Ag (%)	Pb + Zn (%)	LEAD METAL (%)	ZINC METAL (%)	SILVER METAL (%)	TOTAL BENCH METAL NUMBER (%)	TOE ELEV.
23.52	-38.91	-1.21	-24.71	-20.91	-24.61	22.01	-7.11	-2.31	17 3750
212.01	7.61	8.51	20.31	8.11	235.01	238.61	275.41	237.41	18 3730
-21.61	21.41	8.51	17.71	13.71	-4.91	-15.01	-7.01	-10.91	19 3710
45.22	3.11	10.71	0.81	7.71	49.81	60.71	46.31	56.41	20 3690
19.71	-19.71	-11.11	-21.11	-14.61	-3.81	6.41	-5.51	2.21	21 3670
-6.91	-25.81	-17.01	-23.21	-21.01	-32.41	-24.41	-30.01	-28.11	22 3650
7.51	-24.51	-22.41	-25.31	-23.31	-18.81	-16.51	-19.71	-17.51	23 3630
-6.11	-3.51	-14.01	8.01	-10.21	-9.41	-19.31	1.41	-15.71	24 3610
43.31	0.71	-11.41	16.01	-7.71	44.41	27.01	66.31	33.11	25 3590
-2.01	-21.51	-16.61	-25.61	-18.51	-23.11	-10.21	-27.11	-20.11	26 3570
-21.11	-12.71	-9.01	-16.91	-10.31	-30.71	-20.71	-34.41	-29.71	27 3550
-12.31	-7.31	-7.81	-1.81	-7.61	-18.71	-19.11	-13.81	-18.91	28 3530
-11.81	-10.71	-8.71	-6.31	-9.51	-21.71	-19.51	-17.41	-20.21	29 3510
10.01	-9.31	-9.91	-5.51	-9.71	-0.31	-0.91	3.91	-0.71	30 3490
-3.71	-2.41	-0.71	2.01	-1.11	-6.01	-4.01	-1.81	-4.81	31 3470
6.61	-6.11	-4.81	-10.11	-5.31	0.11	1.41	-4.21	0.91	32 3450
19.51	0.81	-0.61	-1.71	-0.11	20.41	18.81	18.01	19.41	33 3430
29.21	0.71	-1.91	6.71	-1.11	29.51	26.81	37.91	27.81	34 3410
-4.31	4.01	-2.01	3.81	0.41	-0.41	-6.21	-0.61	-3.81	35 3390
-0.71	-7.61	-7.21	-5.61	-7.41	-7.81	-7.41	-5.81	-7.51	AVERAGE VARIANCE

-----ACTUAL VARIANCES ((NEW-OLD)/OLD)-----

TONNES +61 ORE	Pb (%)	Zn (%)	Ag (%)	Pb + Zn (%)	LEAD METAL (tonnes)	ZINC METAL (tonnes)	SILVER METAL (ounces)	TOTAL METAL (tonnes)
730	-2.53	-0.07	-25.96	-2.60	-50	41	-672	-9
22,640	0.29	0.41	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	1639	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	-20513	-1241
5,360	-1.02	-1.23	-12.66	-2.25	-560	-646	-20520	-1206
(9,210)	-0.11	-0.79	2.43	-0.91	-458	-1648	1838	-2106
36,660	0.02	-0.61	4.33	-0.59	1083	1221	44233	2304
(11,630)	-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	-0.95	-0.73	-3029	-4563	-81284	-7592
(67,060)	-0.43	-0.50	-3.03	-0.93	-4793	-6375	-137976	-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
(1750)	0.12	-0.08	0.87	0.03	-2	-46	-69	-48
15,830	-0.29	-0.40	-2.64	-0.69	-9475	-13127	-254763	-22603

Table 3.4A

CUSPASH RESOURCES

Comparison of FI and FB608 models - FOR R1 PHASE ONLY

Gregg Jilson

September 30, 1986

NO ADJUSTMENTS FOR DILUTION OR MINING LOSS

*****NEW MODEL (FB608)*****											*****OLD MODEL (FI)*****											
BENCH NUMBER	TDE ELEV.	TONNES +62 ORE	Pb (%)	Zn (%)	Ag (g/t)	Pb + Zn (%)	LEAD METAL (tonnes)	ZINC METAL (tonnes)	SILVER METAL (grams)	TOTAL METAL (tonnes)		BENCH NUMBER	TDE ELEV.	TONNES +62 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		20	3690	0	0.00	0.00	0.00	0.00	0	0	0	0
21	3670	11,930	3.14	7.75	27.81	10.89	374	925	331797	1299		21	3670	0	0.00	0.00	0.00	0.00	0	0	0	0
22	3650	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		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		24	3610	0	0.00	0.00	0.00	0.00	0	0	0	0
25	3590	51,360	3.79	5.76	51.14	9.35	1948	2958	2626756	4906		25	3590	1,420	3.59	4.44	50.00	8.03	51	63	71000	114
26	3570	120,300	3.74	4.83	54.03	8.57	4499	5809	6499328	10309		26	3570	105,810	4.32	5.41	54.45	9.73	4574	5725	5761566	10300
27	3550	176,490	3.93	4.76	56.83	8.69	6931	8403	10029927	15333		27	3550	181,470	4.31	4.73	63.08	9.04	7816	8584	11446220	16399
28	3530	252,270	3.52	4.62	54.13	8.14	8877	11660	13655880	20537		28	3530	267,200	4.40	5.18	69.03	9.58	11757	13833	18444549	25590
29	3510	319,950	4.26	5.60	63.46	9.86	13636	17908	20304987	31544		29	3510	272,490	4.18	5.47	66.52	9.65	11390	14905	19126307	26295
30	3490	471,560	4.03	5.55	55.40	9.59	19018	26181	26125839	45199		30	3490	471,420	4.32	5.77	61.81	10.09	20342	27220	25137999	47562
31	3470	745,720	4.24	5.83	56.23	10.07	31633	43461	41929598	75094		31	3470	751,690	4.13	5.70	56.51	9.83	31045	42846	42474243	73891
32	3450	560,290	4.06	5.54	49.38	9.59	22737	31012	27666000	53749		32	3450	489,640	4.13	5.71	55.03	9.84	20207	27973	26945379	48181
33	3430	529,670	3.61	5.22	40.87	8.83	19116	27638	21648143	46754		33	3430	503,770	3.68	5.34	39.35	9.02	18534	26896	19824861	45430
34	3410	679,850	3.13	4.84	31.59	7.97	21273	32884	21476462	54157		34	3410	706,020	3.30	5.06	34.35	8.36	23299	35746	24252493	59044
35	3390	428,320	3.02	4.65	29.36	7.68	12948	19934	12576332	32882		35	3390	450,190	3.07	4.38	36.45	7.45	13816	19736	16408075	33553
36	3370	305,530	2.96	4.89	26.96	7.85	9053	14943	8236172	23996		36	3370	243,000	2.96	4.79	31.70	7.76	7203	11642	7702128	18845
37	3350	109,330	2.78	5.26	29.86	8.03	3640	5761	3270725	8801		37	3350	148,630	2.88	5.28	34.04	8.17	4286	7849	5059811	12136
38	3330	14,410	2.62	5.20	43.80	7.82	377	750	631086	1127		38	3330	20,830	2.61	4.92	39.32	7.52	751	1417	1133624	2188
TOTALS/AVERAGES:		4,703,620	3.67	5.24	45.42	8.91	175668	250596	217252222	426264				4,623,940	3.79	5.29	49.00	9.08	175,217	244,620	226,932,216	419,837

TABLE 3.4B

-----PERCENT VARIANCES ((NEW-OLD)/OLD)-----

TONNES +62 ORE	Pb (%)	Zn (%)	Ag (%)	Pb + Zn (%)	LEAD METAL (%)	ZINC METAL (%)	SILVER METAL (%)	TOTAL BENCH METAL NUMBER (%)	ICE ELEV.
ERR	ERR	ERR	ERR	ERR	ERR	ERR	ERR	ERR	20 3690
ERR	ERR	ERR	ERR	ERR	ERR	ERR	ERR	ERR	21 3670
-100.01	-100.01	-100.01	-100.01	-100.01	-100.01	-100.01	-100.01	-100.01	22 3650
ERR	ERR	ERR	ERR	ERR	ERR	ERR	ERR	ERR	23 3630
ERR	ERR	ERR	ERR	ERR	ERR	ERR	ERR	ERR	24 3610
3516.91	5.61	29.71	2.31	19.01	3720.41	4592.21	3599.71	4202.41	25 3590
13.71	-13.51	-10.81	-0.81	-12.01	-1.61	1.51	12.81	0.11	26 3570
-2.71	-8.81	0.71	-9.91	-3.91	-11.31	-2.11	-12.41	-6.51	27 3550
-5.61	-20.01	-10.71	-21.61	-15.01	-24.51	-15.71	-26.01	-19.71	28 3530
17.41	2.01	2.31	-4.61	2.21	19.71	20.11	12.01	20.01	29 3510
0.01	-6.51	-3.81	-10.41	-5.01	-6.51	-3.81	-10.31	-5.01	30 3490
-0.81	2.71	2.21	-0.51	2.41	1.91	1.41	-1.31	1.61	31 3470
14.41	-1.71	-3.11	-10.31	-2.51	12.51	10.91	2.71	11.61	32 3450
5.11	-1.91	-2.31	3.91	-2.11	3.11	2.81	9.21	2.91	33 3430
-3.71	-5.21	-4.51	-8.01	-4.71	-8.71	-8.01	-11.41	-8.31	34 3410
-4.91	-1.51	6.21	-19.41	3.01	-6.31	1.01	-23.41	-2.01	35 3390
25.71	-0.81	2.11	-15.01	1.31	25.71	28.41	6.91	27.31	36 3370
-26.31	-3.81	-0.41	-12.31	-1.61	-29.11	-26.61	-35.41	-27.51	37 3350
-50.01	0.41	5.91	11.41	4.01	-49.81	-47.11	-44.31	-48.01	38 3330
3.51	-3.11	-1.01	-7.51	-1.91	0.31	2.41	-4.31	1.51	AVERAGE VARIANCE

-----ACTUAL VARIANCES ((NEW-OLD)/OLD)-----

TONNES +62 ORE	Pb (%)	Zn (%)	Ag (%)	Pb + Zn (%)	LEAD METAL (tonnes)	ZINC METAL (tonnes)	SILVER METAL (grams) oz	TOTAL BENCH METAL NUMBER (tonnes)	ICE 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
(12,360)	-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
(14,980)	-0.38	0.03	-6.25	-0.35	-885	-181	-41336	-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
(15,970)	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	-57241	-3334	37 3350
(14,420)	0.01	0.29	4.47	0.30	-374	-667	-14674	-1041	38 3330
159,680	-0.12	-0.05	-3.66	-0.17	452	5975	-282656	6427	AVERAGE VARIANCE

Table 3.5 A

CURRAGH RESOURCES
FARO DEPOSIT - JB PHASE
COMPARISON OF FI AND F8608 MODELING METHODS IN THE JB PHASE

Gregg Jilson
September 26, 1986

All values are unadjusted for dilution or mining loss in first set of results.
Values are adjusted for dilution as stated in second set, no mining loss taken.

"F8608" hybrid model (the new calculation) is based on interpolation of grade into the FI model blocks using the same geology and composite database as the FI model (the old calculation) but techniques as close as possible to those used in the AY phase for the actual F8608 model which uses smaller blocks and revised geology.

RESULTS OF NEW CALCULATION WITH PCMINE, NO LENGTH WEIGHTING OF COMPOSITES

BENCH NUMBER	TOE ELEV.	TONNES ORE **	Pb (%)	Zn (%)	Ag (g/t)	Au (g/t)	Pb + Zn (%)	LEAD METAL (tonnes)	ZINC METAL (tonnes)	SILVER METAL (grams)	GOLD METAL (grams)	TOTAL METAL (tonnes)
10	3890	32,700	3.57	5.11	51.4	0.06	8.68	1168	1671	1679864	2027	2839
11	3870	5,500	3.63	5.75	47.0	0.09	9.38	199	316	258500	479	516
12	3850	50,850	3.80	5.99	49.3	0.09	9.79	1930	3047	2508125	4526	4977
13	3830	61,150	3.34	5.39	47.4	0.12	8.73	2042	3294	2901017	7521	5335
14	3810	71,470	3.58	6.18	47.3	0.30	9.76	2555	4418	3383676	21084	6973
15	3790	116,960	3.90	6.60	50.1	0.27	10.50	4561	7719	5859696	31579	12281
16	3770	159,650	3.17	5.60	43.0	0.28	8.77	5061	8940	6864950	44702	14001
17	3750	209,740	3.25	4.98	42.2	0.24	8.23	6817	10445	8851028	50338	17262
18	3730	246,450	2.99	4.66	39.4	0.19	7.65	7369	11485	9710130	46826	18853
19	3710	231,110	2.86	4.52	37.6	0.15	7.38	6610	10446	8689736	34667	17056
20	3690	120,530	2.42	4.15	33.6	0.15	6.57	2917	5002	4049808	18080	7919
TOTALS/AVERAGES		1,306,110	3.16	5.11	41.9	0.20	8.27	41229	66784	54756531	261827	108012
3890 to 3770		498,280	3.52	5.90	47.1	0.22	9.42	17517	29406	23455829	111918	46923

** = ORE ABOVE 3790 IS +6% ON 3790 IS MIXED +5 AND +6 AND ON 3750 IS +5%

Table 3.5B

RESULTS OF OLD CALCULATION WITH MINTEC, LENGTH WEIGHTING OF COMPOSITES

BENCH NUMBER	TOE ELEV.	TONNES +6% ORE	Pb (%)	Zn (%)	Ag (g/t)	Au (g/t)	Pb + Zn (%)	LEAD METAL (tonnes)	ZINC METAL (tonnes)	SILVER METAL (grams)	GOLD METAL (grams)	TOTAL METAL (tonnes)
10	3890	28,560	3.71	5.01	54.2	0.07	8.73	1060	1432	1547409	1942	2492
11	3870	10,050	3.18	4.78	46.0	0.06	7.96	319	481	462300	593	800
12	3850	45,500	4.34	7.65	52.1	0.06	11.98	1972	3478	2368366	2821	5451
13	3830	76,000	3.98	6.56	51.9	0.18	10.54	3026	4982	3943564	13528	8007
14	3810	83,390	3.93	6.33	52.2	0.21	10.25	3275	5275	4349122	17345	8550
15	3790	115,150	4.38	7.29	54.5	0.22	11.67	5044	8394	6275675	25333	13438
16	3770	184,900	3.42	5.61	49.0	0.21	9.03	6324	10373	9060100	38829	16696
17	3750	222,400	3.15	4.98	44.5	0.26	8.13	7006	11076	9896800	57824	18081
18	3730	233,320	3.05	5.07	37.3	0.16	8.12	7116	11829	8702836	37331	18946
19	3710	228,710	2.71	4.57	34.6	0.12	7.28	6198	10452	7913366	27445	16650
20	3690	135,410	2.45	4.55	33.7	0.13	7.00	3318	6161	4563317	17603	9479
TOTALS/AVERAGES		1,363,390	3.28	5.42	43.3	0.18	8.70	44656	73934	59082855	240595	118590
3890 to 3770		543,550	3.87	6.33	51.5	0.18	10.20	21019	34416	28006536	100391	55434

Table 3.6 A

BLASTHOLE RESULTS (FROM MINE GEOLOGY DEPARTMENT PRINTOUT)

BENCH NUMBER	TOE ELEV.	TONNES +6% ORE	Pb (%)	Zn (%)	Ag (g/t)	Au (g/t)	Pb + Zn (%)	LEAD METAL (tonnes)	ZINC METAL (tonnes)	SILVER METAL (grams)	GOLD METAL (grams)	TOTAL METAL (tonnes)
10	3890	44,180	3.10	4.70	44.0		7.80	1370	2076	1943920	0	3446
11	3870	20,378	3.30	5.30			8.60	672	1080	0	0	1753
12	3850	29,549	2.41	3.83			6.24	712	1132	0	0	1844
13	3830	81,920	2.79	4.51			7.30	2286	3695	0	0	5980
14	3810	99,840	3.19	4.41			7.60	3185	4403	0	0	7588
15	3790	184,320	3.44	4.94			8.38	6341	9105	0	0	15446
16	3770	217,600	2.73	4.43	35.0		7.16	5940	9640	7616000	0	15580
17	3750	208,249	3.11	4.60	43.0		7.71	6470	9579	8954707	0	16050
18	3730	261,111	2.99	4.67	42.0		7.66	7807	12194	10966662	0	20001
19	3710						0.00	0	0	0	0	0
20	3690						0.00	0	0	0	0	0
TOTALS/AVERAGES		1,147,147	3.03	4.61	25.7	0.00	7.64	34783	52904	29481289	0	87687
3890 to 3770		677,787	3.03	4.59	14.1	0.00	7.62	20506	31131	9559920	0	51637

TABLE 3.5C

PERCENT VARIANCE FI MODEL TO NEW CALCULATION ((NEW-FI)/FI)

BENCH NUMBER	TOE ELEV.	TONNES +6% ORE	Pb (%)	Zn (%)	Ag (g/t)	Au (g/t)	Pb + Zn (%)	LEAD METAL (tonnes)	ZINC METAL (tonnes)	SILVER METAL (grams)	GOLD METAL (grams)	TOTAL METAL (tonnes)
10	3890	14.5%	-3.8%	1.9%	-5.2%	-8.8%	-0.5%	10.2%	16.7%	8.6%	4.4%	13.9%
11	3870	-45.3%	14.2%	20.2%	2.2%	47.5%	17.8%	-37.5%	-34.2%	-44.1%	-19.3%	-35.5%
12	3850	11.8%	-12.4%	-21.6%	-5.2%	43.5%	-18.3%	-2.1%	-12.4%	5.9%	60.4%	-8.7%
13	3830	-19.5%	-16.1%	-17.8%	-8.6%	-30.9%	-17.2%	-32.5%	-33.9%	-26.4%	-44.4%	-33.4%
14	3810	-14.3%	-9.0%	-2.3%	-9.2%	41.8%	-4.8%	-22.0%	-16.2%	-22.2%	21.6%	-18.4%
15	3790	1.6%	-11.0%	-9.5%	-8.1%	22.7%	-10.0%	-9.6%	-8.0%	-6.6%	24.7%	-8.6%
16	3770	-13.7%	-7.3%	-0.2%	-12.2%	33.3%	-2.9%	-20.0%	-13.8%	-24.2%	15.1%	-16.1%
17	3750	-5.7%	3.2%	0.0%	-5.2%	-7.7%	1.2%	-2.7%	-5.7%	-10.6%	-12.9%	-4.5%
18	3730	5.6%	-2.0%	-8.1%	5.6%	18.8%	-5.8%	3.5%	-2.9%	11.6%	25.4%	-0.5%
19	3710	1.0%	5.5%	-1.1%	8.7%	25.0%	1.4%	6.6%	-0.1%	9.8%	26.3%	2.4%
20	3690	-11.0%	-1.2%	-8.8%	-0.3%	15.4%	-6.1%	-12.1%	-18.8%	-11.3%	2.7%	-16.5%
		-4.2%	-3.6%	-5.7%	-3.3%	13.6%	-4.9%	-7.7%	-9.7%	-7.3%	8.8%	-8.9%

ACTUAL VARIANCE FI MODEL TO NEW CALCULATION (NEW-FI)

BENCH NUMBER	TOE ELEV.	TONNES ORE	Pb (%)	Zn (%)	Ag (g/t)	Au (g/t)	Pb + Zn (%)	LEAD METAL (tonnes)	ZINC METAL (tonnes)	SILVER METAL (grams)	GOLD METAL (grams)	TOTAL METAL (tonnes)
10	3890	4,140	-0.14	0.10	-2.8	-0.01	-0.04	108	239	132455	85	347
11	3870	(4,550)	0.45	0.97	1.0	0.03	1.42	-120	-164	-203800	-114	-284
12	3850	5,350	-0.54	-1.65	-2.7	0.03	-2.19	-42	-432	139759	1705	-474
13	3830	(14,850)	-0.64	-1.17	-4.4	-0.06	-1.81	-984	-1688	-1042547	-6007	-2672
14	3810	(11,920)	-0.35	-0.14	-4.8	0.09	-0.50	-720	-857	-965446	3739	-1577
15	3790	1,810	-0.48	-0.69	-4.4	0.05	-1.17	-482	-675	-415979	6246	-1157
16	3770	(25,250)	-0.25	-0.01	-6.0	0.07	-0.26	-1263	-1432	-2195150	5873	-2695
17	3750	(12,660)	0.10	0.00	-2.3	-0.02	0.10	-189	-630	-1045772	-7486	-820
18	3730	13,130	-0.06	-0.41	2.1	0.03	-0.47	253	-345	1007294	9494	-92
19	3710	2,400	0.15	-0.05	3.0	0.03	0.10	412	-6	776370	7221	406
20	3690	(14,880)	-0.03	-0.40	-0.1	0.02	-0.43	-401	-1159	-513509	476	-1560
		(57,280)	-0.12	-0.31	-1.4	0.02	-0.43	-3428	-7150	-4326325	21232	-10578

Table 3.6B

PERCENT VARIANCE BLASTHOLES TO NEW CALCULATION ((BH-NEW)/BH)

BENCH NUMBER	TDE ELEV.	TONNES +6% ORE	Pb (%)	Zn (%)	Ag (g/t)	Au (g/t)	Pb + Zn (%)	LEAD METAL (tonnes)	ZINC METAL (tonnes)	SILVER METAL (grams)	GOLD METAL (grams)	TOTAL METAL (tonnes)
10	3890	26.0%	-15.2%	-8.7%	-16.8%		-11.3%	14.7%	19.5%	13.6%		17.6%
11	3870	73.0%	-9.9%	-8.5%			-9.0%	70.3%	70.7%			70.6%
12	3850	-72.1%	-57.5%	-56.4%			-56.9%	-171.1%	-169.2%			-169.9%
13	3830	25.4%	-19.7%	-19.4%			-19.5%	10.7%	10.9%			10.8%
14	3810	28.4%	-12.1%	-40.2%			-28.4%	19.8%	-0.3%			8.1%
15	3790	36.5%	-13.4%	-33.6%			-25.3%	28.1%	15.2%			20.5%
16	3770	26.6%	-16.1%	-26.4%	-22.9%		-22.5%	14.8%	7.3%	9.9%		10.1%
17	3750	-0.7%	-4.6%	-8.3%	1.9%		-6.8%	-5.4%	-9.0%	1.2%		-7.6%
18	3730	5.6%	0.0%	0.2%	6.2%		0.1%	5.6%	5.8%	11.5%		5.7%
19	3710	ERR	ERR	ERR	ERR		ERR	ERR	ERR	ERR		ERR
20	3690	ERR	ERR	ERR	ERR		ERR	ERR	ERR	ERR		ERR
OVERALL		-13.9%	-4.1%	-10.9%	-63.1%	ERR	-8.2%	-18.5%	-26.2%	-85.7%	ERR	-23.2%
3890 to 3770		26.5%	-16.2%	-28.5%	-233.7%	ERR	-23.6%	14.6%	5.5%	-145.4%	ERR	9.1%

ACTUAL VARIANCE BLASTHOLES TO NEW CALCULATION (BH-NEW)

BENCH NUMBER	TDE ELEV.	TONNES ORE	Pb (%)	Zn (%)	Ag (g/t)	Au (g/t)	Pb + Zn (%)	LEAD METAL (tonnes)	ZINC METAL (tonnes)	SILVER METAL (Kgrams)	GOLD METAL (grams)	TOTAL METAL (tonnes)
10	3890	11,480	-0.47	-0.41	-7.4		-0.88	202	405	264		607
11	3870	14,878	-0.33	-0.45			-0.78	473	764	(259)		1,237
12	3850	(21,301)	-1.39	-2.16			-3.55	(1,218)	(1,915)			(3,133)
13	3830	20,770	-0.55	-0.88			-1.43	244	401			645
14	3810	28,370	-0.39	-1.77			-2.16	630	(15)			615
15	3790	67,360	-0.46	-1.66			-2.12	1,779	1,386			3,165
16	3770	57,950	-0.44	-1.17	-8.0		-1.61	880	699	751		1,579
17	3750											
18	3730											
19	3710											
20	3690											
		*****	-0.12	-0.50	-16.2	-0.20	-0.63	(6,446)	(13,880)	(25,275)	(261,827)	(20,325)

Table 36C

PERCENT VARIANCE BLASTHOLES TO OLD CALCULATION ((BH-OLD)/BH)

BENCH NUMBER	TOE ELEV.	TONNES +6% ORE	Pb (%)	Zn (%)	Ag (g/t)	Au (g/t)	Pb + Zn (%)	LEAD METAL (tonnes)	ZINC METAL (tonnes)	SILVER METAL (grams)	GOLD METAL (grams)	TOTAL METAL (tonnes)
10	3890	35.4%	-19.7%	-6.7%	-23.1%	ERR	-11.9%	22.6%	31.0%	20.4%	ERR	27.7%
11	3870	50.7%	3.8%	9.8%	ERR	ERR	7.5%	52.5%	55.5%	ERR	ERR	54.4%
12	3850	-54.0%	-79.9%	-99.6%	ERR	ERR	-92.0%	-177.0%	-207.4%	ERR	ERR	-195.6%
13	3830	7.2%	-42.7%	-45.3%	ERR	ERR	-44.3%	-32.4%	-34.8%	ERR	ERR	-33.9%
14	3810	16.5%	-23.1%	-43.4%	ERR	ERR	-34.9%	-2.8%	-19.8%	ERR	ERR	-12.7%
15	3790	37.5%	-27.3%	-47.6%	ERR	ERR	-39.3%	20.5%	7.8%	ERR	ERR	13.0%
16	3770	15.0%	-25.3%	-26.6%	-40.0%	ERR	-26.1%	-6.4%	-7.6%	-19.0%	ERR	-7.2%
17	3750	-6.8%	-1.4%	-8.3%	-3.5%	ERR	-5.5%	-8.3%	-15.6%	-10.5%	ERR	-12.7%
18	3730	10.6%	-2.0%	-8.6%	11.2%	ERR	-6.0%	8.9%	3.0%	20.6%	ERR	5.3%
19	3710	ERR	ERR	ERR	ERR	ERR	ERR	ERR	ERR	ERR	ERR	ERR
20	3690	ERR	ERR	ERR	ERR	ERR	ERR	ERR	ERR	ERR	ERR	ERR
OVERALL		-18.9%	-8.0%	-17.6%	-68.6%	ERR	-13.8%	-28.4%	-39.8%	-100.4%	ERR	-35.2%
3890 to 3770		19.8%	-27.8%	-37.9%	-265.3%	ERR	-33.9%	-2.5%	-10.6%	-193.0%	ERR	-7.4%

ACTUAL VARIANCE BLASTHOLES TO OLD CALCULATION (BH-OLD)

BENCH NUMBER	TOE ELEV.	TONNES ORE	Pb (%)	Zn (%)	Ag (g/t)	Au (g/t)	Pb + Zn (%)	LEAD METAL (tonnes)	ZINC METAL (tonnes)	SILVER METAL (Kgrams)	GOLD METAL (grams)	TOTAL METAL (tonnes)
10	3890	15,620	-0.61	-0.31	-10.2		-0.92	310	644	397		954
11	3870	10,328	0.12	0.52			0.64	353	599			953
12	3850	(15,951)	-1.93	-3.82			-5.74	(1,260)	(2,347)			(3,607)
13	3830	5,920	-1.19	-2.05			-3.24	(740)	(1,287)			(2,027)
14	3810	16,450	-0.74	-1.92			-2.65	(90)	(872)			(962)
15	3790	69,170	-0.94	-2.35			-3.29	1,297	711			2,008
16	3770	32,700	-0.69	-1.18	-14.0		-1.87	(383)	(733)	(1,444)		(1,116)
17	3750											
18	3730											
19	3710											
20	3690											
		(216,243)	-0.24	-0.81	-17.6	-0.18	-1.05	(9,873)	(21,029)	(29,602)	(240,595)	(30,903)

Table 3.6 D

Amount of dilution (in %): 10
 Grade of dilutant (%): 0
 Pb/Pb+Zn of dilutant: 0
 Ag grade of dilutant: 0

PERCENT VARIANCE BLASTHOLES TO NEW CALCULATION ((BH-NEW)/BH)

BENCH NUMBER	TOE ELEV.	TONNES ORE **	Pb (%)	Zn (%)	Ag (g/t)	Au (g/t)	Pb + Zn (%)	LEAD METAL (tonnes)	ZINC METAL (tonnes)	SILVER METAL (grams)	GOLD METAL (grams)	TOTAL METAL (tonnes)
10	3890	18.6%	-4.7%	1.1%	-6.1%		-1.2%	14.7%	19.5%	13.6%		17.6%
11	3870	70.3%	0.1%	1.4%			0.9%	70.3%	70.7%			70.6%
12	3850	-89.3%	-43.2%	-42.2%			-42.6%	-171.1%	-169.2%			-169.9%
13	3830	17.9%	-8.8%	-8.6%			-8.7%	10.7%	10.9%			10.8%
14	3810	21.3%	-1.9%	-27.4%			-16.7%	19.8%	-0.3%			8.1%
15	3790	30.2%	-3.1%	-21.5%			-13.9%	28.1%	15.2%			20.5%
16	3770	19.3%	-5.6%	-14.9%	-11.7%		-11.4%	14.8%	7.3%	9.9%		10.1%
17	3750	-10.8%	4.9%	1.6%	10.8%		2.9%	-5.4%	-9.0%	1.2%		-7.6%
18	3730	-3.8%	9.1%	9.3%	14.7%		9.2%	5.6%	5.8%	11.5%		5.7%
19	3710	ERR	ERR	ERR	ERR		ERR	ERR	ERR	ERR		ERR
20	3690	ERR	ERR	ERR	ERR		ERR	ERR	ERR	ERR		ERR

AVERAGES:
 3890 to 3770 19.1% -5.6% -16.8% -203.4% ERR -12.4% 14.6% 5.5% -145.4% ERR 9.1%

ACTUAL VARIANCE BLASTHOLES TO NEW CALCULATION (BH-NEW)

BENCH NUMBER	TOE ELEV.	TONNES ORE **	Pb (%)	Zn (%)	Ag (g/t)	Au (g/t)	Pb + Zn (%)	LEAD METAL (tonnes)	ZINC METAL (tonnes)	SILVER METAL (grams)	GOLD METAL (grams)	TOTAL METAL (tonnes)
10	3890	8,210	-0.15	0.05	-2.7		-0.09	202	405	264056		607
11	3870	14,328	0.00	0.07			0.08	473	764			1237
12	3850	(26,386)	-1.04	-1.62			-2.66	-1218	-1915			-3133
13	3830	14,655	-0.25	-0.39			-0.63	244	401			645
14	3810	21,223	-0.06	-1.21			-1.27	630	-15			615
15	3790	55,664	-0.11	-1.06			-1.17	1779	1386			3165
16	3770	41,985	-0.15	-0.66	-4.1		-0.81	880	699	751050		1579
17	3750	(22,465)	0.15	0.07	4.6		0.23	-346	-866	103679		-1212
18	3730	(9,984)	0.27	0.43	6.2		0.71	438	709	1256532		1148
19	3710											
20	3690											

AVERAGES:
 3890 to 3770 129,679 -0.17 -0.77 -28.7 -0.22 -0.94 2989 1725 ***** -120065 4714

Table 3.6 E

Amount of dilution (in %): 30
 Grade of dilutant (%): 3.5
 Pb/Pb+Zn of dilutant: 0.6
 Ag grade of dilutant: 25

PERCENT VARIANCE BLASTHOLES TO NEW CALCULATION ((BH-NEW)/BH)

BENCH NUMBER	TOE ELEV.	TONNES ORE **	Pb (%)	Zn (%)	Ag (g/t)	Au (g/t)	Pb + Zn (%)	LEAD METAL (tonnes)	ZINC METAL (tonnes)	SILVER METAL (grams)	GOLD METAL (grams)	TOTAL METAL (tonnes)
10	3890	3.8%	-4.2%	9.5%	-2.9%		4.0%	-0.3%	12.9%	1.0%		7.7%
11	3870	64.9%	0.8%	10.4%			6.7%	65.2%	68.6%			67.3%
12	3850	-123.7%	-41.3%	-28.8%			-33.6%	-216.0%	-188.1%			-198.9%
13	3830	3.0%	-9.4%	1.0%			-3.0%	-6.2%	3.9%			0.0%
14	3810	6.9%	-1.4%	-15.2%			-9.4%	5.6%	-7.2%			-1.8%
15	3790	17.5%	-1.3%	-9.3%			-6.0%	16.4%	9.8%			12.5%
16	3770	4.6%	-7.1%	-4.5%	-11.0%		-5.5%	-2.1%	0.3%	-5.9%		-0.6%
17	3750	-30.9%	3.9%	9.7%	11.1%		7.4%	-25.8%	-18.2%	-16.4%		-21.3%
18	3730	-22.7%	6.9%	16.3%	14.1%		12.6%	-14.3%	-2.7%	-5.4%		-7.2%
19	3710	ERR	ERR	ERR	ERR		ERR	ERR	ERR	ERR		ERR
20	3690	ERR	ERR	ERR	ERR		ERR	ERR	ERR	ERR		ERR

AVERAGES:
 3890 to 3770 4.4% -5.4% -5.9% -197.6% ERR -5.7% -0.7% -1.2% -184.4% ERR -1.0%

ACTUAL VARIANCE BLASTHOLES TO NEW CALCULATION (BH-NEW)

BENCH NUMBER	TOE ELEV.	TONNES ORE **	Pb (%)	Zn (%)	Ag (g/t)	Au (g/t)	Pb + Zn (%)	LEAD METAL (tonnes)	ZINC METAL (tonnes)	SILVER METAL (grams)	GOLD METAL (grams)	TOTAL METAL (tonnes)
10	3890	1,670	-0.13	0.45	-1.3		0.31	-4	268	18806		264
11	3870	13,228	0.03	0.55			0.58	438	741			1179
12	3850	(36,556)	-0.99	-1.10			-2.10	-1538	-2129			-3667
13	3830	2,425	-0.26	0.04			-0.22	-141	144			3
14	3810	6,929	-0.04	-0.67			-0.71	180	-316			-136
15	3790	32,272	-0.04	-0.46			-0.50	1042	895			1937
16	3770	10,055	-0.19	-0.20	-3.8		-0.39	-126	29	-446325		-97
17	3750	(64,413)	0.12	0.45	4.8		0.57	-1668	-1747	-1469371		-3414
18	3730	(59,274)	0.21	0.76	5.9		0.97	-1114	-326	-591843		-1440
19	3710											
20	3690											

AVERAGES:
 3890 to 3770 30,023 -0.16 -0.27 -27.9 -0.22 -0.43 -150 -368 ***** -141895 -518

Table J.6F

Amount of dilution (in %): 10
 Grade of dilutant (%): 0
 Pb/Pb+Zn of dilutant: 0
 Ag grade of dilutant: 0

PERCENT VARIANCE BLASTHOLES TO OLD CALCULATION ((BH-OLD)/BH)

BENCH NUMBER	TOE ELEV.	TONNES DRE **	Pb (%)	Zn (%)	Ag (g/t)	Au (g/t)	Pb + Zn (%)	LEAD METAL (tonnes)	ZINC METAL (tonnes)	SILVER METAL (grams)	GOLD METAL (grams)	TOTAL METAL (tonnes)
10	3890	28.9%	-8.8%	3.0%	-11.9%		-1.7%	22.6%	31.0%	20.4%		27.7%
11	3870	45.8%	12.5%	18.0%			15.9%	52.5%	55.5%			54.4%
12	3850	-69.4%	-63.5%	-81.5%			-74.5%	-177.0%	-207.4%			-195.6%
13	3830	-2.1%	-29.7%	-32.1%			-31.2%	-32.4%	-34.8%			-33.9%
14	3810	8.1%	-11.9%	-30.4%			-22.6%	-2.8%	-19.8%			-12.7%
15	3790	31.3%	-15.8%	-34.2%			-26.6%	20.5%	7.8%			13.0%
16	3770	6.5%	-13.9%	-15.1%	-27.3%		-14.7%	-6.4%	-7.6%	-19.0%		-7.2%
17	3750	-17.5%	7.8%	1.6%	5.9%		4.1%	-8.3%	-15.6%	-10.5%		-12.7%
18	3730	1.7%	7.3%	1.3%	19.3%		3.6%	8.9%	3.0%	20.6%		5.3%
19	3710	ERR	ERR	ERR	ERR		ERR	ERR	ERR	ERR		ERR
20	3690	ERR	ERR	ERR	ERR		ERR	ERR	ERR	ERR		ERR
AVERAGES:												
3890 to 3770		11.8%	-16.2%	-25.3%	-232.1%	ERR	-21.7%	-2.5%	-10.6%	-193.0%	ERR	-7.4%

ACTUAL VARIANCE BLASTHOLES TO OLD CALCULATION (BH-OLD)

BENCH NUMBER	TOE ELEV.	TONNES DRE **	Pb (%)	Zn (%)	Ag (g/t)	Au (g/t)	Pb + Zn (%)	LEAD METAL (tonnes)	ZINC METAL (tonnes)	SILVER METAL (grams)	GOLD METAL (grams)	TOTAL METAL (tonnes)
10	3890	12,764	-0.27	0.14	-5.3		-0.13	310	644	396511		954
11	3870	9,323	0.41	0.95			1.36	353	599			953
12	3850	(20,501)	-1.53	-3.12			-4.65	-1260	-2347			-3607
13	3830	(1,680)	-0.83	-1.45			-2.28	-740	-1287			-2027
14	3810	8,111	-0.38	-1.34			-1.72	-90	-872			-962
15	3790	57,655	-0.54	-1.69			-2.23	1297	711			2008
16	3770	14,210	-0.38	-0.67	-9.5		-1.05	-383	-733	-1444100		-1116
17	3750	(36,391)	0.24	0.07	2.5		0.32	-535	-1496	-942093		-2032
18	3730	4,459	0.22	0.06	8.1		0.28	691	365	2263826		1056
19	3710											
20	3690											
AVERAGES:												
3890 to 3770		79,882	-0.49	-1.16	-32.7	-17.7%	-1.65	-513	-3285	*****		-3798

Table 3-6G

Amount of dilution (in %): 20
 Grade of dilutant (%): 0
 Pb/Pb+Zn of dilutant: 0
 Ag grade of dilutant: 0

PERCENT VARIANCE BLASTHOLES TO OLD CALCULATION ((BH-OLD)/BH)

BENCH NUMBER	TOE ELEV.	TONNES DRE **	Pb (%)	Zn (%)	Ag (g/t)	Au (g/t)	Pb + Zn (%)	LEAD METAL (tonnes)	ZINC METAL (tonnes)	SILVER METAL (grams)	GOLD METAL (grams)	TOTAL METAL (tonnes)
10	3890	22.4%	0.2%	11.1%	-2.6%		6.8%	22.6%	31.0%	20.4%		27.7%
11	3870	40.8%	19.8%	24.8%			22.9%	52.5%	55.5%			54.4%
12	3850	-84.8%	-49.9%	-66.3%			-60.0%	-177.0%	-207.4%			-195.6%
13	3830	-11.3%	-18.9%	-21.1%			-20.3%	-32.4%	-34.8%			-33.9%
14	3810	-0.2%	-2.6%	-19.5%			-12.4%	-2.8%	-19.8%			-12.7%
15	3790	25.0%	-6.1%	-23.0%			-16.1%	20.5%	7.8%			13.0%
16	3770	-2.0%	-4.4%	-5.5%	-16.7%		-5.1%	-6.4%	-7.6%	-19.0%		-7.2%
17	3750	-28.2%	15.5%	9.8%	13.8%		12.1%	-8.3%	-15.6%	-10.5%		-12.7%
18	3730	-7.2%	15.0%	9.5%	26.0%		11.7%	8.9%	3.0%	20.6%		5.3%
19	3710	ERR	ERR	ERR	ERR		ERR	ERR	ERR	ERR		ERR
20	3690	ERR	ERR	ERR	ERR		ERR	ERR	ERR	ERR		ERR

AVERAGES:
 3890 to 3770 3.8% -6.5% -14.9% -204.4% ERR -11.6% -2.5% -10.6% -193.0% ERR -7.4%

ACTUAL VARIANCE BLASTHOLES TO OLD CALCULATION (BH-OLD)

BENCH NUMBER	TOE ELEV.	TONNES DRE **	Pb (%)	Zn (%)	Ag (g/t)	Au (g/t)	Pb + Zn (%)	LEAD METAL (tonnes)	ZINC METAL (tonnes)	SILVER METAL (grams)	GOLD METAL (grams)	TOTAL METAL (tonnes)
10	3890	9,908	0.01	0.52	-1.2		0.53	310	644	396511		954
11	3870	8,318	0.65	1.31			1.97	353	599			953
12	3850	(25,051)	-1.20	-2.54			-3.74	-1260	-2347			-3607
13	3830	(9,280)	-0.53	-0.95			-1.48	-740	-1287			-2027
14	3810	(228)	-0.08	-0.86			-0.94	-90	-872			-962
15	3790	46,140	-0.21	-1.13			-1.34	1297	711			2008
16	3770	(4,280)	-0.12	-0.25	-5.8		-0.37	-383	-733	-1444100		-1116
17	3750	(58,631)	0.48	0.45	5.9		0.93	-535	-1496	-942093		-2032
18	3730	(18,873)	0.45	0.44	10.9		0.89	691	365	2263826		1056
19	3710											
20	3690											

AVERAGES:
 3890 to 3770 25,527 -0.20 -0.68 -28.8 -17.7% -0.88 -513 -3285 ***** -3798

3.3 Grum

3.3.1 History

3.3.2 General Geology

3.3.2.1 Lithology

3.3.2.2 Structure

3.3.2.3 Surficial geology

3.3.2.4 Ore deposit geology

3.3.3 Drill definition and information base

3.3.4 Methods and procedure of reserve calculation

3.3.4.1 Introduction

3.3.4.2 Block geology and drillhole information

3.3.4.3 Composite calculation

3.3.4.4 Variogram analysis

3.3.4.5 Interpolation

3.3.4.6 Geological reserve reporting

3.3.5 Results

3.3.6 Comparison of geological reserves

3.3.7 Reliability of geological reserves

3.3.8 Additional work needed

3.3 Grum

3.3.1 History

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

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

Kerr Addison sold the deposit, along with Vangorda and Swim, to Cyprus Anvil Mining Corporation in 1979. From 1980 to 1982 Cyprus Anvil drilled additional holes in and around the deposit and relogged all existing holes in it. All available sulphide intersections were 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 by 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 (fig 3.9) 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° to 30°) 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° trend 11° plunge).

There are several important faults at Grum. The largest displacements occur on moderately (35° - 45°) dipping structures that truncate the deposit at both its northwest and southeast ends (fig.3.10). 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° , passes between sections 70W and 72W and downdrops the deposit about 60 m to the northwest. A myriad of smaller faults were mapped underground by Kerr Addison trending on the average 080° and dipping steeply. Joints mapped underground and on surface tend to strike 060° and dip subvertically.

3.3.2.3 Surficial Geology

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

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. 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 and gradational both at the footwall and laterally. As with all Anvil District deposits overall 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).

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. All deposits other than Faro consist of several distinct, highly

contorted horizons separated by barren phyllite waste, whereas Faro is 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. Grum's higher grade if diluted at 3 times the historical 5% dilution used at Faro still gives Grum a higher average grade. 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 may have higher 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.

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 utilize 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. Similarly, 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, particularly 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 (figure 3.14). 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 total 2900 m (the workings are now flooded). The fans are most complete on even numbered sections (ie: spaced 61 m apart); on odd numbered sections 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 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 N₂ 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 was constructed and new reserves 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 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 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 - .

3.3.4.2 Block Geology and Drillhole information

All drill holes in the portion of the deposit modeled were used with the exception of a few drilled at high grade angle to the cross sections. An example of the cross sections is included in pockets of this report (Plate____). All sections are available in a supporting document available at Curragh's Toronto and Whitehorse offices. The cross sections are 61m apart (200 feet) and provide the only geologic control for the mine model.

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 longitudinal 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 most prominent ore type if the block was more than 50% sulphides, 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 otherwise coded.

The rock types used in the Grum model are summarized in table 3.7. Since there is a large amount of low grade quartzose mineralization in the footwall of mineralized horizons at Grum it was judged desirable to carry a separate code for base metal poor and base metal rich lithologies. This is inherent 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 rows 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 row of blocks one level (southeast of the section) and lowering the 4th row of blocks one level (northwest of the section). The second and third rows of blocks are kept the same.

Table 3.7

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

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 reformatted to suit PC Mine then imported.

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

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 3.8. Intervals with no measured S.G. were assigned an SG depending on rock type as listed in table 3.9. 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 3.8

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 3.9

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

<u>Ore Type</u>	<u>SG</u>
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
4LO	3.11
4L4/4LE	3.29

Table 3.10

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 3.10. Every composite was manually checked against the cross-sections to insure that the codes applied to sectional units were consistent with the composite codes.

3.3.4.4 Variogram analysis:

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 3.15 to 3.17. The variogram analysis shows the range along the deposit is greater than either vertically or across the deposit. The 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 Interpolation:

The geostatistical analysis done was not adequate to use kriging as an interpolation method thus inverse square distance weighting was used following precedent set at Faro. The search volume was an ellipsoid with major axis 150 m parallel to the deposit plunge and diameter in cross section 106 m. The ellipsoid 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 anisotropy of 1.41 was used. This results in samples along trend being weighted twice as heavily as those across trend (with an anisotropy 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 53×1.41 or 75 m from the block center, once this apparent distance is squared the factor of 2 ($= 1.41^2$) appears).

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

TWO DIMENSIONAL VARIOGRAM

EXTRACTION DATA USED :

M
O
M
E
N
T

C
E
N
T
R
E

4.727866 +
4.609670 +
4.491473 +
4.373277 +
4.255080 +
4.136884 +
4.018687 +
3.900491 +
3.782294 +
3.664097 +
3.545900 +
3.427704 +
3.309507 +
3.191310 +
3.073113 +
2.954917 +
2.836720 +
2.718523 +
2.600327 +
2.482130 +
2.363933 +
2.245736 +
2.127540 +
2.009343 +
1.891146 +
1.772950 +
1.654753 +
1.536556 +
1.418360 +
1.300163 +
1.181967 +
1.063770 +
.945573 +
.827377 +
.709180 +
.590983 +
.472787 +
.354590 +
.236393 +
.118197 +

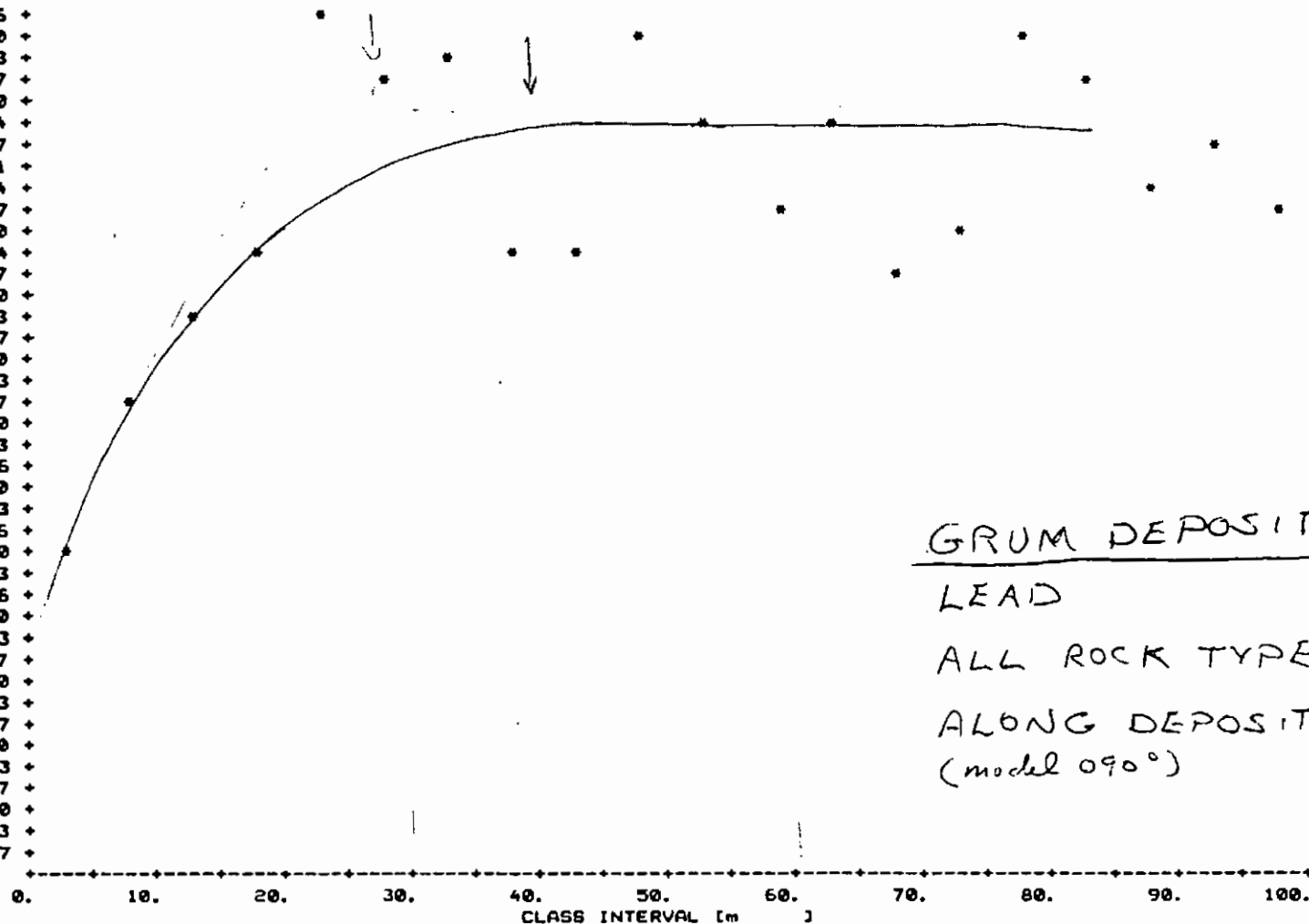


Fig 3.15

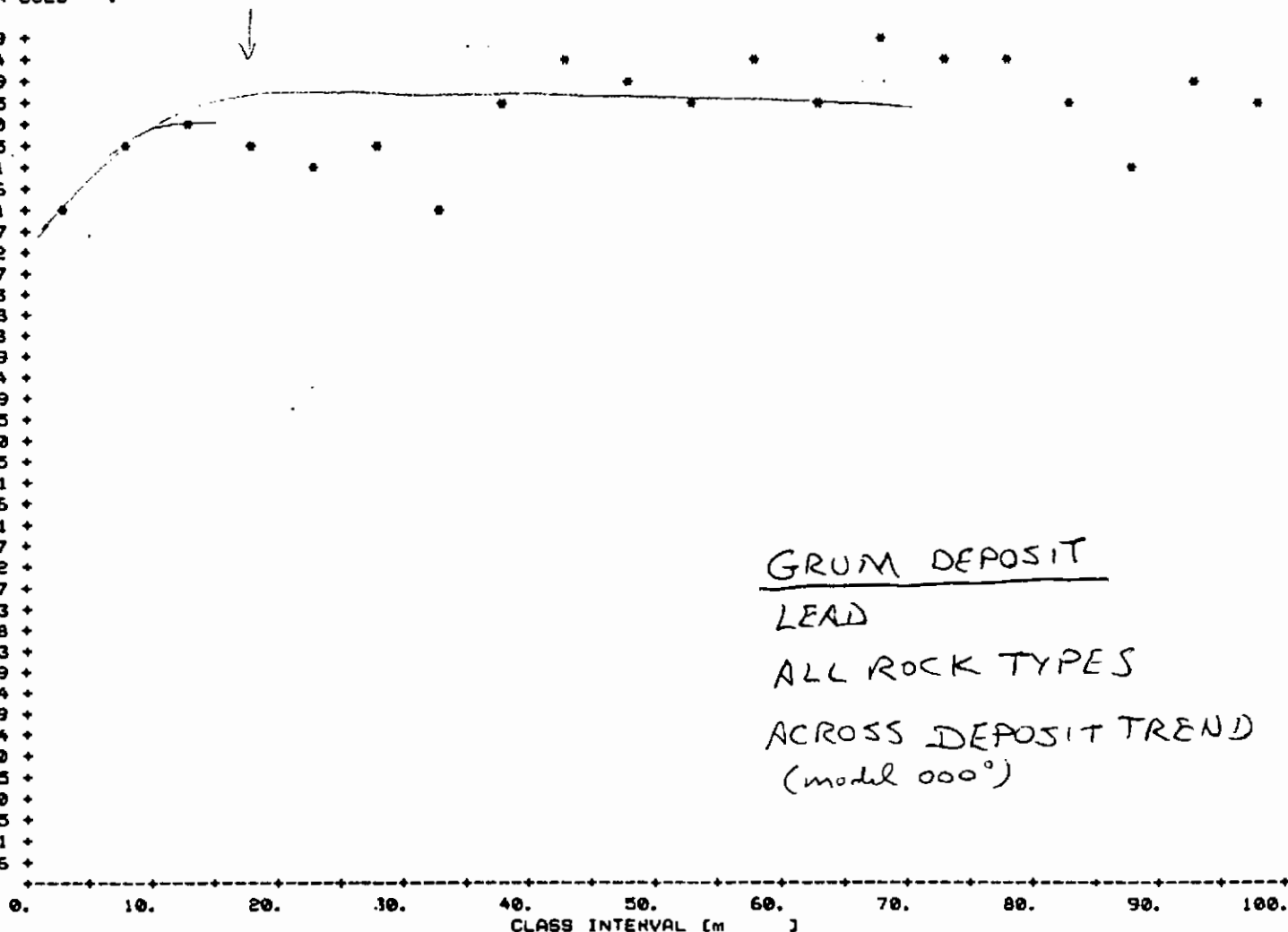
TWO DIMENSIONAL VARIOGRAM

EXTRACTION DATA USED :

M
O
M
E
N
T

C
E
N
T
R
E

4.217789 +
4.112344 +
4.006899 +
3.901455 +
3.796010 +
3.690565 +
3.585121 +
3.479676 +
3.374231 +
3.268787 +
3.163342 +
3.057897 +
2.952453 +
2.847088 +
2.741563 +
2.636119 +
2.530674 +
2.425229 +
2.319785 +
2.214340 +
2.108895 +
2.003451 +
1.898006 +
1.792561 +
1.687117 +
1.581672 +
1.476227 +
1.370783 +
1.265338 +
1.159893 +
1.054449 +
.949004 +
.843559 +
.738114 +
.632670 +
.527225 +
.421780 +
.316335 +
.210891 +
.105446 +



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 %

M
D
M
E
N
T

C
E
N
T
R
E

4.805925 +
4.685777 +
4.565629 +
4.445480 +
4.325332 +
4.205184 +
4.085036 +
3.964888 +
3.844739 +
3.724591 +
3.604443 +
3.484295 +
3.364147 +
3.243999 +
3.123850 +
3.003702 +
2.883554 +
2.763406 +
2.643258 +
2.523109 +
2.402961 +
2.282813 +
2.162665 +
2.042517 +
1.922369 +
1.802220 +
1.682072 +
1.561924 +
1.441776 +
1.321628 +
1.201479 +
1.081331 +
.961183 +
.841035 +
.720887 +
.600739 +
.480591 +
.360443 +
.240294 +
.120146 +

0. 2. 4. 6. 8. 10. 12. 14. 16. 18. 20.
CLASS INTERVAL (m)

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

GRUM DEPOSIT
LEAD
ALL ROCK TYPES
ALONG VERTICAL DDH'S

Fig 3.17

Reserves are 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 ($\pm 30.5\text{m}$ from the section line). The sectional reserves were needed to compare to previous calculations which are largely sectional hand calculations.

3.3.5 Results

The geological reserves calculated for Grum are summarized in table 3.11 for the two constituent models and for the entire deposit. Sectional reserves are summarized in table 3.12 for the entire deposit.

Southeast of section 62W the Champ zone is estimated to contain a additional 1.7 million 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.

Table 3.13. and 3.14 A & B compare the newly calculated reserves (G8606 model) to previous calculations on a whole deposit and sectional basis respectively.

Unfortunately the calculations listed in table 3.13 are not exactly comparable. The first 3 (A-C) are all based on Kerr-Addisons geological interpretation. 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 3.13) are based on the current geological interpretation and the current assay information which nearly completely replaces Kerr-Addisons 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 interpolations 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 3.13).

The current 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 3.13). The current 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 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 on that section (table 3.14B). The distribution of uninterpolated blocks by section in the two constituent models is given in table 3.15.

A further complication in comparing the current model to the results obtained from the same interpolation is that two problems were encountered with the Cyprus Anvil/Dome calculation: a) there were some calculation errors involving volumes 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 rather than the measured SG as used for the rest of the deposit, in retrospect this value was too high as much of the mineralization on those sections is considerably lighter. These points have been adjusted for in Table 3.14A. Table 3.14B 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 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.

Table 3.11

CURRAGH RESOURCES
GRUM DEPOSIT
68606 MODEL FOR TOTAL DEPOSIT

ABOVE GRUM (1336.0 m to 1088.5 m)

| GRADE
CATEGORY | VOLUME
(bcm) | S.G. | ORE
(tonnes) | LEAD
(%) | ZINC
(%) | Pb + Zn
(%) | SILVER
(g/t) | GOLD
(g/t) |
|-------------------|-----------------|------|-----------------|-------------|-------------|----------------|-----------------|---------------|
| +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)

| GRADE
CATEGORY | VOLUME
(bcm) | S.G. | ORE
(tonnes) | LEAD
(%) | ZINC
(%) | Pb + Zn
(%) | SILVER
(g/t) | GOLD
(g/t) |
|-------------------|-----------------|------|-----------------|-------------|-------------|----------------|-----------------|---------------|
| +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)

| GRADE
CATEGORY | VOLUME
(bcm) | S.G. | ORE
(tonnes) | LEAD
(%) | ZINC
(%) | Pb + Zn
(%) | SILVER
(g/t) | GOLD
(g/t) |
|-------------------|-----------------|------|-----------------|-------------|-------------|----------------|-----------------|---------------|
| +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 |

CYPRUS ANVIL (SIMPSON-ADAMSON)
COMPARISON TO SIMPSON-ADAMSON/DOME HAND CALCULATION (uncorrected, see table -)

| | | | | | | | | |
|------|-----------|------|------------|------|------|------|----|--|
| +4 % | 8,225,911 | 3.96 | 32,611,000 | 3.48 | 5.72 | 9.20 | 59 | |
|------|-----------|------|------------|------|------|------|----|--|

VARIANCE 9.7%_x -14.3%_x -6.0%_x -2.2%_x -2.5%_x -2.4%_x -3.0%_x
new to old

NOTES:

- * Part of this difference in tonnage is due to uninterpolated blocks. There are 707 such blocks in under grum and ~~316~~ ³¹⁶ number in above grum not all of this material will be ore however. Each block represents about 1900 tonnes of material thus there is about 1,000,000 tonnes of unaccounted for sulphides.
- x The large variance ~~between volume and specific gravity between~~ the two calculations is ~~presently unexplained.~~ *explained in table -*

Table 3.12

68606 COMPUTER MODEL, 1986
TOTAL DEPOSIT

| 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%) between this ^{total} ~~figure~~ and that on table -
small amount of material outside the area limits set for
the sections when computing these reserves.

Table 3.13

CURRAGH RESOURCES

GRUM DEPOSIT

comparison of ^{previous} ~~various~~ calculations of geological reserves for 62W to 86W to the current model.

| Calculation | ore
(tonnes) | Pb
(%) | Zn
(%) | Pb+Zn
(%) | Ag
(g/t) | total
metal
(tonnes) | ^{total}
~~metal~~
variance
from avg.
(tonnes) | ^{total}
~~metal~~
percent
variance
from avg. |
|---|-----------------|-----------|-----------|--------------|-------------|----------------------------|---|--|
| A Kerr Addison - 1977
sectional hand calculation | 26,083,000 | 4.1 | 6.4 | 10.5 | 62 | 2,738,715 | 104,352 | 4.0% |
| B Kerr Addison/Noranda - 1978
3D block computer model | 27,650,000 | 3.1 | 4.9 | 8.0 | 48 | 2,212,000 | (422,363) | -16.0% |
| C Cyprus Anvil - 1981
"61" 3D block computer model | 30,781,000 | 3.1 | 4.9 | 8.0 | 49 | 2,462,480 | (171,883) | -6.5% |
| D ^{Cyprus-Anvil}
(Simpson-Adams/Dome - 1983) *
sectional hand calculation
(uncorrected see table -) | 32,611,000 | 3.5 | 5.7 | 9.2 | 59 | 3,000,212 | 365,849 | 13.9% |
| E Curragh Resources - 1986 *
"68606" 3D block computer model | 30,649,000 | 3.4 | 5.6 | 9.0 | 57 | 2,758,410 | 124,047 | 4.7% |
| Average of all the models | 29,554,800 | | | | | 2,634,363 | | |

NOTES:

- * includes on the order of 3,000,000 tonnes in the northwest part of the deposit drilled off in 1982 after the other calculations were done.

Table 3.14A

KERR ADDISON SECTIONAL CALCULATION, 1977
TOTAL DEPOSIT

| SECTION | VOLUME | SPECIFIC GRAVITY | TONNAGE | LEAD | ZINC | SILVER | GOLD | Pb+Zn | TOTAL METAL |
|------------|--------|------------------|------------|------|------|--------|-------|-------|-------------|
| | (bcm) | | (tonnes) | (%) | (%) | (g/t) | (g/t) | (%) | (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 |

CYPRUS ANVIL/DOME SECTIONAL CALCULATION (SIMPSON-ADAMSON), 1983
TOTAL DEPOSIT

| SECTION | VOLUME | SPECIFIC GRAVITY | TONNAGE | LEAD | ZINC | SILVER | GOLD | Pb+Zn | TOTAL METAL |
|------------|-----------|------------------|------------|------|------|--------|-------|-------|-------------|
| | (bcm) | | (tonnes) | (%) | (%) | (g/t) | (g/t) | (%) | (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 |

A assumed value for ⁸⁶W was ~~10.10~~ 10.10. This is used on these sections, ~~was~~ too high and has been reduced by 10% for comparison purposes.

* Volumes above cutoff have been recalculated due to an error in the previous calculation

Table 3.1#4B

PERCENT VARIANCE OF 68608 TO KERR ADDISON ((NEW-OLD)/OLD)
TOTAL DEPOSIT

| SECTION | VOLUME | SPECIFIC GRAVITY | TONNAGE | LEAD | ZINC | SILVER | GOLD | Pb+Zn | TOTAL METAL |
|------------|--------|------------------|----------|--------|--------|--------|-------|--------|-------------|
| | (bcm) | | (tonnes) | (%) | (%) | (g/t) | (g/t) | (%) | (tonnes) |
| 86 W | | | 26.1% | -27.6% | -14.0% | -16.4% | | -19.2% | 11.9% |
| 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% |

PERCENT VARIANCE OF 68608 TO SIMPSON -ADAMSON/DOME ((NEW-OLD)/OLD)
TOTAL DEPOSIT

| SECTION | VOLUME | SPECIFIC GRAVITY | TONNAGE | LEAD | ZINC | SILVER | GOLD | Pb+Zn | TOTAL METAL |
|------------|--------|------------------|----------|-------|--------|--------|-------|-------|-------------|
| | (bcm) | | (tonnes) | (%) | (%) | (g/t) | (g/t) | (%) | (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% |

Table 3.15

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 |

3.3.6 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 $\pm 10\%$ 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 and geology matching.

Based on the reproductibility 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 reliable within $\pm 10\%$, 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 probably reasonably accurate however towards the periphery there is considerably more possible variance. On a bench basis variances of $\pm 50\%$ 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 troughlike pit following the subcrop of the ore horizons. This part of the deposit is the most densely drilled from the surface but only the part southeast of 72W has been drilled from underground. The drill density is at least one hole every 100' on 200' 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 predicted 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 apparently 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 with those found having 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.

3.3.8 Additional Work needed

The complex geology of the Grum deposit and its multi-horizon nature creates a great deal of difficulty in producing an interpolation that it is consistent from section to section. Horizons are indistinguishable 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 spacial 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 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 hydrogeology. If the overburden is amenable 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. The northwest half of the ore in the first years pit will be almost entirely 4A.

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
 - 3.4.1 History
 - 3.4.2 General Geology
 - 3.4.2.1 Stratigraphy and Lithology
 - 3.4.2.2 Structure
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 - 3.4.3 Drilling density and information base
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 - 3.4.8 Additional work required

3.4 Vangorda

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 metallurgical 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 Stratigraphy and lithology

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 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 Structure

The Vangorda deposit occurs in the hinge of a large second phase fold. Overall the deposit has the shape of a reclining M or a 3 in cross section, however there is 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° towards the northwest but the southeast half is sub-horizontal. 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 subcropping 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 prominent. In the interpreted 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. 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 delineated 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. 90% of the mineralization exceeding 6% Pb & Zn is barite bearing massive sulphides (4EG). Most pyritic quartzite (4C and 4EC) is sulphide waste on the basis of lead zinc content.

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 (figure 3.18) 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 and 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.

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 PC MINE software package was used.

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 consistent interpretation. The resulting fold shapes are consistent with the deformation style observed in nearby outcrops on a small scale and with the inferred shapes of more closely controlled folds at Grum. An example of the sections is included in pocket - in this report. 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 $\pm 10\%$ to 15% may be possible due to interpretation) thus the current interpretation is adequate for long term planning even if there is disagreement on the treatment of structural detail.

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. 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 directly 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 value.

Assays were weighted by length but not by 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 values to arbitrary maxima prior to compositing calculations.

3.4.4.4 Interpolation

Experimental variograms calculated from these composites showed only nugget effect due to the relatively large drill hole spacing. For this reason the anisotropy 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 anisotropy: 1.41
- vertical anisotropy: 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.

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

Geological reserves at a 4% and 6% Pb+Zn cutoff grade are summarized in Table 3.16. 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 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 3.17 compares these results to the present model.

The oldest one by Prospector Airways is based on the triangular 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 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 3.18 and table 3.19.

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.

3.4.7 Reliability of Geological Reserves

As a check on model computations, the areas of geological units output during digitizing of unit outlines was used to compute the volume of the overall deposit. 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 (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 anisotropy 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 anisotropy parameters from isotropic to about the same anisotropy used for 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 $\pm 10\%$ to $\pm 20\%$. This is especially true in the southeast part of the deposit where $\pm 50\%$ might be a more realistic estimate of possible variance to be expected.

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

3.4.8 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 commitment to this project is made. A staggered triangular pattern should be drilled rather than the current rectangular 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.

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.

VANGORDA DEPOSIT - GEOLOGICAL RESERVES - 86-07 MODEL

in-situ reserves - no adjustments

entire deposit, irrespective of pit outlines

plus 6 %

| rock type | | % of
ore type | tonnes
(x1000) | density
(tn/bcm) | Pb
(%) | Zn
(%) | Pb + Zn
(%) | Ag
(g/t) | Au
(g/t) |
|-----------|---|------------------|-------------------|---------------------|-----------|-----------|----------------|-------------|-------------|
| 4A | 1 | 7.1 | 382.33 | 2.81 | 3.17 | 4.66 | 7.83 | 39.74 | 0.68 |
| 4C | 2 | 0.0 | 1.42 | 3.50 | 6.40 | 1.01 | 7.41 | 57.51 | 1.77 |
| 4EC | 3 | 0.4 | 21.41 | 3.65 | 2.97 | 3.79 | 6.75 | 46.65 | 0.80 |
| 4E | 4 | 0.1 | 4.16 | 3.43 | 2.57 | 4.12 | 6.69 | 46.50 | 0.63 |
| 4EG | 5 | 90.8 | 4,917.88 | 3.96 | 4.51 | 5.83 | 10.34 | 64.03 | 0.75 |
| 4EH | 6 | 1.7 | 90.23 | 3.65 | 6.82 | 5.05 | 11.87 | 83.15 | 0.54 |
| TOTAL | | 100.0 | 5,417.43 | 3.88 | 4.45 | 5.72 | 10.17 | 62.55 | 0.74 |

4.1 to 6 %

| rock type | | % of
ore type | tonnes
(x1000) | density
(tn/bcm) | Pb
(%) | Zn
(%) | Pb + Zn
(%) | Ag
(g/t) | Au
(g/t) |
|-----------|---|------------------|-------------------|---------------------|-----------|-----------|----------------|-------------|-------------|
| 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 %

| rock type | | % of
ore type | tonnes
(x1000) | density
(tn/bcm) | Pb
(%) | Zn
(%) | Pb + Zn
(%) | Ag
(g/t) | Au
(g/t) |
|-----------|---|------------------|-------------------|---------------------|-----------|-----------|----------------|-------------|-------------|
| 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)

| rock type | | % of
ore type | tonnes
(x1000) | density
(tn/bcm) | Pb
(%) | Zn
(%) | Pb + Zn
(%) | Ag
(g/t) | Au
(g/t) |
|-----------|---|------------------|-------------------|---------------------|-----------|-----------|----------------|-------------|-------------|
| 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 %

| rock type | | % of
ore type | tonnes
(x1000) | density
(tn/bcm) | Pb
(%) | Zn
(%) | Pb + Zn
(%) | Ag
(g/t) | Au
(g/t) |
|-----------|---|------------------|-------------------|---------------------|-----------|-----------|----------------|-------------|-------------|
| 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 |

OTHER GEOLOGICAL RESERVE ESTIMATES FOR THE ENTIRE DEPOSIT

Table 3.17

PROSPECTOR AIRWAYS / CHISHOLM et al. (extent unknown - may be 4W to 40E)

| | tonnes (x1000) | density (tn/bcm) | Pb (%) | Zn (%) | Pb+Zn (%) | Ag (g/t) | Au (g/t) | total metal (tonnes) | % variance (new-old)/old |
|------------------|----------------|--------------------------|--------|--------|-----------|----------|----------|----------------------|--------------------------|
| high grade (+4%) | 8,528 | 4.0 ? | 3.16 | 4.96 | 8.12 | 60.33 | 0.69 | 692,474 | -6.2 |
| 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----- | 8.67 | -----nd----- | | | 601,871 | 7.9 |
| 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 3.8

 OTHER GEOLOGICAL RESERVE ESTIMATES FOR THE NORTHWEST PART OF THE DEPOSIT

79-09 MODEL (i.e. the Vangorda Plateau AFE model)

| | tonnes
(x1000) | density
(tn/bcm) | Pb
(%) | Zn
(%) | Pb+Zn
(%) | Ag
(g/t) | Au
(g/t) | total metal
(tonnes) | % variance
(new-old)/old |
|------------------|-------------------|---------------------|-----------|-----------|--------------|-------------|-------------|-------------------------|-----------------------------|
| 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 3M to 13E)

| | | | | | | | | | |
|---------------|--------|------|--|--|-----|--|--|---------|------|
| high grade | 4,906 | 4.00 | | | 8.6 | | | 420,796 | 15.2 |
| low grade | 5,831 | | | | | | | | |
| total deposit | 10,738 | | | | | | | | |

THIS CALCULATION (3M 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 3.19

VANGORDA MODEL COMPARISON FOR NORTHWEST PART OF DEPOSIT (3W TO 13E)

86-07 TO 79-12 MODEL

The 79-12 model was a computer based 3d block model based on geologic cross sections. 19m X 19m X 6m blocks.
An inordinately high S.G. of 4.2 was used for all sulphides.

| 5.0 % CUTOFF | 79-12 MODEL | 86-07 MODEL | VARIANCE
(NEW-OLD)/OLD |
|--------------|-------------|-------------|---------------------------|
| TONNES | 6134 | 4710 | -23.2 |
| S.G. | 4.2 | 3.7 | -11.9 |
| Pb (%) | 3.7 | 4.3 | 16.7 |
| Zn (%) | 4.8 | 5.5 | 14.8 |
| Ag (g/t) | 52.8 | 61.5 | 16.5 |
| metal | 521.4 | 462.8 | -11.2 |
| VOLUME bcm | 1460.5 | 1273.3 | -12.8 |
| 4.5 % CUTOFF | | | |
| TONNES | 6437 | 5067 | -21.3 |
| S.G. | 4.2 | 3.6 | -13.3 |
| Pb (%) | 3.6 | 4.2 | 15.4 |
| Zn (%) | 4.7 | 5.3 | 13.0 |
| Ag (g/t) | 51.7 | 59.3 | 14.7 |
| metal | 534.3 | 479.7 | -10.2 |
| VOLUME bcm | 1532.6 | 1392.4 | -9.1 |
| 4.0 % CUTOFF | | | |
| TONNES | 6751 | 5471 | -19.0 |
| S.G. | 4.2 | 3.6 | -14.6 |
| Pb (%) | 3.5 | 4.0 | 13.9 |
| Zn (%) | 4.6 | 5.1 | 10.8 |
| Ag (g/t) | 50.7 | 57.0 | 12.4 |
| metal | 546.8 | 496.9 | -9.1 |
| VOLUME bcm | 1607.4 | 1524.6 | -5.1 |
| 3.5 % CUTOFF | | | |
| TONNES | 7090 | 5994 | -15.5 |
| S.G. | 4.2 | 3.5 | -15.7 |
| Pb (%) | 3.4 | 3.8 | 11.3 |
| Zn (%) | 4.5 | 4.8 | 7.4 |
| Ag (g/t) | 49.6 | 54.3 | 9.4 |
| metal | 560.1 | 516.4 | -7.8 |
| VOLUME bcm | 1688.1 | 1693.5 | 0.3 |

 86-07 TO 81-10 MODEL

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 of these comparisons 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 models 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%.

4.0 METALLURGY

4.0 Metallurgy

Available metallurgical data was compiled and analyzed by CAMC for their report The Development of the Vangorda Plateau Orebodies. The results of their analysis to provide the metallurgical parameters used in this report. For a more detailed review of the Grum and Vangorda metallurgies, appropriate sections of the CAMC report have been included as appendix C to this report.

4.1 Vangorda Metallurgy

The Vangorda orebody has been categorized into three types: baritic, pyritic and quartzitic. The predicted metallurgical response as determined by CAMC is as shown in table 4.1-1 below.

| Ore Type | Recovery, % | | | |
|------------|-------------|----|----|----|
| | Pb | Zn | Ag | Au |
| Baritic | 84 | 80 | 65 | 40 |
| Pyritic | 77 | 72 | 50 | 50 |
| Quartzitic | 81 | 79 | 55 | 15 |

Table 4.1-1
Predicted Recoveries of Vangorda Ore

In addition to the above recoveries, a head grade effect, derived from current Faro mill operations, is applied:

$$\text{Pb Factor} = 0.181 \times \ln (\text{Pb head grade, \%}) + 0.767$$

$$\text{Zn Factor} = 0.162 \times \ln (\text{Zn head grade, \%}) + 0.733$$

$$\text{Ag Factor} = 0.18 \times \ln (\text{Wt \% to Pb conc.}) + 0.733$$

$$\text{Au factor} = 0.18 \times \ln (\text{Wt \% to Pb conc.}) + 0.733$$

The southeast end of the Vangorda orebody is known to be oxidized to some degree, but the extent and degree of oxidation have not been defined. Although this oxidation is almost certain to have a detrimental effect on recovery, no such allowance has been made in this report. Also, much of the low grade ore is expected to be stockpiled for a number of years. Again, no allowance has been made for oxidation of stockpiled ore.

4.2 Grum Metallurgy

Although the Grum orebody is composed of several different ore types, metallurgical parameters have been defined without regard to the various ore types. Since the metallurgical results are expected to vary with ore type, more work is required in this area.

The predicted metallurgical response as determined by CAMC is as shown in Table 4.2-1 below.

| Ore Type | Recovery, % | | | |
|-------------|-------------|----|----|----|
| | Pb | Zn | Ag | Au |
| Average Ore | 80 | 83 | 65 | 33 |

Table 4.2-1
Predicted Recoveries of Grum Ore

In addition to the above recoveries, a head grade effect, derived from current Faro mill operations, is applied:

$$\text{Pb Factor} = 0.181 \times \ln (\text{Pb head grade, \%}) + 0.767$$

$$\text{Zn Factor} = 0.162 \times \ln (\text{Zn head grade, \%}) + 0.733$$

$$\text{Ag Factor} = 0.18 \times \ln (\text{Wt \% to Pb conc.}) + 0.733$$

$$\text{Au factor} = 0.18 \times \ln (\text{Wt \% to Pb conc.}) + 0.733$$

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 Vangorda Pit

5.1.1 Geotechnical

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

5.1.1.1 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 occurring 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.

5.1.1.2 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.

5.1.2 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.

5.1.2.1 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 5.1-1 and 5.1-2.

5.1.2.2 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 = $0.181 \times \ln (\text{Pb head grade, \%}) + 0.767$

Zn factor = $0.162 \times \ln (\text{Zn head grade, \%}) + 0.733$

Ag factor = $0.18 \ln (\text{Wt. \% to Pb conc.}) + 0.733$

Au factor = $0.18 \times \ln (\text{Wt. \% to Pb conc.}) + 0.733$

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

MINING COST DATA

REC STAT ROCK DESCRIPTION
CODE

COST DATA (CDN \$ PER BCM)

| DRILLING | BLASTING | LOADING | FIXED
HAULAGE | MINING
SERVICES |
|----------|----------|---------|------------------|--------------------|
|----------|----------|---------|------------------|--------------------|

| | | | | | | | |
|----|---|--|-------|-------|-------|-------|-------|
| 1 | 1 | 1 4A Ribbon banded graphitic pyritic quartzite | .1748 | .3865 | .4018 | .3533 | .9817 |
| 2 | 1 | 2 4C Pyritic quartzite | .1748 | .3865 | .4018 | .3533 | .9817 |
| 3 | 1 | 3 4C Quartz rich massive or nearly massive sulphides | .1748 | .3865 | .4018 | .3533 | .9817 |
| 4 | 1 | 4 4E Pyritic Massive Sulphides | .1748 | .3865 | .4018 | .3533 | .9817 |
| 5 | 1 | 5 4EG Variably baritic pyritic massive sulphides | .1748 | .3865 | .4018 | .3533 | .9817 |
| 6 | 1 | 6 4H Pyrrhotitic massive sulphides | .1748 | .3865 | .4018 | .3533 | .9817 |
| 7 | 1 | 10 Waste | .1748 | .3865 | .4018 | .3533 | .9817 |
| 8 | 1 | 11 Overburden | .0175 | .0387 | .4018 | .3533 | .9817 |
| 9 | 1 | 0 Air | .0000 | .0000 | .0000 | .0000 | .0000 |
| 10 | 1 | 12 Partially Above Topography | .0175 | .0387 | .4018 | .3533 | .9817 |

TABLE 5.1-11

VOLUMETRIC MINING COSTS - VANGORDA PIT

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

Curragh Resources
Vangorda 8607 Geological Model

SOFTWARE BY GEMCOM SERVICES INC
MODULE 1.06
PAGE 1

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

DETAILED PRINTOUT FOR RECORD [2]

DATE CAPTURED : 4/11/1986

RECORD DESCRIPTION : November 86 Model

GENERAL COST INFORMATION

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

HAULAGE INFORMATION

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

TABLE 5.15-22

ORE BASED COSTS AND HAULAGE COSTS - VANGORDA PIT

Table 5.1-3 shows the calculation of revenue estimates for entry into the PC-MINE system; tables 5.1-4 and 5.1-5 show the PC-MINE referenced metallurgical and revenue parameters and table 5.1-6 shows a sample calculation for one ore block.

5.1.3 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 stabilization operations are underway.

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

Figure 5.2-1 shows the ultimate pit.

5.1.4 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:

$$\begin{aligned}\text{Recovered ore tonnes} &= (\text{recovery}) \times (\text{in-situ ore tonnes}) \\ \text{Dilution tonnes} &= (\text{recovered ore tonnes}) \times (\text{dilution}) \\ \text{Mined ore tonnes} &= \text{Recovered ore tonnes} + \text{dilution tonnes} \\ &= (\text{recovered ore tonnes}) \times (1 + \text{dilution}) \\ &= (\text{in-situ ore tonnes}) \times (\text{recovery}) \times (1 + \text{dilution})\end{aligned}$$

The dilution is assumed to contain zero metal:

Recovered ore grade = in-situ ore grade

Dilution grade = zero

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

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 5.1 - 3

REVENUE CALCULATIONS - VANGORDA PIT

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

Curragh Resources
Vangorda B607 Geological Model

SOFTWARE BY GEMCOM SERVICES INC
MODULE 1.04
PAGE 1

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

METALLURGICAL DATA

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

TABLE 5.1-4

METALLURGICAL PARAMETERS - VANGORDA PIT

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

Curragh Resources
 Vangorda B607 Geological Model

SOFTWARE BY GENCOM SERVICES INC
 MODULE 1.06
 PAGE 2

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)

DESCRIPTION : Lead Revenue

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

| 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

DESCRIPTION : Silver Revenue

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

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

TABLE 5.15, 5

REVENUE/HEAD GRADE INFORMATION - VANGORDA PIT

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

Curragh Resources
Vangorda 8607 Geological Model

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MODULE 1.06
PAGE 3

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

DETAILED PRINTOUT FOR RECORD [2]

DESCRIPTION : Gold Revenue

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

TABLE 5.1-75 (CONTINUED)

REVENUE/HEAD GRADE INFORMATION - VANGORDA PIT

PC-MINE VERSION 1.10
 SERIAL NO : 20320
 29/11/1986

Curragh Resources
 Vangorda 8607 Geological Model

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 MODULE 3.06
 PAGE 1

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

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

| LABEL: | UNITS: | GRADE: | REVENUE: | RECOVERY: | BLOCK REVENUE: | CUM. REVENUE: |
|--------|--------|--------|----------|-----------|----------------|---------------|
| 1 | ZPb+Zn | 8.10 | .00 | .00 | .00 | .00 |
| 2 | ZPb | 3.28 | .89 | 81.00 | 509.19 | 509.19 |
| 3 | ZZn | 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 5.1 - 6
 SAMPLE CALCULATION

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

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

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 5.1-7 details the mining reserves for the pit. In addition, a detailed bench by bench listing is given in appendix E.

5.1.5 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 5.1-2.

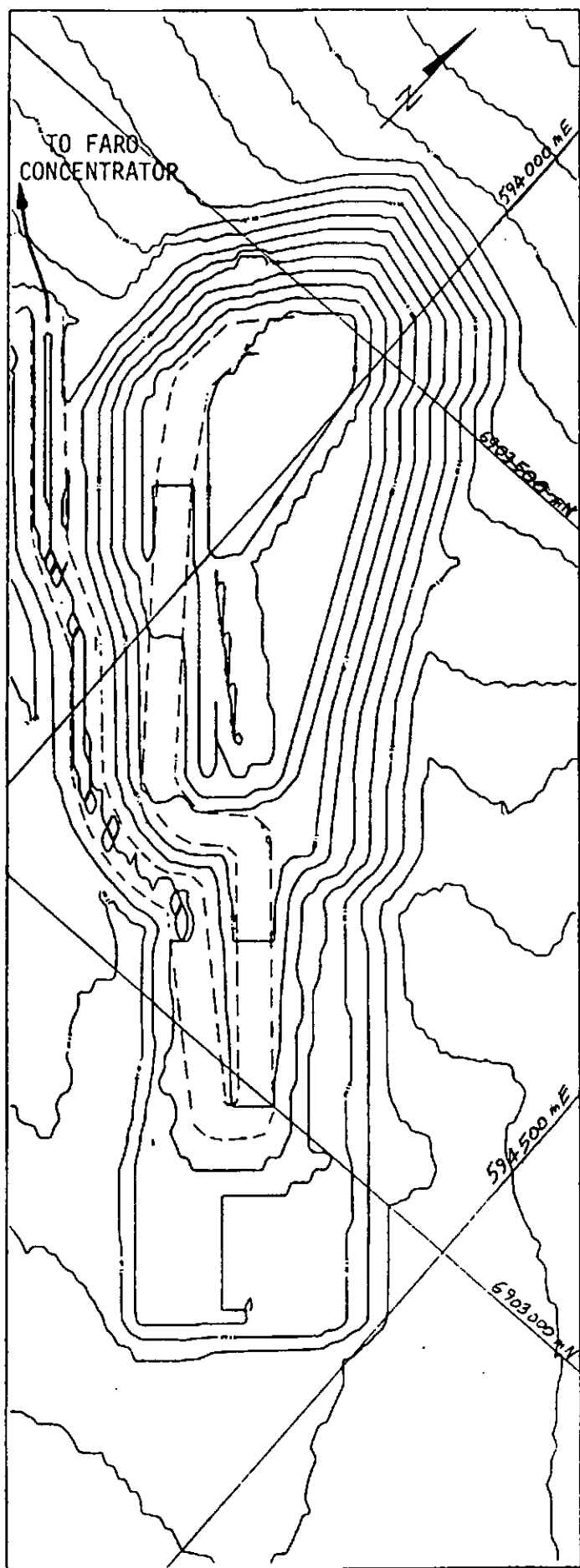


FIGURE 5.1 - 1
VANGORDA ULTIMATE PIT

Mining Reserves
Vangorda Pit

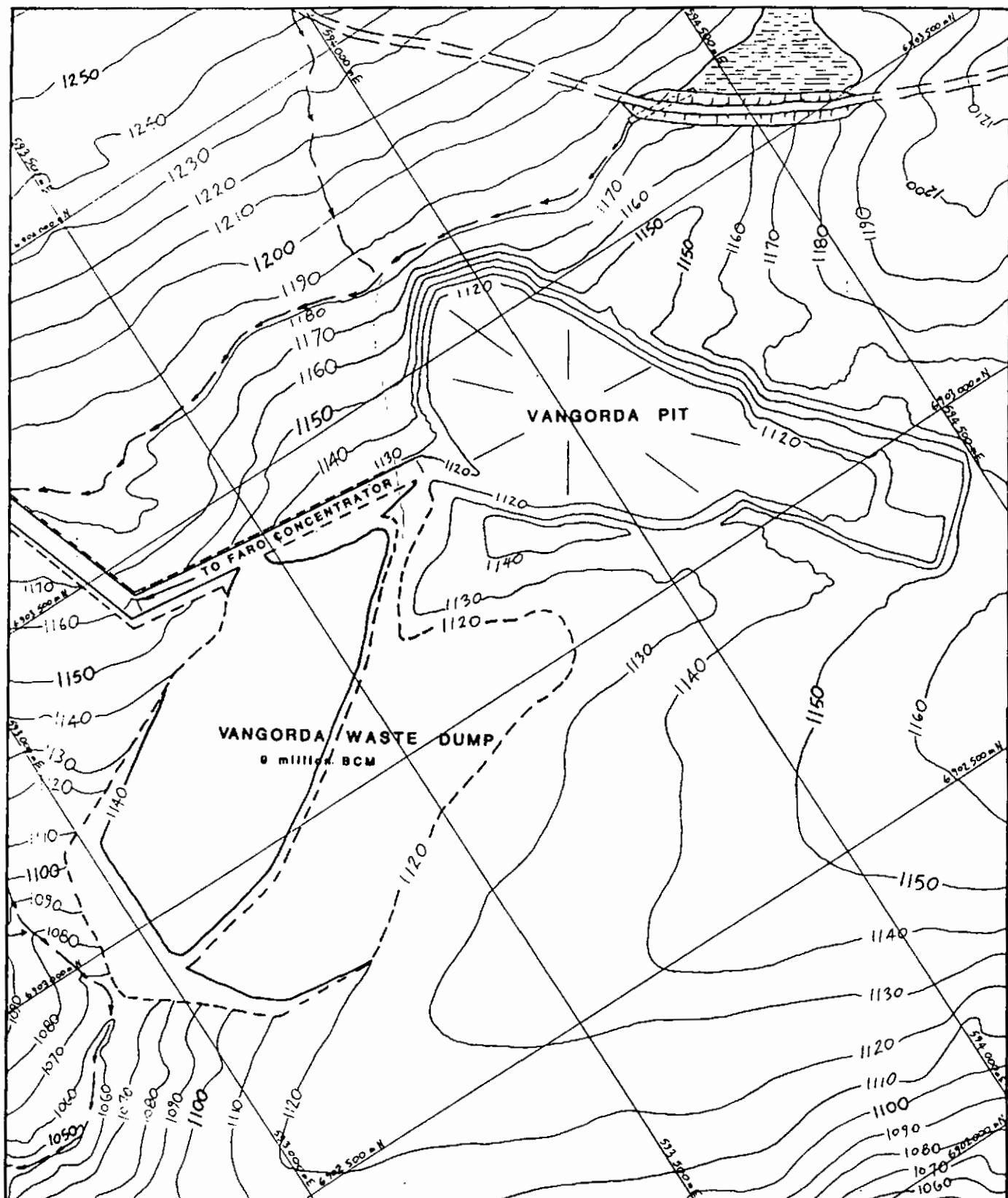
Tonnes

| | |
|----------------|----------|
| Rock | 8593620 |
| Overburden | 9069372 |
| Sulphide Waste | 3824375 |
| | 7 |
| 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. | .88 |
| Baritic | 5.0 + | 45750546. | 9.08 | 3.96 | 5.12 | 57. | .63 |
| Pyritic | 4.0-5.0 | 138091. | 3.84 | 1.92 | 1.92 | 28. | .50 |
| Pyritic | 5.0 + | 163821. | 7.39 | 3.94 | 3.44 | 57. | .20 |
| Quartzitic | 4.0-5.0 | 482793. | 3.92 | 1.61 | 2.31 | 25. | .48 |
| Quartzitic | 5.0 + | 910336. | 5.69 | 2.30 | 3.38 | 31. | .53 |

Strip Ratio 2.9

TABLE 5.1-7
VANDORDA PIT RESERVES



CURRAGH RESOURCES

VANGORDA WASTE DUMP
GENERAL AREA

DR. BY: CGY

NOV. /86

5.1 - 2

0 100 200 300 400 500 METRES

5.2 Grum Pit

5.2.1 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).

5.2.1.1 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.

5.2.1.2 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 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.

5.2.2 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.

5.2.2.1 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.

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

Curragh Resources
Grum 8606 Geological Model

SOFTWARE BY GEMCON SERVICES INC
MODULE 1.04
PAGE 1

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

MINING COST DATA

REC STAT ROCK DESCRIPTION
CODE

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 B 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 5.2-1

VOLUMETRIC MINING COSTS - GRUM PIT

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

Curragh Resources
Grum 8606 Geological Model

SOFTWARE BY GEMCOM SERVICES INC
MODULE 1.06
PAGE 1

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

DETAILED PRINTOUT FOR RECORD [1]

DATE CAPTURED : 4/11/1986

RECORD DESCRIPTION : November 86 Model

GENERAL COST INFORMATION

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

HAULAGE INFORMATION

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

TABLE 5.2--2

ORE BASED COSTS AND HAULAGE COSTS - GRUM PIT

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 = $0.181 \times \ln (\text{Pb head grade, \%}) + 0.767$

Zn factor = $0.162 \times \ln (\text{Zn head grade, \%}) + 0.733$

Ag factor = $0.18 \ln (\text{Wt \% to Pb conc.}) + .733$

Au factor = $0.18 \times \ln (\text{Wt \% to Pb conc.}) + .733$

Table 5.2-3 shows the calculation of revenue estimates for entry into the PC-MINE system; tables 5.2-4 and 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 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 stabilization 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

Revenue Calculations Grum Pit

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

| | | | |
|--------|----------|--------|---------|
| | | | Smelter |
| | \$ US | \$ CDN | 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 5.2 - 3

REVENUE CALCULATIONS - GRUM PIT

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

Curragh Resources
Grum 8606 Geological Model

SOFTWARE BY GEMCOM SERVICES INC
MODULE 1.04
PAGE 1

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

METALLURGICAL DATA

ROCK DESCRIPTION
CODE

TYPE PRIMARY CUT OFF GRADES RECOVERIES (PERCENT)
MINERAL 0 - S/P S/P - W %Pb+Zn %Pb %Zn Ag g/tAu g/t

| | | | | | | | | | |
|------------------------------------|------------|-------|-------|----|------|------|------|------|--|
| 0 Air | air | | | | | | | | |
| 1 A Type / 2nd code A, A4, C, E, L | ore %Pb+Zn | 5.000 | 4.000 | .0 | 80.0 | 83.0 | 65.0 | 33.0 | |
| 2 A4 Type | ore %Pb+Zn | 5.000 | 4.000 | .0 | 80.0 | 83.0 | 65.0 | 33.0 | |
| 3 C Type | ore %Pb+Zn | 5.000 | 4.000 | .0 | 80.0 | 83.0 | 65.0 | 33.0 | |
| 4 D Type | ore %Pb+Zn | 5.000 | 4.000 | .0 | 80.0 | 83.0 | 65.0 | 33.0 | |
| 5 E Type | ore %Pb+Zn | 5.000 | 4.000 | .0 | 80.0 | 83.0 | 64.0 | 33.0 | |
| 6 E4 Type | ore %Pb+Zn | 5.000 | 4.000 | .0 | 80.0 | 83.0 | 65.0 | 33.0 | |
| 7 G Type | ore %Pb+Zn | 5.000 | 4.000 | .0 | 80.0 | 83.0 | 65.0 | 33.0 | |
| 8 H Type | ore %Pb+Zn | 5.000 | 4.000 | .0 | 80.0 | 83.0 | 65.0 | 33.0 | |
| 9 L Type | ore %Pb+Zn | 5.000 | 4.000 | .0 | 80.0 | 83.0 | 65.0 | 33.0 | |
| 10 Waste | waste | | | | | | | | |
| 11 Unconsolidated Overburden | waste | | | | | | | | |
| 12 Partially Above Topography | waste | | | | | | | | |

TABLE 5.2 -4

METALLURGICAL PARAMETERS - GRUM PIT

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

Curragh Resources
Grum 6606 Geological Model

SOFTWARE BY GEMCOM SERVICES INC
MODULE 1.06
PAGE 2

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)

DESCRIPTION : Lead Revenue

| POINT | HEAD GRADE
[%Pb+Zn] | REVENUE
[CDN \$ / %Pb+Zn] |
|-------|--------------------------|--------------------------------|
|-------|--------------------------|--------------------------------|

| | | |
|---|---------|------|
| 1 | .000 | .000 |
| 2 | 100.000 | .000 |

| 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

DESCRIPTION : Silver 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 |

| POINT | HEAD GRADE
[Ag g/t] | REVENUE
[CDN \$ / Ag g/t] |
|-------|--------------------------|--------------------------------|
|-------|--------------------------|--------------------------------|

| | | |
|---|---------|--------|
| 1 | .000 | .000 |
| 2 | 300.000 | 70.059 |

TABLE 5.244 5

REVENUE/HEAD GRADE INFORMATION - GRUM PIT

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

Curragh Resources
Grum 8606 Geological Model

SOFTWARE BY GEMCOM SERVICES INC
MODULE 1.06
PAGE 3

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

DETAILED PRINTOUT FOR RECORD [1]

DESCRIPTION : Gold Revenue

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

TABLE 5.2-45 (CONTINUED)

REVENUE/HEAD GRADE INFORMATION - GRUM PIT

PC-MINE VERSION 1.10
 SERIAL NO : 20320
 29/11/1986

Curragh Resources
 Grun 8606 Geological Model

SOFTWARE BY GENCON SERVICES INC
 MODULE 3.06
 PAGE 1

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

DRE-STOCKPILE CUT-OFF GRADE : 5.000 [ZPb+Zn]
 STOCKPILE-WASTE CUT-OFF GRADE : 4.000 [ZPb+Zn]

| LABEL: | UNITS: | GRADE: | REVENUE: | RECOVERY: | BLOCK REVENUE: | CUM. REVENUE: |
|--------|--------|--------|----------|-----------|----------------|---------------|
| 1 | ZPb+Zn | 12.58 | .00 | .00 | .00 | .00 |
| 2 | ZPb | 4.47 | 5.58 | 80.00 | 9361.47 | 9361.47 |
| 3 | ZZn | 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 |

DRE 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 DRE BASED MINING COST : 22969.03 -

TOTAL BLOCK MINING COST : 26158.61

BLOCK ECONOMIC VALUE : 109670.40

TABLE 5.2 - 6
 Sample Calculation

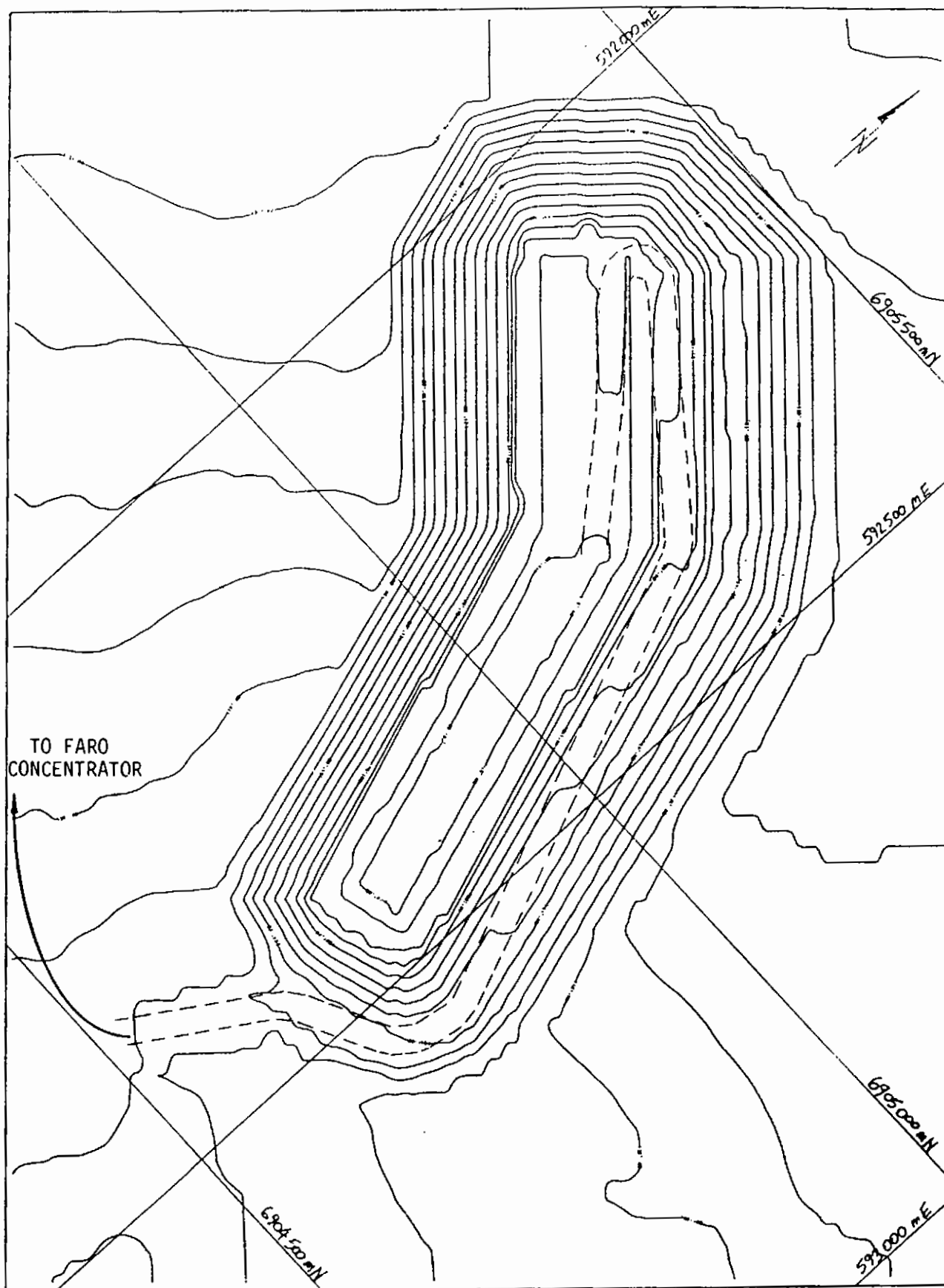


FIGURE 5.2 - 1
GRUM STAGE ONE PIT

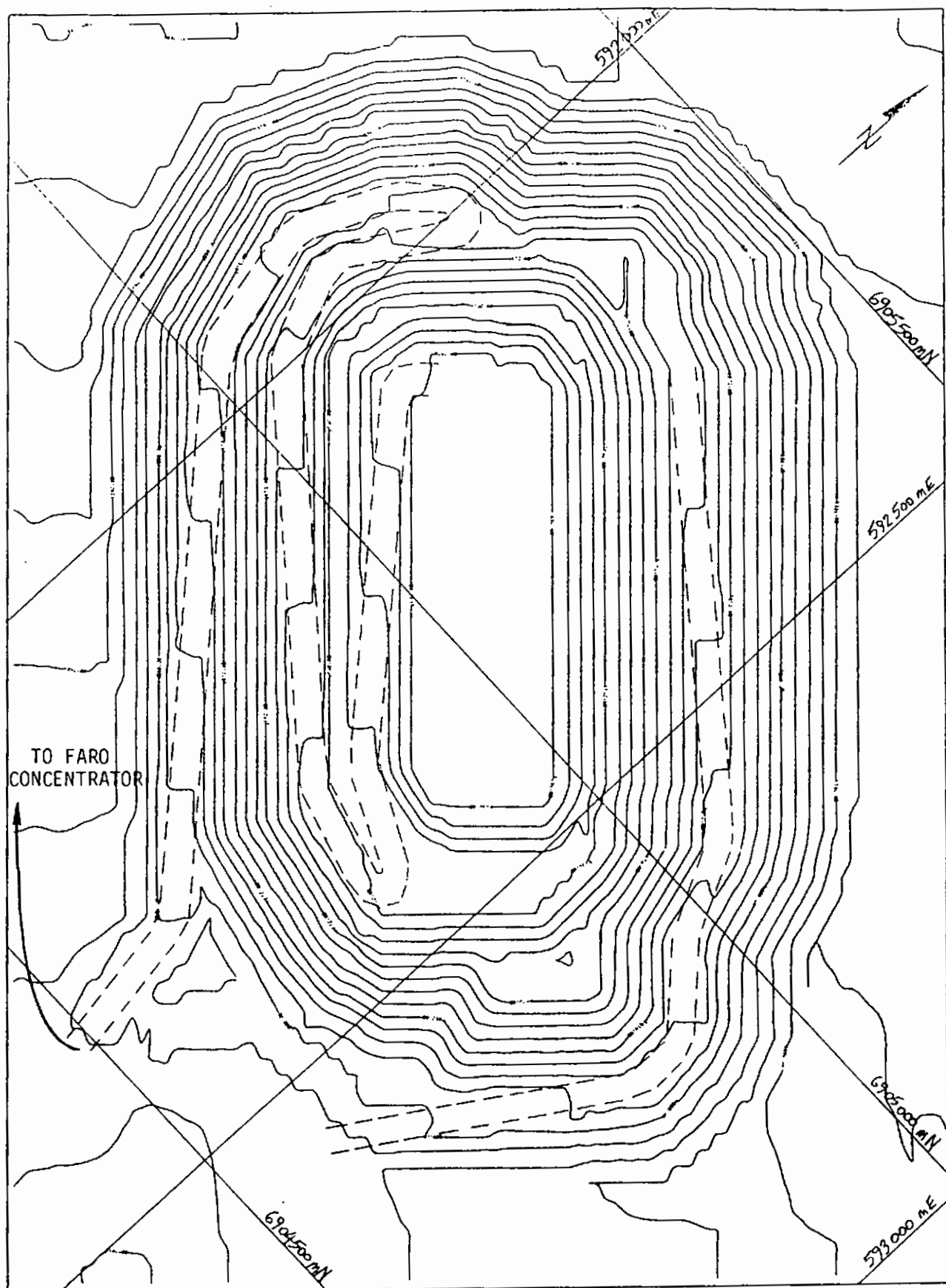


FIGURE 5.2 - 2
GRUM STAGE TWO PIT

as possible, the pit has been designed in three stages.

The first stage of the pit, shown in figure 5.2-1, 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 5.2-2. 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 5.2-3.

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

5.1.4 Pit Reserves

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

Mining reserves are adjusted to reflect a 95% mining recovery and a 15% waste dilution. Recovery is defined as the tonnes of ore mined, expressed as a percentage of the in-situ tonnes. Dilution is defined as the tonnes of waste (at zero grade) mined with the ore and reported as ore, expressed as a percentage of recovered ore tonnes.

Adjustments to the ore quantities are as follows:

| | |
|----------------------|--|
| Recovered ore tonnes | = (mining recovery)x(in-situ ore tonnes) |
| Dilution tonnes | = (recovered ore tonnes)x(dilution) |
| Mined ore tonnes | = Recovered ore tonnes + dilution tonnes |
| | = (recovered ore tonnes)x(1 + dilution) |
| | = (in-situ ore tonnes)x(recovery)x(1 + dilution) |

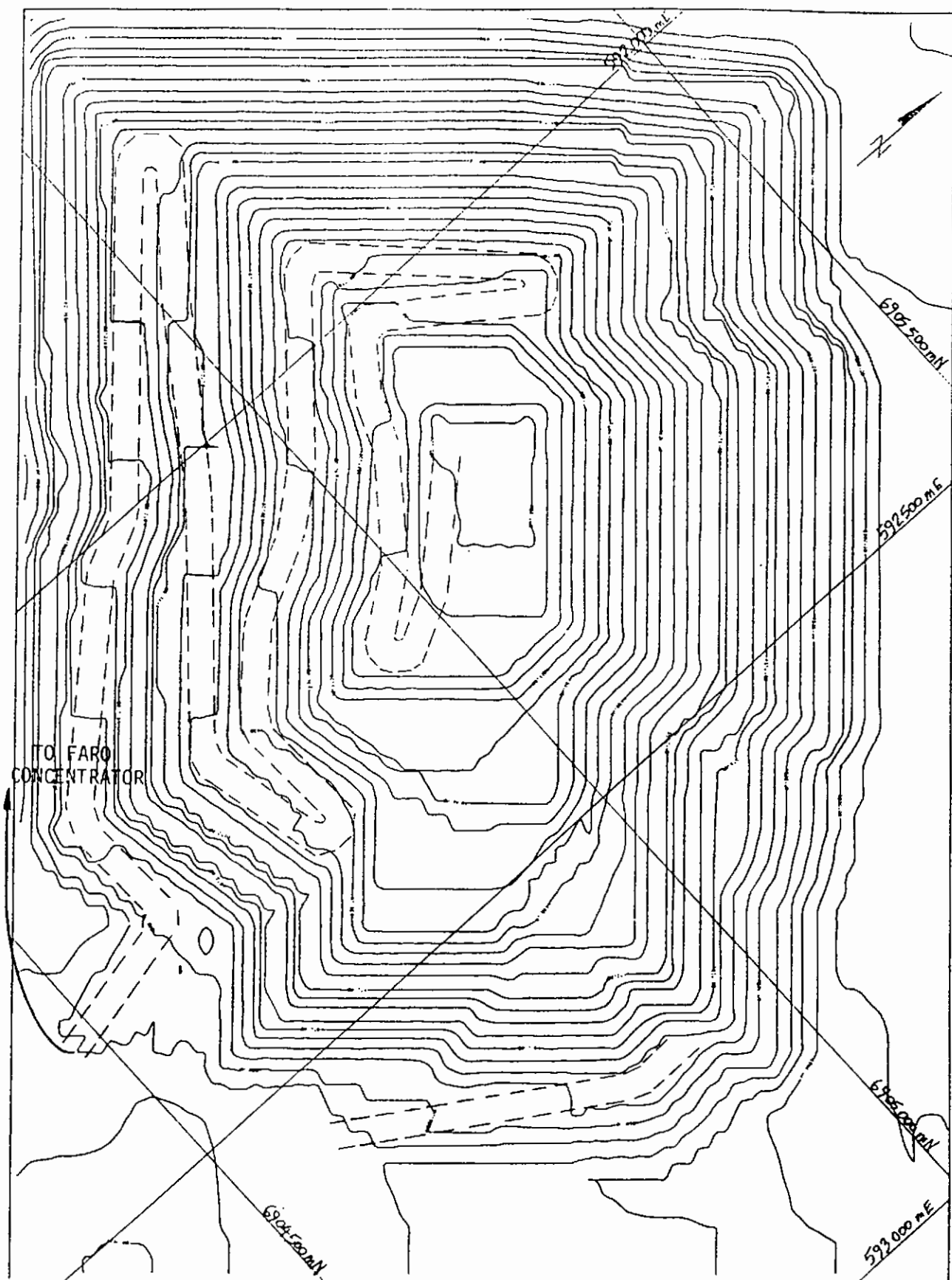


FIGURE 5.2 - 3

GRUM STAGE THREE PIT

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 |
|----------------|---------|----------|------|------|--------|--------|
| - 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 |
|----------------|--------|----------|------|------|--------|--------|
| - 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 |
|----------------|--------|----------|------|------|--------|--------|
| - 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 5.2 - 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 5.2 - 7 (CONTINUED)

Mining Reserves
Grum Total Pit

Tonnes

| | |
|----------------|-----------|
| Waste Rock | 194942010 |
| Overburden | 31580416 |
| Sulphide Waste | 4139521 |
| All Waste | 230661947 |

| | | %(Pb+Zn) | % Pb | % Zn | Ag g/t | Au g/t |
|------------------|----------|----------|------|------|--------|--------|
| 4 - 5 % combined | 3253263 | 3.89 | 1.45 | 2.44 | 26 | 0.61 |
| + 5 % combined | 21740126 | 8.55 | 3.17 | 5.38 | 54 | 0.84 |
| Strip Ratio | 9.2 | | | | | |

TABLE 5.2 - 7 (CONTINUED)

The dilution is assumed to contain zero metal:

Recovered ore grade = in-situ ore grade

Dilution grade = zero

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 5.2-7 details the mining reserves for each stage of the pit.

5.1.5 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.

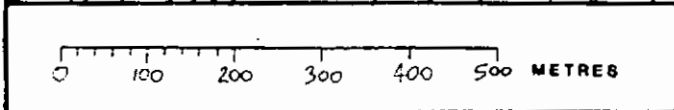
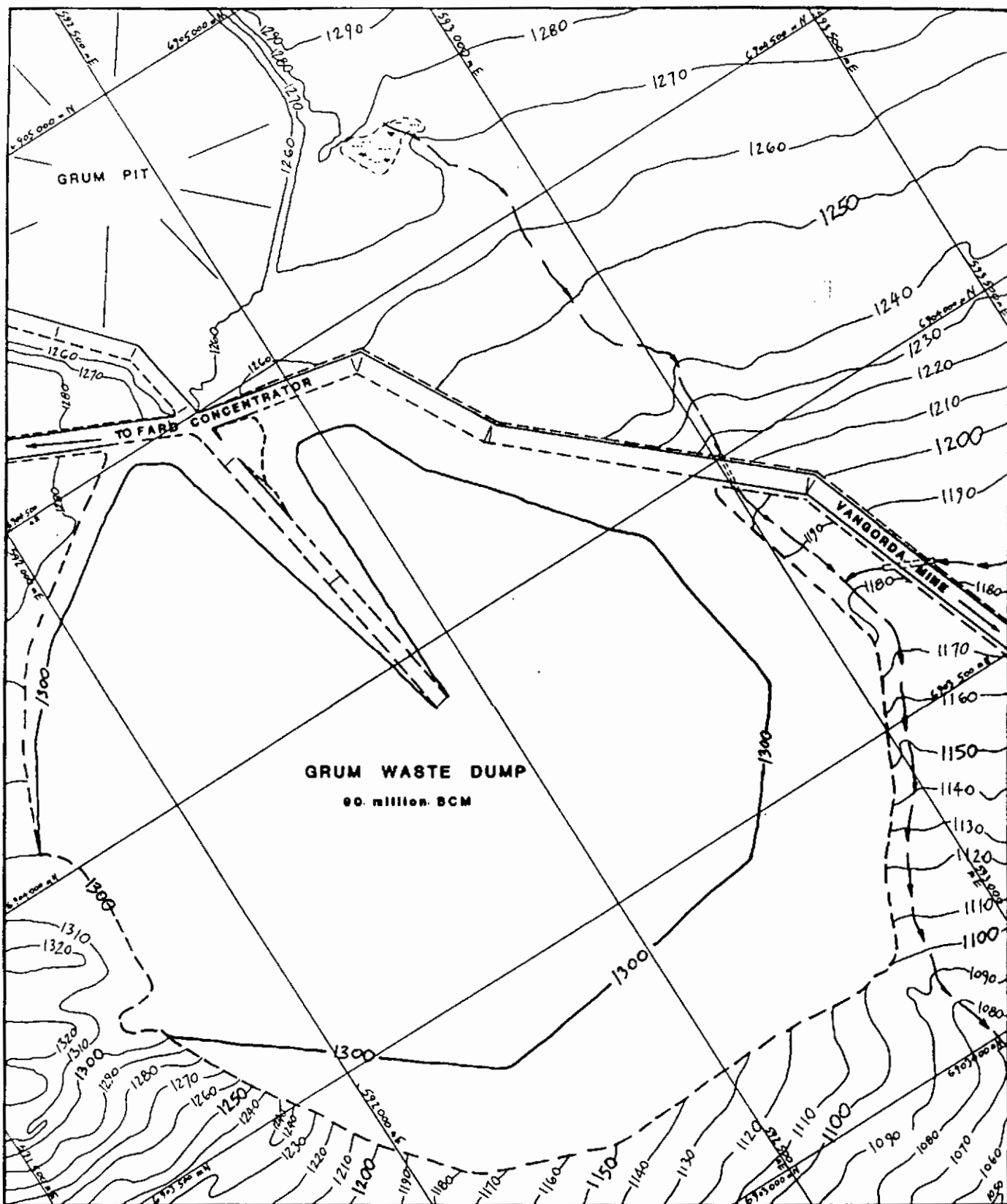
The dump will be constructed in a small valley south of the pit initially at the 1260 metre elevation, later being built up to the 1300 metre elevation.

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 5.1-5.



| | | |
|--------------------------|--|-------------|
| CURRAGH RESOURCES | | DR. BY: CGY |
| GRUM WASTE DUMP | | NOV./86 |
| GENERAL AREA | | 5.2-4 |

6.0 INFRASTRUCTURE

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.

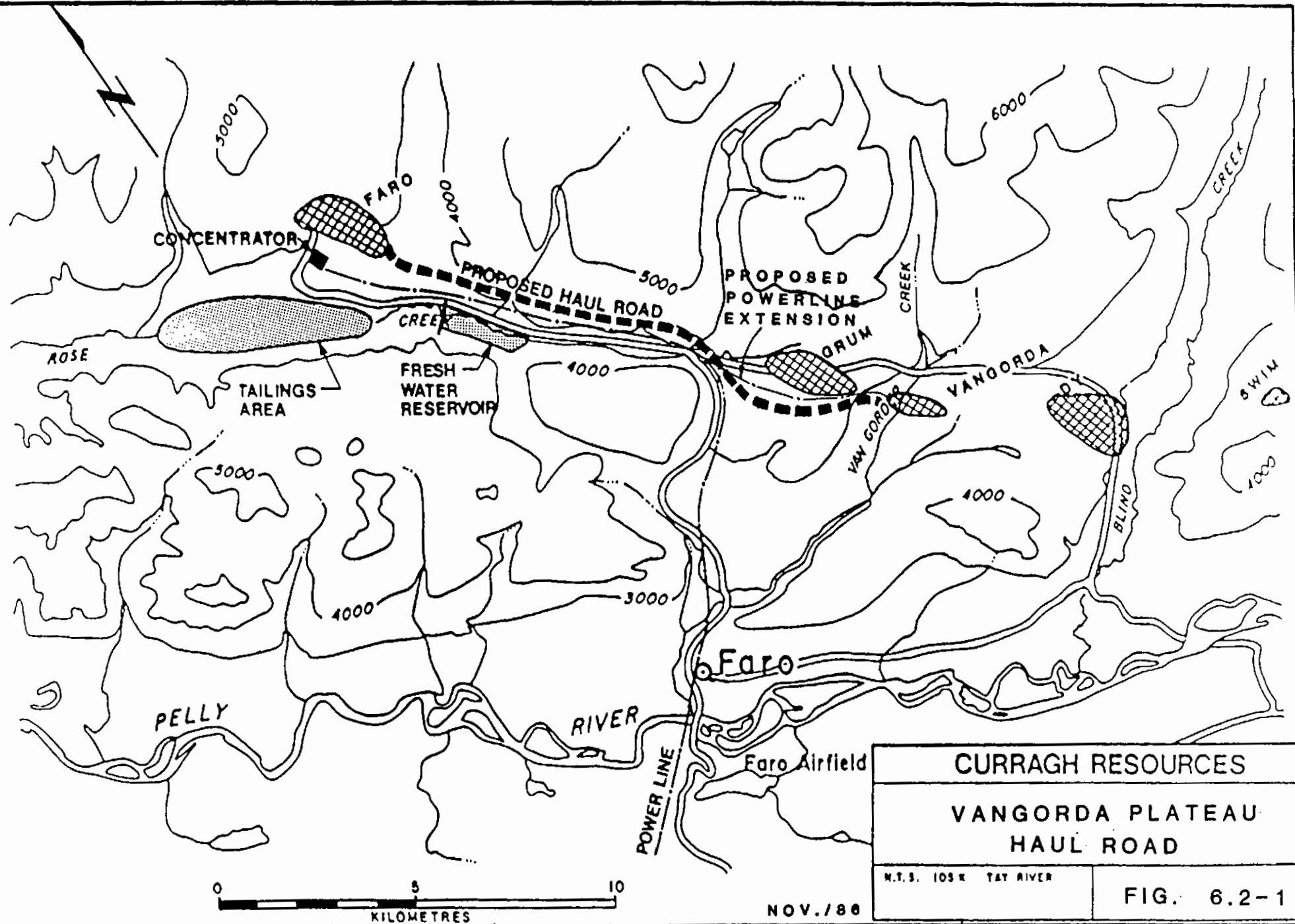
Construction of the haul road has already begun. Beginning in October 1986 waste rock from the pit is being used to construct the first stage of the haul road; the crossing of the North Fork of Rose Creek. Design of the rock drain and creek crossing at this point was done by Golder Associates in September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

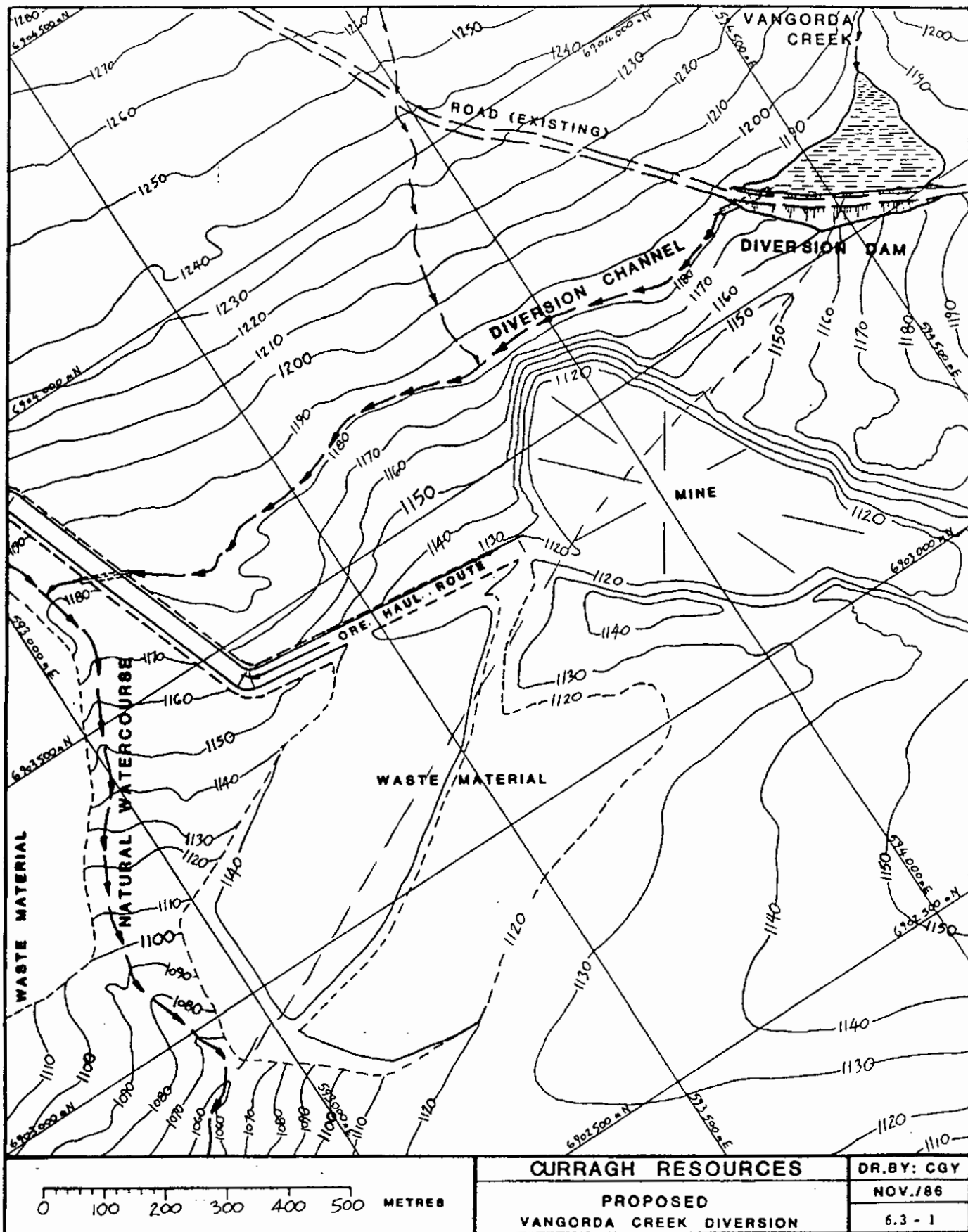
Alignment of the haul road is essentially that developed by Stanley Associates for CAMC in December 1980 (ref. 5). The first 3000 metres of the Stanley design has been realigned in order to reduce the adverse grade on that portion of the haul road. Also, the last 3000 metres will be realigned to bring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





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

GOLDER DESIGN 820m

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 m³/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.

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 Summary

The total pit production summary is given in Table 7.1-1.

This schedule shows a very high strip ratio and low ore production in the years 1993 and 1994. It is expected that the recommended redesign of the Grum ultimate pit and its stages will give a more consistent ore release through this period.

Operating equipment is summarized in Table 7.1-2.

Pit Production Summary: Grum plus Vangorda

| | 1988 | 1989 | 1990 | 1991 | 1992 | 1993 | 1994 | 1995 | 1996 | 1997 | 1998 | 1999 | Total |
|------------------------|---------|----------|----------|----------|----------|----------|----------|----------|----------|----------|----------|------|-----------|
| Waste rock | 1286331 | 11532924 | 12044170 | 17556870 | 15261520 | 22145240 | 30859340 | 27787600 | 26121767 | 31323420 | 7616420 | | 203533602 |
| Overburden | 5857089 | 10544202 | 5351051 | 2384716 | 5351719 | 6451748 | 2082219 | 1731284 | 826249 | 69507 | 0 | | 40649784 |
| Sulphide Waste | 274726 | 863116 | 2704810 | 710510 | 525661 | 6022 | 147937 | 471145 | 1220875 | 475840 | 563253 | | 7963895 |
| All Waste | 7418146 | 22940242 | 20100031 | 20652096 | 21138900 | 28603010 | 33089496 | 29990029 | 28168891 | 31868767 | 8179673 | | 252149281 |
| + 5 % tonnes | 501642 | 1151052 | 4441940 | 3540616 | 3451647 | 74225 | 621260 | 3435351 | 4828997 | 1755428 | 3762671 | | 27564829 |
| 1Pb+Zn | 7.31 | 7.93 | 8.62 | 7.26 | 7.36 | 12.49 | 10.87 | 9.15 | 9.48 | 8.32 | 8.99 | | 8.54 |
| 1Pb | 3.12 | 3.36 | 3.76 | 2.60 | 2.63 | 5.17 | 4.43 | 3.47 | 3.47 | 3.29 | 3.35 | | 3.29 |
| 1Zn | 4.19 | 4.57 | 4.86 | 4.66 | 4.73 | 7.32 | 6.44 | 5.68 | 6.01 | 5.03 | 5.64 | | 5.26 |
| Ag g/t | 44 | 49 | 53 | 42 | 43 | 87 | 73 | 58 | 59 | 54 | 59 | | 53 |
| Au g/t | 0.49 | 0.57 | 0.62 | 0.63 | 0.68 | 0.75 | 0.87 | 0.88 | 0.96 | 0.79 | 1.01 | | 0.79 |
| Concentrate Equivalent | | | | | | | | | | | | | |
| Pb Conc DMT | 24159 | 60178 | 264825 | 117074 | 115608 | 5560 | 38549 | 160408 | 224984 | 76698 | 167253 | | 1255296 |
| 1Pb | 53.06 | 53.34 | 53.43 | 60.00 | 60.00 | 60.00 | 60.00 | 60.00 | 60.00 | 60.00 | 60.00 | | 58.16 |
| Ag g/t | 608 | 622 | 606 | 800 | 812 | 840 | 817 | 826 | 844 | 813 | 876 | | 771 |
| Au g/t | 4.04 | 4.34 | 3.95 | 5.97 | 6.40 | 3.57 | 4.85 | 6.22 | 6.86 | 5.91 | 7.51 | | 5.82 |
| 1Zn Conc DMT | 29647 | 75649 | 313209 | 246695 | 245643 | 8695 | 63151 | 303740 | 453999 | 134456 | 326840 | | 2201724 |
| 1Zn | 54.90 | 54.72 | 54.71 | 55.00 | 55.00 | 55.00 | 55.00 | 55.00 | 55.00 | 55.00 | 55.00 | | 54.95 |
| 1 - 5 % tonnes | 98574 | 143439 | 445322 | 794575 | 396738 | | 137270 | 422650 | 850135 | 223828 | 374795 | | 3887326 |
| 1Pb+Zn | 3.86 | 3.90 | 3.89 | 3.89 | 3.97 | | 4.06 | 3.88 | 3.87 | 3.84 | 3.88 | | 3.90 |
| 1Pb | 1.60 | 1.64 | 1.66 | 1.28 | 1.29 | | 1.73 | 1.53 | 1.54 | 1.55 | 1.55 | | 1.49 |
| 1Zn | 2.26 | 2.26 | 2.23 | 2.61 | 2.68 | | 2.33 | 2.35 | 2.33 | 2.29 | 2.33 | | 2.41 |
| Ag g/t | 27 | 29 | 24 | 25 | 24 | | 31 | 27 | 26 | 30 | 27 | | 26 |
| Au g/t | 0.35 | 0.45 | 0.62 | 0.46 | 0.45 | | 0.82 | 0.63 | 0.64 | 0.76 | 0.79 | | 0.59 |
| Concentrate Equivalent | | | | | | | | | | | | | |
| Pb Conc DMT | 1984 | 2876 | 9212 | 10983 | 5546 | | 2763 | 7281 | 14838 | 3918 | 6587 | | 65988 |
| 1Pb | 54.23 | 56.51 | 55.62 | 60.00 | 60.00 | | 60.00 | 60.00 | 60.00 | 60.00 | 60.00 | | 59.06 |
| Ag g/t | 616 | 664 | 549 | 912 | 871 | | 864 | 834 | 816 | 917 | 842 | | 799 |
| Au g/t | 3.44 | 3.05 | 6.64 | 8.76 | 8.42 | | 11.67 | 10.11 | 10.02 | 12.05 | 12.39 | | 9.14 |
| 1Zn Conc DMT | 2828 | 4151 | 12744 | 27870 | 14341 | | 4208 | 13079 | 25985 | 6714 | 11461 | | 123381 |
| 1Zn | 52.45 | 52.69 | 53.06 | 55.00 | 55.00 | | 55.00 | 55.00 | 55.00 | 55.00 | 55.00 | | 54.66 |
| Total Waste Mined | 7418146 | 22940242 | 20100031 | 20652096 | 21138900 | 28603010 | 33089496 | 29990029 | 28168891 | 31868767 | 8179673 | | 252149281 |
| Total Ore Mined | 600216 | 1294491 | 4887262 | 4335191 | 3846385 | 74225 | 758530 | 3858001 | 5679132 | 1979256 | 4137466 | | 31452155 |
| Total Tonnes Mined | 8018362 | 24234733 | 24987293 | 24987287 | 24987285 | 28677235 | 33848026 | 33848030 | 33848023 | 33848023 | 12317139 | | 283601436 |
| Strip Ratio | 12.36 | 17.72 | 4.11 | 4.76 | 5.49 | 385.36 | 43.62 | 7.77 | 4.96 | 16.10 | 1.98 | | 8.02 |

TABLE 7.1 - 1
PIT PRODUCTION SUMMARY

7.2 Production Scheduling Parameters

Equipment productivity parameters are based on those currently being experienced at the Faro operation.

The mining schedule is driven by shovel productivity and truck requirements are then determined as a consequence of the amount of material moved.

Shovel productivity rates are detailed in Table 7.2-1.

Ore production targets are based on a maximum concentrator feed rate of 11,000 tonnes per day or a maximum concentrate production rate of 1,600 tonnes per day.

Pit production schedules were produced under the following guidelines:

1. Ore production to supplement Faro ore production so that concentrator targets are met.
2. Pre-production waste stripping is minimized.
3. Production to come from equipment surplus to the Faro pit operations; shovel release dates are as follows:

| | |
|---------------|--------------------|
| Marion shovel | - January 1, 1988 |
| P & H 2100 | - December 1, 1988 |
| P & H 2100 | - February 1, 1989 |
| P & H 2100 | - August 1, 1993 |

As a consequence of these guidelines, production plans for the Vangorda Plateau orebodies are closely linked to the Faro pit production plans. The Faro Pit development plan is described in Appendix D of this report.

Operating Equipment - Grum and Vangorda Pits

| | 1988 | 1989 | 1990 | 1991 | 1992 | 1993 | 1994 | 1995 | 1996 | 1997 | 1998 | 1999 |
|-------------------------|------|------|------|------|------|------|------|------|------|------|------|------|
| Production Equipment: | | | | | | | | | | | | |
| Shovel | 1.1 | 2.9 | 3 | 3 | 3 | 3.4 | 4 | 4 | 4 | 4 | 1.5 | 0 |
| Drill | 0.1 | 0.7 | 1.0 | 1.1 | 1.0 | 1.1 | 1.6 | 1.6 | 1.7 | 1.7 | 0.6 | 0.0 |
| Haul Truck | 2.1 | 11.5 | 18.9 | 25.1 | 21.7 | 20.2 | 30.2 | 38.6 | 39.0 | 39.4 | 18.7 | 1.4 |
| Front End Loader - L800 | 0.0 | 0.5 | 1.9 | 1.7 | 0.9 | 1.0 | 0.5 | 1.9 | 1.9 | 1.9 | 1.9 | 0.6 |
| Support Equipment | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| Dozer - D8 w/ ripper | 0 | 1 | 1 | 1 | 1 | 2 | 2 | 2 | 2 | 2 | 1 | 0 |
| Dozer - D9 w/ ripper | 1.5 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 1 | 0 |
| Rubber Tired Dozer | 1 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 0 |
| Grader | | | | | | | | | | | | |

TABLE 7.1 - 2
OPERATING EQUIPMENT SUMMARY

Shovel Productivity

| | P & H | Marion |
|---------------------------|-----------|-----------|
| tonnes/manned hr (Avg) | 1400 | 1400 |
| 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 | 1016 | 833 |
| Tonnes/calendar day | 24,276 | 19,992 |
| Tonnes/365 day year | 8,860,740 | 7,297,080 |

Table 7.2-1
Shovel Productivity

7.3.1 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 7.3-1.

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 7.3-2.

1990

During 1990, the Vangorda Pit is to supply the total feed to the Faro concentrator. In order to meet the production requirements, 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 7.3-3.

Table 7.3-1 details the annual ore production for the Vangorda Pit.

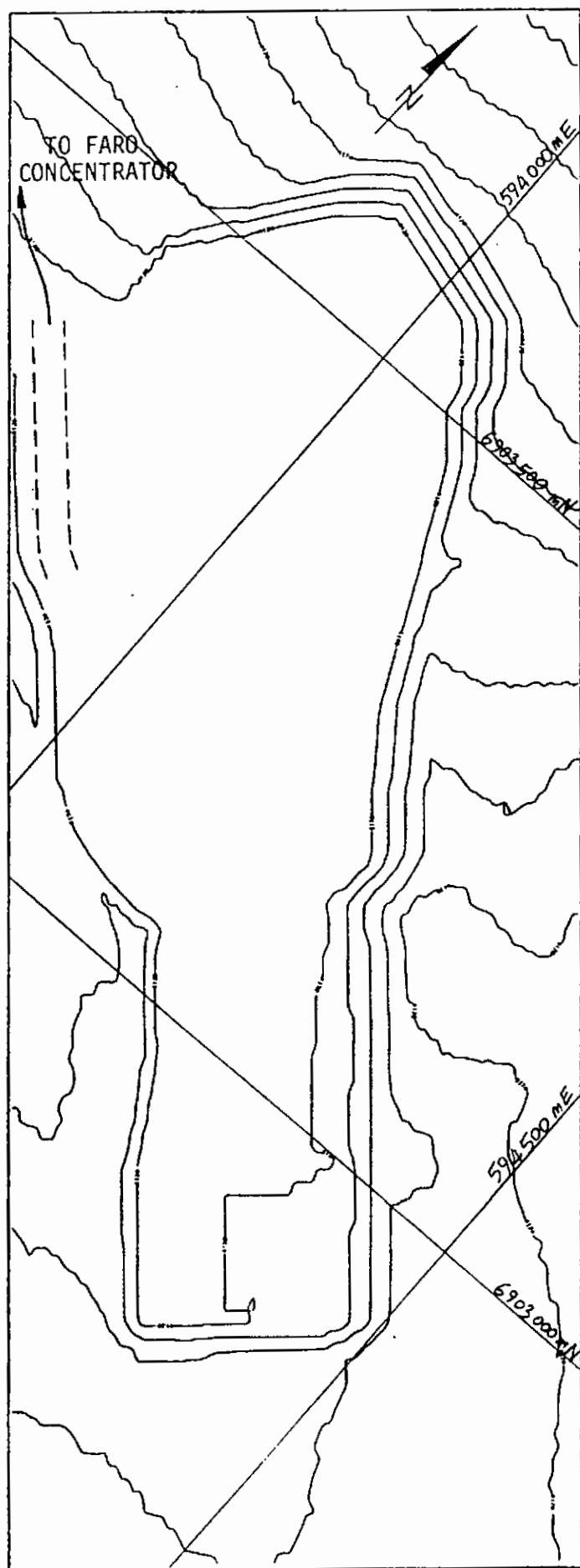


FIGURE 7.3 - 1
VANGORDA PIT END OF 1988

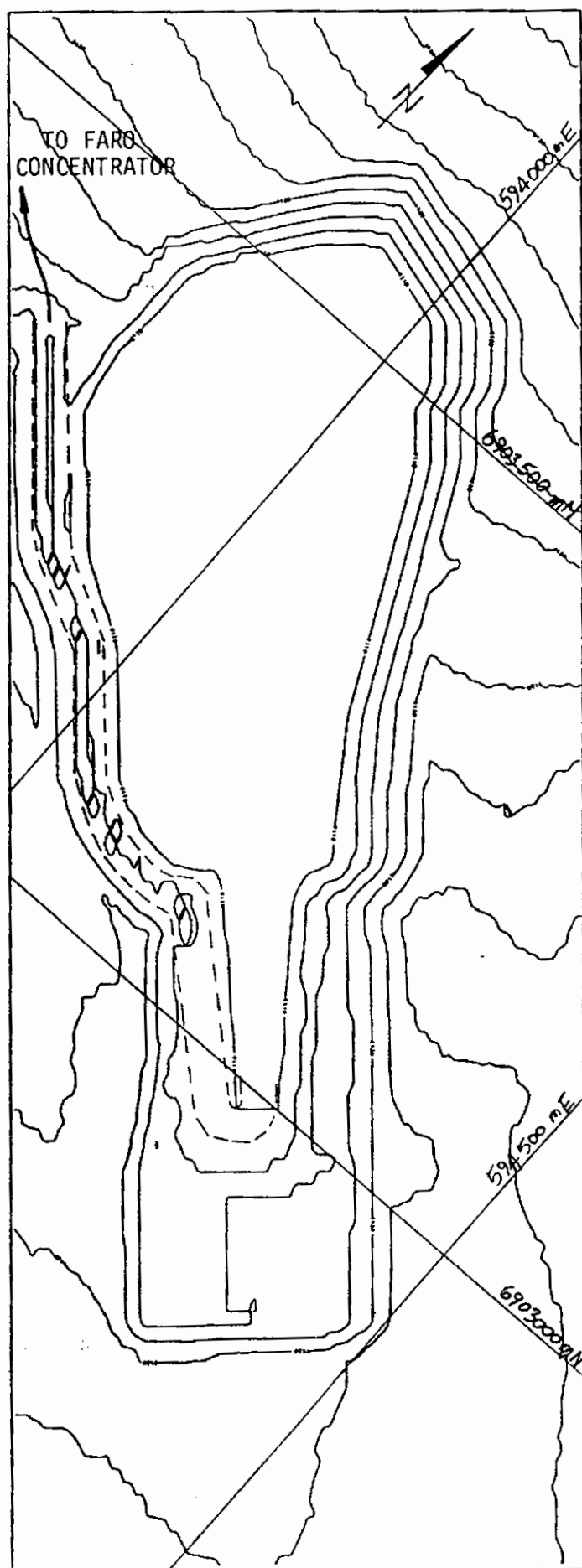


FIGURE 7.3 - 2
VANGORDA PIT END OF 1989

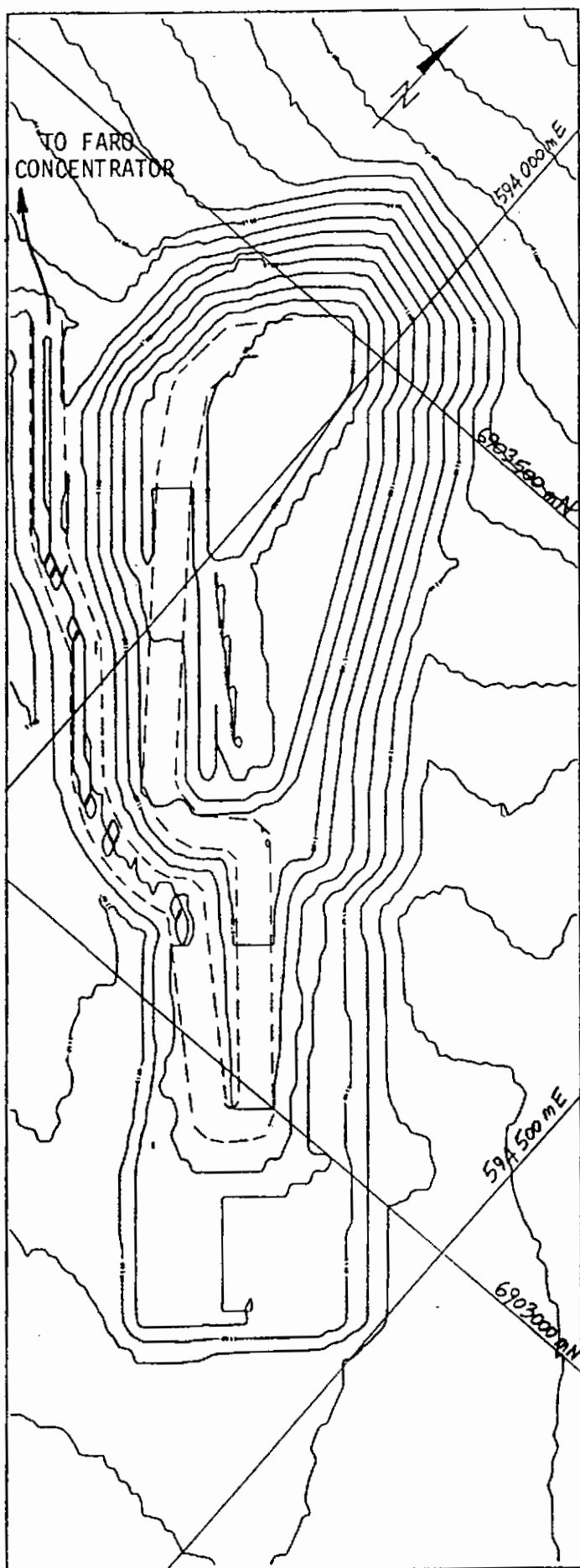


FIGURE 7.3 - 3
VANGORDA PIT END OF 1990

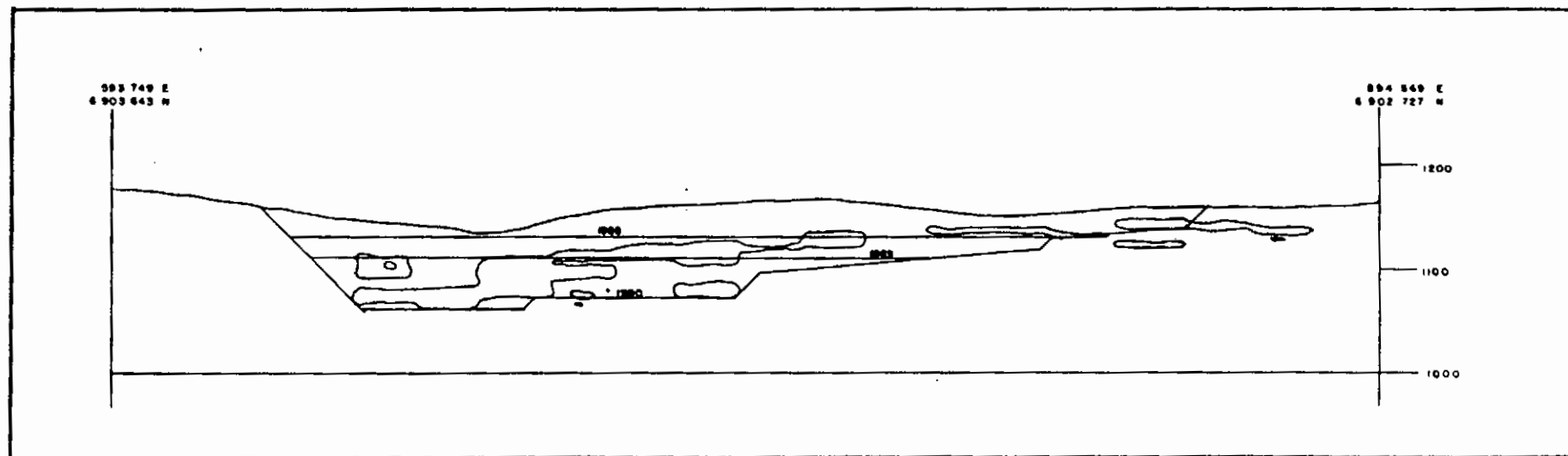


FIGURE 7.3 - 4
Longitudinal Section Through Vangorda Pit

Pit Production Summary: Vangorda

| | 1988 | 1989 | 1990 | 1991 | 1992 | 1993 | 1994 | 1995 | 1996 | 1997 | 1998 | 1999 | Total |
|------------------------|---------|---------|----------|------|------|------|------|------|------|------|------|------|----------|
| Waste rock | 1176277 | 1972163 | 3445180 | | | | | | | | | | 8593620 |
| Overburden | 5214587 | 3196395 | 638416 | | | | | | | | | | 9069398 |
| Sulphide Waste | 274726 | 860474 | 2689175 | | | | | | | | | | 3824375 |
| All Waste | 6665590 | 6029032 | 8792771 | | | | | | | | | | 21487393 |
| + 5 % tonnes | 501642 | 1093336 | 4229724 | | | | | | | | | | 5824702 |
| ZPb+Zn | 7.31 | 8.04 | 8.77 | | | | | | | | | | 8.51 |
| ZPb | 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 |
| ZPb | 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 |
| ZPb+Zn | 3.86 | 3.9 | 3.91 | | | | | | | | | | 3.90 |
| ZPb | 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 | 0.33 | 0.45 | 0.6 | | | | | | | | | | 0.53 |
| Concentrate Equivalent | | | | | | | | | | | | | |
| Pb Conc DMT | 1984 | 2876 | 8486 | | | | | | | | | | 13346 |
| ZPb | 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 7.3 - 1
VANGORDA PIT PRODUCTION

7.3.2 Equipment Schedule - Vangorda Pit

The operating equipment schedule for the Vangorda Pit is given in Table 7.3-2. Note that truck requirements have been determined separately and are presented in in Section 7.5. Note also the production equipment (shovels and drills) is surplus to the Faro Pit while support equipment is additional to the Faro Pit.

Operating Equipment - Vangorda Pit

| | 1988 | 1989 | 1990 | 1991 | 1992 | 1993 | 1994 | 1995 | 1996 | 1997 | 1998 | 1999 |
|--------------------------|------|------|------|------|------|------|------|------|------|------|------|------|
| | | | | | | | | | | | | |
| Production Equipment: | | | | | | | | | | | | |
| Shovel | 1 | 1 | 1.9 | | | | | | | | | |
| Drill | NA | NA | NA | | | | | | | | | |
| Haul Truck | NA | NA | NA | | | | | | | | | |
| Front End Loader - LB800 | NA | NA | NA | | | | | | | | | |
| Support Equipment | | | | | | | | | | | | |
| Dozer - D8 w/ ripper | | | 1 | | | | | | | | | |
| Dozer - D9 w/ ripper | 1 | 1 | 1 | | | | | | | | | |
| Rubber Tired Dozer | 1 | 1 | 1 | | | | | | | | | |
| Grader | 1 | 0.5 | 1 | | | | | | | | | |

TABLE 7.3 - 2
VANGORDA EQUIPMENT SCHEDULE

7.4.1 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 7.4-1.

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 7.4-2.

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 million tonnes of waste.

Pit status at the end of 1991 is shown in figure 7.4-3.

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 million tonnes of waste will be mined.

Pit status at the end of 1992 is shown in figure 7.4-4.

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 7.4-5.

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 7.4-6.

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 7.4-7.

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 7.4-8.

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 7.4-9.

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 7.4-10.

Tables 7.4-1 through 7.4-4 detail the annual production schedule for the Grum Pit.

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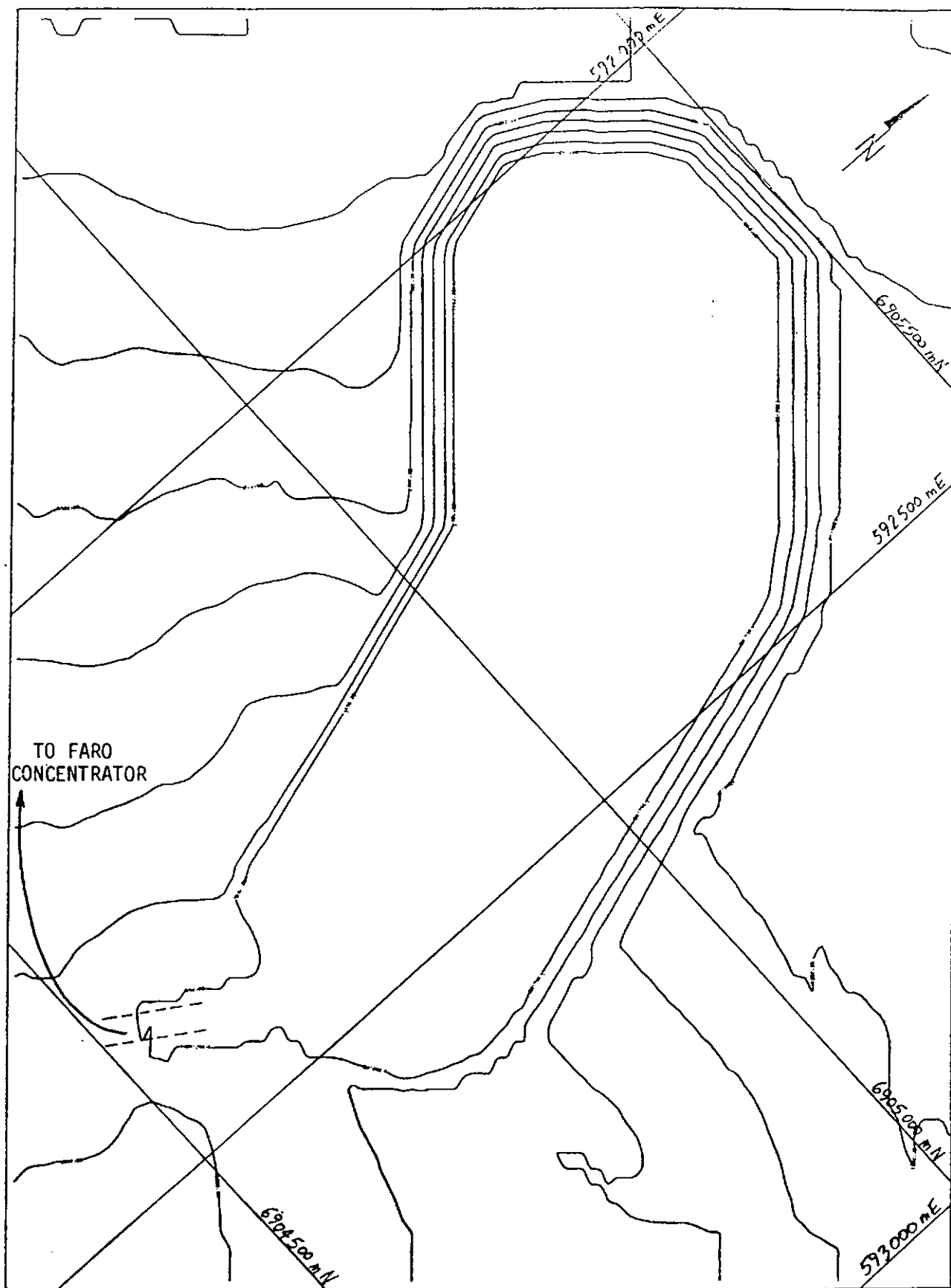


FIGURE 7.4 - 1
GRUM PIT END OF 1989

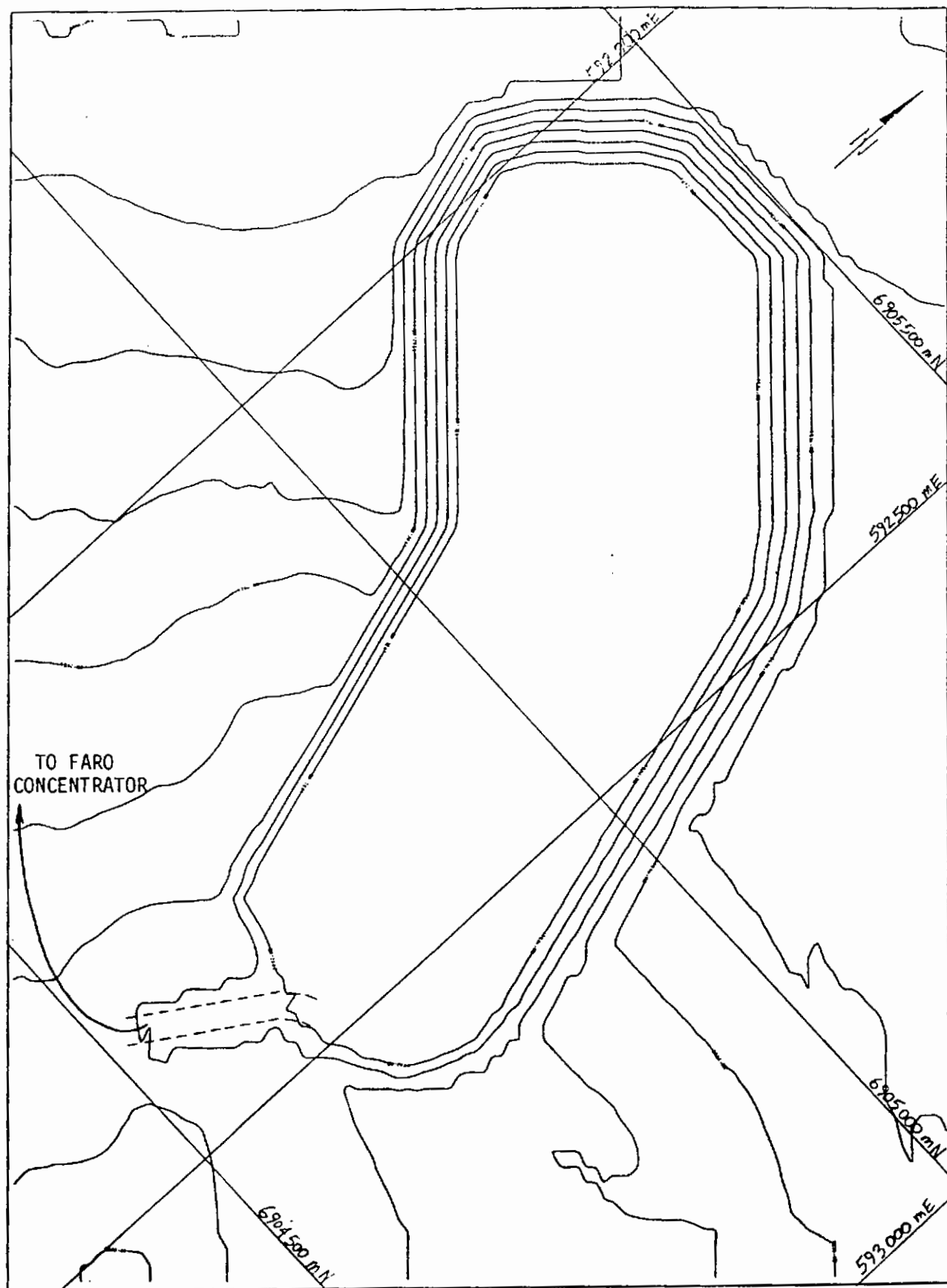


FIGURE 7.4 - 2
GRUM PIT END OF 1990

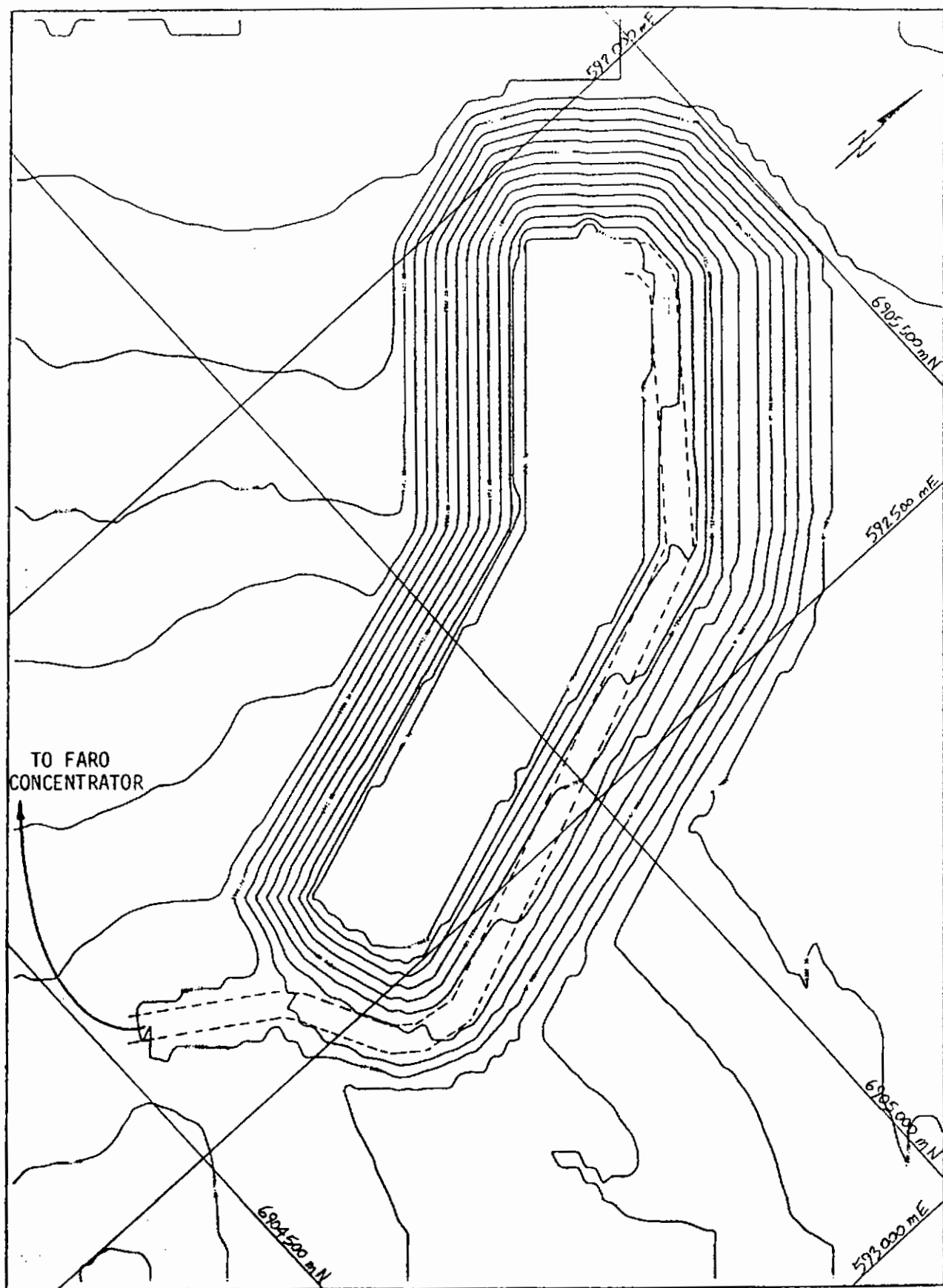


FIGURE 7.4 - 3

GRUM PIT END OF 1991

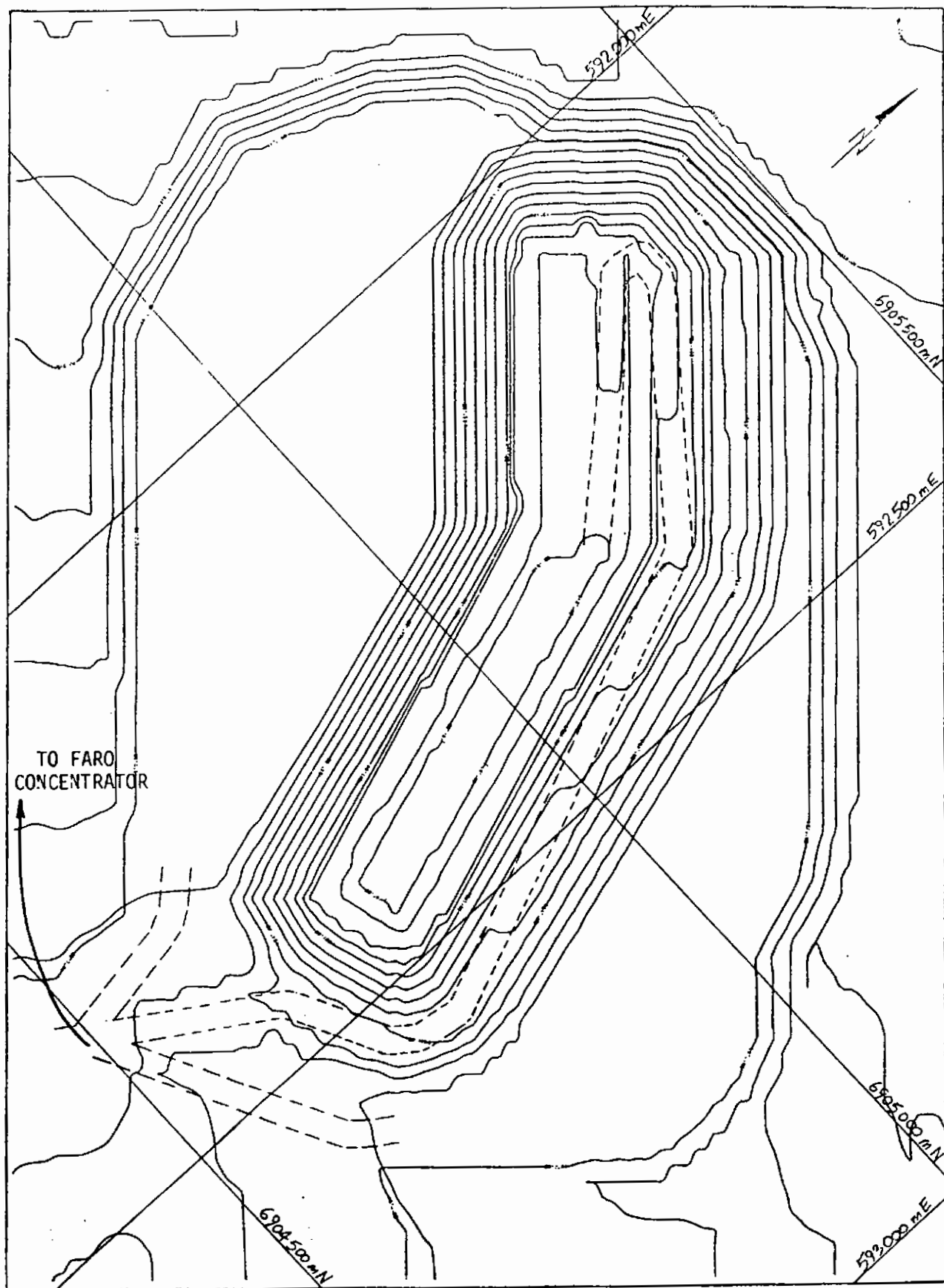


FIGURE 7.4 - 4

GRUM PIT END OF 1992

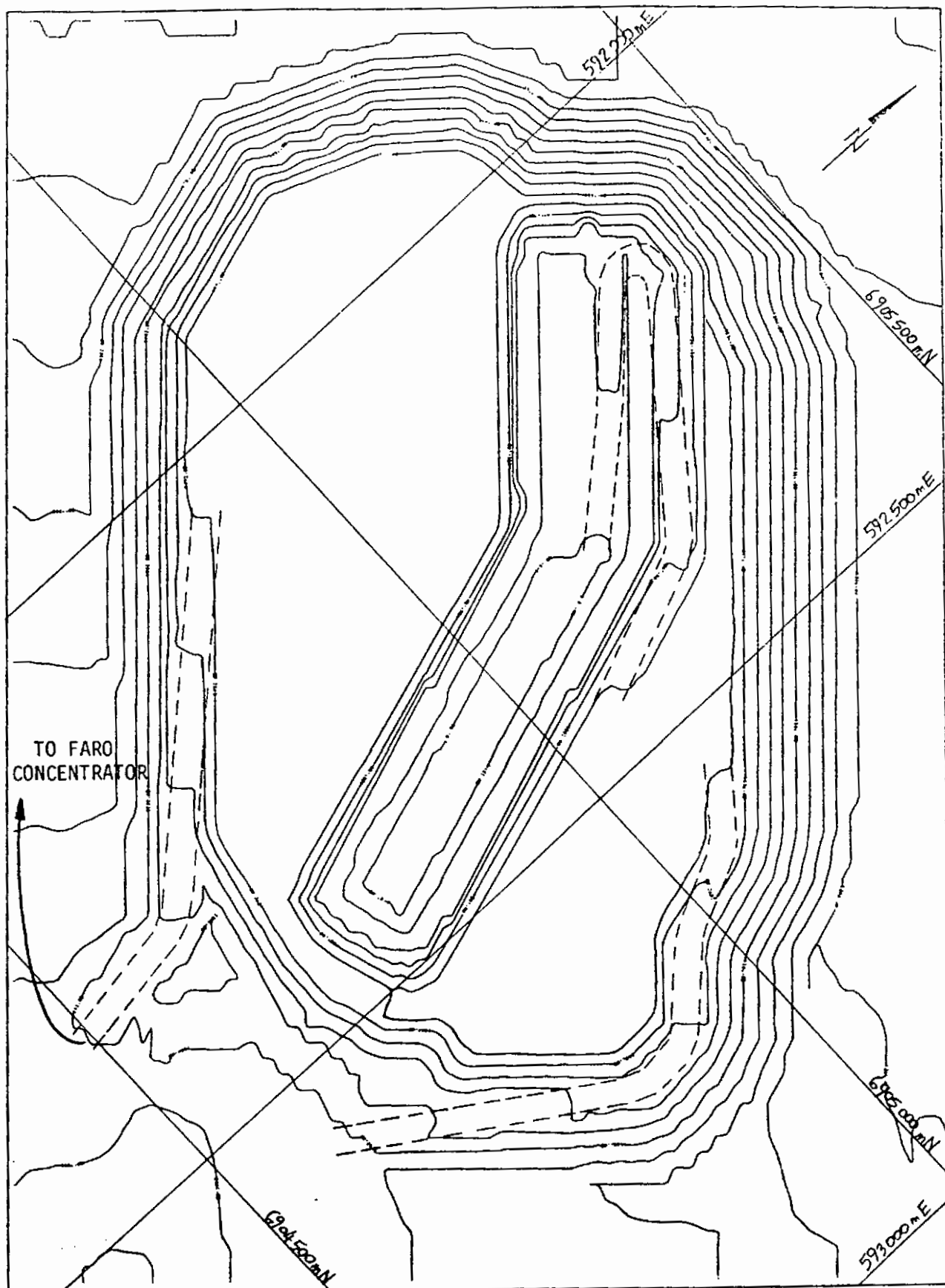


FIGURE 7.4 - 5

GRUM PIT END OF 1993

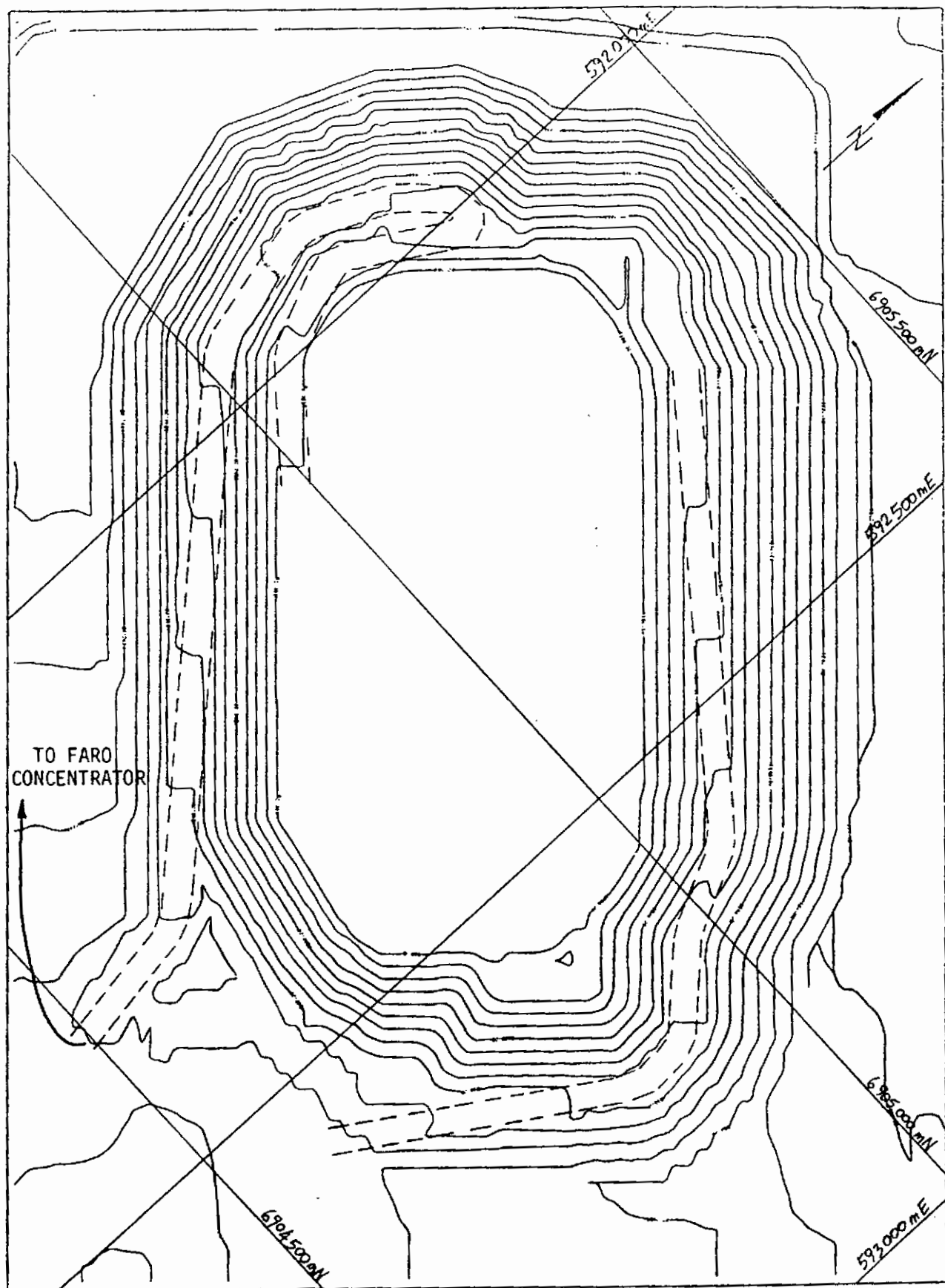


FIGURE 7.4 - 6
GRUM PIT END OF 1994

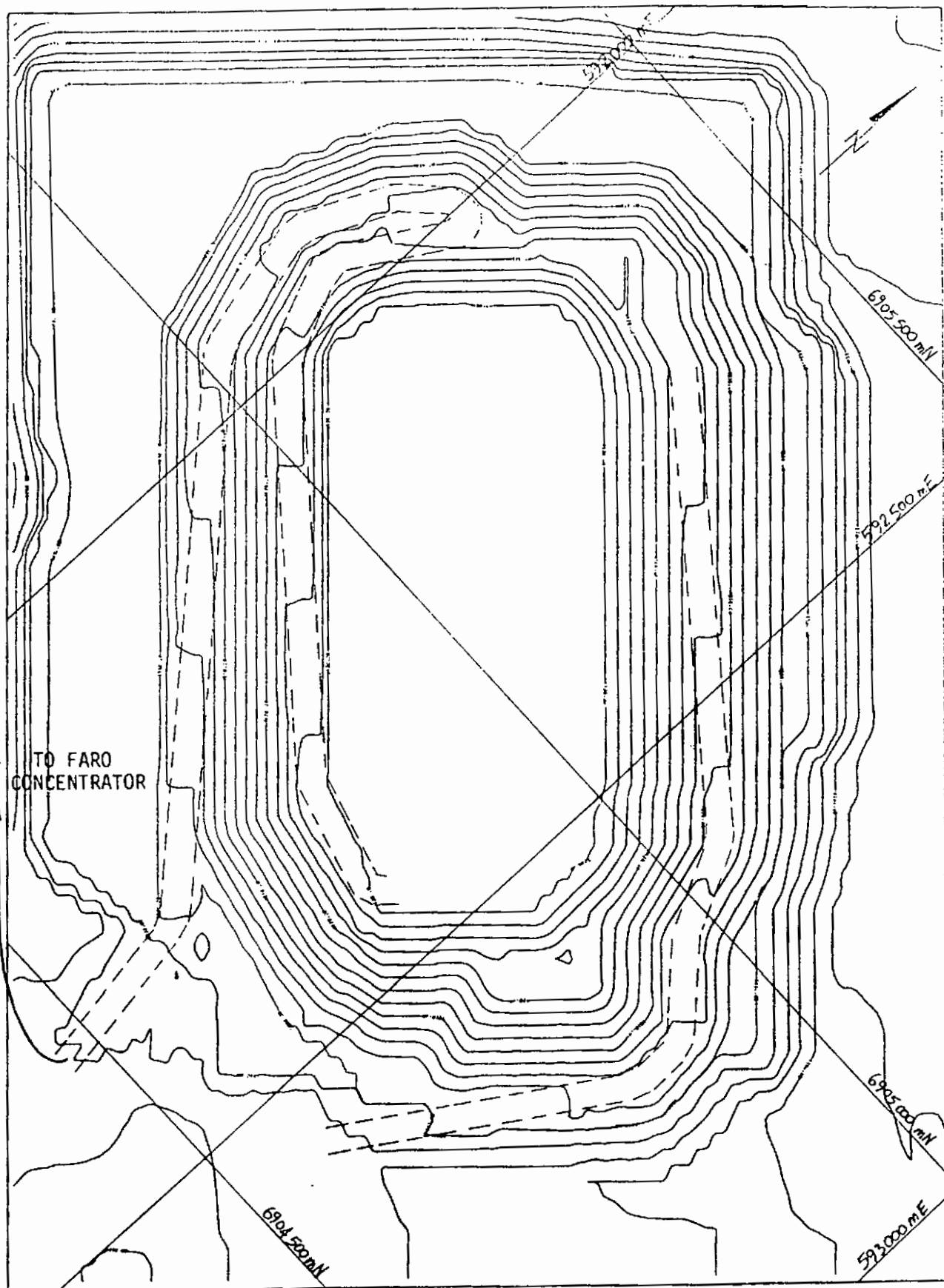


FIGURE 7. 4 - 7
GRUM PIT END OF 1995

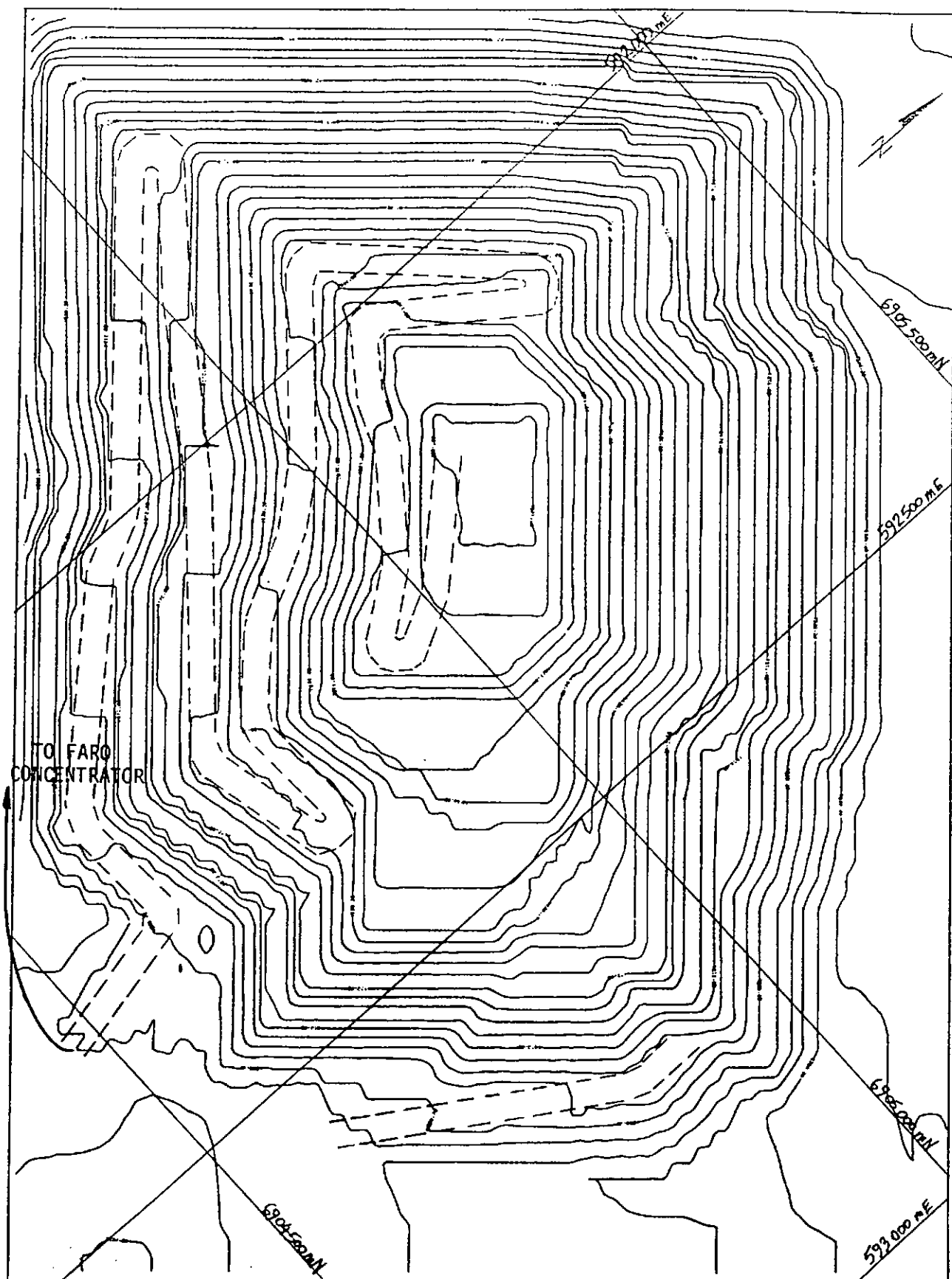


FIGURE 7.4 - 10
GRUM PIT END OF 1998

Pit Production Summary: Grue 1

| | 1988 | 1989 | 1990 | 1991 | 1992 | 1993 | 1994 | 1995 | 1996 | 1997 | 1998 | 1999 | Total |
|------------------------|--------|----------|----------|----------|---------|------|------|------|------|------|------|------|----------|
| Waste rock | 110054 | 9560761 | 6598990 | 17536870 | 5180593 | | | | | | | | 39207268 |
| Overburden | 642502 | 7347807 | 4692635 | 2384716 | | | | | | | | | 15067660 |
| Sulphide Waste | | 2642 | 15635 | 710510 | 525661 | | | | | | | | 1254448 |
| All Waste | 752556 | 16911210 | 11307260 | 20652096 | 5906254 | | | | | | | | 55529376 |
| + 5 t tonnes | | 57716 | 212216 | 3540616 | 3451647 | | | | | | | | 7262195 |
| IPb+Zn | | 5.89 | 5.55 | 7.26 | 7.36 | | | | | | | | 7.25 |
| IPb | | 2.21 | 1.93 | 2.6 | 2.63 | | | | | | | | 2.59 |
| IZn | | 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 |
| IPb | | 60 | 60 | 60 | 60 | | | | | | | | 60.00 |
| Ag g/t | | 810 | 828 | 800 | 812 | | | | | | | | 806 |
| Au g/t | | 5.31 | 8.42 | 5.97 | 6.4 | | | | | | | | 6.22 |
| Zn Conc DMT | | 3055 | 10942 | 246695 | 245643 | | | | | | | | 506335 |
| IZn | | 55 | 55 | 55 | 55 | | | | | | | | 55.00 |
| 4 - 5 t tonnes | | | 53271 | 794575 | 396738 | | | | | | | | 1244584 |
| IPb+Zn | | | 3.78 | 3.89 | 3.97 | | | | | | | | 3.91 |
| IPb | | | 1.26 | 1.28 | 1.29 | | | | | | | | 1.28 |
| IZn | | | 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 | | | | | | | | 17253 |
| IPb | | | 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 |
| IZn | | | 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 7.4 - 1
PRODUCTION SUMMARY GRUM STAGE ONE

Pit Production Summary: Grue 2

| | 1988 | 1989 | 1990 | 1991 | 1992 | 1993 | 1994 | 1995 | 1996 | 1997 | 1998 | 1999 | Total |
|------------------------|------|------|------|------|----------|----------|----------|----------|----------|------|------|------|-----------|
| Waste rock | | | | | 9880927 | 22145240 | 30859340 | 13363200 | 8184547 | | | | 84433254 |
| 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 |
| ZPb+Zn | | | | | | 12.49 | 10.87 | 9.15 | 9.46 | | | | 9.46 |
| ZPb | | | | | | 5.17 | 4.43 | 3.47 | 3.47 | | | | 3.55 |
| ZZn | | | | | | 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 |
| ZPb | | | | | | 60 | 60 | 60 | 60 | | | | 60.00 |
| Ag g/t | | | | | | 840 | 817 | 826 | 844 | | | | 835 |
| Au g/t | | | | | | 3.57 | 4.85 | 6.22 | 6.87 | | | | 6.40 |
| Zn Conc DMT | | | | | | 8695 | 63151 | 303740 | 449539 | | | | 825125 |
| ZZn | | | | | | 55 | 55 | 55 | 55 | | | | 55.00 |
| 4 - 5 % tonnes | | | | | | | 137270 | 422650 | 848849 | | | | 1408769 |
| ZPb+Zn | | | | | | | 4.06 | 3.88 | 3.87 | | | | 3.89 |
| ZPb | | | | | | | 1.73 | 1.53 | 1.54 | | | | 1.56 |
| ZZn | | | | | | | 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 |
| ZPb | | | | | | | 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 |
| ZZn | | | | | | | 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 7.4 - 2
PRODUCTION SUMMARY GRUM STAGE TWO

Pit Production Summary: Grue 3

| | 1988 | 1989 | 1990 | 1991 | 1992 | 1993 | 1994 | 1995 | 1996 | 1997 | 1998 | 1999 | Total |
|------------------------|------|------|------|------|------|------|---------|----------|----------|----------|----------|------|----------|
| Waste rock | | | | | | | | 14424400 | 17937220 | 31323420 | 7616420 | | 71301460 |
| Overburden | | | | | | | 1660482 | 1702149 | 821836 | 69507 | | | 4253974 |
| Sulphide Waste | | | | | | | | | 11827 | 475840 | 563253 | | 1050920 |
| All Waste | | | | | | | 1660482 | 16126549 | 18770883 | 31868767 | 8179673 | | 76606354 |
| + 5 % tonnes | | | | | | | | | 32018 | 1755428 | 3762671 | | 5550117 |
| 2Pb+Zn | | | | | | | | | 12.54 | 8.32 | 8.99 | | 8.80 |
| 2Pb | | | | | | | | | 4.07 | 3.29 | 3.35 | | 3.34 |
| 2Zn | | | | | | | | | 8.47 | 5.03 | 5.64 | | 5.46 |
| Ag g/t | | | | | | | | | 69 | 54 | 59 | | 57 |
| Au g/t | | | | | | | | | 0.93 | 0.79 | 1.01 | | 0.94 |
| Concentrate Equivalent | | | | | | | | | | | | | |
| Pb Conc DMT | | | | | | | | | 1791 | 76698 | 167253 | | 245742 |
| 2Pb | | | | | | | | | 60 | 60 | 60 | | 60.00 |
| Ag g/t | | | | | | | | | 842 | 813 | 876 | | 856 |
| Au g/t | | | | | | | | | 5.75 | 5.91 | 7.51 | | 7.00 |
| Zn Conc DMT | | | | | | | | | 4460 | 134456 | 326840 | | 465756 |
| 2Zn | | | | | | | | | 55 | 55 | 55 | | 55.00 |
| 4 - 5 % tonnes | | | | | | | | | 1286 | 223828 | 374795 | | 599909 |
| 2Pb+Zn | | | | | | | | | 3.99 | 3.84 | 3.88 | | 3.87 |
| 2Pb | | | | | | | | | 1.76 | 1.55 | 1.55 | | 1.55 |
| 2Zn | | | | | | | | | 2.23 | 2.29 | 2.33 | | 2.31 |
| Ag g/t | | | | | | | | | 26 | 30 | 27 | | 28 |
| Au g/t | | | | | | | | | 0.42 | 0.76 | 0.79 | | 0.78 |
| Concentrate Equivalent | | | | | | | | | | | | | |
| Pb Conc DMT | | | | | | | | | 26 | 3918 | 6587 | | 10531 |
| 2Pb | | | | | | | | | 60 | 60 | 60 | | 60.00 |
| Ag g/t | | | | | | | | | 717 | 917 | 842 | | 870 |
| Au g/t | | | | | | | | | 5.8 | 12.05 | 12.39 | | 12.25 |
| Zn Conc DMT | | | | | | | | | 37 | 6714 | 11461 | | 18212 |
| 2Zn | | | | | | | | | 55 | 55 | 55 | | 55.00 |
| Total Waste Mined | | | | | | | 1660482 | 16126549 | 18770883 | 31868767 | 8179673 | | 76606354 |
| Total Ore Mined | | | | | | | 0 | 0 | 33304 | 1979256 | 4137466 | | 6150026 |
| Total Tonnes Mined | | | | | | | 1660482 | 16126549 | 18804187 | 33848023 | 12317139 | | 82756380 |
| Strip Ratio | | | | | | | | | 563.62 | 16.10 | 1.98 | | 12.46 |

TABLE 7.4 - 3
PRODUCTION SUMMARY GRUM STAGE THREE

Pit Production Summary: Grum Total

| | 1988 | 1989 | 1990 | 1991 | 1992 | 1993 | 1994 | 1995 | 1996 | 1997 | 1998 | 1999 | Total |
|------------------------|--------|----------|----------|----------|----------|----------|----------|----------|----------|----------|----------|------|-----------|
| Waste rock | 110054 | 9560761 | 6598990 | 17556870 | 15261520 | 22145240 | 30859340 | 27787600 | 26121767 | 31323420 | 7616420 | | 194941982 |
| Overburden | 642502 | 7347807 | 4692633 | 2384716 | 5331719 | 6451748 | 2082219 | 1731284 | 826249 | 69507 | 0 | | 31580386 |
| Sulphide Waste | 0 | 2642 | 15633 | 710310 | 525661 | 6022 | 147937 | 471145 | 1220875 | 475840 | 563253 | | 4139520 |
| All Waste | 752556 | 16911210 | 11307260 | 20652096 | 21138900 | 28603010 | 33089496 | 29990029 | 28168891 | 31868767 | 8179673 | | 230661888 |
| + 5 % tonnes | | 57716 | 212216 | 3540616 | 3451647 | 74225 | 621260 | 3435351 | 4828997 | 1755428 | 3762671 | | 21740127 |
| Pb+Zn | | 5.89 | 5.53 | 7.26 | 7.36 | 12.49 | 10.87 | 9.15 | 9.48 | 8.32 | 8.99 | | 8.55 |
| Pb | | 2.21 | 1.93 | 2.60 | 2.63 | 5.17 | 4.43 | 3.47 | 3.47 | 3.29 | 3.35 | | 3.18 |
| Zn | | 3.68 | 3.62 | 4.66 | 4.73 | 7.32 | 6.44 | 5.68 | 6.01 | 5.03 | 5.64 | | 5.38 |
| Ag g/t | | 37 | 33 | 42 | 43 | 87 | 73 | 58 | 59 | 54 | 59 | | 53 |
| Au g/t | | 0.48 | 0.67 | 0.63 | 0.68 | 0.75 | 0.87 | 0.88 | 0.96 | 0.79 | 1.01 | | 0.84 |
| Concentrate Equivalent | | | | | | | | | | | | | |
| Pb Conc DMT | | 1564 | 4856 | 117074 | 115608 | 5560 | 38549 | 160408 | 224984 | 76698 | 167253 | | 912554 |
| Pb | | 60.00 | 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 | 812 | 840 | 817 | 826 | 844 | 813 | 876 | | 833 |
| Au g/t | | 5.31 | 8.42 | 5.97 | 6.40 | 3.57 | 4.85 | 6.22 | 6.86 | 5.91 | 7.51 | | 6.52 |
| Zn Conc DMT | | 3055 | 10942 | 246695 | 245643 | 8695 | 63151 | 303740 | 453999 | 134456 | 326840 | | 1797216 |
| Zn | | 55.00 | 55.00 | 55.00 | 55.00 | 55.00 | 55.00 | 55.00 | 55.00 | 55.00 | 55.00 | | 55.00 |
| - 5 % tonnes | | | 53271 | 794575 | 396738 | | 137270 | 422650 | 850135 | 223828 | 374795 | | 3253262 |
| Pb+Zn | | | 3.78 | 3.89 | 3.97 | | 4.06 | 3.88 | 3.87 | 3.84 | 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 | 2.29 | 2.33 | | 2.44 |
| Ag g/t | | | 24 | 25 | 24 | | 31 | 27 | 26 | 30 | 27 | | 26 |
| Au g/t | | | 0.77 | 0.46 | 0.45 | | 0.82 | 0.63 | 0.64 | 0.76 | 0.79 | | 0.61 |
| Concentrate Equivalent | | | | | | | | | | | | | |
| Pb Conc DMT | | | 726 | 10983 | 5546 | | 2763 | 7281 | 14838 | 3918 | 6587 | | 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 | 917 | 842 | | 859 |
| Au g/t | | | 14.66 | 8.76 | 8.42 | | 11.67 | 10.11 | 10.02 | 12.05 | 12.39 | | 10.20 |
| Zn Conc DMT | | | 1787 | 27870 | 14341 | | 4208 | 13079 | 25985 | 6714 | 11461 | | 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 | 20652096 | 21138900 | 28603010 | 33089496 | 29990029 | 28168891 | 31868767 | 8179673 | | 230661888 |
| Total Ore Mined | 0 | 57716 | 265487 | 4335191 | 3848385 | 74225 | 758530 | 3858001 | 5679132 | 1979256 | 4137466 | | 24993389 |
| Total Tonnes Mined | 752556 | 16968926 | 11572747 | 24987287 | 24987285 | 28677235 | 33848026 | 33848030 | 33848023 | 33848023 | 12317139 | | 255655277 |
| Strip Ratio | | 293.01 | 42.59 | 4.76 | 5.49 | 385.36 | 43.62 | 7.77 | 4.96 | 16.10 | 1.98 | | 9.23 |

TABLE 7.4 - 4
PRODUCTION SUMMARY GRUM PIT TOTAL

7.4.2 Equipment Schedule - Grum Pit

The operating equipment schedule for the Grum Pit is given in Table 7.4-5. Note that truck requirements have been determined separately and are presented in in Section 7.5. Note also the production equipment (shovels and drills) is surplus to the Faro Pit while support equipment is additional to the Faro Pit.

:

Operating Equipment - Grum Pit

| | 1988 | 1989 | 1990 | 1991 | 1992 | 1993 | 1994 | 1995 | 1996 | 1997 | 1998 | 1999 |
|-------------------------|------|------|------|------|------|------|------|------|------|------|------|------|
| | | | | | | | | | | | | |
| Production Equipment: | | | | | | | | | | | | |
| Shovel | 0.1 | 1.9 | 1.1 | 3 | 3 | 3.4 | 4 | 4 | 4 | 4 | 1.5 | |
| Drill | NA | NA | NA | NA | NA | NA | NA | NA | NA | NA | NA | |
| Haul Truck | NA | NA | NA | NA | NA | NA | NA | NA | NA | NA | NA | |
| Front End Loader - L800 | NA | NA | NA | NA | NA | NA | NA | NA | NA | NA | NA | |
| Support Equipaent | | | | | | | | | | | | |
| Dozer - DB w/ ripper | | 1 | | 1 | 1 | 2 | 2 | 2 | 2 | 2 | 1 | |
| Dozer - D9 w/ ripper | 0.5 | 1 | 1 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 1 | |
| Rubber Tired Dozer | | 1 | 1 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | |
| Grader | 1 | 1 | 1 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 1 | |

TABLE 7.4 - 5
GRUM EQUIPMENT SCHEDULE

7.5 Truck Requirements

Truck requirements have been calculated for both pits and are presented in table 7.5-1 with the total number of units required listed at the bottom of the table.

During the years 1994 through 1997, more trucks are required than are in the total fleet. These trucks are not included in the capital cost forecast as it is expected that contractors could be used to carry out the haulage.

The operating cost estimate for this contract haulage requirement has been taken at 125% of Curragh operating cost.

| A) TRUCK FACTORS t/LOAD | | WST | ORE | | | | | | | | | | |
|---|-------|---|-----------|----------|-----------|-------------|--------|--------|--------|--------|--------|--------|--------|
| EUCLID 170T | | 134 | 156 | | | | | | | | | | |
| WABCO 120T | | 95 | 108 | | | | | | | | | | |
| B) TRUCK HAUL SPEEDS Km/HR | | FLAT SHORT | FLAT LONG | UP RAMP | DOWN RAMP | AVG. RAMP | | | | | | | |
| EUCLID 170T | | 25 | 40 | 12 | 15 | 13.5 | | | | | | | |
| WABCO 120T | | 25 | 35 | 10 | 15 | 12.5 | | | | | | | |
| C) HAUL PROFILES, Km | | 1988 | 1989 | 1990 | 1991 | 1992 | 1993 | 1994 | 1995 | 1996 | 1997 | 1998 | 1999 |
| WASTE; TO DUMP | | | | | | | | | | | | | |
| FLAT TO | | 0.590 | 1.105 | 0.790 | 1.020 | 1.090 | 0.380 | 0.580 | 0.650 | 0.650 | 0.650 | 0.650 | 0.650 |
| FLAT RETURN | | 0.590 | 1.105 | 0.790 | 1.020 | 1.090 | 0.380 | 0.580 | 0.650 | 0.650 | 0.650 | 0.650 | 0.650 |
| RAMP UP | | 0.125 | 0.251 | 0.140 | 1.063 | 1.063 | 1.074 | 1.826 | 2.248 | 2.248 | 2.248 | 2.248 | 2.248 |
| RAMP DOWN | | 0.125 | 0.251 | 0.140 | 1.063 | 1.063 | 1.074 | 1.826 | 2.248 | 2.248 | 2.248 | 2.248 | 2.248 |
| ORE; TO MILL | | | | | | | | | | | | | |
| FLAT TO | | 15.730 | 16.150 | 15.780 | 15.330 | 15.330 | 15.190 | 15.140 | 15.190 | 15.320 | 15.010 | 15.010 | 15.010 |
| FLAT RETURN | | 15.730 | 16.150 | 15.780 | 15.330 | 15.330 | 15.190 | 15.140 | 15.190 | 15.320 | 15.010 | 15.010 | 15.010 |
| RAMP UP | | 3.488 | 3.739 | 4.338 | 2.816 | 2.816 | 2.325 | 3.077 | 3.499 | 4.070 | 4.561 | 4.561 | 4.561 |
| RAMP DOWN | | 3.488 | 3.739 | 4.338 | 2.816 | 2.816 | 2.325 | 3.077 | 3.499 | 4.070 | 4.561 | 4.561 | 4.561 |
| D) HAUL TIMES, MINUTES (TRAVEL TIME ONLY) | | 1988 | 1989 | 1990 | 1991 | 1992 | 1993 | 1994 | 1995 | 1996 | 1997 | 1998 | 1999 |
| EUCLID 170T: | WASTE | 3.9 | 7.5 | 5.0 | 14.3 | 14.7 | 11.4 | 19.0 | 23.1 | 23.1 | 23.1 | 23.1 | 23.1 |
| | ORE | 78.2 | 81.7 | 85.9 | 71.0 | 71.0 | 66.2 | 72.8 | 76.7 | 82.1 | 85.6 | 85.6 | 85.6 |
| WABCO 120T: | WASTE | 4.0 | 7.7 | 5.1 | 15.1 | 15.4 | 12.1 | 20.3 | 24.7 | 24.7 | 24.7 | 24.7 | 24.7 |
| | ORE | 87.4 | 91.3 | 95.7 | 79.6 | 79.6 | 74.4 | 81.4 | 85.7 | 91.6 | 95.2 | 95.2 | 95.2 |
| E) TRUCK CYCLE TIME, MINUTES | | SPOT/SHV | LOAD | SPOT/DUM | DUMP | TOTAL FIXED | | | | | | | |
| EUCLID 170T: | | 0.5 | 4.5 | 0.5 | 0.5 | 6.0 | | | | | | | |
| WABCO 120T: | | 0.5 | 3.0 | 0.5 | 0.5 | 4.5 | | | | | | | |
| TOTAL CYCLE = FIXED + HAUL | | ORE TO STOCKPILE = WASTE TO DUMP
STOCKPILE TO MILL = ORE - WASTE TO DUMP TRAVEL TIME | | | | | | | | | | | |
| | | 1988 | 1989 | 1990 | 1991 | 1992 | 1993 | 1994 | 1995 | 1996 | 1997 | 1998 | 1999 |
| EUCLID 170T: | WASTE | 9.9 | 13.5 | 11.0 | 20.3 | 20.7 | 17.4 | 25.0 | 29.1 | 29.1 | 29.1 | 29.1 | 29.1 |
| | ORE | 93.4 | 97.3 | 101.7 | 85.6 | 85.6 | 80.4 | 87.4 | 91.7 | 97.6 | 101.2 | 101.2 | 101.2 |
| | STK | 89.5 | 89.7 | 96.7 | 71.2 | 70.9 | 69.0 | 68.4 | 68.6 | 74.5 | 78.1 | 78.1 | 78.1 |
| WABCO 120T: | WASTE | 8.5 | 12.2 | 9.6 | 19.6 | 19.9 | 16.6 | 24.8 | 29.2 | 29.2 | 29.2 | 29.2 | 29.2 |
| | ORE | 91.9 | 95.8 | 100.2 | 84.1 | 84.1 | 78.9 | 85.9 | 90.2 | 96.1 | 99.7 | 99.7 | 99.7 |
| | STK | 87.9 | 88.1 | 95.1 | 69.0 | 68.7 | 66.8 | 65.6 | 65.5 | 71.4 | 75.0 | 75.0 | 75.0 |

TABLE 7.5 - 1
TRUCK REQUIREMENTS

F) TRUCK PRODUCTIVITIES, tonnes/HR

HAULAGE EFFICIENCY, WASTE = 0.85
ORE = 0.95
STK = 0.95

| | | 1988 | 1989 | 1990 | 1991 | 1992 | 1993 | 1994 | 1995 | 1996 | 1997 | 1998 | 1999 |
|--------------|-------|------|------|------|------|------|------|------|------|------|------|------|------|
| EUCLID 170T; | WASTE | 687 | 505 | 619 | 336 | 330 | 393 | 273 | 235 | 235 | 235 | 235 | 235 |
| | ORE | 95 | 91 | 87 | 104 | 104 | 111 | 102 | 97 | 91 | 88 | 88 | 88 |
| | STK | 99 | 99 | 92 | 125 | 125 | 129 | 130 | 130 | 119 | 114 | 114 | 114 |
| WABCO 120T; | WASTE | 568 | 397 | 503 | 247 | 243 | 291 | 195 | 166 | 166 | 166 | 166 | 166 |
| | ORE | 67 | 64 | 61 | 73 | 73 | 78 | 72 | 68 | 64 | 62 | 62 | 62 |
| | STK | 70 | 70 | 65 | 89 | 90 | 92 | 94 | 94 | 86 | 82 | 82 | 82 |

G) SURPLUS TRUCK UNITS, FARD ONLY

| | | | | | | | | | | | | | |
|-------------------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|
| EUCLID 170T UNITS | 2.0 | 8.0 | 6.3 | 7.0 | 8.6 | 10.0 | 11.0 | 11.0 | 11.0 | 11.0 | 11.0 | 11.0 | 11.0 |
| EUCLID 170T HRS | 11046 | 44661 | 35094 | 39042 | 47766 | 55417 | 61261 | 61261 | 61429 | 61261 | 61261 | 61261 | 61261 |
| WABCO 120T UNITS | 6.8 | 11.5 | 18.0 | 17.5 | 13.7 | 14.7 | 17.0 | 19.0 | 19.0 | 19.0 | 19.0 | 19.0 | 19.0 |
| WABCO 120T HRS | 33590 | 56482 | 88059 | 85501 | 62470 | 66757 | 83167 | 92951 | 93206 | 92951 | 92951 | 92951 | 92951 |

H) TRUCK REQUIREMENTS; VANGORDA/GRUM

| VANGORDA
GRUM PITS | | 1988 | 1989 | 1990 | 1991 | 1992 | 1993 | 1994 | 1995 | 1996 | 1997 | 1998 | 1999 |
|-----------------------|--|---------|---------|---------|---------|---------|---------|---------|---------|---------|---------|---------|-------|
| WORK DAYS | | 365 | 364 | 364 | 364 | 365 | 364 | 364 | 364 | 365 | 364 | 364 | 364 |
| MINED (t x '000) | | | | | | | | | | | | | |
| Total Waste | | 7418.14 | 22940.2 | 20100.0 | 20652.0 | 21138.9 | 28603.0 | 33089.4 | 29990.0 | 28168.8 | 31868.7 | 8179.67 | 0 |
| Ore to Stk | | 600.216 | 1294.49 | 4887.26 | 4335.19 | 3848.38 | 74.225 | 758.53 | 3858.00 | 5679.13 | 1979.25 | 4137.46 | 0 |
| Stk to Mill | | 0 | 1012 | 4015 | 3718 | 2035 | 2189 | 1188 | 4015 | 4015 | 4014 | 4025 | 896 |
| Total Mined | | 8018 | 24233 | 24987 | 24987 | 24987 | 28677 | 33848 | 33848 | 33848 | 33848 | 12317 | 0 |
| Total Moved | | 8018 | 25247 | 29002 | 28705 | 27022 | 30866 | 35036 | 37863 | 37863 | 37862 | 16342 | 896 |
| Mined tpd | | 21.968 | 66.579 | 68.646 | 68.646 | 68.458 | 78.784 | 92.989 | 92.989 | 92.734 | 92.989 | 33.838 | 0.000 |
| Moved tpd | | 21.968 | 69.339 | 79.677 | 78.860 | 74.034 | 84.797 | 96.252 | 104.019 | 103.734 | 104.016 | 44.895 | 2.462 |

TABLE 7.5 - 1 (CONTINUED)

TRUCK REQUIREMENTS

=====

EUCLID 170 TON FLEET

| | | | | | | | | | | | | |
|--------------------|---------|---------|---------|---------|---------|---------|---------|---------|---------|---------|---------|--------|
| # OF UNITS AVAIL | 2.0 | 8.0 | 6.3 | 7.0 | 8.6 | 10.0 | 11.0 | 11.0 | 11.0 | 11.0 | 11.0 | 11.0 |
| PHYSIAL 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 |
| UTAL | 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 UTAL. | 0.6375 | 0.6375 | 0.6375 | 0.6375 | 0.6375 | 0.6375 | 0.6375 | 0.6375 | 0.6375 | 0.6375 | 0.6375 | 0.6375 |
| OP.HRS/YR AVAIL. | 11046 | 44661 | 35094 | 39042 | 47766 | 55417 | 61261 | 61261 | 61429 | 61261 | 61261 | 61261 |
| OP.HRS/UNIT/YR | 5585 | 5569 | 5569 | 5569 | 5584 | 5569 | 5569 | 5569 | 5584 | 5569 | 5569 | 5569 |
| PROD. WST+ORE/STK | 687 | 505 | 619 | 336 | 330 | 393 | 273 | 235 | 235 | 235 | 235 | 235 |
| PROD. STK TO MILL | 99 | 99 | 92 | 125 | 125 | 129 | 130 | 130 | 119 | 114 | 114 | 114 |
| STK TO MILL, t | 0 | 1012 | 4015 | 3718 | 2035 | 2189 | 1188 | 4015 | 4015 | 4014 | 4025 | 896 |
| WST+ORE TO STK, t | 8018.36 | 24234.7 | 24987.2 | 24987.2 | 24987.2 | 28677.2 | 33848.0 | 33848.0 | 33848.0 | 33848.0 | 12317.1 | 0 |
| HRS STK TO MILL | 0 | 10212 | 35094 | 29789 | 16229 | 16993 | 9140 | 30961 | 33637 | 35275 | 35370 | 7875 |
| HRS WST+ORE TO STK | 11046 | 34449 | 0 | 9254 | 31537 | 38424 | 52121 | 30301 | 27793 | 25987 | 25891 | 0 |
| STK TO MILL REM | 0 | 0 | 788 | 0 | 0 | 0 | 0 | -0 | -0 | -0 | -0 | -0 |
| WST+ORE TO STK REM | 426 | 6841 | 24987 | 21879 | 14566 | 13560 | 19609 | 26733 | 27322 | 27746 | 6237 | 0 |

WABCO 120 TON FLEET

| | | | | | | | | | | | | |
|---------------------|-------|-------|-------|--------|--------|--------|--------|--------|--------|--------|-------|-------|
| # UNITS AVAIL | 6.8 | 11.5 | 18.0 | 17.5 | 13.7 | 14.7 | 17.0 | 19.0 | 19.0 | 19.0 | 19.0 | 19.0 |
| PHYSIAL AVAIL | 0.7 | 0.7 | 0.7 | 0.7 | 0.65 | 0.65 | 0.7 | 0.7 | 0.7 | 0.7 | 0.7 | 0.7 |
| UTAL | 0.8 | 0.8 | 0.8 | 0.8 | 0.8 | 0.8 | 0.8 | 0.8 | 0.8 | 0.8 | 0.8 | 0.8 |
| EFFECTIVE UTAL. | 0.56 | 0.56 | 0.56 | 0.56 | 0.52 | 0.52 | 0.56 | 0.56 | 0.56 | 0.56 | 0.56 | 0.56 |
| OP.HRS/YR PER UNIT | 4906 | 4892 | 4892 | 4892 | 4555 | 4543 | 4892 | 4892 | 4906 | 4892 | 4892 | 4892 |
| HRS AVAIL | 33590 | 56482 | 88059 | 85501 | 62470 | 66757 | 83167 | 92951 | 93206 | 92951 | 92951 | 92951 |
| PROD. WST+ORE/STK | 568 | 397 | 503 | 247 | 243 | 291 | 195 | 166 | 166 | 166 | 166 | 166 |
| PROD. STK TO MILL | 70 | 70 | 65 | 89 | 90 | 92 | 94 | 94 | 86 | 82 | 82 | 82 |
| HRS WST+ORE/STK | 751 | 17246 | 49696 | 88513 | 59937 | 46557 | 83167 | 92951 | 93206 | 92951 | 37592 | 0 |
| HRS ORE TO MILL | 0 | 0 | 12180 | 0 | 0 | 0 | 0 | -0 | -0 | -0 | -0 | -0 |
| WST+ORE TO STK REM | 0 | 0 | 0 | -0 | -0 | -0 | 3370 | 11310 | 11857 | 12323 | -0 | 0 |
| TOTAL WABCO HRS | 751 | 17246 | 61876 | 88513 | 59937 | 46557 | 83167 | 92951 | 93206 | 92951 | 37592 | -0 |
| WABCO UNITS UTAL. | 0.2 | 3.5 | 12.6 | 18.1 | 13.2 | 10.2 | 17.0 | 19.0 | 19.0 | 19.0 | 7.7 | -0.0 |
| TOTAL TRUCK HRS | 11797 | 61907 | 96970 | 127553 | 107703 | 101975 | 144428 | 154212 | 154636 | 154212 | 98853 | 7875 |
| ===== | ===== | ===== | ===== | ===== | ===== | ===== | ===== | ===== | ===== | ===== | ===== | ===== |
| # TOTAL FLEET AVAIL | 2.1 | 11.3 | 18.9 | 25.1 | 21.7 | 20.2 | 28.0 | 30.0 | 30.0 | 30.0 | 18.7 | 11.0 |

| | | | | | | | | | | | | |
|--------------------|-----|-----|-----|------|------|------|-------|-------|-------|-------|------|-----|
| WST+ORE TO STK REM | 0 | 0 | 0 | -0 | -0 | -0 | 3370 | 11310 | 11857 | 12323 | -0 | 0 |
| 170T TRUCK HRS REQ | 0 | 0 | 0 | -0 | -0 | -0 | 12336 | 48164 | 50491 | 52478 | -0 | 0 |
| # 170T UNITS REQ.D | 0.0 | 0.0 | 0.0 | -0.0 | -0.0 | -0.0 | 2.2 | 8.6 | 9.0 | 9.4 | -0.0 | 0.0 |

TABLE 7.5 - 1 (CONTINUED)

SUMMARY; TRUCK HRS VANGORDA/GRUM REQUIRED

| | 1988 | 1989 | 1990 | 1991 | 1992 | 1993 | 1994 | 1995 | 1996 | 1997 | 1998 | 1999 |
|----------------|-------|-------|-------|--------|--------|--------|--------|--------|--------|--------|-------|-------|
| | ----- | ----- | ----- | ----- | ----- | ----- | ----- | ----- | ----- | ----- | ----- | ----- |
| 170T TRUCK HRS | 11046 | 44661 | 35094 | 39042 | 47766 | 55417 | 73597 | 109425 | 111921 | 113739 | 61261 | 7875 |
| 120T TRUCK HRS | 751 | 17246 | 61876 | 88513 | 59937 | 46557 | 83167 | 92951 | 93206 | 92951 | 37592 | -0 |
| | ----- | ----- | ----- | ----- | ----- | ----- | ----- | ----- | ----- | ----- | ----- | ----- |
| TOTAL HRS | 11797 | 61907 | 96970 | 127555 | 107703 | 101975 | 156764 | 202376 | 205127 | 206690 | 98853 | 7875 |
| | ----- | ----- | ----- | ----- | ----- | ----- | ----- | ----- | ----- | ----- | ----- | ----- |
| # 170T UNITS | 2.0 | 8.0 | 6.3 | 7.0 | 8.6 | 10.0 | 13.2 | 19.6 | 20.0 | 20.4 | 11.0 | 1.4 |
| # 120T UNITS | 0.2 | 3.5 | 12.6 | 18.1 | 13.2 | 10.2 | 17.0 | 19.0 | 19.0 | 19.0 | 7.7 | -0.0 |
| | ----- | ----- | ----- | ----- | ----- | ----- | ----- | ----- | ----- | ----- | ----- | ----- |
| # TOTAL UNITS | 2.1 | 11.5 | 18.9 | 25.1 | 21.7 | 20.2 | 30.2 | 38.6 | 39.0 | 39.4 | 18.7 | 1.4 |

TABLE 7.5 - 1 (CONTINUED)

8.0 PRODUCTION SCHEDULE - MILLING

8.0 Production Schedule - Milling

The mill production schedule is based on the following:

- 11,000 tonne per day mill feed
- mill feed from Faro Pit as outlined in Appendix D
- metallurgical recoveries as outlined in Section 4

The production schedule was determined with the requirement of providing a constant supply of ore to the Faro concentrator. A cutoff grade of 5% (combined lead plus zinc) was the target, with 4-5% ore being fed as required to make up shortfalls.

This forecast does not make any allowance for reduced recovery of stockpiled ore as not enough information on the rate or the effect of stockpile oxidation is available at this time. It should be kept in mind however that a negative effect on recovery should be expected.

The mill production summary (including Faro Pit) is given in Table 8-1, while the production details are given in Tables 8-2 through 8-4.

Mill Production Summary (Including Faro Pit)

| | 1988 | 1989 | 1990 | 1991 | 1992 | 1993 | 1994 | 1995 | 1996 | 1997 | 1998 | 1999 | Total |
|-------------------|---------|---------|---------|---------|---------|---------|---------|---------|---------|---------|---------|---------|----------|
| <hr/> | | | | | | | | | | | | | |
| Faro | | | | | | | | | | | | | |
| + 5 % tonnes feed | 4759869 | 3685500 | | 364455 | 2432421 | 2242948 | | | | | | | 13485193 |
| Pb conc DMT | 194584 | 129437 | | 10935 | 76074 | 84693 | | | | | | | 495723 |
| Zn conc DMT | 338612 | 243399 | | 19817 | 170068 | 193198 | | | | | | | 985094 |
| + 4 % tonnes feed | | | | | | | 3741336 | | | | | | 3741336 |
| Pb conc DMT | | | | | | | 66092 | | | | | | 66092 |
| Zn conc DMT | | | | | | | 127121 | | | | | | 127121 |
| Vangorda | | | | | | | | | | | | | |
| + 5 % tonnes feed | | 1012000 | 4015000 | 797701 | | | | | | | | | 5824701 |
| Pb conc DMT | | 52519 | 242119 | 48104 | | | | | | | | | 342742 |
| Zn conc DMT | | 64871 | 283342 | 56295 | | | | | | | | | 404508 |
| + 4 % tonnes feed | | | | | | | | | | | 634064 | | 634064 |
| Pb conc DMT | | | | | | | | | | | 13346 | | 13346 |
| Zn conc DMT | | | | | | | | | | | 17936 | | 17936 |
| Grue | | | | | | | | | | | | | |
| + 5 % tonnes feed | | | | 2920000 | 2035000 | 2189000 | 813680 | 3435000 | 4015000 | 2569776 | 3762671 | | 21740127 |
| Pb conc DMT | | | | 94633 | 67707 | 75671 | 45201 | 160392 | 187060 | 114639 | 167253 | | 912554 |
| Zn conc DMT | | | | 199767 | 143675 | 157724 | 77015 | 303709 | 377469 | 211017 | 326840 | | 1797216 |
| + 4 % tonnes feed | | | | | | | 374000 | 580000 | | 1444000 | 262000 | 593262 | 3253262 |
| Pb conc DMT | | | | | | | 5418 | 8872 | | 23835 | 4447 | 10070 | 52642 |
| Zn conc DMT | | | | | | | 13047 | 19558 | | 46055 | 8205 | 18580 | 105445 |
| <hr/> | | | | | | | | | | | | | |
| Total | | | | | | | | | | | | | |
| Tonnes Feed | 4759869 | 4697500 | 4015000 | 4082156 | 4467421 | 4431948 | 4929016 | 4015000 | 4015000 | 4013776 | 4024671 | 1227326 | 48678683 |
| Pb conc DMT | 194584 | 181956 | 242119 | 153672 | 143781 | 160364 | 116711 | 169263 | 187060 | 138473 | 171700 | 23416 | 1883099 |
| Zn conc DMT | 338612 | 308270 | 283342 | 275878 | 313743 | 350922 | 217183 | 323267 | 377469 | 257072 | 335045 | 36516 | 3437320 |
| Total conc DMT | 553196 | 490226 | 525461 | 429550 | 457524 | 511286 | 333894 | 492530 | 564529 | 395545 | 506746 | 59932 | 5320419 |

TABLE 8 - 1
MILL PRODUCTION SUMMARY

| | 1988 | 1989 | 1990 | 1991 | 1992 | 1993 | 1994 | 1995 | 1996 | 1997 | 1998 | 1999 | Total |
|----------------------------------|--------|---------|---------|--------|--------|--------|--------|--------|--------|--------|--------|--------|---------|
| Vangorda Mill Feed | | | | | | | | | | | | | |
| * 5 1 tonnes | | 1012000 | 4015000 | 797701 | | | | | | | | | 5824701 |
| ZPb+Zn | | 7.81 | 8.65 | 8.65 | | | | | | | | | 8.51 |
| ZPb | | 3.33 | 3.79 | 3.79 | | | | | | | | | 3.71 |
| ZZn | | 4.48 | 4.87 | 4.87 | | | | | | | | | 4.80 |
| Ag g/t | | 48.11 | 53.29 | 53.29 | | | | | | | | | 52 |
| Au g/t | | 0.55 | 0.61 | 0.61 | | | | | | | | | 0.60 |
| Concentrate | | | | | | | | | | | | | |
| Pb Conc DMT | | 52519 | 242119 | 48104 | | | | | | | | | 342742 |
| ZPb | | 53.13 | 53.29 | 53.29 | | | | | | | | | 53.27 |
| Ag g/t | | 614.37 | 603.29 | 603.29 | | | | | | | | | 603 |
| Au g/t | | 4.23 | 3.91 | 3.91 | | | | | | | | | 3.96 |
| Zn Conc DMT | | 64871 | 283342 | 56295 | | | | | | | | | 404508 |
| ZZn | | 54.77 | 54.71 | 54.71 | | | | | | | | | 54.72 |
| 4 - 5 1 tonnes | | | | | | | | | | | | | |
| ZPb+Zn | | | | | | | | | | | 634064 | | 634064 |
| ZPb | | | | | | | | | | | 3.90 | | 3.90 |
| ZZn | | | | | | | | | | | 1.68 | | 1.68 |
| Ag g/t | | | | | | | | | | | 2.22 | | 2.22 |
| Au g/t | | | | | | | | | | | 25.60 | | 26 |
| Au g/t | | | | | | | | | | | 0.53 | | 0.53 |
| Concentrate | | | | | | | | | | | | | |
| Pb Conc DMT | | | | | | | | | | | 13346 | | 13346 |
| ZPb | | | | | | | | | | | 55.37 | | 55.37 |
| Ag g/t | | | | | | | | | | | 565 | | 565 |
| Au g/t | | | | | | | | | | | 4.95 | | 4.95 |
| Zn Conc DMT | | | | | | | | | | | 17936 | | 17936 |
| ZZn | | | | | | | | | | | 52.68 | | 52.68 |
| Vangorda Stockpile Status | | | | | | | | | | | | | |
| * 5 1 tonnes | 501642 | 562978 | 797702 | | | | | | | | | | |
| ZPb+Zn | 7.31 | 7.81 | 8.65 | | | | | | | | | | |
| ZPb | 3.12 | 3.33 | 3.79 | | | | | | | | | | |
| ZZn | 4.19 | 4.48 | 4.87 | | | | | | | | | | |
| Ag g/t | 44.00 | 48.11 | 53.29 | | | | | | | | | | |
| Au g/t | 0.49 | 0.55 | 0.61 | | | | | | | | | | |
| Concentrate Equivalent | | | | | | | | | | | | | |
| Pb Conc DMT | 24159 | 30254 | 48104 | | | | | | | | | | |
| ZPb | 53.06 | 53.13 | 53.29 | | | | | | | | | | |
| Ag g/t | 608.00 | 614.37 | 603.29 | | | | | | | | | | |
| Au g/t | 4.04 | 4.23 | 3.91 | | | | | | | | | | |
| Zn Conc DMT | 29647 | 37370 | 56295 | | | | | | | | | | |
| ZZn | 54.90 | 54.77 | 54.71 | | | | | | | | | | |
| 4 - 5 1 tonnes | | | | | | | | | | | | | |
| ZPb+Zn | 98574 | 242013 | 634064 | 634064 | 634064 | 634064 | 634064 | 634064 | 634064 | 634064 | 634064 | 634064 | |
| ZPb | 3.86 | 3.88 | 3.90 | 3.90 | 3.90 | 3.90 | 3.90 | 3.90 | 3.90 | 3.90 | 3.90 | 3.90 | |
| ZPb | 1.60 | 1.62 | 1.68 | 1.68 | 1.68 | 1.68 | 1.68 | 1.68 | 1.68 | 1.68 | 1.68 | 1.68 | |
| ZZn | 2.26 | 2.26 | 2.22 | 2.22 | 2.22 | 2.22 | 2.22 | 2.22 | 2.22 | 2.22 | 2.22 | 2.22 | |
| Ag g/t | 27.00 | 28.19 | 25.60 | 25.60 | 25.60 | 25.60 | 25.60 | 25.60 | 25.60 | 25.60 | 25.60 | 25.60 | |
| Au g/t | 0.35 | 0.41 | 0.53 | 0.53 | 0.53 | 0.53 | 0.53 | 0.53 | 0.53 | 0.53 | 0.53 | 0.53 | |
| Concentrate Equivalent | | | | | | | | | | | | | |
| Pb Conc DMT | 1984 | 4860 | 13346 | 13346 | 13346 | 13346 | 13346 | 13346 | 13346 | 13346 | 13346 | 13346 | |
| ZPb | 54.23 | 55.58 | 55.37 | 55.37 | 55.37 | 55.37 | 55.37 | 55.37 | 55.37 | 55.37 | 55.37 | 55.37 | |
| Ag g/t | 616.00 | 644.40 | 564.67 | 564.67 | 564.67 | 564.67 | 564.67 | 564.67 | 564.67 | 564.67 | 564.67 | 564.67 | |
| Au g/t | 3.44 | 3.21 | 4.95 | 4.95 | 4.95 | 4.95 | 4.95 | 4.95 | 4.95 | 4.95 | 4.95 | 4.95 | |
| Zn Conc DMT | 2828 | 6979 | 17936 | 17936 | 17936 | 17936 | 17936 | 17936 | 17936 | 17936 | 17936 | 17936 | |
| ZZn | 52.45 | 52.59 | 52.68 | 52.68 | 52.68 | 52.68 | 52.68 | 52.68 | 52.68 | 52.68 | 52.68 | 52.68 | |

TABLE 8 - 2

VANGORDA ORE SCHEDULE

Ore Schedule

| | 1988 | 1989 | 1990 | 1991 | 1992 | 1993 | 1994 | 1995 | 1996 | 1997 | 1998 | 1999 | Total |
|-------------------------------|--------|--------|---------|---------|---------|---------|---------|---------|---------|---------|--------|--------|----------|
| Grass Mill Feed | | | | | | | | | | | | | |
| + 5 % tonnes | | | 2920000 | 2035000 | 2189000 | 813680 | 3435000 | 4015000 | 2569776 | 3762671 | | | 21740127 |
| IPb+Zn | | | 7.14 | 7.32 | 7.48 | 10.07 | 9.15 | 9.48 | 8.69 | 8.99 | | | 8.55 |
| IPb | | | 2.56 | 2.61 | 2.69 | 4.02 | 3.47 | 3.47 | 3.35 | 3.35 | | | 3.18 |
| IZn | | | 4.59 | 4.70 | 4.78 | 6.05 | 5.68 | 6.01 | 5.34 | 5.64 | | | 5.38 |
| Ag g/t | | | 41.42 | 42.68 | 44.06 | 66.16 | 58.00 | 59.07 | 55.61 | 59.00 | | | 53 |
| Au g/t | | | 0.63 | 0.67 | 0.67 | 0.82 | 0.88 | 0.96 | 0.84 | 1.01 | | | 0.84 |
| Concentrate | | | | | | | | | | | | | |
| Pb Conc DMT | | | 94633 | 67707 | 75671 | 45201 | 160392 | 187060 | 114639 | 167253 | | | 912554 |
| IPb | | | 60.00 | 60.00 | 60.00 | 60.00 | 60.00 | 60.00 | 60.00 | 60.00 | | | 60.00 |
| Ag g/t | | | 801.23 | 809.85 | 811.88 | 816.25 | 826.00 | 843.98 | 823.25 | 876.00 | | | 833 |
| Au g/t | | | 6.06 | 6.33 | 6.15 | 5.04 | 6.22 | 6.86 | 6.22 | 7.51 | | | 6.52 |
| Zn Conc DMT | | | 199767 | 143675 | 157724 | 77015 | 303709 | 377469 | 211017 | 326840 | | | 1797216 |
| IZn | | | 55.00 | 55.00 | 55.00 | 55.00 | 55.00 | 55.00 | 55.00 | 55.00 | | | 55.00 |
| 4 - 5 % tonnes | | | | | | 374000 | 580000 | | 1444000 | 262000 | 593262 | | 3253262 |
| IPb+Zn | | | | | | 3.93 | 3.91 | | 3.89 | 3.88 | 3.88 | | 3.89 |
| IPb | | | | | | 1.33 | 1.39 | | 1.47 | 1.51 | 1.51 | | 1.45 |
| IZn | | | | | | 2.60 | 2.53 | | 2.41 | 2.38 | 2.38 | | 2.44 |
| Ag g/t | | | | | | 25.27 | 25.78 | | 26.37 | 26.65 | 26.65 | | 26 |
| Au g/t | | | | | | 0.50 | 0.54 | | 0.61 | 0.69 | 0.69 | | 0.61 |
| Concentrate | | | | | | | | | | | | | |
| Pb Conc DMT | | | | | | 5418 | 8872 | | 23835 | 4447 | 10070 | | 52642 |
| IPb | | | | | | 60.00 | 60.00 | | 60.00 | 60.00 | 60.00 | | 60.00 |
| Ag g/t | | | | | | 893.47 | 873.68 | | 852.00 | 847.46 | 847.46 | | 859 |
| Au g/t | | | | | | 9.28 | 9.56 | | 10.08 | 11.13 | 11.13 | | 10.20 |
| Zn Conc DMT | | | | | | 13047 | 19558 | | 46055 | 8205 | 18580 | | 105445 |
| IZn | | | | | | 55.00 | 55.00 | | 55.00 | 55.00 | 55.00 | | 55.00 |
| Grass Stockpile Status | | | | | | | | | | | | | |
| + 5 % tonnes | 57716 | 269932 | 890548 | 2307195 | 192420 | | 351 | 814348 | | | | | |
| IPb+Zn | 5.89 | 5.62 | 7.14 | 7.32 | 7.48 | | 9.15 | 9.48 | | | | | |
| IPb | 2.21 | 1.99 | 2.56 | 2.61 | 2.69 | | 3.47 | 3.47 | | | | | |
| IZn | 3.68 | 3.63 | 4.59 | 4.70 | 4.78 | | 5.68 | 6.01 | | | | | |
| Ag g/t | 37.00 | 33.86 | 41.42 | 42.68 | 44.06 | | 58.00 | 59.07 | | | | | |
| Au g/t | 0.48 | 0.63 | 0.63 | 0.67 | 0.67 | | 0.88 | 0.96 | | | | | |
| Concentrate Equivalent | | | | | | | | | | | | | |
| Pb Conc DMT | 1564 | 6420 | 28861 | 76763 | 6652 | | 16 | 37941 | | | | | |
| IPb | 60.00 | 60.00 | 60.00 | 60.00 | 60.00 | | 60.00 | 60.00 | | | | | |
| Ag g/t | 810.00 | 823.61 | 801.23 | 809.85 | 811.88 | | 826.00 | 843.98 | | | | | |
| Au g/t | 5.31 | 7.66 | 6.06 | 6.33 | 6.15 | | 6.22 | 6.86 | | | | | |
| Zn Conc DMT | 3055 | 13997 | 60925 | 162893 | 12864 | | 31 | 76561 | | | | | |
| IZn | 55.00 | 55.00 | 55.00 | 55.00 | 55.00 | | 55.00 | 55.00 | | | | | |
| 4 - 5 % tonnes | | 53271 | 847846 | 1244584 | 1244584 | 1007854 | 850504 | 1700639 | 480467 | 593262 | | 0 | |
| IPb+Zn | | 3.78 | 3.88 | 3.91 | 3.91 | 3.93 | 3.91 | 3.89 | 3.89 | 3.88 | | 3.88 | |
| IPb | | 1.26 | 1.28 | 1.28 | 1.28 | 1.33 | 1.39 | 1.46 | 1.47 | 1.51 | | 1.51 | |
| IZn | | 2.52 | 2.60 | 2.63 | 2.63 | 2.60 | 2.53 | 2.43 | 2.41 | 2.38 | | 2.38 | |
| Ag g/t | | 24.00 | 24.94 | 24.64 | 24.64 | 25.27 | 25.78 | 25.89 | 26.37 | 26.65 | | 26.65 | |
| Au g/t | | 0.77 | 0.48 | 0.47 | 0.47 | 0.50 | 0.54 | 0.59 | 0.61 | 0.69 | | 0.69 | |
| Concentrate Equivalent | | | | | | | | | | | | | |
| Pb Conc DMT | | 726 | 11707 | 17255 | 17255 | 14600 | 13009 | 27847 | 7931 | 10070 | | 0 | |
| IPb | | 60.00 | 60.00 | 60.00 | 60.00 | 60.00 | 60.00 | 60.00 | 60.00 | 60.00 | | 60.00 | |
| Ag g/t | | 877.00 | 911.07 | 898.19 | 898.19 | 893.47 | 873.68 | 842.85 | 852.00 | 847.46 | | 847.46 | |
| Au g/t | | 14.66 | 9.13 | 8.70 | 8.70 | 7.29 | 9.56 | 9.81 | 10.08 | 11.13 | | 11.13 | |
| Zn Conc DMT | | 1737 | 29557 | 43998 | 43998 | 35157 | 28680 | 54665 | 15324 | 18580 | | 0 | |
| IZn | | 55.00 | 55.00 | 55.00 | 55.00 | 55.00 | 55.00 | 55.00 | 55.00 | 55.00 | | 55.00 | |

TABLE 8 - 3
GRUM ORE SCHEDULE

Ore Schedule

| | 1988 | 1989 | 1990 | 1991 | 1992 | 1993 | 1994 | 1995 | 1996 | 1997 | 1998 | 1999 | Total |
|-------------------|---------|---------|---------|---------|---------|---------|---------|---------|---------|---------|---------|----------|-------|
| <hr/> | | | | | | | | | | | | | |
| Mill Feed Summary | | | | | | | | | | | | | |
| Tonnes | 1012000 | 4015000 | 3717701 | 2035000 | 2189000 | 1187680 | 4015000 | 4015000 | 4013776 | 4024671 | 1227326 | 31452154 | |
| % Pb + Zn | 7.81 | 8.65 | 7.47 | 7.32 | 7.48 | 8.13 | 8.39 | 9.48 | 6.96 | 8.66 | 3.89 | 7.97 | |
| % Pb | 3.33 | 3.79 | 2.82 | 2.61 | 2.69 | 3.17 | 3.17 | 3.47 | 2.67 | 3.23 | 1.60 | 3.07 | |
| % Zn | 4.48 | 4.87 | 4.65 | 4.70 | 4.78 | 4.96 | 5.22 | 6.01 | 4.29 | 5.43 | 2.29 | 4.90 | |
| Ag g/t | 48 | 53 | 44 | 43 | 44 | 53 | 53 | 59 | 45 | 57 | 26 | 50 | |
| Au g/t | 0.55 | 0.61 | 0.63 | 0.67 | 0.67 | 0.72 | 0.83 | 0.96 | 0.76 | 0.99 | 0.61 | 0.76 | |
| Concentrate | | | | | | | | | | | | | |
| Pb Conc DMT | 52519 | 242119 | 142737 | 67707 | 75671 | 50619 | 169263 | 187060 | 138473 | 171700 | 23416 | 1321284 | |
| % Pb | 53.13 | 53.29 | 57.74 | 60.00 | 60.00 | 60.00 | 60.00 | 60.00 | 60.00 | 60.00 | 57.36 | 58.21 | |
| Ag g/t | 614.37 | 603.29 | 734.52 | 809.85 | 811.88 | 824.51 | 828.50 | 843.98 | 828.20 | 875.26 | 686.28 | 772.22 | |
| Au g/t | 4.23 | 3.91 | 5.33 | 6.33 | 6.15 | 5.49 | 6.39 | 6.86 | 6.89 | 7.60 | 7.61 | 5.98 | |
| Zn Conc DMT | 64871 | 283342 | 256061 | 143675 | 157724 | 90062 | 323267 | 377469 | 257072 | 335045 | 36516 | 2325105 | |
| % Zn | 54.77 | 54.71 | 54.94 | 55.00 | 55.00 | 55.00 | 55.00 | 55.00 | 55.00 | 55.00 | 53.86 | 54.93 | |
| Total Conc DMT | 117390 | 525461 | 398798 | 211382 | 233395 | 140681 | 492530 | 564529 | 395545 | 506746 | 59932 | 3646389 | |
| Recoveries | | | | | | | | | | | | | |
| Pb | 82.91 | 84.91 | 78.61 | 76.31 | 77.01 | 80.61 | 79.81 | 80.51 | 77.41 | 79.21 | 68.51 | 79.81 | |
| Zn | 78.31 | 79.31 | 81.41 | 82.61 | 82.91 | 84.11 | 84.81 | 86.11 | 82.21 | 84.41 | 69.91 | 82.81 | |
| Ag | 66.31 | 68.31 | 64.11 | 63.11 | 63.71 | 66.01 | 65.51 | 66.61 | 63.41 | 65.61 | 50.21 | 65.21 | |
| Au | 39.81 | 38.51 | 32.71 | 31.51 | 31.61 | 32.41 | 32.41 | 33.31 | 31.31 | 32.81 | 24.01 | 32.91 | |

TABLE 8 - 4
MILL FEED SUMMARY

9.0 MANPOWER

9.0 Manpower

Manpower estimates have been prepared using several techniques. Mine equipment operators have been forecasted based on the number of operating units. Additional mine operations manpower forecasts are prorated from the 1987 Curragh Resources Operating Budget, based on total tonnes mined. Mine maintenance manpower has also been prorated based on total tonnes mined.

Mill operations and maintenance manpower has been prorated based on total days of mill feed.

All other manpower forecasts are prorated based on total tonnes mined.

Manpower forecasts are listed in tables 9-1 and 9-2.

| | 1988 | 1989 | 1990 | 1991 | 1992 | 1993 | 1994 | 1995 | 1996 | 1997 | 1998 | 1999 |
|-----------------------------|-------------|--------------|--------------|--------------|--------------|--------------|--------------|--------------|--------------|--------------|--------------|-------------|
| Hourly Manpower | | | | | | | | | | | | |
| Mine Operations | | | | | | | | | | | | |
| Shovel Operator | 3.3 | 8.7 | 9.0 | 9.0 | 9.0 | 10.2 | 12.0 | 12.0 | 12.0 | 12.0 | 4.4 | |
| Driller | 1.4 | 4.3 | 4.4 | 4.4 | 4.4 | 5.1 | 6.0 | 6.0 | 6.0 | 6.0 | 2.2 | |
| Mine Utility Blaster | 0.7 | 2.1 | 2.2 | 2.2 | 2.2 | 2.5 | 3.0 | 3.0 | 3.0 | 3.0 | 1.1 | |
| Mine Utility Blaster Helper | 0.5 | 1.4 | 1.5 | 1.5 | 1.5 | 1.7 | 2.0 | 2.0 | 2.0 | 2.0 | 0.7 | |
| Haulage Truck Driver | 6.4 | 34.6 | 56.8 | 75.3 | 65.1 | 60.6 | 90.6 | 115.9 | 117.1 | 118.3 | 56.1 | 4.2 |
| Equipment Operator | 10.5 | 22.5 | 21 | 21 | 21 | 24 | 24 | 24 | 24 | 24 | 12 | 1 |
| Mine Utility Worker | 1.4 | 4.3 | 4.4 | 4.4 | 4.4 | 5.1 | 6.0 | 6.0 | 6.0 | 6.0 | 2.2 | 0.0 |
| Pumpman | 1.2 | 3.6 | 3.7 | 3.7 | 3.7 | 4.2 | 5.0 | 5.0 | 5.0 | 5.0 | 1.8 | 0.0 |
| Trainer Operator Shovel | 0.2 | 0.7 | 0.7 | 0.7 | 0.7 | 0.8 | 1.0 | 1.0 | 1.0 | 1.0 | 0.4 | 0.0 |
| Trainer Operator Truck | 0.2 | 0.7 | 0.7 | 0.7 | 0.7 | 0.8 | 1.0 | 1.0 | 1.0 | 1.0 | 0.4 | 0.0 |
| Total | 25.8 | 83.1 | 104.6 | 123.0 | 112.9 | 115.2 | 150.6 | 175.9 | 177.1 | 178.3 | 81.2 | 5.2 |
| Mine Maintenance | | | | | | | | | | | | |
| H D Mechanic | 7.6 | 22.9 | 23.6 | 23.6 | 23.6 | 27.1 | 32.0 | 32.0 | 32.0 | 32.0 | 11.6 | 0.0 |
| Welder | 3.3 | 10.0 | 10.3 | 10.3 | 10.3 | 11.9 | 14.0 | 14.0 | 14.0 | 14.0 | 5.1 | 0.0 |
| Electrician | 2.8 | 8.6 | 8.9 | 8.9 | 8.9 | 10.2 | 12.0 | 12.0 | 12.0 | 12.0 | 4.4 | 0.0 |
| Machinist | 0.7 | 2.1 | 2.2 | 2.2 | 2.2 | 2.5 | 3.0 | 3.0 | 3.0 | 3.0 | 1.1 | 0.0 |
| Carpenter | 0.2 | 0.7 | 0.7 | 0.7 | 0.7 | 0.8 | 1.0 | 1.0 | 1.0 | 1.0 | 0.4 | 0.0 |
| Plumber | 0.2 | 0.7 | 0.7 | 0.7 | 0.7 | 0.8 | 1.0 | 1.0 | 1.0 | 1.0 | 0.4 | 0.0 |
| Cableman | 0.2 | 0.7 | 0.7 | 0.7 | 0.7 | 0.8 | 1.0 | 1.0 | 1.0 | 1.0 | 0.4 | 0.0 |
| Equipment Operator | 1.4 | 4.3 | 4.4 | 4.4 | 4.4 | 5.1 | 6.0 | 6.0 | 6.0 | 6.0 | 2.2 | 0.0 |
| Lubeman | 2.1 | 6.4 | 6.6 | 6.6 | 6.6 | 7.6 | 9.0 | 9.0 | 9.0 | 9.0 | 3.3 | 0.0 |
| Utility Operator | 0.2 | 0.7 | 0.7 | 0.7 | 0.7 | 0.8 | 1.0 | 1.0 | 1.0 | 1.0 | 0.4 | 0.0 |
| Tool Crib | 0.7 | 2.1 | 2.2 | 2.2 | 2.2 | 2.5 | 3.0 | 3.0 | 3.0 | 3.0 | 1.1 | 0.0 |
| Labourers | 1.4 | 4.3 | 4.4 | 4.4 | 4.4 | 5.1 | 6.0 | 6.0 | 6.0 | 6.0 | 2.2 | 0.0 |
| Total | 21.1 | 63.7 | 65.7 | 65.7 | 65.7 | 75.4 | 89.0 | 89.0 | 89.0 | 89.0 | 32.4 | 0.0 |
| Mine Total | 46.9 | 146.8 | 170.3 | 188.7 | 178.6 | 190.6 | 239.6 | 264.9 | 266.1 | 267.3 | 113.6 | 5.2 |
| Mill Operations | | | | | | | | | | | | |
| Flotation Operator | 0.0 | 1.0 | 4.0 | 3.7 | 2.0 | 2.2 | 1.2 | 4.0 | 4.0 | 4.0 | 4.0 | 1.2 |
| Flotation Helper | 0.0 | 0.3 | 1.0 | 0.9 | 0.5 | 0.5 | 0.3 | 1.0 | 1.0 | 1.0 | 1.0 | 0.3 |
| Grinding Operator | 0.0 | 0.8 | 3.0 | 2.8 | 1.5 | 1.6 | 0.9 | 3.0 | 3.0 | 3.0 | 3.0 | 0.9 |
| Grinding Helper | 0.0 | 0.3 | 1.0 | 0.9 | 0.5 | 0.5 | 0.3 | 1.0 | 1.0 | 1.0 | 1.0 | 0.3 |
| Crusher Operator | 0.0 | 0.8 | 3.0 | 2.8 | 1.5 | 1.6 | 0.9 | 3.0 | 3.0 | 3.0 | 3.0 | 0.9 |
| Crusher Helper | 0.0 | 0.8 | 3.0 | 2.8 | 1.5 | 1.6 | 0.9 | 3.0 | 3.0 | 3.0 | 3.0 | 0.9 |
| Filter Operator | 0.0 | 0.8 | 3.0 | 2.8 | 1.5 | 1.6 | 0.9 | 3.0 | 3.0 | 3.0 | 3.0 | 0.9 |
| Coal Crusher/Filter Op. | 0.0 | 0.8 | 3.0 | 2.8 | 1.5 | 1.6 | 0.9 | 3.0 | 3.0 | 3.0 | 3.0 | 0.9 |
| Trainee | 0.0 | 1.0 | 4.0 | 3.7 | 2.0 | 2.2 | 1.2 | 4.0 | 4.0 | 4.0 | 4.0 | 1.2 |
| Labourer | 0.0 | 3.8 | 15.0 | 13.9 | 7.6 | 8.2 | 4.4 | 15.0 | 15.0 | 15.0 | 15.0 | 4.6 |
| Heat Plant Operator | 0.0 | 1.0 | 4.0 | 3.7 | 2.0 | 2.2 | 1.2 | 4.0 | 4.0 | 4.0 | 4.0 | 1.2 |
| Heat Plant Helper | 0.0 | 1.0 | 4.0 | 3.7 | 2.0 | 2.2 | 1.2 | 4.0 | 4.0 | 4.0 | 4.0 | 1.2 |
| Reagent Operator | 0.0 | 0.5 | 2.0 | 1.9 | 1.0 | 1.1 | 0.6 | 2.0 | 2.0 | 2.0 | 2.0 | 0.6 |
| Reagent Helper | 0.0 | 0.3 | 1.0 | 0.9 | 0.5 | 0.5 | 0.3 | 1.0 | 1.0 | 1.0 | 1.0 | 0.3 |
| Tailings Operator | 0.0 | 0.3 | 1.0 | 0.9 | 0.5 | 0.5 | 0.3 | 1.0 | 1.0 | 1.0 | 1.0 | 0.3 |
| Equipment Operator | 0.0 | 0.5 | 2.0 | 1.9 | 1.0 | 1.1 | 0.6 | 2.0 | 2.0 | 2.0 | 2.0 | 0.6 |
| Steel Crew | 0.0 | 0.5 | 2.0 | 1.9 | 1.0 | 1.1 | 0.6 | 2.0 | 2.0 | 2.0 | 2.0 | 0.6 |
| Filter Sector | 0.0 | 0.3 | 1.0 | 0.9 | 0.5 | 0.5 | 0.3 | 1.0 | 1.0 | 1.0 | 1.0 | 0.3 |
| Sample Bucher | 0.0 | 0.8 | 3.0 | 2.8 | 1.5 | 1.6 | 0.9 | 3.0 | 3.0 | 3.0 | 3.0 | 0.9 |
| Total | 0.0 | 15.1 | 60.0 | 55.6 | 30.1 | 32.7 | 17.7 | 60.0 | 60.0 | 60.0 | 60.1 | 18.7 |

TABLE 9 - 1
HOURLY MANPOWER

| | 1988 | 1989 | 1990 | 1991 | 1992 | 1993 | 1994 | 1995 | 1996 | 1997 | 1998 | 1999 |
|----------------------|------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|------|
| Mill Maintenance | | | | | | | | | | | | |
| Mill Mechanic | 0.0 | 8.6 | 34.0 | 31.5 | 17.2 | 18.5 | 10.1 | 34.0 | 34.0 | 34.0 | 34.1 | 10.4 |
| Electrician | 0.0 | 2.3 | 9.0 | 8.3 | 4.6 | 4.9 | 2.7 | 9.0 | 9.0 | 9.0 | 9.0 | 2.8 |
| Instrument Mechanic | 0.0 | 1.3 | 5.0 | 4.6 | 2.5 | 2.7 | 1.5 | 5.0 | 5.0 | 5.0 | 5.0 | 1.5 |
| Total | 0.0 | 12.1 | 48.0 | 44.4 | 24.3 | 26.2 | 14.2 | 48.0 | 48.0 | 48.0 | 48.1 | 14.7 |
| Mill Total | 0.0 | 27.2 | 108.0 | 100.0 | 54.7 | 58.9 | 31.9 | 108.0 | 108.0 | 108.0 | 108.3 | 33.0 |
| Materials Management | | | | | | | | | | | | |
| Warehouse Person A | 0.5 | 1.4 | 1.5 | 1.5 | 1.5 | 1.7 | 2.0 | 2.0 | 2.0 | 2.0 | 0.7 | 0.0 |
| Warehouse Person B | 0.7 | 2.1 | 2.2 | 2.2 | 2.2 | 2.5 | 3.0 | 3.0 | 3.0 | 3.0 | 1.1 | 0.0 |
| Warehouse Person C | 0.5 | 1.4 | 1.5 | 1.5 | 1.5 | 1.7 | 2.0 | 2.0 | 2.0 | 2.0 | 0.7 | 0.0 |
| Warehouse/First Aid | 0.7 | 2.1 | 2.2 | 2.2 | 2.2 | 2.5 | 3.0 | 3.0 | 3.0 | 3.0 | 1.1 | 0.0 |
| Total | 2.4 | 7.2 | 7.4 | 7.4 | 7.4 | 8.5 | 10.0 | 10.0 | 10.0 | 10.0 | 3.6 | 0.0 |
| Mine Total | 46.9 | 146.8 | 170.3 | 188.7 | 178.6 | 190.6 | 239.6 | 264.9 | 266.1 | 267.3 | 113.6 | 5.2 |
| Mill Total | 0.0 | 27.2 | 108.0 | 100.0 | 54.7 | 58.9 | 31.9 | 108.0 | 108.0 | 108.0 | 108.3 | 33.0 |
| Total Hourly | 49.3 | 181.2 | 285.6 | 296.1 | 240.7 | 257.9 | 281.6 | 382.9 | 384.1 | 385.2 | 225.5 | 38.3 |

TABLE 9 - 1 (CONTINUED)

| | 1988 | 1989 | 1990 | 1991 | 1992 | 1993 | 1994 | 1995 | 1996 | 1997 | 1998 | 1999 |
|----------------------------|------|------|------|------|------|------|------|------|------|------|------|------|
| Salaried Manpower | | | | | | | | | | | | |
| Mine Operations | | | | | | | | | | | | |
| General Foreman | 0.2 | 0.7 | 0.7 | 0.7 | 0.7 | 0.8 | 1.0 | 1.0 | 1.0 | 1.0 | 0.4 | 0.0 |
| Drill & Blast Foreman | 0.2 | 0.7 | 0.7 | 0.7 | 0.7 | 0.8 | 1.0 | 1.0 | 1.0 | 1.0 | 0.4 | 0.0 |
| Mine Training Foreman | 0.5 | 1.4 | 1.5 | 1.5 | 1.5 | 1.7 | 2.0 | 2.0 | 2.0 | 2.0 | 0.7 | 0.0 |
| Mine Foreman | 0.7 | 2.1 | 2.2 | 2.2 | 2.2 | 2.5 | 3.0 | 3.0 | 3.0 | 3.0 | 1.1 | 0.0 |
| Mine Clerk | 0.2 | 0.7 | 0.7 | 0.7 | 0.7 | 0.8 | 1.0 | 1.0 | 1.0 | 1.0 | 0.4 | 0.0 |
| Mine Engineering | | | | | | | | | | | | |
| Chief Engineer | 0.2 | 0.7 | 0.7 | 0.7 | 0.7 | 0.8 | 1.0 | 1.0 | 1.0 | 1.0 | 0.4 | 0.0 |
| Senior Mine Engineer | 0.2 | 0.7 | 0.7 | 0.7 | 0.7 | 0.8 | 1.0 | 1.0 | 1.0 | 1.0 | 0.4 | 0.0 |
| Intermediate Engineer | 0.2 | 0.7 | 0.7 | 0.7 | 0.7 | 0.8 | 1.0 | 1.0 | 1.0 | 1.0 | 0.4 | 0.0 |
| Mine Engineer | 0.2 | 0.7 | 0.7 | 0.7 | 0.7 | 0.8 | 1.0 | 1.0 | 1.0 | 1.0 | 0.4 | 0.0 |
| Senior Geologist | 0.2 | 0.7 | 0.7 | 0.7 | 0.7 | 0.8 | 1.0 | 1.0 | 1.0 | 1.0 | 0.4 | 0.0 |
| Geologist | 0.7 | 2.1 | 2.2 | 2.2 | 2.2 | 2.5 | 3.0 | 3.0 | 3.0 | 3.0 | 1.1 | 0.0 |
| Surveyor | 0.7 | 2.1 | 2.2 | 2.2 | 2.2 | 2.5 | 3.0 | 3.0 | 3.0 | 3.0 | 1.1 | 0.0 |
| Technician | 0.2 | 0.7 | 0.7 | 0.7 | 0.7 | 0.8 | 1.0 | 1.0 | 1.0 | 1.0 | 0.4 | 0.0 |
| Clerk | 0.2 | 0.7 | 0.7 | 0.7 | 0.7 | 0.8 | 1.0 | 1.0 | 1.0 | 1.0 | 0.4 | 0.0 |
| Mine Maintenance | | | | | | | | | | | | |
| General Foreman | 0.2 | 0.7 | 0.7 | 0.7 | 0.7 | 0.8 | 1.0 | 1.0 | 1.0 | 1.0 | 0.4 | 0.0 |
| Shop Foreman | 0.7 | 2.1 | 2.2 | 2.2 | 2.2 | 2.5 | 3.0 | 3.0 | 3.0 | 3.0 | 1.1 | 0.0 |
| Field Maintenance Foreman | 0.5 | 1.4 | 1.5 | 1.5 | 1.5 | 1.7 | 2.0 | 2.0 | 2.0 | 2.0 | 0.7 | 0.0 |
| Mine Electrical Foreman | 0.2 | 0.7 | 0.7 | 0.7 | 0.7 | 0.8 | 1.0 | 1.0 | 1.0 | 1.0 | 0.4 | 0.0 |
| Senior Maintenance Planner | 0.2 | 0.7 | 0.7 | 0.7 | 0.7 | 0.8 | 1.0 | 1.0 | 1.0 | 1.0 | 0.4 | 0.0 |
| Maintenance Engineer | 0.2 | 0.7 | 0.7 | 0.7 | 0.7 | 0.8 | 1.0 | 1.0 | 1.0 | 1.0 | 0.4 | 0.0 |
| Yards & Services Foreman | 0.2 | 0.7 | 0.7 | 0.7 | 0.7 | 0.8 | 1.0 | 1.0 | 1.0 | 1.0 | 0.4 | 0.0 |
| Planning Clerk | 0.5 | 1.4 | 1.5 | 1.5 | 1.5 | 1.7 | 2.0 | 2.0 | 2.0 | 2.0 | 0.7 | 0.0 |
| Mine Total | 7.8 | 23.6 | 24.4 | 24.4 | 24.4 | 28.0 | 33.0 | 33.0 | 33.0 | 33.0 | 12.0 | 0.0 |
| Mill Operations | | | | | | | | | | | | |
| General Foreman | 0.0 | 0.3 | 1.0 | 0.9 | 0.5 | 0.5 | 0.3 | 1.0 | 1.0 | 1.0 | 1.0 | 0.3 |
| Training Foreman | 0.0 | 0.3 | 1.0 | 0.9 | 0.5 | 0.5 | 0.3 | 1.0 | 1.0 | 1.0 | 1.0 | 0.3 |
| Labour/Surface Foreman | 0.0 | 0.3 | 1.0 | 0.9 | 0.5 | 0.5 | 0.3 | 1.0 | 1.0 | 1.0 | 1.0 | 0.3 |
| Shift Foreman | 0.0 | 0.8 | 3.0 | 2.8 | 1.5 | 1.6 | 0.9 | 3.0 | 3.0 | 3.0 | 3.0 | 0.9 |
| Mill Metallurgical | | | | | | | | | | | | |
| Chief Metallurgist | 0.0 | 0.3 | 1.0 | 0.9 | 0.5 | 0.5 | 0.3 | 1.0 | 1.0 | 1.0 | 1.0 | 0.3 |
| Senior Metallurgist | 0.0 | 0.3 | 1.0 | 0.9 | 0.5 | 0.5 | 0.3 | 1.0 | 1.0 | 1.0 | 1.0 | 0.3 |
| Metallurgist | 0.0 | 0.3 | 1.0 | 0.9 | 0.5 | 0.5 | 0.3 | 1.0 | 1.0 | 1.0 | 1.0 | 0.3 |
| Process Control Engineer | 0.0 | 0.3 | 1.0 | 0.9 | 0.5 | 0.5 | 0.3 | 1.0 | 1.0 | 1.0 | 1.0 | 0.3 |
| Chief Assayer | 0.0 | 0.3 | 1.0 | 0.9 | 0.5 | 0.5 | 0.3 | 1.0 | 1.0 | 1.0 | 1.0 | 0.3 |
| Assayer | 0.0 | 1.5 | 6.0 | 5.6 | 3.0 | 3.3 | 1.8 | 6.0 | 6.0 | 6.0 | 6.0 | 1.8 |
| Met Technician | 0.0 | 1.0 | 4.0 | 3.7 | 2.0 | 2.2 | 1.2 | 4.0 | 4.0 | 4.0 | 4.0 | 1.2 |
| Mill Maintenance | | | | | | | | | | | | |
| General Foreman | 0.0 | 0.3 | 1.0 | 0.9 | 0.5 | 0.5 | 0.3 | 1.0 | 1.0 | 1.0 | 1.0 | 0.3 |
| Mechanical Foreman | 0.0 | 0.8 | 3.0 | 2.8 | 1.5 | 1.6 | 0.9 | 3.0 | 3.0 | 3.0 | 3.0 | 0.9 |
| Chief Heat Plant Engineer | 0.0 | 0.3 | 1.0 | 0.9 | 0.5 | 0.5 | 0.3 | 1.0 | 1.0 | 1.0 | 1.0 | 0.3 |
| Electrical Foreman | 0.0 | 0.3 | 1.0 | 0.9 | 0.5 | 0.5 | 0.3 | 1.0 | 1.0 | 1.0 | 1.0 | 0.3 |
| Instrument/Electrical | 0.0 | 0.3 | 1.0 | 0.9 | 0.5 | 0.5 | 0.3 | 1.0 | 1.0 | 1.0 | 1.0 | 0.3 |
| Maintenance Planner | 0.0 | 0.3 | 1.0 | 0.9 | 0.5 | 0.5 | 0.3 | 1.0 | 1.0 | 1.0 | 1.0 | 0.3 |
| Maintenance Planner | 0.0 | 0.3 | 1.0 | 0.9 | 0.5 | 0.5 | 0.3 | 1.0 | 1.0 | 1.0 | 1.0 | 0.3 |
| Mill Total | 0.0 | 7.6 | 30.0 | 27.8 | 15.2 | 16.4 | 8.9 | 30.0 | 30.0 | 30.0 | 30.1 | 9.2 |

TABLE 9 - 2
SALARIED MANPOWER

| | 1988 | 1989 | 1990 | 1991 | 1992 | 1993 | 1994 | 1995 | 1996 | 1997 | 1998 | 1999 |
|------------------------------------|------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|------|
| <hr/> | | | | | | | | | | | | |
| Controllers Group | | | | | | | | | | | | |
| Chief Accountant | 0.2 | 0.7 | 0.7 | 0.7 | 0.7 | 0.8 | 1.0 | 1.0 | 1.0 | 1.0 | 0.4 | 0.0 |
| Information Services | 0.2 | 0.7 | 0.7 | 0.7 | 0.7 | 0.8 | 1.0 | 1.0 | 1.0 | 1.0 | 0.4 | 0.0 |
| Payroll Supervisor | 0.2 | 0.7 | 0.7 | 0.7 | 0.7 | 0.8 | 1.0 | 1.0 | 1.0 | 1.0 | 0.4 | 0.0 |
| Accounting Supervisor | 0.5 | 1.4 | 1.5 | 1.5 | 1.5 | 1.7 | 2.0 | 2.0 | 2.0 | 2.0 | 0.7 | 0.0 |
| Programmer | 0.2 | 0.7 | 0.7 | 0.7 | 0.7 | 0.8 | 1.0 | 1.0 | 1.0 | 1.0 | 0.4 | 0.0 |
| Computer Operator | 0.2 | 0.7 | 0.7 | 0.7 | 0.7 | 0.8 | 1.0 | 1.0 | 1.0 | 1.0 | 0.4 | 0.0 |
| Junior Accountant | 0.2 | 0.7 | 0.7 | 0.7 | 0.7 | 0.8 | 1.0 | 1.0 | 1.0 | 1.0 | 0.4 | 0.0 |
| Accounts Payable Clerk | 0.7 | 2.1 | 2.2 | 2.2 | 2.2 | 2.5 | 3.0 | 3.0 | 3.0 | 3.0 | 1.1 | 0.0 |
| Payroll Clerk | 0.5 | 1.4 | 1.5 | 1.5 | 1.5 | 1.7 | 2.0 | 2.0 | 2.0 | 2.0 | 0.7 | 0.0 |
| Total | 3.1 | 9.3 | 9.6 | 9.6 | 9.6 | 11.0 | 13.0 | 13.0 | 13.0 | 13.0 | 4.7 | 0.0 |
| Materials Management | | | | | | | | | | | | |
| Chief Warehouseman | 0.2 | 0.7 | 0.7 | 0.7 | 0.7 | 0.8 | 1.0 | 1.0 | 1.0 | 1.0 | 0.4 | 0.0 |
| Buyer/Expiditer | 0.5 | 1.4 | 1.5 | 1.5 | 1.5 | 1.7 | 2.0 | 2.0 | 2.0 | 2.0 | 0.7 | 0.0 |
| Warehouse Supervisor | 0.2 | 0.7 | 0.7 | 0.7 | 0.7 | 0.8 | 1.0 | 1.0 | 1.0 | 1.0 | 0.4 | 0.0 |
| Clerk | 0.2 | 0.7 | 0.7 | 0.7 | 0.7 | 0.8 | 1.0 | 1.0 | 1.0 | 1.0 | 0.4 | 0.0 |
| Total | 1.2 | 3.6 | 3.7 | 3.7 | 3.7 | 4.2 | 5.0 | 5.0 | 5.0 | 5.0 | 1.8 | 0.0 |
| Human Resources | | | | | | | | | | | | |
| Personnel Supervisor | 0.2 | 0.7 | 0.7 | 0.7 | 0.7 | 0.8 | 1.0 | 1.0 | 1.0 | 1.0 | 0.4 | 0.0 |
| Safety & Training Supervisor | 0.2 | 0.7 | 0.7 | 0.7 | 0.7 | 0.8 | 1.0 | 1.0 | 1.0 | 1.0 | 0.4 | 0.0 |
| Secretary | 0.5 | 1.4 | 1.5 | 1.5 | 1.5 | 1.7 | 2.0 | 2.0 | 2.0 | 2.0 | 0.7 | 0.0 |
| Recruiter | 0.2 | 0.7 | 0.7 | 0.7 | 0.7 | 0.8 | 1.0 | 1.0 | 1.0 | 1.0 | 0.4 | 0.0 |
| Safety Officer | 0.2 | 0.7 | 0.7 | 0.7 | 0.7 | 0.8 | 1.0 | 1.0 | 1.0 | 1.0 | 0.4 | 0.0 |
| Clerk | 0.2 | 0.7 | 0.7 | 0.7 | 0.7 | 0.8 | 1.0 | 1.0 | 1.0 | 1.0 | 0.4 | 0.0 |
| Receptionist | 0.2 | 0.7 | 0.7 | 0.7 | 0.7 | 0.8 | 1.0 | 1.0 | 1.0 | 1.0 | 0.4 | 0.0 |
| Total | 1.9 | 5.7 | 5.9 | 5.9 | 5.9 | 6.8 | 8.0 | 8.0 | 8.0 | 8.0 | 2.9 | 0.0 |
| Salaried Total | 14.0 | 49.8 | 73.6 | 71.3 | 58.8 | 66.3 | 67.9 | 89.0 | 89.0 | 89.0 | 51.5 | 9.2 |
| Project Total Excluding Contractor | 63.3 | 231.0 | 359.2 | 367.4 | 299.4 | 324.3 | 349.5 | 471.9 | 473.1 | 474.2 | 277.0 | 47.4 |

TABLE 9 - 2 (CONTINUED)

10.0 CAPITAL COSTS

10.1 Structures and Equipment

Cost estimates for structures and equipment are based on industry experience. In preparing the estimate, assistance was provided by Kliborn Engineering Ltd.

Major capital items in this category include:

- a 5000 metre powerline extension
- a 138 kV substation
- a lube and fuel station
- a small maintenance facility
- support equipment
- Vangorda haul road

Although previous design work on the Vangorda haul road was done in some detail, the alignment has been changed enough so that the previous construction quantity estimates are no longer valid. Also at question at this time is whether the construction can be completed by Curragh Resources or if it will be let out to contract. A total of \$10 million has been allocated for the road construction; this amount is exclusive of the section currently under construction, the crossing of the North Fork of Rose Creek. This cost has been estimated at \$1.24 million.

No detailed design work has been done on any of the above structures, so the costs presented here are only order of magnitude estimates.

10.2 Equipment Replacement

Support equipment has been scheduled for replacement after the following schedule.

| Equipment | Hours | Years |
|------------------------|--------|-------|
| Tracked dozers | 23,000 | 4 |
| Rubber-tired equipment | 40,000 | 6 |

Scheduling equipment replacement costs for production equipment is more difficult as the replacement schedule is dependent on Faro pit operations. Since it would be unreasonable to assume that replacement of production equipment would not be required, a replacement schedule has been included.

Major equipment replacement costs include:

- replacement of the shovel fleet, one unit annually, commencing in 1990
- replacement of the loader fleet, one unit annually, commencing in 1990
- an annual haul truck replacement allowance of 7.5 million for the years 1992 through 1996 inclusive. This represents an annual replacement of five haul trucks.
- replacement of one blasthole drill in 1990 and 1993

10.3 Contingencies

Other recurring capital items to be expected over the life of the project would include:

- upgrading of electric power distribution systems
- replacement of service equipment
- building repairs and modifications
- replacement of electrical and mechanical tools
- technological upgrades of production, maintenance, and technical equipment
- tailing systems upgrades
- mill modifications

An allowance for these items has been made based on the 1987 proposed Curragh Resources capital expenditures, prorated on tonnes mined.

Capital Forecast

| Item | 1987 | 1988 | 1989 | 1990 | 1991 | 1992 | 1993 | 1994 | 1995 | 1996 | 1997 | 1998 | 1999 |
|-----------------------------------|-----------|------------|-----------|-----------|-----------|------------|------------|------------|------------|------------|-----------|-----------|------|
| Structures and Equipment | | | | | | | | | | | | | |
| Power | | | | | | | | | | | | | |
| 5000 metre powerline | 220,000 | | | | | | | | | | | | |
| Substation: 138 kV | 200,000 | | | | | | | | | | | | |
| Equipment | | | | | | | | | | | | | |
| Dozer: Cat D8 with ripper | | 560,000 | | | | 560,000 | | | | | | | |
| Dozer: Cat D9 with ripper | | 1,260,000 | | | | | | | | | | | |
| Rubber tired Dozer: Cat 824 | | 530,000 | 530,000 | | | | | | | | | | |
| Grader: Cat 166 | | 630,000 | 630,000 | | | | | | | | | | |
| 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 | | | | | | | | | | | |
| Shovel Moves | 25,000 | 25,000 | 25,000 | | | | 25,000 | | | | | | |
| Subtotal Structures and Equipment | 5,995,000 | 9,305,000 | 985,000 | 0 | 0 | 560,000 | 25,000 | 0 | 0 | 0 | 0 | 0 | 0 |
| Equipment replacement | | | | | | | | | | | | | |
| Shovel | | | 4,000,000 | 4,000,000 | 4,000,000 | 4,000,000 | | | | | | | |
| Haul Truck | | | | | 7,500,000 | 7,500,000 | 7,500,000 | 7,500,000 | 7,500,000 | | | | |
| Blasthole Drill | | | 500,000 | | | 500,000 | | | | | | | |
| Dozer: Cat D8 with ripper | | | | | 1,120,000 | | | | | 1,120,000 | | | |
| Dozer: Cat D9 with ripper | | | | | 1,260,000 | | | | | 1,260,000 | | | |
| Rubber tired Dozer: Cat 824 | | | | | | | | 1,120,000 | | | | | |
| Grader: Cat 166 | | | | | | | | 860,000 | | | | | |
| Front End Loader | | | 1,100,000 | 1,100,000 | | | | | | | | | |
| Subtotal Equipment Replacement | 0 | 0 | 0 | 5,600,000 | 5,100,000 | 13,880,000 | 12,000,000 | 9,480,000 | 7,500,000 | 9,880,000 | 0 | 0 | 0 |
| Contingencies | | | | | | | | | | | | | |
| All Items | 1,165,535 | 3,522,718 | 3,632,109 | 3,632,108 | 3,632,107 | 4,168,472 | 4,920,089 | 4,920,090 | 4,920,089 | 4,920,089 | 1,790,397 | 0 | |
| Total Capital Cost | 5,995,000 | 10,470,535 | 4,507,718 | 9,232,109 | 8,732,108 | 18,072,107 | 16,193,472 | 14,400,089 | 12,420,090 | 14,800,089 | 4,920,089 | 1,790,397 | 0 |

TABLE 10 - 1
CAPITAL COST SCHEDULE

11.0 OPERATING COSTS

11.0 Operating Costs

Operating cost estimates are based on the 1987 Curragh Resources Operating Budget and are expressed in constant (fourth quarter 1986) Canadian Dollars.

The operating cost estimate is based strictly on the Vangorda Plateau mining, milling and concentrate handling activities and does not include the cost of independent Faro operations which may be operating concurrently.

Mining, milling and concentrate handling costs have been determined from unit costs, while general and administrative costs have been prorated based on total mined tonnages. Contract hauling, when required, has been costed at 125% of Curragh operating costs.

Because the existence of the Whitehorse and Toronto offices is expected to be independent of Vangorda Plateau mining activities (with the exception of the Whitehorse transportation function) costs for Whitehorse administration and technical services and for Toronto office are not included in this estimate.

Unit costs for all activities are listed in table 11-1, and the annual operating cost estimate is listed in table 11-2.

| Activity | Cost | Basis |
|---|--------|----------------------|
| <u>Mining Operations</u> | | |
| Drill and Blast - Rock | .200 | \$/tonne mined |
| Drill and Blast - Overburden | .020 | \$/tonne mined |
| Loading - Marion Shovel | 227.06 | \$/operating hour |
| Loading - P & H Shovel | 151.13 | \$/operating hour |
| Hauling - Wabco | 107.76 | \$/operating hour |
| Hauling - Euclid | 126.90 | \$/operating hour |
| Hauling - Contract | 158.63 | \$/operating hour |
| Equipment - L800 | 148.21 | \$/operating hour |
| Equipment - Dozer | 73.23 | \$/operating hour |
| Equipment - RTD | 66.23 | \$/operating hour |
| Equipment - Grader | 64.18 | \$/operating hour |
| Pit Dewatering | .012 | \$/tonne mined |
| General Services | .249 | \$/tonne mined |
| Technical Services | .042 | \$/tonne mined |
| <u>Milling Operations</u> | | |
| Primary Crushing | .133 | \$/tonne feed |
| Secondary Crushing | .216 | \$/tonne feed |
| Grinding | 1.562 | \$/tonne feed |
| Flotation | .258 | \$/tonne feed |
| Dewatering | 1.513 | \$/tonne concentrate |
| Drying | 2.576 | \$/tonne concentrate |
| Reagent Area | 1.672 | \$/tonne feed |
| Other | 2.254 | \$/tonne feed |
| Faro G & A | .152 | \$/tonne mined |
| Concentrate Handling | 50.147 | \$/tonne concentrate |
| Freight Backhaul
(reagents and grinding media) | 1.156 | \$/tonne feed |

Table 11-1
Unit Operating Costs

Operating Cost Estimate

| | 1988 | 1989 | 1990 | 1991 | 1992 | 1993 | 1994 | 1995 | 1996 | 1997 | 1998 | 1999 |
|-------------|---------|----------|----------|----------|----------|----------|----------|----------|----------|----------|----------|---------|
| Production | | | | | | | | | | | | |
| Waste O/B | 5857089 | 10544202 | 5351051 | 2384716 | 5351719 | 6451748 | 2082219 | 1731284 | 826249 | 69507 | 0 | |
| Mined | 8018362 | 24234733 | 24987293 | 24987287 | 24987285 | 28677233 | 33848026 | 33848030 | 33848023 | 33848023 | 12317139 | |
| Feed | 0 | 1012000 | 4015000 | 3717701 | 2035000 | 2189000 | 1187680 | 4015000 | 4015000 | 4013776 | 4024671 | 1227326 |
| Concentrate | 0 | 117390 | 525461 | 398798 | 211382 | 233395 | 140681 | 492530 | 564529 | 395345 | 506746 | 59932 |

| Cost | 1988 | 1989 | 1990 | 1991 | 1992 | 1993 | 1994 | 1995 | 1996 | 1997 | 1998 | 1999 |
|------|------|------|------|------|------|------|------|------|------|------|------|------|
|------|------|------|------|------|------|------|------|------|------|------|------|------|

| Unit Costs | \$/DMT
Mined | \$/DMT
Feed | \$/DMT
Conc | | | | | | | | | | | | |
|------------------------|-----------------|----------------|----------------|-----------|------------|------------|------------|------------|------------|------------|------------|------------|------------|------------|------------|
| Mining Operations | | | | | | | | | | | | | | | |
| Drill and Blast - Rock | 0.200 | | | 432,255 | 2,738,106 | 3,927,248 | 4,520,514 | 3,927,113 | 4,445,097 | 6,353,161 | 6,423,349 | 6,604,355 | 6,753,703 | 2,463,428 | 0 |
| Drill and Blast - O/B | 0.020 | | | 117,142 | 210,884 | 107,021 | 47,694 | 107,034 | 129,035 | 41,644 | 34,626 | 16,525 | 1,390 | 0 | 0 |
| Pit Dewatering | 0.012 | | | 96,220 | 290,817 | 299,848 | 299,847 | 299,847 | 344,127 | 406,176 | 406,176 | 406,176 | 406,176 | 147,806 | 0 |
| General Services | 0.249 | | | 1,996,572 | 6,034,449 | 6,221,836 | 6,221,834 | 6,221,834 | 7,140,632 | 8,428,158 | 8,428,158 | 8,428,158 | 8,428,158 | 3,066,968 | 0 |
| Technical Services | 0.042 | | | 336,771 | 1,017,859 | 1,049,466 | 1,049,466 | 1,049,466 | 1,204,444 | 1,421,617 | 1,421,617 | 1,421,617 | 1,421,617 | 517,320 | 0 |
| Milling Operations | | | | | | | | | | | | | | | |
| Primary Crushing | | 0.133 | | 0 | 134,596 | 533,995 | 494,454 | 270,655 | 291,137 | 157,961 | 533,995 | 533,995 | 533,832 | 535,281 | 163,234 |
| Secondary Crushing | | 0.216 | | 0 | 218,592 | 867,240 | 803,023 | 439,560 | 472,824 | 256,539 | 867,240 | 867,240 | 866,976 | 869,329 | 265,102 |
| Grinding | | 1.562 | | 0 | 1,580,744 | 6,271,430 | 5,807,049 | 3,178,670 | 3,419,218 | 1,853,156 | 6,271,430 | 6,271,430 | 6,269,518 | 6,286,536 | 1,917,083 |
| Flotation | | 0.258 | | 0 | 261,096 | 1,035,870 | 959,167 | 525,030 | 564,762 | 306,421 | 1,035,870 | 1,035,870 | 1,035,554 | 1,038,365 | 316,650 |
| Dewatering | | | 1.513 | 0 | 177,611 | 795,023 | 603,382 | 319,821 | 353,126 | 212,850 | 745,199 | 854,132 | 598,460 | 766,706 | 90,677 |
| Drying | | | 2.576 | 0 | 302,396 | 1,353,588 | 1,027,304 | 544,520 | 601,224 | 362,394 | 1,268,759 | 1,454,227 | 1,018,924 | 1,305,377 | 154,385 |
| Reagent Area | | 1.672 | | 0 | 1,692,064 | 6,713,080 | 6,215,996 | 3,402,520 | 3,660,008 | 1,985,801 | 6,713,080 | 6,713,080 | 6,711,033 | 6,729,250 | 2,052,089 |
| Other | | 2.254 | | 0 | 2,281,048 | 9,049,810 | 8,379,698 | 4,586,890 | 4,934,006 | 2,677,031 | 9,049,810 | 9,049,810 | 9,047,051 | 9,071,608 | 2,766,393 |
| Faro 6 & A | 0.152 | | | 1,218,791 | 3,683,679 | 3,798,069 | 3,798,068 | 3,798,067 | 4,258,940 | 5,144,900 | 5,144,901 | 5,144,899 | 5,144,899 | 1,872,205 | 0 |
| Concentrate Handling | | | 50.147 | 0 | 5,886,747 | 26,350,308 | 19,998,532 | 10,600,169 | 11,704,037 | 7,054,730 | 24,698,926 | 28,309,437 | 19,835,395 | 25,411,771 | 3,005,411 |
| Freight Backhaul | | 1.156 | | 0 | 1,169,872 | 4,641,340 | 4,297,662 | 2,352,460 | 2,530,484 | 1,372,958 | 4,641,340 | 4,641,340 | 4,639,925 | 4,652,520 | 1,418,789 |
| Subtotal Unit Costs | | | | | | | | | | | | | | | |
| | | | | 4,197,751 | 27,680,560 | 73,015,172 | 64,523,692 | 41,623,657 | 46,153,100 | 58,037,500 | 77,684,476 | 81,752,292 | 72,714,612 | 64,734,469 | 12,149,813 |

TABLE 11 - 2
OPERATING COST ESTIMATE

Hour Based Costs

| | 1988 | 1989 | 1990 | 1991 | 1992 | 1993 | 1994 | 1995 | 1996 | 1997 | 1998 | 1999 |
|-----------------------------|--------|-----------|------------|------------|------------|------------|------------|------------|-------------|-------------|-------------|------------|
| <hr/> | | | | | | | | | | | | |
| Operating Hours | | | | | | | | | | | | |
| Loading - Marion Shovel | 5212 | 5212 | 5212 | 5212 | 5212 | 5212 | 5212 | 5212 | 5212 | 5212 | 2335 | |
| Loading - P & H Shovel | 538 | 12121 | 12658 | 12658 | 12658 | 15294 | 18987 | 18987 | 18987 | 18987 | 6421 | |
| Hauling - Mabco | 751 | 17246 | 61876 | 88513 | 59937 | 46537 | 83167 | 92951 | 93206 | 92951 | 37592 | -0 |
| Hauling - Euclid | 11046 | 44661 | 35094 | 39042 | 47766 | 55417 | 61261 | 61261 | 61429 | 61261 | 61261 | 7875 |
| Hauling - Contract | | | | | | | 12336 | 48164 | 50491 | 52478 | | |
| Equipment - LB800 | 0 | 2439 | 9675 | 8958 | 4904 | 5275 | 2862 | 9675 | 9675 | 9672 | 9698 | 2957 |
| Equipment - Dozer | 9494 | 12658 | 12658 | 18987 | 18987 | 25316 | 25316 | 25316 | 25316 | 25316 | 12658 | |
| Equipment - RTD | 6329 | 12658 | 12658 | 12658 | 12658 | 12658 | 12658 | 12658 | 12658 | 12658 | 6329 | |
| Equipment - Grader | 6329 | 9494 | 12658 | 12658 | 12658 | 12658 | 12658 | 12658 | 12658 | 12658 | 6329 | 3165 |
| <hr/> | | | | | | | | | | | | |
| Operating Cost \$/Hour | | | | | | | | | | | | |
| Loading - Marion Shovel | 227.06 | 1,183,437 | 1,183,437 | 1,183,437 | 1,183,437 | 1,183,437 | 1,183,437 | 1,183,437 | 1,183,437 | 1,183,437 | 530,185 | 0 |
| Loading - P & H Shovel | 151.13 | 81,308 | 1,831,847 | 1,913,004 | 1,913,004 | 2,311,382 | 2,869,505 | 2,869,505 | 2,869,505 | 2,869,505 | 970,406 | 0 |
| Hauling - Mabco | 107.76 | 80,882 | 1,858,390 | 6,667,738 | 9,538,124 | 6,458,824 | 5,017,017 | 8,962,046 | 10,016,404 | 10,043,922 | 4,050,902 | (0) |
| Hauling - Euclid | 126.90 | 1,401,759 | 5,667,506 | 4,453,433 | 4,954,487 | 6,061,527 | 7,032,443 | 7,774,046 | 7,774,046 | 7,774,046 | 7,774,046 | 999,331 |
| Hauling - Contract | 158.63 | 0 | 0 | 0 | 0 | 0 | 1,956,838 | 7,440,209 | 8,009,436 | 8,324,537 | 0 | 0 |
| Equipment - LB800 | 148.21 | 0 | 361,418 | 1,433,887 | 1,327,712 | 726,763 | 781,763 | 424,159 | 1,433,887 | 1,433,887 | 1,433,450 | 438,318 |
| Equipment - Dozer | 73.23 | 695,246 | 926,945 | 926,945 | 1,390,418 | 1,390,418 | 1,853,891 | 1,853,891 | 1,853,891 | 1,853,891 | 926,945 | 0 |
| Equipment - RTD | 66.23 | 419,170 | 838,339 | 838,339 | 838,339 | 838,339 | 838,339 | 838,339 | 838,339 | 838,339 | 419,170 | 0 |
| Equipment - Grader | 64.18 | 406,195 | 609,325 | 812,390 | 812,390 | 812,390 | 812,390 | 812,390 | 812,390 | 812,390 | 406,195 | 203,130 |
| <hr/> | | | | | | | | | | | | |
| Subtotal - Hour Based Costs | | 4,267,995 | 13,277,208 | 18,229,174 | 21,957,911 | 19,384,703 | 19,830,663 | 26,674,651 | 34,422,109 | 34,840,211 | 35,106,000 | 16,515,190 |
| <hr/> | | | | | | | | | | | | |
| Total Operating Cost | | 8,465,746 | 40,957,768 | 91,244,345 | 86,481,603 | 61,008,360 | 65,983,763 | 64,712,151 | 112,106,586 | 116,592,503 | 107,820,612 | 81,249,660 |

TABLE 11 - 2 (CONTINUED)

Appendix A

REFERENCES

1. Golder Associates; A report to Cyprus Anvil Mining Corporation on Engineering Recommendations for the Proposed Diversion of Vangorda Creek; December, 1979.
2. Cyprus Anvil Mining Corporation; The Development of the Vangorda Plateau Ore Deposits; 1981.
3. Montreal Engineering Co. Ltd; Cyprus Anvil Mining Corporation Ltd., Grum Deposit, Phase Two Geotechnical Studies, Draft Report; December 1979.
4. Cyprus Anvil Mining Corporation; Vangorda Geotechnical Study, Feasability Stage, October 1980.
5. Stanley Associates Engineering Ltd., Vangorda Plateau Haul Road Preliminary Design Study, December 1980.
6. Golder Associates; "Proposed Rock Drain North Fork of Rose Creek, Faro, Yukon", September 1986.

Appendix B

Recommendations for Future Work

This study, while endeavoring to be complete, suffers from limitations in time and available information. It is therefore recommended that the following work be done in order to provide a more realistic production plan:

- carry out geotechnical studies on overburden slope dump stability
- examine alternate means of overburden removal
- examine the usefulness of overburden dewatering
- accurately define the orientation of foliation surfaces in the rock
- redesign the Grum Pit and its stages in order to levelize the waste stripping
- finalize the Vangorda Haul Road alignment
- undertake more metallurgical test work of the Grum ore in order to define (a) the metallurgical response of the individual ore types, and (b) the effect of oxidation on stockpiled ore
- define the degree and extent of oxidation in the southeast end of the Vangorda orebody
- assess the Champ Zone of the Grum orebody for possible inclusion into the mine plan
- complete exploration drilling in the area of the waste dumps to ensure that no reserves are sterilized
- fill-in drilling as required on both orebodies
- remodel both orebodies

APPENDIX C
VANGORDA PLATEAU METALLURGY

(Reprinted from the CAMC report The Development of the
Vangorda Plateau Orebodies)

2.3 GRUM METALLURGY

2.3 GRUM METALLURGY

2.31 Preliminary Laboratory Testwork

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

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

2.32 Critical Results from Laboratory Testwork

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

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

TABLE 15
EFFECT OF FINENESS OF PRIMARY GRIND

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

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

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

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

TABLE 16
EFFECT OF LEAD CONCENTRATE REGRIND ON LEAD CLEANING

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

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

b) Sample: Pilot plant composite.

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

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

TABLE 17
EFFECT OF ZINC REGRIND ON ZINC CLEANING

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

Note: a) Data source LR2027.

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

2.33 Pilot Plant Test Program

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

mlwl

TABLE 18
HEAD ASSAY - PILOT PLANT COMPOSITE

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

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

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

TABLE 19
LOCKED CYCLE TEST DATA - GRUM COMPOSITES

| TEST NO. | COMPOSITE NO. | PRODUCT | WEIGHT % | ASSAYS % | | % DISTRIBUTION | |
|----------|---------------|------------------|----------|----------|-------|----------------|-------|
| | | | | Pb | Zn | Pb | Zn |
| 207 | 1 | Pb Cleaner Conc. | 4.77 | 66.7 | 4.56 | 78.4 | 3.2 |
| | | Zn Cleaner Conc. | 9.18 | 0.58 | 57.40 | 1.3 | 78.5 |
| | | Zn Flot. Tailing | 86.05 | 0.95 | 1.43 | 20.3 | 18.3 |
| | | Head (Calc.) | 100.0 | 4.06 | 6.71 | 100.0 | 100.0 |
| 217 | 1 | Pb Cleaner Conc. | 6.12 | 62.2 | 5.67 | 91.0 | 5.0 |
| | | Zn Cleaner Conc. | 8.91 | 0.40 | 59.70 | 0.9 | 76.7 |
| | | Zn Flot. Tailing | 84.07 | 0.40 | 1.49 | 8.1 | 16.3 |
| | | Head (Calc.) | 100.0 | 4.18 | 6.93 | 100.0 | 100.0 |
| 230 | 3 | Pb Cleaner Conc. | 5.82 | 62.5 | 6.16 | 80.6 | 3.9 |
| | | Zn Cleaner Conc. | 13.75 | 0.39 | 57.80 | 1.2 | 85.4 |
| | | Zn Flot. Tailing | 80.43 | 1.02 | 1.25 | 18.2 | 10.7 |
| | | Head (Calc.) | 100.0 | 4.51 | 9.31 | 100.0 | 100.0 |
| 237 | 2 | Pb Cleaner Conc. | 9.12 | 59.7 | 7.47 | 93.3 | 7.4 |
| | | Zn Cleaner Conc. | 14.31 | 0.56 | 55.7 | 1.4 | 86.2 |
| | | Zn Flot. Tailing | 76.57 | 0.41 | 0.78 | 5.3 | 6.4 |
| | | Head (Calc.) | 100.0 | 5.84 | 9.25 | 100.0 | 100.0 |
| 241 | 2 | Pb Cleaner Conc. | 9.06 | 59.1 | 7.24 | 91.4 | 6.9 |
| | | Zn Cleaner Conc. | 14.49 | 0.46 | 56.6 | 1.2 | 86.0 |
| | | Zn Flot. Tailing | 75.87 | 0.44 | 0.57 | 5.7 | 4.5 |
| | | Head (Calc.) | 100.0 | 5.86 | 9.53 | 100.0 | 100.0 |

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

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

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

- a) Initially it appeared that lead regrind was not quite as critical as the laboratory work had indicated. Presumably this was because much of the laboratory work was carried out with samples averaging 9-10% combined metal; the higher grade pilot plant samples would naturally require less regrinding.
- b) The collector consumption in the pilot plant was considerably lower than had been observed in the laboratory. Again this could be related to the relatively coarse mineralization associated with the high grade sample.
- c) The zinc circuit first cleaner pH was found to be critical. Even very small variations (± 0.5 units at pH 11.5) exercised enormous effects on the zinc concentrate grade.
- d) High cyanide additions, in total about 250 g/tonne were found to be critical to attainment of good metallurgy.

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

TABLE 20
GRUM PILOT PLANT TEST DATA

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

Notes: a) Data source progress report No. 11
Vol. 1-IV L.R. 2027.

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

2.34 Analysis of Pilot Plant Results

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

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

TABLE 21
PREDICTED METALLURGY FROM PILOT PLANT RESULTS

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

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

b) Au and Ag in g/tonne.

2.35 Gold, Silver and Minor Elements

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

TABLE 22
GOLD & SILVER METALLURGY - GRUM PILOT PLANT

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

Notes: a) Data source - Progress Report No.11 Vol.1
L.R.2027. Gold & Silver assays reported only
on these tests.

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

2.36 Predicted Plant Metallurgy

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

- a) Lead grade was difficult to achieve in the pilot plant, and lead recovery was quite low in those tests in which concentrate grades in excess of 60% Pb were obtained.
- b) Silver recovery at 72% probably reflects the anomalously high silver content in the pilot plant feed. Silver recovery into the lead concentrate at Cyprus Anvil averages somewhat less than 60% and is expected to reach 65% only at very high lead recoveries.

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

reduced to 65% to reflect lower silver grade expected in the diluted ore from the Grum deposit.

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

TABLE 23
PREDICTED METALLURGY - GRUM ORE

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

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

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

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

2.4 VANGORDA METALLURGY

2.4 VANGORDA METALLURGY

2.41 Preliminary Laboratory Work

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

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

The Table 24 below is a synopsis of the early published work on Vangorda ore - unfortunately very few of the investigators noted the ore type being tested. The testwork covers a wide range of metallurgical conditions varying from lime to soda ash modulated circuits, various collectors and depressants and several cleaning schemes. All work exhibited one common factor however - Vangorda ores required an extremely fine primary grind of the order 30-50 μ P₈₀ to permit any sort of separation to be achieved.

TABLE 24
PRELIMINARY LABORATORY TEST DATA - VANGOKDA ORE

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

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

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

2.42 Detailed Type Testing

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

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

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

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

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

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

TABLE 25
VANGORDA METALLURGY BY ORE TYPE

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

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

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

c) Grind time refers to time in the laboratory test mill: Product P₈₀ is proportional to grind time.

2.43 Gold, Silver and Minor Elements

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

TABLE 26
GOLD AND SILVER METALLURGY - VANGORDA ORE TYPES

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

Notes: a) Recoveries calculated from composite head assay and unadjusted test distribution data.

b) It is estimated that silver and gold recoveries are accurate to within $\pm 10\%$.

c) Au and Ag in g/tonne.

2.44 Predicted Plant Metallurgy

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

TABLE 27
VANGORDA METALLURGY BY ORE TYPE

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

- Notes: a) Data from best adjusted test data from type testing program.
- b) Relative occurrence data calculated from the mine model available in 1980.
- c) Au, Ag, As, and Hg, in g/tonne.

TABLE 2B
PREDICTED PLANT METALLURGY - VANGORDA ORE

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

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

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

2.5 COMPATABILITY TESTING

2.5 COMPATABILITY TESTING

2.51 Laboratory Testwork

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

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

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

TABLE 29
HEAD ASSAYS OF COMPOSITES

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

- Notes: a) Grum sample originated from materials stored at Lakefield.
- b) Cyprus Anvil sample was provided by W. Muir plant metallurgist and comprised principally Type 4E Pyritic species.

2.52 Base Metallurgy by Ore Type

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

TABLE 30
BASE METALLURGY FOR GRUM & CYPRUS ANVIL SAMPLES

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

2.53 Tests with Ore Mixtures

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

TABLE 31
METALLURGY OF MIXTURE 50:50 GRUM & ANVIL

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

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

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

TABLE 32
SUMMARY OF TEST DATA

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

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

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

TABLE 33
COMPARISON OF
ANTICIPATED AND ACTUAL METALLURGY FOR BLENDED ORES

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

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

2.6 SPECIAL CHARACTERISTICS OF THE CONCENTRATES

2.6 SPECIAL CHARACTERISTICS OF CONCENTRATES

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

2.61 Flow Moistures

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

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

TABLE 34
FLOW MOISTURE

| | NORMAL | FINE GRIND |
|------|--------|------------|
| Lead | 8.1 | 9.7 |
| Zinc | 9.8 | 11.2 |

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

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

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

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

2.62 Moisture Contained in the Concentrates

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

TABLE 35
MOISTURE CONTENTS OF CONCENTRATES

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

2.63 Spectrographic Analyses

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

TABLE 36
SPECTROGRAPHIC DATA - FINAL CONCENTRATES
(Results from Can-Test Vancouver)

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

- Notes: a) Percentages of the various elements expressed in these analyses may be considered accurate to within plus or minus 35 to 50% of the amount present.
- b) Semi-quantitative spectrographic analytical results for gold and silver are normally not of a sufficient degree of precision to enable calculation of the true value of ores. Therefore, should exact values be required, it is recommended that these elements be assayed by the conventional Fire Assay Method. Quantitative and Fire Assays may be carried out on the retained pulp samples.
- c) Silicon, aluminum, magnesium, calcium and iron are normal components of complex silicates.
- d) MATRIX - Major constituent
MAJOR - Above normal spectrographic range
TRACE - Detected by minor amounts
N.D. - Not detected
* - Suggest assay (above 0.3%)

APPENDIX D
FARO PIT PLAN

Faro Mine Schedule

1987

At the start of 1987 three (3) P & H shovels are in A phase middle benches and the Marion shovel plus the contract hoe are in B phase upper benches. One (1) P & H shovel is moved from A phase to B phase at the end of March, 1987. The contract hoe is utilized in B phase until the end of April, 1987. A second P & H shovel is moved from A phase to B phase in early May, 1987. In mid August, 1987 the Marion shovel is parked. Upon completion of A phase mining at the end of October, 1987, the final A phase shovel is moved to B phase for one month and then to the start of C phase in December, 1987.

1988

At the start of 1988 two shovels are in B phase and one shovel is in C phase. One shovel is moved from B phase to C phase in mid February, 1988. B phase mining is completed by the end of September, 1988 and the final B phase shovel is then parked. Two shovels continue to operate in C phase throughout the year.

1989

Two shovels continue to mine C phase until mid January, 1989 at which time one shovel is parked. C phase mining is complete by the end of July, 1989 at which time the sole remaining Faro pit shovel starts D phase mining. This shovel continues to mine D Phase throughout the rest of the year.

1990

One shovel continues to mine D phase throughout the year.

1991

One shovel continues to mine D phase throughout the year.

1992

One shovel continues to mine D phase throughout the year.

1993

The Faro pit is mined out by mid July, 1993.

Table 1 -- Faro Pit Summary
(tonnes X 1000)

| | 1987 | 1988 | 1989 | 1990 | 1991 | 1992 | 1993 |
|----------------|---------|---------|---------|------|-------|--------|---------|
| First Quarter | | | | | | | |
| - Waste | 8127 | 3840 | 855 | 2287 | 2028 | 1320 | 831 |
| - Ore | 1062 | 2189 | 1131 | - | - | 529 | 998 |
| Tot mined | 9189 | 6029 | 1986 | 2287 | 2028 | 1849 | 1829 |
| - S.R. | 7.65 | 1.75 | 0.76 | - | - | 2.50 | 0.83 |
| - Millfeed | 1215 | 1107 | 1215 | - | - | 412 | 879 |
| - %Pb+Zn | 7.07 | 8.83 | 7.77 | - | - | 6.76 | 8.69 |
| - Pb conc | 32.276 | 53.402 | 49.266 | - | - | 12.919 | 33.541 |
| - g/t Ag | 516 | 605 | 596 | - | - | 426 | 340 |
| - Zn conc | 83.156 | 90.604 | 86.889 | - | - | 26.789 | 81.246 |
| Tot. conc | 120.432 | 144.006 | 136.155 | - | - | 39.708 | 114.787 |
| Second Quarter | | | | | | | |
| - Waste | 6791 | 4325 | 65 | 2312 | 1824 | 1120 | 610 |
| - Ore | 1379 | 1673 | 1785 | - | 26 | 729 | 1239 |
| Tot mined | 8170 | 5998 | 1850 | 2312 | 1850 | 1849 | 1849 |
| - S.R. | 4.92 | 2.59 | 0.04 | - | 70.15 | 1.54 | 0.49 |
| - Millfeed | 1172 | 1229 | 1229 | - | - | 548 | 1060 |
| - %Pb+Zn | 8.19 | 7.82 | 8.12 | - | - | 7.00 | 7.49 |
| - Pb conc | 48.175 | 47.911 | 49.110 | - | - | 16.782 | 35.647 |
| - g/t Ag | 537 | 478 | 543 | - | - | 444 | 342 |
| - Zn conc | 92.151 | 90.768 | 96.497 | - | - | 38.398 | 81.605 |
| Tot conc | 140.326 | 138.679 | 145.607 | - | - | 55.180 | 117.252 |

Table 1 - Faro Pit Summary (Continued)
(tonnes X 1000)

| | 1987 | 1988 | 1989 | 1990 | 1991 | 1992 | 1993 |
|----------------|---------|---------|--------|------|--------|--------|--------|
| Third Quarter | | | | | | | |
| - Waste | 5772 | 4005 | 2163 | 2318 | 1742 | 895 | 21 |
| - Ore | 1486 | 1905 | 126 | - | 127 | 974 | 313 |
| Tot mined | 7258 | 5910 | 2289 | 2318 | 1869 | 1869 | 334 |
| - S.R. | 3.88 | 2.10 | 17.17 | - | 13.72 | 0.92 | 0.07 |
| - Millfeed | 1219 | 1242 | 1242 | - | - | 622 | 304 |
| - %Pb+Zn | 8.06 | 7.04 | 5.42 | - | - | 7.06 | 9.34 |
| - Pb conc | 53.973 | 41.802 | 31.061 | - | - | 20.113 | 15.505 |
| - g/t Ag | 533 | 497 | 659 | - | - | 413 | 282 |
| - Zn conc | 92.190 | 83.705 | 60.013 | - | - | 43.664 | 30.348 |
| Tot conc | 146.163 | 125.507 | 91.074 | - | - | 63.777 | 45.853 |
| Fourth Quarter | | | | | | | |
| - Waste | 4620 | 2946 | 2312 | 1849 | 1516 | 765 | - |
| - Ore | 1358 | 1031 | - | - | 333 | 1084 | - |
| Tot mined | 5978 | 3977 | 2312 | 1849 | 1849 | 1849 | - |
| - S.R. | 3.40 | 2.86 | - | - | 4.55 | 0.71 | - |
| - Millfeed | 1229 | 1182 | 0 | - | 364 | 850 | - |
| - %Pb+Zn | 7.94 | 8.39 | 0 | - | 6.12 | 7.06 | - |
| - Pb conc | 53.232 | 51.468 | 0 | - | 10.935 | 26.261 | - |
| - g/t Ag | 620 | 595 | 0 | - | 511 | 406 | - |
| - Zn conc | 87.635 | 93.536 | 0 | - | 19.817 | 61.218 | - |
| Tot conc | 140.867 | 145.004 | 0 | - | 30.752 | 87.479 | - |

Table 1 - Faro Pit Summary (Continued)
(tonnes X 1000)

| | 1987 | 1988 | 1989 | 1990 | 1991 | 1992 | 1993 |
|--------------|---------|---------|---------|------|--------|---------|---------|
| Annual Total | | | | | | | |
| - Waste | 25310 | 15115 | 5395 | 8766 | 7110 | 4099 | 1461 |
| - Ore | 5285 | 6798 | 3042 | 0 | 486 | 3319 | 2552 |
| Tot mined | 30594 | 21914 | 8437 | 8766 | 7596 | 7418 | 4013 |
| - S.R. | 4.79 | 2.22 | 1.77 | - | 14.63 | 1.24 | 0.57 |
| - Millfeed | 4885 | 4760 | 3686 | 0 | 364 | 2432 | 2243 |
| - %Pb+Zn | 7.94 | 7.99 | 7.10 | 0 | 6.12 | 7.00 | 8.21 |
| - Pb conc | 187.656 | 194.583 | 129.437 | 0 | 10.935 | 76.075 | 84.693 |
| - g/t Ag | 555 | 548 | 591 | 0 | 511 | 420 | 330 |
| - Zn conc | 355.132 | 358.613 | 243.399 | 0 | 19.817 | 170.069 | 193.199 |
| Tot conc | 547.788 | 553.196 | 372.836 | 0 | 30.752 | 246.144 | 277.892 |

Remaining in Stk 1994
@ end 1993: Faro Feed

| | |
|------------|---------|
| Q | |
| - Waste | 0 |
| - Ore | - |
| Tot mined | - |
| - S.R. | - |
| - Millfeed | 3471 |
| - %Pb+Zn | 4.19 |
| - Pb conc | 66.092 |
| - g/t Ag | 585 |
| - Zn conc | 127.121 |
| Tot conc | 193.213 |

APPENDIX E
DETAILED RESERVE LISTING

CURRAGH RESOURCES

Reserve Listing: VP 1-1 Mining Reserves

Page: 1

| Block
No. | Bench
Desc | Block | Dig
Rate | Material | Grade
ZPb+Zn | Volume
BCM | Tonnes | HEAD GRADE | | | | | CONCENTRATE | | | | | | |
|--------------|---------------|---------|-------------|----------|--|---|--|------------|------|------|--------|--------|-------------|-------|--------|--------|--------|-------|--|
| | | | | | | | | ZPb+Zn | ZPb | ZZn | Ag g/t | Au g/t | Pb Tns | ZPb | Ag g/t | Au g/t | Zn tns | ZZn | |
| 3 | 1312. | Grum 1- | 2 | A | Overburd | 262116. | 550444. | | | | | | | | | | | | |
| 4 | 1300. | Grum 1- | 3 | A | Rock
Overburd | 600960.
646320. | 1622592.
1357272. | | | | | | | | | | | | |
| 5 | 1296. | Grum 1- | 4 | A | Rock
Overburd | 387264.
565728. | 1045613.
1188029. | | | | | | | | | | | | |
| 6 | 1291. | Grum 1- | 5 | A | Rock
Overburd
Sulph Wast
Ore | 486031.
494820.
27.
5.0+**** | 1312269.
1039122.
78.
540. | | | | | | | | | | | | |
| | | | | | | | 1707. | 5.10 | 1.91 | 3.19 | 30. | .43 | 38. | 60.00 | 750. | 5.53 | 76. | 55.00 | |
| 7 | 1287. | Grum 1- | 6 | A | Rock
Overburd
Sulph Wast
Ore | 555225.
457872.
81.
5.0+**** | 1499063.
961531.
235.
1620. | | | | | | | | | | | | |
| | | | | | | | 5125. | 5.11 | 1.93 | 3.18 | 30. | .43 | 117. | 60.00 | 757. | 5.43 | 226. | 55.00 | |
| 8 | 1282. | Grum 1- | 7 | A | Rock
Overburd
Sulph Wast
Ore | 569975.
484812.
135.
5.0+**** | 1538854.
1018105.
392.
2700. | | | | | | | | | | | | |
| | | | | | | | 8576. | 5.00 | 1.87 | 3.13 | 30. | .43 | 189. | 60.00 | 764. | 5.62 | 372. | 55.00 | |
| 9 | 1278. | Grum 1- | 8 | A | Rock
Overburd
Sulph Wast
Ore | 579366.
490308.
297.
5.0+**** | 1564038.
1029647.
889.
5940. | | | | | | | | | | | | |
| | | | | | | | 19428. | 6.23 | 2.34 | 3.90 | 39. | .49 | 563. | 60.00 | 821. | 5.18 | 1100. | 55.00 | |
| 10 | 1273. | Grum 1- | 9 | A | Rock
Overburd
Sulph Wast
Ore | 590174.
589764.
513.
5.0+**** | 1593049.
1238505.
1533.
10260. | | | | | | | | | | | | |
| | | | | | | | 33489. | 6.18 | 2.32 | 3.86 | 39. | .50 | 960. | 60.00 | 827. | 5.31 | 1876. | 55.00 | |
| 11 | 1269. | Grum 1- | 10 | A | Rock
Overburd
Sulph Wast
Ore
Ore | 591190.
622248.
729.
4.0- 5.0
5.0+**** | 1595688.
1306721.
2153.
1080.
13500. | | | | | | | | | | | | |
| | | | | | | | 43436. | 5.47 | 1.96 | 3.52 | 33. | .65 | 1011. | 60.00 | 820. | 8.04 | 2168. | 55.00 | |
| 12 | 1264. | Grum 1- | 11 | A | Rock
Overburd
Sulph Wast
Ore
Ore | 606250.
579432.
864.
4.0- 5.0
5.0+**** | 1636290.
1216807.
2538.
2160.
15120. | | | | | | | | | | | | |
| | | | | | | | 7134. | 3.73 | 1.19 | 2.54 | 25. | .90 | 90. | 60.00 | 935. | 18.22 | 242. | 55.00 | |
| | | | | | | | 48319. | 5.32 | 1.84 | 3.48 | 31. | .68 | 1038. | 60.00 | 828. | 8.95 | 2381. | 55.00 | |
| 13 | 1260. | Grum 1- | 12 | A | Rock
Overburd
Sulph Wast
Ore
Ore | 618135.
544620.
1161.
4.0- 5.0
5.0+**** | 1668096.
1143702.
3440.
7020.
16200. | | | | | | | | | | | | |
| | | | | | | | 23161. | 3.72 | 1.25 | 2.46 | 25. | .76 | 313. | 60.00 | 884. | 14.58 | 758. | 55.00 | |
| | | | | | | | 51996. | 5.44 | 1.86 | 3.58 | 32. | .70 | 1136. | 60.00 | 827. | 9.10 | 2647. | 55.00 | |

CURRAGH RESOURCES

Reserve Listing: VP 1-1 Mining Reserves

Page: 2

| Block
No. | Bench
Desc | Dig
Rate | Material
Grade
ZPb+Zn | Volume
BCM | Tonnes | HEAD GRADE | | | | | CONCENTRATE | | | | | | |
|--------------|---------------|-------------|-----------------------------|---------------|----------|------------|---------|------|--------|--------|-------------|-----|--------|--------|--------|-------|--------|
| | | | | | | ZPb+Zn | ZPb | Zn | Ag g/t | Au g/t | Pb Tns | ZPb | Ag g/t | Au g/t | Zn tns | Zn | |
| 14 | 1255. | Grum 1- 13 | A | Rock | 645936. | 1742755. | | | | | | | | | | | |
| | | | | Overburd | 443304. | 930938. | | | | | | | | | | | |
| | | | | Sulph Wast | 3321. | 10027. | | | | | | | | | | | |
| | | | | Ore | 4.0- 5.0 | 8640. | 28260. | 3.91 | 1.33 | 2.58 | 25. | .70 | 410. | 60.00 | 889. | 12.72 | 976. |
| | | | | Ore | 5.0+**** | 25380. | 81847. | 5.72 | 1.96 | 3.76 | 34. | .69 | 1908. | 60.00 | 835. | 8.56 | 4410. |
| 15 | 1251. | Grum 1- 14 | A | Rock | 635672. | 1714600. | | | | | | | | | | | |
| | | | | Overburd | 377556. | 792868. | | | | | | | | | | | |
| | | | | Sulph Wast | 4536. | 13647. | | | | | | | | | | | |
| | | | | Ore | 4.0- 5.0 | 8640. | 27938. | 4.02 | 1.33 | 2.69 | 25. | .72 | 405. | 60.00 | 892. | 13.03 | 1016. |
| | | | | Ore | 5.0+**** | 38880. | 125391. | 6.38 | 2.22 | 4.16 | 38. | .64 | 3401. | 60.00 | 837. | 7.05 | 7621. |
| 16 | 1246. | Grum 1- 15 | A | Rock | 705524. | 1902098. | | | | | | | | | | | |
| | | | | Overburd | 225672. | 473911. | | | | | | | | | | | |
| | | | | Sulph Wast | 7614. | 23479. | | | | | | | | | | | |
| | | | | Ore | 4.0- 5.0 | 14580. | 46751. | 3.98 | 1.27 | 2.71 | 25. | .65 | 644. | 60.00 | 926. | 12.26 | 1710. |
| | | | | Ore | 5.0+**** | 51300. | 169176. | 7.27 | 2.65 | 4.62 | 45. | .67 | 5751. | 60.00 | 831. | 6.13 | 11621. |
| 17 | 1242. | Grum 1- 16 | A | Rock | 687643. | 1852637. | | | | | | | | | | | |
| | | | | Overburd | 170712. | 358495. | | | | | | | | | | | |
| | | | | Sulph Wast | 12987. | 40019. | | | | | | | | | | | |
| | | | | Ore | 4.0- 5.0 | 19980. | 64164. | 3.95 | 1.26 | 2.69 | 25. | .57 | 873. | 60.00 | 939. | 10.91 | 2329. |
| | | | | Ore | 5.0+**** | 66960. | 222946. | 7.23 | 2.66 | 4.57 | 44. | .68 | 7629. | 60.00 | 815. | 6.12 | 15170. |
| 18 | 1237. | Grum 1- 17 | A | Rock | 657717. | 1770739. | | | | | | | | | | | |
| | | | | Overburd | 101628. | 213419. | | | | | | | | | | | |
| | | | | Sulph Wast | 18773. | 57082. | | | | | | | | | | | |
| | | | | Ore | 4.0- 5.0 | 27000. | 87137. | 3.93 | 1.30 | 2.63 | 26. | .50 | 1229. | 60.00 | 941. | 9.38 | 3083. |
| | | | | Ore | 5.0+**** | 78456. | 262997. | 7.54 | 2.76 | 4.78 | 45. | .68 | 9387. | 60.00 | 799. | 5.95 | 18847. |
| 19 | 1233. | Grum 1- 18 | A | Rock | 627693. | 1688807. | | | | | | | | | | | |
| | | | | Overburd | 68976. | 144850. | | | | | | | | | | | |
| | | | | Sulph Wast | 20329. | 61153. | | | | | | | | | | | |
| | | | | Ore | 4.0- 5.0 | 28620. | 92732. | 3.89 | 1.28 | 2.61 | 25. | .47 | 1283. | 60.00 | 936. | 8.79 | 3254. |
| | | | | Ore | 5.0+**** | 86352. | 292125. | 7.86 | 2.83 | 5.02 | 45. | .66 | 10756. | 60.00 | 786. | 5.69 | 22188. |
| 20 | 1228. | Grum 1- 19 | A | Rock | 590628. | 1587843. | | | | | | | | | | | |
| | | | | Overburd | 38952. | 81799. | | | | | | | | | | | |
| | | | | Sulph Wast | 29111. | 87356. | | | | | | | | | | | |
| | | | | Ore | 4.0- 5.0 | 22140. | 71908. | 3.87 | 1.27 | 2.60 | 24. | .39 | 987. | 60.00 | 898. | 7.38 | 2506. |
| | | | | Ore | 5.0+**** | 100476. | 342302. | 7.53 | 2.66 | 4.87 | 43. | .63 | 11639. | 60.00 | 793. | 5.78 | 25060. |
| 21 | 1224. | Grum 1- 20 | A | Rock | 544149. | 1460218. | | | | | | | | | | | |
| | | | | Overburd | 10248. | 21521. | | | | | | | | | | | |
| | | | | Sulph Wast | 27417. | 81656. | | | | | | | | | | | |
| | | | | Ore | 4.0- 5.0 | 21600. | 70670. | 3.85 | 1.26 | 2.59 | 24. | .38 | 961. | 60.00 | 893. | 7.23 | 2457. |
| | | | | Ore | 5.0+**** | 127140. | 436947. | 7.35 | 2.62 | 4.73 | 43. | .61 | 14570. | 60.00 | 794. | 5.77 | 30946. |
| 22 | 1219. | Grum 1- 21 | A | Rock | 498579. | 1336410. | | | | | | | | | | | |
| | | | | Sulph Wast | 27017. | 81194. | | | | | | | | | | | |
| | | | | Ore | 4.0- 5.0 | 28620. | 93932. | 3.83 | 1.26 | 2.58 | 24. | .43 | 1273. | 60.00 | 921. | 8.19 | 3238. |
| | | | | Ore | 5.0+**** | 127488. | 441212. | 7.46 | 2.72 | 4.74 | 44. | .63 | 15330. | 60.00 | 791. | 5.77 | 31376. |

CURRAGH RESOURCES

Page: 3

Reserve Listing: VP 1-1 Mining Reserves

| Block No. | Bench Desc | Block | Dig Rate | Material | Grade
%Pb+Zn | Volume
BCM | Tonnes | HEAD GRADE | | | | | CONCENTRATE | | | | |
|-----------|------------|---------|----------|----------|-----------------|---------------|----------|------------|------|------|--------|--------|-------------|--------|--------|--------|--------|
| | | | | | | | | %Pb+Zn | %Pb | %Zn | Ag g/t | Au g/t | Pb lbs | %Pb | Ag g/t | Au g/t | Zn lbs |
| 23 | 1215. | Grum 1- | 22 | A | Rock | 477075. | 1280118. | | | | | | | | | | |
| | | | | | Sulph | 31715. | 95921. | | | | | | | | | | |
| | | | | | Wast | | | | | | | | | | | | |
| | | | | | Ore | 4.0- 5.0 | 25920. | 85715. | 3.82 | 1.26 | 2.56 | 24. | .41 | 1165. | 60.00 | 913. | 7.76 |
| | | | | | Ore | 5.0+**** | 111576. | 381075. | 7.25 | 2.58 | 4.67 | 42. | .62 | 12470. | 60.00 | 801. | 5.95 |
| 24 | 1210. | Grum 1- | 23 | A | Rock | 455800. | 1223737. | | | | | | | | | | |
| | | | | | Sulph | 30703. | 91631. | | | | | | | | | | |
| | | | | | Wast | | | | | | | | | | | | |
| | | | | | Ore | 4.0- 5.0 | 25920. | 85928. | 3.91 | 1.32 | 2.59 | 24. | .40 | 1240. | 60.00 | 872. | 7.38 |
| | | | | | Ore | 5.0+**** | 105504. | 354818. | 6.97 | 2.43 | 4.54 | 40. | .61 | 10773. | 60.00 | 807. | 6.18 |
| 25 | 1206. | Grum 1- | 24 | A | Rock | 404270. | 1084322. | | | | | | | | | | |
| | | | | | Sulph | 24210. | 72745. | | | | | | | | | | |
| | | | | | Wast | | | | | | | | | | | | |
| | | | | | Ore | 4.0- 5.0 | 16740. | 54208. | 3.85 | 1.26 | 2.60 | 23. | .40 | 735. | 60.00 | 885. | 7.69 |
| | | | | | Ore | 5.0+**** | 121860. | 409872. | 6.80 | 2.39 | 4.40 | 39. | .61 | 12217. | 60.00 | 799. | 6.27 |
| 26 | 1201. | Grum 1- | 25 | A | Rock | 363842. | 974190. | | | | | | | | | | |
| | | | | | Sulph | 16774. | 50364. | | | | | | | | | | |
| | | | | | Wast | | | | | | | | | | | | |
| | | | | | Ore | 4.0- 5.0 | 14400. | 46696. | 3.85 | 1.24 | 2.61 | 23. | .42 | 625. | 60.00 | 894. | 8.16 |
| | | | | | Ore | 5.0+**** | 133884. | 453438. | 6.92 | 2.47 | 4.45 | 40. | .60 | 14103. | 60.00 | 787. | 5.98 |
| 27 | 1197. | Grum 1- | 26 | A | Rock | 315825. | 843611. | | | | | | | | | | |
| | | | | | Sulph | 17693. | 53393. | | | | | | | | | | |
| | | | | | Wast | | | | | | | | | | | | |
| | | | | | Ore | 4.0- 5.0 | 12960. | 41856. | 3.96 | 1.24 | 2.72 | 24. | .42 | 558. | 60.00 | 920. | 8.15 |
| | | | | | Ore | 5.0+**** | 135696. | 466520. | 7.31 | 2.65 | 4.66 | 42. | .64 | 15823. | 60.00 | 785. | 6.00 |
| 28 | 1192. | Grum 1- | 27 | A | Rock | 290484. | 774958. | | | | | | | | | | |
| | | | | | Sulph | 18435. | 55972. | | | | | | | | | | |
| | | | | | Wast | | | | | | | | | | | | |
| | | | | | Ore | 4.0- 5.0 | 17688. | 56749. | 3.97 | 1.27 | 2.71 | 23. | .41 | 777. | 60.00 | 873. | 7.77 |
| | | | | | Ore | 5.0+**** | 121092. | 424286. | 7.60 | 2.77 | 4.83 | 45. | .68 | 15217. | 60.00 | 803. | 6.04 |
| 29 | 1188. | Grum 1- | 28 | A | Rock | 271243. | 724094. | | | | | | | | | | |
| | | | | | Sulph | 21956. | 66323. | | | | | | | | | | |
| | | | | | Wast | | | | | | | | | | | | |
| | | | | | Ore | 4.0- 5.0 | 19980. | 64092. | 4.00 | 1.28 | 2.72 | 23. | .39 | 689. | 60.00 | 848. | 7.29 |
| | | | | | Ore | 5.0+**** | 105696. | 369966. | 7.47 | 2.66 | 4.81 | 44. | .66 | 12553. | 60.00 | 818. | 6.17 |
| 30 | 1183. | Grum 1- | 29 | A | Rock | 257189. | 686829. | | | | | | | | | | |
| | | | | | Sulph | 21033. | 63598. | | | | | | | | | | |
| | | | | | Wast | | | | | | | | | | | | |
| | | | | | Ore | 4.0- 5.0 | 19440. | 63264. | 4.03 | 1.31 | 2.72 | 23. | .41 | 904. | 60.00 | 834. | 7.54 |
| | | | | | Ore | 5.0+**** | 109620. | 375356. | 6.97 | 2.46 | 4.51 | 42. | .62 | 11586. | 60.00 | 829. | 6.24 |
| 31 | 1179. | Grum 1- | 30 | A | Rock | 213094. | 567174. | | | | | | | | | | |
| | | | | | Sulph | 21532. | 65594. | | | | | | | | | | |
| | | | | | Wast | | | | | | | | | | | | |
| | | | | | Ore | 4.0- 5.0 | 14580. | 47522. | 3.97 | 1.32 | 2.65 | 23. | .52 | 687. | 60.00 | 836. | 9.52 |
| | | | | | Ore | 5.0+**** | 124464. | 425338. | 7.07 | 2.51 | 4.56 | 42. | .63 | 13412. | 60.00 | 827. | 6.27 |
| 32 | 1174. | Grum 1- | 31 | A | Rock | 121875. | 321570. | | | | | | | | | | |
| | | | | | Sulph | 20537. | 62047. | | | | | | | | | | |
| | | | | | Wast | | | | | | | | | | | | |
| | | | | | Ore | 4.0- 5.0 | 9660. | 32283. | 3.92 | 1.36 | 2.56 | 25. | .57 | 483. | 60.00 | 868. | 10.05 |
| | | | | | Ore | 5.0+**** | 109476. | 376580. | 7.11 | 2.55 | 4.56 | 43. | .69 | 12109. | 60.00 | 824. | 6.72 |
| 33 | 1170. | Grum 1- | 32 | A | Rock | 117880. | 312166. | | | | | | | | | | |
| | | | | | Sulph | 13996. | 42544. | | | | | | | | | | |
| | | | | | Wast | | | | | | | | | | | | |
| | | | | | Ore | 4.0- 5.0 | 5436. | 17721. | 3.87 | 1.34 | 2.53 | 23. | .52 | 259. | 60.00 | 897. | 9.37 |
| | | | | | Ore | 5.0+**** | 89436. | 308976. | 7.11 | 2.54 | 4.57 | 42. | .76 | 9918. | 60.00 | 817. | 7.36 |

CURRAGH RESOURCES

Reserve Listing: VP 1-1 Mining Reserves

Page: 4

[illegible]

CURRAGH RESOURCES

Reserve Listing: VP 1-1 Mining Reserves

Page: 5

| Block | Bench | Block Desc | Dig Rate | Material | Grade | Volume BCM | Tonnes | HEAD GRADE | | | | CONCENTRATE | | | | | | |
|-------|-------|------------|----------|------------|----------|------------|----------|------------|------|------|--------|-------------|--------|-------|--------|--------|--------|-------|
| | | | | | | | | TPb+Zn | TPb | ZZn | Ag g/t | Au g/t | Pb tns | TPb | Ag g/t | Au g/t | Zn tns | ZZn |
| 52 | 1237. | Grum 25-17 | A | Rock | | 700188. | 1890508. | | | | | | | | | | | |
| | | | | Overburd | | 8844. | 18572. | | | | | | | | | | | |
| 53 | 1233. | Grum 25-18 | A | Rock | | 686916. | 1854673. | | | | | | | | | | | |
| | | | | Overburd | | 8328. | 17489. | | | | | | | | | | | |
| 54 | 1228. | Grum 25-19 | A | Rock | | 674820. | 1822014. | | | | | | | | | | | |
| | | | | Overburd | | 9000. | 18900. | | | | | | | | | | | |
| 55 | 1224. | Grum 25-20 | A | Rock | | 665676. | 1797326. | | | | | | | | | | | |
| | | | | Overburd | | 3900. | 8190. | | | | | | | | | | | |
| | | | | Sulph Wast | | 1056. | 3674. | | | | | | | | | | | |
| 56 | 1219. | Grum 25-21 | A | Rock | | 656988. | 1773868. | | | | | | | | | | | |
| | | | | Sulph Wast | | 768. | 2672. | | | | | | | | | | | |
| 57 | 1215. | Grum 25-22 | A | Rock | | 646790. | 1746260. | | | | | | | | | | | |
| | | | | Sulph Wast | | 1405. | 4891. | | | | | | | | | | | |
| | | | | Ore | 5.0+**** | 492. | 2015. | 13.02 | 4.48 | 8.55 | 75. | .96 | 126. | 60.00 | 837. | 5.40 | 282. | 55.00 |
| 58 | 1210. | Grum 25-23 | A | Rock | | 628341. | 1696206. | | | | | | | | | | | |
| | | | | Sulph Wast | | 1953. | 6801. | | | | | | | | | | | |
| | | | | Ore | 5.0+**** | 1620. | 7197. | 8.53 | 3.76 | 4.77 | 64. | .92 | 364. | 60.00 | 839. | 6.16 | 513. | 55.00 |
| 59 | 1206. | Grum 25-24 | A | Rock | | 616258. | 1663358. | | | | | | | | | | | |
| | | | | Sulph Wast | | 1781. | 6588. | | | | | | | | | | | |
| | | | | Ore | 4.0- 5.0 | 1620. | 6507. | 4.14 | 1.76 | 2.38 | 35. | .83 | 133. | 60.00 | 953. | 11.57 | 204. | 55.00 |
| | | | | Ore | 5.0+**** | 1596. | 7106. | 9.53 | 3.88 | 5.65 | 65. | .99 | 373. | 60.00 | 837. | 6.41 | 617. | 55.00 |
| 60 | 1201. | Grum 25-25 | A | Rock | | 602192. | 1625363. | | | | | | | | | | | |
| | | | | Sulph Wast | | 2702. | 9053. | | | | | | | | | | | |
| | | | | Ore | 4.0- 5.0 | 2880. | 11590. | 4.01 | 1.74 | 2.27 | 34. | .83 | 233. | 60.00 | 941. | 11.66 | 344. | 55.00 |
| | | | | Ore | 5.0+**** | 768. | 3430. | 10.89 | 4.05 | 6.84 | 66. | .97 | 189. | 60.00 | 816. | 6.02 | 370. | 55.00 |
| 61 | 1197. | Grum 25-26 | A | Rock | | 586134. | 1581497. | | | | | | | | | | | |
| | | | | Sulph Wast | | 4625. | 15382. | | | | | | | | | | | |
| | | | | Ore | 4.0- 5.0 | 3240. | 13145. | 4.11 | 1.81 | 2.30 | 35. | .87 | 277. | 60.00 | 933. | 11.86 | 396. | 55.00 |
| | | | | Ore | 5.0+**** | 2868. | 13033. | 12.37 | 4.76 | 7.61 | 77. | 1.01 | 849. | 60.00 | 802. | 5.37 | 1590. | 55.00 |
| 62 | 1192. | Grum 25-27 | A | Rock | | 562472. | 1516311. | | | | | | | | | | | |
| | | | | Sulph Wast | | 5790. | 18859. | | | | | | | | | | | |
| | | | | Ore | 4.0- 5.0 | 2700. | 10870. | 4.00 | 1.86 | 2.14 | 35. | .81 | 237. | 60.00 | 903. | 10.72 | 301. | 55.00 |
| | | | | Ore | 5.0+**** | 10380. | 45843. | 11.31 | 4.52 | 6.78 | 73. | .98 | 2896. | 60.00 | 803. | 5.40 | 4933. | 55.00 |
| 63 | 1188. | Grum 25-28 | A | Rock | | 551395. | 1431632. | | | | | | | | | | | |
| | | | | Sulph Wast | | 4028. | 13340. | | | | | | | | | | | |
| | | | | Ore | 4.0- 5.0 | 2160. | 8912. | 4.03 | 2.02 | 2.01 | 35. | 1.02 | 214. | 60.00 | 836. | 12.50 | 229. | 55.00 |
| | | | | Ore | 5.0+**** | 13596. | 61592. | 11.48 | 5.01 | 6.47 | 78. | .97 | 4368. | 60.00 | 777. | 4.86 | 6255. | 55.00 |
| 64 | 1183. | Grum 25-29 | A | Rock | | 513833. | 1384875. | | | | | | | | | | | |
| | | | | Overburd | | 540. | 1458. | | | | | | | | | | | |
| | | | | Sulph Wast | | 2263. | 7544. | | | | | | | | | | | |
| | | | | Ore | 4.0- 5.0 | 2160. | 9132. | 4.08 | 2.26 | 1.62 | 35. | 1.22 | 251. | 60.00 | 760. | 13.41 | 209. | 55.00 |
| | | | | Ore | 5.0+**** | 10692. | 47742. | 11.36 | 5.25 | 6.11 | 81. | .91 | 3573. | 60.00 | 772. | 4.42 | 4531. | 55.00 |

CURRAGH RESOURCES

Reserve Listings: VP 1-1 Mining Reserves

Page: 6

| Block | Bench | Block Desc | Dig Material | Grade | Volume BCM | Tonnes | H E A D G R A D E | | | | | C O N C E N T R A T E | | | | |
|-------|-------|------------|--------------|----------|------------|----------|-------------------|------|------|--------|--------|-----------------------|-------|--------|--------|---------|
| | | | | | | | 1Pb+1n | 2Pb | 1Zn | Ag g/t | Au g/t | Pb lbs | 1Pb | Ag g/t | Au g/t | 1Zn lbs |
| 65 | 1179. | Grus 2S-30 | A Rock | | 388476. | 1047235. | | | | | | | | | | |
| | | | Overburd | | 744. | 2009. | | | | | | | | | | |
| | | | Sulph Mast | | 3420. | 10984. | | | | | | | | | | |
| | | | Ore | 4.0- 5.0 | 2160. | 9264. | 4.03 | 2.38 | 1.65 | 35. | 1.33 | 272. | 60.00 | 724. | 13.85 | 187. |
| | | | Ore | 5.0+**** | 7440. | 31706. | 10.73 | 5.02 | 5.71 | 79. | .86 | 2254. | 60.00 | 791. | 4.32 | 2787. |
| 68 | 1300. | Grus 2N- 3 | A Overburd | | 45924. | 96440. | | | | | | | | | | |
| 69 | 1296. | Grus 2N- 4 | A Rock | | 4056. | 10951. | | | | | | | | | | |
| | | | Overburd | | 181776. | 381730. | | | | | | | | | | |
| 70 | 1291. | Grus 2N- 5 | A Rock | | 12120. | 32724. | | | | | | | | | | |
| | | | Overburd | | 225168. | 472853. | | | | | | | | | | |
| 71 | 1287. | Grus 2N- 6 | A Rock | | 28644. | 77339. | | | | | | | | | | |
| | | | Overburd | | 256120. | 542052. | | | | | | | | | | |
| 72 | 1282. | Grus 2N- 7 | A Rock | | 24900. | 67230. | | | | | | | | | | |
| | | | Overburd | | 309288. | 649505. | | | | | | | | | | |
| 73 | 1278. | Grus 2N- 8 | A Rock | | 84888. | 229198. | | | | | | | | | | |
| | | | Overburd | | 278328. | 584489. | | | | | | | | | | |
| 74 | 1273. | Grus 2N- 9 | A Rock | | 86952. | 234770. | | | | | | | | | | |
| | | | Overburd | | 333900. | 701190. | | | | | | | | | | |
| 75 | 1269. | Grus 2N-10 | A Rock | | 89616. | 241963. | | | | | | | | | | |
| | | | Overburd | | 353244. | 741812. | | | | | | | | | | |
| 76 | 1264. | Grus 2N-11 | A Rock | | 107064. | 289073. | | | | | | | | | | |
| | | | Overburd | | 346776. | 728230. | | | | | | | | | | |
| 77 | 1260. | Grus 2N-12 | A Rock | | 109740. | 296298. | | | | | | | | | | |
| | | | Overburd | | 340476. | 714999. | | | | | | | | | | |
| 78 | 1255. | Grus 2N-13 | A Rock | | 120072. | 324194. | | | | | | | | | | |
| | | | Overburd | | 318204. | 668229. | | | | | | | | | | |
| 79 | 1251. | Grus 2N-14 | A Rock | | 128664. | 347393. | | | | | | | | | | |
| | | | Overburd | | 303408. | 637157. | | | | | | | | | | |
| 80 | 1246. | Grus 2N-15 | A Rock | | 157884. | 426287. | | | | | | | | | | |
| | | | Overburd | | 233748. | 490871. | | | | | | | | | | |
| 81 | 1242. | Grus 2N-16 | A Rock | | 175946. | 474870. | | | | | | | | | | |
| | | | Overburd | | 200844. | 421772. | | | | | | | | | | |
| | | | Sulph Mast | | 54. | 211. | | | | | | | | | | |
| | | | Ore | 5.0+**** | 1080. | 4600. | 15.77 | 7.26 | 8.51 | 120. | .87 | 501. | 60.00 | 832. | 3.06 | 628. |
| 82 | 1237. | Grus 2N-17 | A Rock | | 198422. | 535394. | | | | | | | | | | |
| | | | Overburd | | 180012. | 378025. | | | | | | | | | | |
| | | | Sulph Mast | | 100. | 392. | | | | | | | | | | |
| | | | Ore | 5.0+**** | 1904. | 8568. | 15.68 | 7.28 | 8.40 | 120. | .88 | 937. | 60.00 | 832. | 3.07 | 1179. |

CURRAGH RESOURCES

Reserve Listing: VP 1-1 Mining Reserves

Page: 7

| Block No. | Bench Desc | Block | Dig Rate | Material | Grade IPb+Zn | Volume BCM | Tonnes | HEAD GRADE | | | | | CONCENTRATE | | | | | |
|-----------|------------|------------|----------|------------|--------------|------------|---------|------------|------|------|--------|--------|-------------|-------|--------|--------|--------|-------|
| | | | | | | | | IPb+Zn | IPb | Zn | Ag g/t | Au g/t | Pb Tns | IPb | Ag g/t | Au g/t | Zn tns | Zn |
| 83 | 1233. | Grus 2M-18 | A | Rock | | 205389. | 554195. | | | | | | | | | | | |
| | | | | Overburd | | 165012. | 346525. | | | | | | | | | | | |
| | | | | Sulph Wast | | 110. | 423. | | | | | | | | | | | |
| | | | | Ore | 5.0+**** | 2208. | 9246. | 15.60 | 7.16 | 8.44 | 118. | .86 | 991. | 60.00 | 829. | 3.05 | 1269. | 55.00 |
| 84 | 1228. | Grus 2M-19 | A | Rock | | 233001. | 628545. | | | | | | | | | | | |
| | | | | Overburd | | 134040. | 281484. | | | | | | | | | | | |
| | | | | Sulph Wast | | 481. | 1530. | | | | | | | | | | | |
| | | | | Ore | 5.0+**** | 4824. | 18495. | 11.44 | 4.67 | 6.76 | 79. | .73 | 1243. | 60.00 | 840. | 3.77 | 1985. | 55.00 |
| 85 | 1224. | Grus 2M-20 | A | Rock | | 258799. | 698023. | | | | | | | | | | | |
| | | | | Overburd | | 95088. | 199685. | | | | | | | | | | | |
| | | | | Sulph Wast | | 866. | 2677. | | | | | | | | | | | |
| | | | | Ore | 5.0+**** | 6516. | 24859. | 10.97 | 4.12 | 6.86 | 71. | .67 | 1436. | 60.00 | 854. | 4.00 | 2706. | 55.00 |
| 86 | 1219. | Grus 2M-21 | A | Rock | | 272184. | 733974. | | | | | | | | | | | |
| | | | | Overburd | | 71652. | 150469. | | | | | | | | | | | |
| | | | | Sulph Wast | | 970. | 3022. | | | | | | | | | | | |
| | | | | Ore | 5.0+**** | 8592. | 32411. | 10.76 | 3.87 | 6.89 | 66. | .69 | 1730. | 60.00 | 844. | 4.42 | 3550. | 55.00 |
| 87 | 1215. | Grus 2M-22 | A | Rock | | 279376. | 753261. | | | | | | | | | | | |
| | | | | Overburd | | 53076. | 111460. | | | | | | | | | | | |
| | | | | Sulph Wast | | 575. | 1924. | | | | | | | | | | | |
| | | | | Ore | 5.0+**** | 11508. | 42029. | 9.62 | 3.45 | 6.17 | 60. | .70 | 1951. | 60.00 | 855. | 4.99 | 4073. | 55.00 |
| 88 | 1210. | Grus 2M-23 | A | Rock | | 280477. | 756240. | | | | | | | | | | | |
| | | | | Overburd | | 33780. | 70938. | | | | | | | | | | | |
| | | | | Sulph Wast | | 619. | 2038. | | | | | | | | | | | |
| | | | | Ore | 4.0- 5.0 | 1620. | 5346. | 4.19 | 1.40 | 2.79 | 27. | .78 | 83. | 60.00 | 919. | 13.50 | 202. | 55.00 |
| | | | | Ore | 5.0+**** | 10752. | 39185. | 9.54 | 3.46 | 6.08 | 60. | .76 | 1833. | 60.00 | 854. | 5.29 | 3738. | 55.00 |
| 89 | 1206. | Grus 2M-24 | A | Rock | | 290580. | 783757. | | | | | | | | | | | |
| | | | | Overburd | | 21300. | 44730. | | | | | | | | | | | |
| | | | | Sulph Wast | | 535. | 1728. | | | | | | | | | | | |
| | | | | Ore | 4.0- 5.0 | 1620. | 5329. | 4.10 | 1.36 | 2.73 | 27. | .84 | 80. | 60.00 | 933. | 14.86 | 197. | 55.00 |
| | | | | Ore | 5.0+**** | 9072. | 32419. | 8.74 | 3.21 | 5.53 | 54. | .81 | 1394. | 60.00 | 834. | 6.08 | 2781. | 55.00 |
| 90 | 1201. | Grus 2M-25 | A | Rock | | 294097. | 793310. | | | | | | | | | | | |
| | | | | Overburd | | 15132. | 31777. | | | | | | | | | | | |
| | | | | Sulph Wast | | 412. | 1377. | | | | | | | | | | | |
| | | | | Ore | 4.0- 5.0 | 420. | 1382. | 4.34 | 1.47 | 2.87 | 28. | .91 | 23. | 60.00 | 908. | 15.14 | 54. | 55.00 |
| | | | | Ore | 5.0+**** | 7824. | 28706. | 9.83 | 3.72 | 6.11 | 64. | .97 | 1474. | 60.00 | 849. | 6.16 | 2774. | 55.00 |
| 91 | 1197. | Grus 2M-26 | A | Rock | | 301477. | 813110. | | | | | | | | | | | |
| | | | | Sulph Wast | | 362. | 1285. | | | | | | | | | | | |
| | | | | Ore | 5.0+**** | 7236. | 28070. | 13.47 | 5.15 | 8.32 | 89. | 1.12 | 2092. | 60.00 | 858. | 5.28 | 3841. | 55.00 |
| 92 | 1192. | Grus 2M-27 | A | Rock | | 298875. | 806358. | | | | | | | | | | | |
| | | | | Sulph Wast | | 220. | 835. | | | | | | | | | | | |
| | | | | Ore | 4.0- 5.0 | 132. | 415. | 3.62 | 1.14 | 2.48 | 22. | .40 | 5. | 60.00 | 899. | 8.28 | 14. | 55.00 |
| | | | | Ore | 5.0+**** | 4476. | 17819. | 14.99 | 5.71 | 9.28 | 97. | 1.25 | 1486. | 60.00 | 851. | 5.47 | 2764. | 55.00 |

CURRAGH RESOURCES

Keserve Listings: VP 1-1 Mining Reserves

Page: 8

| Block No. | Bench Desc | Block | Dig Rate | Material | Grade | Volume BCM | Tonnes | HEAD GRADE | | | | CONCENTRATE | | | | | | |
|-----------|------------|------------|----------|------------|----------|------------|----------|------------|------|------|--------|-------------|--------|-------|--------|--------|--------|-------|
| | | | | | | | | Pb+Zn | Pb | Zn | Ag g/t | Au g/t | Pb tns | Pb | Ag g/t | Au g/t | Zn tns | Zn |
| 93 | 1188. | Grum 2N-28 | A | Rock | | 291094. | 785518. | | | | | | | | | | | |
| | | | | Sulph Wast | | 781. | 2370. | | | | | | | | | | | |
| | | | | Ore | 4.0- 5.0 | 2148. | 7014. | 4.19 | 1.45 | 2.73 | 28. | .55 | 114. | 60.00 | 923. | 9.22 | 259. | 55.00 |
| | | | | Ore | 5.0+**** | 2916. | 11277. | 12.42 | 4.94 | 7.48 | 84. | 1.21 | 795. | 60.00 | 847. | 6.16 | 1366. | 55.00 |
| 94 | 1183. | Grum 2N-29 | A | Rock | | 283313. | 764267. | | | | | | | | | | | |
| | | | | Sulph Wast | | 374. | 1302. | | | | | | | | | | | |
| | | | | Ore | 4.0- 5.0 | 3552. | 11716. | 4.18 | 1.44 | 2.74 | 29. | .59 | 187. | 60.00 | 951. | 9.90 | 434. | 55.00 |
| | | | | Ore | 5.0+**** | 4332. | 16733. | 11.14 | 4.64 | 6.50 | 78. | .85 | 1103. | 60.00 | 836. | 4.50 | 1718. | 55.00 |
| 95 | 1179. | Grum 2N-30 | A | Rock | | 381564. | 1028604. | | | | | | | | | | | |
| | | | | Overburd | | 336. | 907. | | | | | | | | | | | |
| | | | | Sulph Wast | | 2585. | 8665. | | | | | | | | | | | |
| | | | | Ore | 4.0- 5.0 | 4860. | 15883. | 4.04 | 1.42 | 2.62 | 27. | .55 | 250. | 60.00 | 891. | 9.35 | 559. | 55.00 |
| | | | | Ore | 5.0+**** | 10128. | 40864. | 11.04 | 4.74 | 6.30 | 77. | .73 | 2745. | 60.00 | 815. | 3.82 | 4048. | 55.00 |
| 96 | 1174. | Grum 2C-31 | A | Rock | | 779550. | 2100028. | | | | | | | | | | | |
| | | | | Overburd | | 540. | 1458. | | | | | | | | | | | |
| | | | | Sulph Wast | | 8309. | 26302. | | | | | | | | | | | |
| | | | | Ore | 4.0- 5.0 | 5928. | 21010. | 3.98 | 1.72 | 2.26 | 28. | .76 | 421. | 60.00 | 788. | 11.10 | 625. | 55.00 |
| | | | | Ore | 5.0+**** | 30408. | 122645. | 10.59 | 4.43 | 6.16 | 73. | .75 | 7620. | 60.00 | 817. | 4.15 | 11820. | 55.00 |
| 97 | 1170. | Grum 2C-32 | A | Rock | | 739947. | 1991166. | | | | | | | | | | | |
| | | | | Sulph Wast | | 7587. | 24257. | | | | | | | | | | | |
| | | | | Ore | 4.0- 5.0 | 9660. | 32689. | 3.77 | 1.46 | 2.31 | 25. | .57 | 533. | 60.00 | 827. | 9.54 | 992. | 55.00 |
| | | | | Ore | 5.0+**** | 46320. | 183727. | 9.98 | 4.08 | 5.90 | 67. | .76 | 10355. | 60.00 | 818. | 4.59 | 16938. | 55.00 |
| 98 | 1165. | Grum 2C-33 | A | Rock | | 737795. | 1982437. | | | | | | | | | | | |
| | | | | Sulph Wast | | 11293. | 35843. | | | | | | | | | | | |
| | | | | Ore | 4.0- 5.0 | 13008. | 44014. | 3.80 | 1.53 | 2.27 | 26. | .58 | 760. | 60.00 | 810. | 9.29 | 1308. | 55.00 |
| | | | | Ore | 5.0+**** | 83256. | 313602. | 8.93 | 3.57 | 5.37 | 58. | .70 | 15115. | 60.00 | 811. | 4.85 | 25912. | 55.00 |
| 99 | 1161. | Grum 2C-34 | A | Rock | | 691748. | 1856258. | | | | | | | | | | | |
| | | | | Overburd | | 2700. | 7290. | | | | | | | | | | | |
| | | | | Sulph Wast | | 13837. | 43385. | | | | | | | | | | | |
| | | | | Ore | 4.0- 5.0 | 19224. | 65462. | 3.89 | 1.61 | 2.28 | 27. | .65 | 1207. | 60.00 | 817. | 9.91 | 1955. | 55.00 |
| | | | | Ore | 5.0+**** | 105360. | 389872. | 8.33 | 3.28 | 5.06 | 53. | .74 | 17043. | 60.00 | 806. | 5.55 | 30098. | 55.00 |
| 100 | 1156. | Grum 2C-35 | A | Rock | | 675572. | 1809012. | | | | | | | | | | | |
| | | | | Overburd | | 5400. | 14580. | | | | | | | | | | | |
| | | | | Sulph Wast | | 25483. | 78695. | | | | | | | | | | | |
| | | | | Ore | 4.0- 5.0 | 16032. | 53269. | 3.86 | 1.50 | 2.37 | 25. | .59 | 897. | 60.00 | 812. | 9.63 | 1662. | 55.00 |
| | | | | Ore | 5.0+**** | 150912. | 554389. | 8.78 | 3.29 | 5.49 | 55. | .89 | 24359. | 60.00 | 822. | 6.63 | 47243. | 55.00 |
| 101 | 1152. | Grum 2C-36 | A | Rock | | 635579. | 1700388. | | | | | | | | | | | |
| | | | | Overburd | | 2604. | 7031. | | | | | | | | | | | |
| | | | | Sulph Wast | | 21310. | 66232. | | | | | | | | | | | |
| | | | | Ore | 4.0- 5.0 | 15360. | 50688. | 3.86 | 1.47 | 2.37 | 25. | .64 | 851. | 60.00 | 816. | 10.40 | 1584. | 55.00 |
| | | | | Ore | 5.0+**** | 151644. | 562082. | 9.17 | 3.37 | 5.80 | 57. | .93 | 25344. | 60.00 | 839. | 6.79 | 51029. | 55.00 |
| 102 | 1147. | Grum 2C-37 | A | Rock | | 597190. | 1595882. | | | | | | | | | | | |
| | | | | Sulph Wast | | 24025. | 74743. | | | | | | | | | | | |
| | | | | Ore | 4.0- 5.0 | 15660. | 51553. | 3.96 | 1.54 | 2.42 | 27. | .65 | 898. | 60.00 | 850. | 10.26 | 1647. | 55.00 |
| | | | | Ore | 5.0+**** | 148992. | 560649. | 7.47 | 3.51 | 5.96 | 59. | .94 | 26542. | 60.00 | 826. | 6.62 | 52469. | 55.00 |

CURRAGH RESOURCES

Reserve Listing: VP 1-1 Mining Reserves

Page: 9

| Block No. | Block Desc | Dig Rate | Material | Grade | Volume BCM | Tonnes | HEAD GRADE | | | | | CONCENTRATE | | | | |
|-----------|------------|------------|----------|-------|------------|----------|------------|------|------|-----|------|-------------|-------|------|-------|--------|
| | | | | | | | Zn | Pb | Ag | Au | | Pb | Zn | Ag | Au | |
| 103 | 1143. | Grum 2C-38 | A | Rock | 553912. | 1478566. | | | | | | | | | | |
| | | | | Sulph | 26867. | 84306. | | | | | | | | | | |
| | | | | Wast | | | | | | | | | | | | |
| | | | | Ore | 4.0- 5.0 | 21060. | 3.93 | 1.52 | 2.40 | 27. | .66 | 1191. | 60.00 | 864. | 10.52 | 2202. |
| | | | | Ore | 5.0+**** | 145488. | 9.38 | 3.52 | 5.85 | 59. | .94 | 26267. | 60.00 | 831. | 6.57 | 50565. |
| 104 | 1138. | Grum 2C-39 | A | Rock | 500429. | 1334286. | | | | | | | | | | |
| | | | | Sulph | 32469. | 102745. | | | | | | | | | | |
| | | | | Wast | | | | | | | | | | | | |
| | | | | Ore | 4.0- 5.0 | 27684. | 3.86 | 1.50 | 2.36 | 27. | .67 | 1542. | 60.00 | 869. | 10.83 | 2867. |
| | | | | Ore | 5.0+**** | 138096. | 9.43 | 3.54 | 5.89 | 59. | .95 | 25190. | 60.00 | 827. | 6.57 | 48566. |
| 105 | 1134. | Grum 2C-40 | A | Rock | 469347. | 1250597. | | | | | | | | | | |
| | | | | Sulph | 45313. | 143168. | | | | | | | | | | |
| | | | | Wast | | | | | | | | | | | | |
| | | | | Ore | 4.0- 5.0 | 28620. | 3.89 | 1.49 | 2.40 | 27. | .69 | 1588. | 60.00 | 864. | 11.14 | 3004. |
| | | | | Ore | 5.0+**** | 127152. | 9.90 | 3.75 | 6.15 | 62. | .98 | 25104. | 60.00 | 826. | 6.47 | 47732. |
| 106 | 1129. | Grum 2C-41 | A | Rock | 429469. | 1144485. | | | | | | | | | | |
| | | | | Sulph | 51842. | 162376. | | | | | | | | | | |
| | | | | Wast | | | | | | | | | | | | |
| | | | | Ore | 4.0- 5.0 | 24732. | 3.91 | 1.53 | 2.38 | 27. | .67 | 1411. | 60.00 | 838. | 10.65 | 2571. |
| | | | | Ore | 5.0+**** | 120024. | 10.00 | 3.73 | 6.28 | 62. | .97 | 23362. | 60.00 | 826. | 6.40 | 45773. |
| 107 | 1125. | Grum 2C-42 | A | Rock | 331796. | 882506. | | | | | | | | | | |
| | | | | Sulph | 41189. | 125651. | | | | | | | | | | |
| | | | | Wast | | | | | | | | | | | | |
| | | | | Ore | 4.0- 5.0 | 27540. | 3.94 | 1.52 | 2.43 | 26. | .63 | 1522. | 60.00 | 836. | 10.06 | 2871. |
| | | | | Ore | 5.0+**** | 116248. | 9.49 | 3.48 | 6.01 | 58. | .97 | 20717. | 60.00 | 833. | 6.83 | 41880. |
| 108 | 1120. | Grum 2C-43 | A | Rock | 302676. | 804096. | | | | | | | | | | |
| | | | | Sulph | 33756. | 102137. | | | | | | | | | | |
| | | | | Wast | | | | | | | | | | | | |
| | | | | Ore | 4.0- 5.0 | 34380. | 3.88 | 1.53 | 2.34 | 26. | .58 | 1919. | 60.00 | 806. | 9.25 | 3425. |
| | | | | Ore | 5.0+**** | 122580. | 9.17 | 3.35 | 5.82 | 57. | .95 | 20209. | 60.00 | 840. | 6.96 | 41144. |
| 109 | 1116. | Grum 2C-44 | A | Rock | 257606. | 683304. | | | | | | | | | | |
| | | | | Sulph | 34307. | 102075. | | | | | | | | | | |
| | | | | Wast | | | | | | | | | | | | |
| | | | | Ore | 4.0- 5.0 | 29100. | 3.82 | 1.53 | 2.28 | 25. | .58 | 1613. | 60.00 | 788. | 9.28 | 2783. |
| | | | | Ore | 5.0+**** | 123048. | 9.02 | 3.29 | 5.72 | 56. | .92 | 19661. | 60.00 | 843. | 6.92 | 40004. |
| 110 | 1111. | Grum 2C-45 | A | Rock | 228029. | 602719. | | | | | | | | | | |
| | | | | Sulph | 36187. | 107143. | | | | | | | | | | |
| | | | | Wast | | | | | | | | | | | | |
| | | | | Ore | 4.0- 5.0 | 24300. | 3.78 | 1.56 | 2.22 | 26. | .60 | 1370. | 60.00 | 783. | 9.35 | 2241. |
| | | | | Ore | 5.0+**** | 121512. | 9.59 | 3.49 | 6.10 | 60. | .96 | 21176. | 60.00 | 853. | 6.80 | 43261. |
| 111 | 1107. | Grum 2C-46 | A | Rock | 206619. | 543531. | | | | | | | | | | |
| | | | | Sulph | 33525. | 99433. | | | | | | | | | | |
| | | | | Wast | | | | | | | | | | | | |
| | | | | Ore | 4.0- 5.0 | 22680. | 3.81 | 1.57 | 2.24 | 27. | .65 | 1307. | 60.00 | 827. | 10.01 | 2141. |
| | | | | Ore | 5.0+**** | 120540. | 9.72 | 3.54 | 6.17 | 61. | .98 | 21835. | 60.00 | 854. | 6.86 | 44420. |
| 112 | 1102. | Grum 2C-47 | A | Rock | 180677. | 474412. | | | | | | | | | | |
| | | | | Sulph | 33261. | 100117. | | | | | | | | | | |
| | | | | Wast | | | | | | | | | | | | |
| | | | | Ore | 4.0- 5.0 | 19440. | 3.91 | 1.55 | 2.36 | 26. | .65 | 1097. | 60.00 | 805. | 10.23 | 1943. |
| | | | | Ore | 5.0+**** | 109620. | 9.76 | 3.53 | 6.23 | 61. | 1.01 | 19931. | 60.00 | 857. | 7.15 | 41116. |
| 113 | 1098. | Grum 2C-48 | A | Rock | 171888. | 452776. | | | | | | | | | | |
| | | | | Sulph | 24565. | 75226. | | | | | | | | | | |
| | | | | Wast | | | | | | | | | | | | |
| | | | | Ore | 4.0- 5.0 | 16404. | 3.86 | 1.51 | 2.35 | 26. | .61 | 905. | 60.00 | 810. | 9.84 | 1645. |
| | | | | Ore | 5.0+**** | 99048. | 9.65 | 3.47 | 6.18 | 60. | 1.02 | 17352. | 60.00 | 855. | 7.28 | 36159. |

CURRAGH RESOURCES

Reserve Listing: VP 1-1 Mining Reserves

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[illegible]

CURRAGH RESOURCES

Reserve Listing: VP 1-1 Mining Reserves

Page: 11

| Block | Bench | Block | Dig | Material | Grade | Volume | Tonnes | H E A D G R A D E | | | | | C O N C E N T R A T E | | | | | |
|-------|-------|------------|-----|------------|----------|---------|----------|---------------------|------|------|--------|--------|-----------------------|-------|--------|--------|--------|-------|
| | | | | | | | | TPb+Zn | TPb | ZN | Ag g/t | Au g/t | Pb Tns | TPb | Ag g/t | Au g/t | Zn tns | ZN |
| 131 | 1242. | Grum 3- 16 | A | Rock | | 585000. | 1579500. | | | | | | | | | | | |
| | | | | Overburd | | 39132. | 82177. | | | | | | | | | | | |
| 132 | 1237. | Grum 3- 17 | A | Rock | | 590976. | 1595635. | | | | | | | | | | | |
| | | | | Overburd | | 32496. | 68242. | | | | | | | | | | | |
| 133 | 1233. | Grum 3- 18 | A | Rock | | 581844. | 1570979. | | | | | | | | | | | |
| | | | | Overburd | | 30480. | 64008. | | | | | | | | | | | |
| 134 | 1228. | Grum 3- 19 | A | Rock | | 557952. | 1506470. | | | | | | | | | | | |
| | | | | Overburd | | 26160. | 54936. | | | | | | | | | | | |
| 135 | 1224. | Grum 3- 20 | A | Rock | | 561190. | 1515166. | | | | | | | | | | | |
| | | | | Overburd | | 34032. | 71467. | | | | | | | | | | | |
| | | | | Sulph Wast | | 577. | 2015. | | | | | | | | | | | |
| | | | | Ore | 5.0+**** | 264. | 1150. | 4.62 | .95 | 3.67 | 19. | .52 | 11. | 60.00 | 942. | 12.98 | 60. | 55.00 |
| 136 | 1219. | Grum 3- 21 | A | Rock | | 575399. | 1553206. | | | | | | | | | | | |
| | | | | Overburd | | 25824. | 54230. | | | | | | | | | | | |
| | | | | Sulph Wast | | 675. | 2374. | | | | | | | | | | | |
| | | | | Ore | 5.0+**** | 2700. | 10813. | 11.25 | 3.68 | 7.57 | 62. | .91 | 536. | 60.00 | 836. | 6.20 | 1328. | 55.00 |
| 137 | 1215. | Grum 3- 22 | A | Rock | | 551343. | 1488171. | | | | | | | | | | | |
| | | | | Overburd | | 14016. | 29434. | | | | | | | | | | | |
| | | | | Sulph Wast | | 1471. | 4817. | | | | | | | | | | | |
| | | | | Ore | 5.0+**** | 3012. | 12365. | 13.75 | 4.34 | 9.41 | 75. | .92 | 742. | 60.00 | 856. | 5.37 | 1934. | 55.00 |
| 138 | 1210. | Grum 3- 23 | A | Rock | | 552278. | 1490448. | | | | | | | | | | | |
| | | | | Overburd | | 13824. | 28030. | | | | | | | | | | | |
| | | | | Sulph Wast | | 1771. | 6224. | | | | | | | | | | | |
| | | | | Ore | 4.0- 5.0 | 540. | 1959. | 3.98 | 1.75 | 2.23 | 26. | .41 | 40. | 60.00 | 719. | 5.76 | 57. | 55.00 |
| | | | | Ore | 5.0+**** | 4164. | 17299. | 14.39 | 4.73 | 9.66 | 80. | 1.01 | 1149. | 60.00 | 840. | 5.39 | 2786. | 55.00 |
| 139 | 1206. | Grum 3- 24 | A | Rock | | 590971. | 1594929. | | | | | | | | | | | |
| | | | | Overburd | | 10800. | 22680. | | | | | | | | | | | |
| | | | | Sulph Wast | | 1879. | 6578. | | | | | | | | | | | |
| | | | | Ore | 4.0- 5.0 | 540. | 1965. | 3.99 | 1.77 | 2.23 | 26. | .42 | 40. | 60.00 | 713. | 5.88 | 57. | 55.00 |
| | | | | Ore | 5.0+**** | 4632. | 18601. | 12.51 | 4.25 | 8.26 | 71. | 1.00 | 1099. | 60.00 | 832. | 5.89 | 2517. | 55.00 |
| 140 | 1201. | Grum 3- 25 | A | Rock | | 624401. | 1685083. | | | | | | | | | | | |
| | | | | Overburd | | 12168. | 25553. | | | | | | | | | | | |
| | | | | Sulph Wast | | 3655. | 12877. | | | | | | | | | | | |
| | | | | Ore | 4.0- 5.0 | 120. | 395. | 4.34 | 1.47 | 2.87 | 28. | .91 | 6. | 60.00 | 908. | 15.14 | 15. | 55.00 |
| | | | | Ore | 5.0+**** | 8184. | 30228. | 9.84 | 3.55 | 6.29 | 60. | .87 | 1457. | 60.00 | 828. | 6.09 | 3006. | 55.00 |
| 141 | 1197. | Grum 3- 26 | A | Rock | | 562834. | 1519056. | | | | | | | | | | | |
| | | | | Sulph Wast | | 5252. | 18643. | | | | | | | | | | | |
| | | | | Ore | 5.0+**** | 7848. | 27712. | 8.39 | 3.03 | 5.36 | 51. | .82 | 1115. | 60.00 | 828. | 6.44 | 2297. | 55.00 |
| 142 | 1192. | Grum 3- 27 | A | Rock | | 523968. | 1414106. | | | | | | | | | | | |
| | | | | Sulph Wast | | 3667. | 13043. | | | | | | | | | | | |
| | | | | Ore | 5.0+**** | 6144. | 22786. | 9.66 | 3.51 | 6.15 | 60. | .96 | 1097. | 60.00 | 836. | 6.44 | 2211. | 55.00 |

CURRAGH RESOURCES

Reserve Listing: VP 1-1 Mining Reserves

Page: 12

| Block | Bench | Block Desc | Dig Material | Grade | Volume BCM | Tonnes | HEAD GRADE | | | | | CONCENTRATE | | | | |
|-------|-------|------------|--------------|----------|------------|----------|------------|------|------|--------|--------|-------------|-------|--------|--------|--------|
| | | | | | | | ZPb+Zn | ZPb | ZZn | Ag g/t | Au g/t | Pb Tns | ZPb | Ag g/t | Au g/t | Zn Tns |
| 143 | 1188. | Grum 3- 28 | A Rock | | 538797. | 1454145. | | | | | | | | | | |
| | | | Sulph Wast | | 785. | 2758. | | | | | | | | | | |
| | | | Ore | 5.0+**** | 4908. | 19123. | 10.24 | 3.79 | 6.45 | 66. | .91 | 987. | 60.00 | 868. | 5.87 | 1949. |
| 144 | 1183. | Grum 3- 29 | A Rock | | 509204. | 1374495. | | | | | | | | | | |
| | | | Sulph Wast | | 1241. | 3911. | | | | | | | | | | |
| | | | Ore | 4.0- 5.0 | 540. | 1827. | 4.19 | 1.75 | 2.43 | 26. | .73 | 37. | 60.00 | 719. | 10.22 | 59. |
| | | | Ore | 5.0+**** | 2688. | 10425. | 8.67 | 3.38 | 5.29 | 62. | .57 | 467. | 60.00 | 910. | 4.20 | 837. |
| 145 | 1179. | Grum 3- 30 | A Rock | | 528272. | 1426066. | | | | | | | | | | |
| | | | Sulph Wast | | 1878. | 5735. | | | | | | | | | | |
| | | | Ore | 4.0- 5.0 | 2232. | 7438. | 3.96 | 1.44 | 2.51 | 27. | .64 | 119. | 60.00 | 878. | 10.70 | 249. |
| | | | Ore | 5.0+**** | 1248. | 4886. | 8.64 | 3.45 | 5.20 | 60. | .50 | 222. | 60.00 | 869. | 3.63 | 383. |
| 146 | 1174. | Grum 3- 31 | A Rock | | 560788. | 1513566. | | | | | | | | | | |
| | | | Sulph Wast | | 5178. | 15582. | | | | | | | | | | |
| | | | Ore | 4.0- 5.0 | 4608. | 15226. | 3.75 | 1.35 | 2.40 | 25. | .61 | 225. | 60.00 | 881. | 10.93 | 482. |
| | | | Ore | 5.0+**** | 12792. | 41948. | 6.41 | 2.33 | 4.09 | 36. | .31 | 1203. | 60.00 | 768. | 3.27 | 2491. |
| 147 | 1170. | Grum 3- 32 | A Rock | | 592570. | 1598684. | | | | | | | | | | |
| | | | Sulph Wast | | 3713. | 11688. | | | | | | | | | | |
| | | | Ore | 4.0- 5.0 | 5364. | 17576. | 3.72 | 1.25 | 2.48 | 26. | .56 | 235. | 60.00 | 975. | 10.75 | 579. |
| | | | Ore | 5.0+**** | 17304. | 58914. | 7.15 | 2.64 | 4.51 | 44. | .43 | 1985. | 60.00 | 820. | 4.02 | 3935. |
| 148 | 1165. | Grum 3- 33 | A Rock | | 489112. | 1319135. | | | | | | | | | | |
| | | | Sulph Wast | | 3405. | 10534. | | | | | | | | | | |
| | | | Ore | 4.0- 5.0 | 4080. | 13279. | 3.73 | 1.32 | 2.42 | 25. | .49 | 190. | 60.00 | 910. | 8.97 | 425. |
| | | | Ore | 5.0+**** | 16020. | 57250. | 7.41 | 2.86 | 4.55 | 48. | .48 | 2116. | 60.00 | 818. | 4.16 | 3863. |
| 149 | 1161. | Grum 3- 34 | A Rock | | 480750. | 1296422. | | | | | | | | | | |
| | | | Sulph Wast | | 4320. | 13037. | | | | | | | | | | |
| | | | Ore | 4.0- 5.0 | 4476. | 14634. | 3.77 | 1.44 | 2.32 | 26. | .49 | 235. | 60.00 | 847. | 8.35 | 447. |
| | | | Ore | 5.0+**** | 17124. | 61374. | 6.98 | 2.69 | 4.30 | 45. | .46 | 2107. | 60.00 | 814. | 4.22 | 3866. |
| 150 | 1156. | Grum 3- 35 | A Rock | | 459849. | 1239613. | | | | | | | | | | |
| | | | Sulph Wast | | 4551. | 14007. | | | | | | | | | | |
| | | | Ore | 4.0- 5.0 | 4440. | 14580. | 3.71 | 1.44 | 2.27 | 25. | .51 | 235. | 60.00 | 842. | 8.65 | 433. |
| | | | Ore | 5.0+**** | 20340. | 73692. | 7.23 | 2.85 | 4.38 | 47. | .51 | 2712. | 60.00 | 815. | 4.41 | 4750. |
| 151 | 1152. | Grum 3- 36 | A Rock | | 452487. | 1219873. | | | | | | | | | | |
| | | | Overburd | | 96. | 259. | | | | | | | | | | |
| | | | Sulph Wast | | 2051. | 6700. | | | | | | | | | | |
| | | | Ore | 4.0- 5.0 | 3000. | 9846. | 3.69 | 1.40 | 2.30 | 25. | .47 | 152. | 60.00 | 855. | 8.16 | 296. |
| | | | Ore | 5.0+**** | 16428. | 61596. | 7.77 | 3.17 | 4.69 | 52. | .59 | 2571. | 60.00 | 806. | 4.58 | 4269. |
| 152 | 1147. | Grum 3- 37 | A Rock | | 451400. | 1216332. | | | | | | | | | | |
| | | | Sulph Wast | | 2764. | 8933. | | | | | | | | | | |
| | | | Ore | 4.0- 5.0 | 2160. | 7206. | 3.84 | 1.54 | 2.29 | 28. | .48 | 126. | 60.00 | 863. | 7.48 | 216. |
| | | | Ore | 5.0+**** | 16844. | 67041. | 7.63 | 3.29 | 4.34 | 55. | .81 | 2922. | 60.00 | 818. | 5.98 | 4304. |
| 153 | 1140. | Grum 3- 38 | A Rock | | 419362. | 1128579. | | | | | | | | | | |
| | | | Sulph Wast | | 6478. | 20673. | | | | | | | | | | |
| | | | Ore | 4.0- 5.0 | 324. | 1064. | 3.95 | 1.77 | 2.17 | 30. | .45 | 23. | 60.00 | 824. | 6.17 | 30. |
| | | | Ore | 5.0+**** | 22680. | 95129. | 6.77 | 3.17 | 4.63 | 53. | 1.00 | 2911. | 60.00 | 829. | 8.00 | 4991. |

Reserve Listing: UP 1-1 Mining Reserves

| Block No. | Bench Desc | Block | Dig Rate | Material Grade | Volume BCM | Tonnes | HEAD GRADE | | | | | CONCENTRATE | | | | | | |
|-----------|------------|------------|----------|----------------------------------|--------------------------|--------------------------------------|---|-----------------------|----------------------|----------------------|--------------------|---------------------|-----------------------|------------------------|-----------------------|-----------------------|------------------------|------------------------|
| | | | | | | | IPb+Zn | IPb | Zn | Ag g/t | Au g/t | Pb lbs | IPb | Ag g/t | Au g/t | Zn lbs | Zn | |
| 154 | 1138. | Grum 3- 39 | A | Rock
Sulph Wast
Ore
Ore |
4.0- 5.0
5.0+**** | 398887.
6843.
396.
24144. | 1073259.
22587.
1592.
99439. |

3.79
6.28 |

1.36
2.93 |

2.43
3.35 |

30.
49. |

.89
.99 |

24.
3780. |

60.00
60.00 |

1044.
818. |

15.87
8.29 |

51.
4719. |

55.00
55.00 |
| 155 | 1134. | Grum 3- 40 | A | Rock
Sulph Wast
Ore
Ore |
4.0- 5.0
5.0+**** | 389057.
6056.
540.
17868. | 1048082.
20765.
2198.
70283. |

4.17
6.13 |

1.55
2.79 |

2.62
3.35 |

33.
44. |

.99
.84 |

38.
2503. |

60.00
60.00 |

1026.
776. |

15.59
7.40 |

77.
3310. |

55.00
55.00 |
| 156 | 1129. | Grum 3- 41 | A | Rock
Sulph Wast
Ore
Ore |
4.0- 5.0
5.0+**** | 378331.
10592.
648.
14880. | 1019508.
35039.
2636.
58390. |

3.86
7.12 |

1.41
3.32 |

2.45
3.79 |

32.
49. |

.96
.77 |

41.
2569. |

60.00
60.00 |

1066.
733. |

16.60
5.74 |

86.
3178. |

55.00
55.00 |
| 157 | 1125. | Grum 3- 42 | A | Rock
Sulph Wast
Ore
Ore |
4.0- 5.0
5.0+**** | 399386.
17240.
1080.
23328. | 1074856.
56584.
4543.
94176. |

3.88
7.62 |

1.83
3.33 |

2.05
4.29 |

39.
50. |

1.08
.78 |

97.
4178. |

60.00
60.00 |

1019.
740. |

14.36
5.75 |

120.
5950. |

55.00
55.00 |
| 158 | 1120. | Grum 3- 43 | A | Rock
Sulph Wast
Ore
Ore |
4.0- 5.0
5.0+**** | 377021.
12536.
4500.
29736. | 1013172.
39927.
17424.
120257. |

3.93
7.90 |

1.80
3.25 |

2.13
4.65 |

35.
50. |

.89
.75 |

365.
5192. |

60.00
60.00 |

948.
759. |

12.13
5.55 |

483.
8376. |

55.00
55.00 |
| 159 | 1116. | Grum 3- 44 | A | Rock
Sulph Wast
Ore
Ore |
4.0- 5.0
5.0+**** | 376980.
9239.
7452.
30444. | 1013158.
28762.
29225.
118495. |

3.87
7.84 |

1.74
2.99 |

2.13
4.85 |

35.
48. |

.98
.78 |

590.
4624. |

60.00
60.00 |

961.
784. |

13.81
6.31 |

804.
8694. |

55.00
55.00 |
| 160 | 1111. | Grum 3- 45 | A | Rock
Sulph Wast
Ore
Ore |
4.0- 5.0
5.0+**** | 344994.
9731.
8028.
22920. | 928071.
29909.
32324.
85127. |

3.85
8.48 |

1.66
3.16 |

2.19
5.32 |

33.
50. |

1.00
.66 |

615.
3528. |

60.00
60.00 |

961.
779. |

14.67
5.14 |

923.
6951. |

55.00
55.00 |
| 161 | 1107. | Grum 3- 46 | A | Rock
Sulph Wast
Ore
Ore |
4.0- 5.0
5.0+**** | 324727.
10931.
1620.
32436. | 872796.
34418.
6869.
124003. |

3.99
9.62 |

1.63
3.59 |

2.37
6.03 |

29.
59. |

1.09
.83 |

128.
6024. |

60.00
60.00 |

868.
810. |

16.42
5.73 |

214.
11740. |

55.00
55.00 |
| 162 | 1102. | Grum 3- 47 | A | Rock
Sulph Wast
Ore
Ore |
4.0- 5.0
5.0+**** | 323458.
7457.
540.
38208. | 868492.
23765.
2367.
149085. |

4.17
11.89 |

1.79
4.35 |

2.38
7.54 |

29.
73. |

1.13
.99 |

49.
9055. |

60.00
60.00 |

775.
834. |

15.51
5.71 |

74.
18237. |

55.00
55.00 |
| 163 | 1098. | Grum 3- 48 | A | Rock
Sulph Wast
Ore
Ore |
4.0- 5.0
5.0+**** | 288297.
5653.
2436.
37668. | 773515.
18410.
9041.
146728. |

3.92
10.89 |

1.62
3.96 |

2.30
6.73 |

30.
70. |

.98
.97 |

167.
7975. |

60.00
60.00 |

901.
879. |

14.74
6.13 |

272.
15694. |

55.00
55.00 |
| 164 | 1093. | Grum 3- 49 | A | Rock
Sulph Wast
Ore
Ore |
4.0- 5.0
5.0+**** | 251393.
5174.
1820.
40428. | 673046.
17256.
10978.
160411. |

3.84
10.61 |

1.46
4.01 |

2.38
6.57 |

32.
71. |

1.26
.98 |

178.
8847. |

60.00
60.00 |

1054.
275. | < | | |

CORRAGH RESOURCES

Reserve Listing: VP 1-1 Mining Reserves

Page: 14

| Block
No. | Bench
Desc. | Block | Dig
Rate | Material
Type | Grade
Zn Pb+Zn | Volume
BCM | Tonnes | HEAD GRADE | | | | | CONCENTRATE | | | | |
|--------------|----------------|------------|-------------|------------------|-------------------|---------------|---------|------------|------|------|-----|------|-------------|-------|------|-------|--------|
| | | | | | | | | Zn | Pb | Ag | Cu | Mo | Pb | Zn | Ag | Cu | Mo |
| 165 | 1089. | Grum J- 50 | A | Rock | | 235445. | 627938. | | | | | | | | | | |
| | | | | Sulph | | 6656. | 22083. | | | | | | | | | | |
| | | | | Wast | | | | | | | | | | | | | |
| | | | | Ore | 4.0- 5.0 | 3780. | 13808. | 4.06 | 1.62 | 2.45 | 32. | 1.09 | 254. | 60.00 | 964. | 16.35 | 448. |
| | | | | Ore | 5.0+**** | 57828. | 227438. | 9.99 | 3.86 | 6.13 | 67. | 1.00 | 11961. | 60.00 | 861. | 6.41 | 21802. |
| 166 | 1084. | Grum J- 51 | A | Rock | | 330060. | 877114. | | | | | | | | | | |
| | | | | Sulph | | 21940. | 67604. | | | | | | | | | | |
| | | | | Wast | | | | | | | | | | | | | |
| | | | | Ore | 4.0- 5.0 | 19980. | 66276. | 3.83 | 1.58 | 2.25 | 27. | .65 | 1187. | 60.00 | 813. | 10.06 | 1947. |
| | | | | Ore | 5.0+**** | 120984. | 457244. | 8.93 | 3.33 | 5.60 | 59. | .97 | 20171. | 60.00 | 872. | 7.24 | 39453. |
| 167 | 1080. | Grum J- 52 | A | Rock | | 296892. | 787914. | | | | | | | | | | |
| | | | | Sulph | | 25409. | 78472. | | | | | | | | | | |
| | | | | Wast | | | | | | | | | | | | | |
| | | | | Ore | 4.0- 5.0 | 15660. | 51132. | 3.83 | 1.54 | 2.29 | 25. | .71 | 888. | 60.00 | 772. | 11.21 | 1535. |
| | | | | Ore | 5.0+**** | 114528. | 437891. | 8.71 | 3.25 | 5.46 | 57. | .97 | 18756. | 60.00 | 871. | 7.46 | 36593. |
| 168 | 1075. | Grum J- 53 | A | Rock | | 265713. | 704518. | | | | | | | | | | |
| | | | | Sulph | | 30200. | 94182. | | | | | | | | | | |
| | | | | Wast | | | | | | | | | | | | | |
| | | | | Ore | 4.0- 5.0 | 12768. | 41945. | 3.89 | 1.58 | 2.31 | 26. | .72 | 753. | 60.00 | 784. | 11.04 | 1271. |
| | | | | Ore | 5.0+**** | 95880. | 377507. | 8.73 | 3.26 | 5.48 | 58. | 1.01 | 16234. | 60.00 | 880. | 7.73 | 31742. |
| 169 | 1071. | Grum J- 54 | A | Rock | | 242041. | 642619. | | | | | | | | | | |
| | | | | Sulph | | 17507. | 55477. | | | | | | | | | | |
| | | | | Wast | | | | | | | | | | | | | |
| | | | | Ore | 4.0- 5.0 | 7608. | 24982. | 3.83 | 1.58 | 2.25 | 25. | .68 | 446. | 60.00 | 771. | 10.48 | 734. |
| | | | | Ore | 5.0+**** | 77100. | 308388. | 8.70 | 3.25 | 5.45 | 58. | 1.00 | 13160. | 60.00 | 883. | 7.71 | 25721. |
| 170 | 1066. | Grum J- 55 | A | Rock | | 224040. | 595023. | | | | | | | | | | |
| | | | | Sulph | | 11446. | 37675. | | | | | | | | | | |
| | | | | Wast | | | | | | | | | | | | | |
| | | | | Ore | 4.0- 5.0 | 7020. | 23392. | 3.84 | 1.52 | 2.31 | 26. | .74 | 401. | 60.00 | 813. | 11.74 | 709. |
| | | | | Ore | 5.0+**** | 70704. | 281667. | 8.43 | 3.13 | 5.31 | 55. | .97 | 11504. | 60.00 | 866. | 7.73 | 22764. |
| 171 | 1062. | Grum J- 56 | A | Rock | | 200692. | 531991. | | | | | | | | | | |
| | | | | Sulph | | 10449. | 34158. | | | | | | | | | | |
| | | | | Wast | | | | | | | | | | | | | |
| | | | | Ore | 4.0- 5.0 | 7020. | 24046. | 3.89 | 1.44 | 2.45 | 28. | .89 | 384. | 60.00 | 925. | 15.07 | 782. |
| | | | | Ore | 5.0+**** | 72360. | 285843. | 8.15 | 3.00 | 5.15 | 52. | .92 | 11123. | 60.00 | 861. | 7.65 | 22299. |
| 172 | 1057. | Grum J- 57 | A | Rock | | 168662. | 445335. | | | | | | | | | | |
| | | | | Sulph | | 11040. | 36508. | | | | | | | | | | |
| | | | | Wast | | | | | | | | | | | | | |
| | | | | Ore | 4.0- 5.0 | 8640. | 30762. | 3.92 | 1.52 | 2.40 | 28. | .91 | 527. | 60.00 | 884. | 14.65 | 976. |
| | | | | Ore | 5.0+**** | 68400. | 272015. | 8.28 | 3.03 | 5.24 | 54. | .92 | 10731. | 60.00 | 884. | 7.55 | 21714. |
| 173 | 1053. | Grum J- 58 | A | Rock | | 147677. | 388670. | | | | | | | | | | |
| | | | | Sulph | | 7399. | 25432. | | | | | | | | | | |
| | | | | Wast | | | | | | | | | | | | | |
| | | | | Ore | 4.0- 5.0 | 8100. | 27807. | 3.87 | 1.49 | 2.38 | 29. | .80 | 464. | 60.00 | 927. | 13.08 | 872. |
| | | | | Ore | 5.0+**** | 64284. | 262825. | 8.81 | 3.24 | 5.57 | 59. | 1.04 | 11203. | 60.00 | 897. | 7.94 | 22526. |
| 174 | 1048. | Grum J- 59 | A | Rock | | 131509. | 343851. | | | | | | | | | | |
| | | | | Sulph | | 7819. | 26656. | | | | | | | | | | |
| | | | | Wast | | | | | | | | | | | | | |
| | | | | Ore | 4.0- 5.0 | 5940. | 21245. | 3.87 | 1.60 | 2.27 | 30. | .89 | 387. | 60.00 | 900. | 13.68 | 630. |
| | | | | Ore | 5.0+**** | 64032. | 271202. | 9.42 | 3.50 | 5.91 | 63. | 1.14 | 12661. | 60.00 | 896. | 8.08 | 24869. |
| 175 | 1044. | Grum J- 60 | A | Rock | | 95184. | 255690. | | | | | | | | | | |
| | | | | Sulph | | 9460. | 31029. | | | | | | | | | | |
| | | | | Wast | | | | | | | | | | | | | |
| | | | | Ore | 4.0- 5.0 | 2760. | 9862. | 3.63 | 1.57 | 2.31 | 29. | .94 | 176. | 60.00 | 871. | 14.56 | 298. |
| | | | | Ore | 5.0+**** | 50184. | 218248. | 9.78 | 3.64 | 6.14 | 66. | 1.17 | 10677. | 60.00 | 899. | 7.98 | 20718. |

CURRAGH RESOURCES

Reserve Listing: UP 1-1 Mining Reserves

Page: 15

| Block No. | Bench Desc | Block | Dig Rate | Material | Grade | Volume BCM | Tonnes | H E A D G R A D E | | | | | C O N C E N T R A T E | | | | | |
|-----------|------------|------------|----------|------------|----------|------------|----------|-------------------|------|------|--------|--------|-----------------------|-------|--------|--------|--------|-------|
| | | | | | | | | ZPb+Zn | ZPb | ZZn | Ag g/t | Au g/t | Pb Tns | ZPb | Ag g/t | Au g/t | Zn tns | ZZn |
| 176 | 1039. | Brue 3- 61 | A | Rock | | 67029. | 176041. | | | | | | | | | | | |
| | | | | Sulph Wast | | 4650. | 15498. | | | | | | | | | | | |
| | | | | Ore | 4.0- 5.0 | 3780. | 12538. | 3.97 | 1.61 | 2.36 | 29. | .81 | 230. | 60.00 | 861. | 12.14 | 389. | 55.00 |
| | | | | Ore | 5.0+**** | 25380. | 111327. | 9.36 | 3.56 | 5.80 | 62. | 1.18 | 5292. | 60.00 | 864. | 8.29 | 9957. | 55.00 |
| 177 | 1035. | Brue 3- 62 | A | Rock | | 54854. | 144441. | | | | | | | | | | | |
| | | | | Sulph Wast | | 2645. | 9070. | | | | | | | | | | | |
| | | | | Ore | 4.0- 5.0 | 6240. | 20602. | 4.02 | 1.62 | 2.41 | 28. | .85 | 379. | 60.00 | 846. | 12.76 | 656. | 55.00 |
| | | | | Ore | 5.0+**** | 18336. | 79986. | 9.30 | 3.60 | 5.70 | 60. | 1.19 | 3848. | 60.00 | 824. | 8.29 | 7014. | 55.00 |
| 178 | 1030. | Brue 3-63 | A | Rock | | 36052. | 95786. | | | | | | | | | | | |
| | | | | Sulph Wast | | 2999. | 9523. | | | | | | | | | | | |
| | | | | Ore | 4.0- 5.0 | 1584. | 5301. | 4.04 | 1.63 | 2.41 | 29. | .84 | 99. | 60.00 | 853. | 12.59 | 169. | 55.00 |
| | | | | Ore | 5.0+**** | 9672. | 39818. | 8.61 | 3.33 | 5.28 | 60. | 1.13 | 1749. | 60.00 | 889. | 8.49 | 3202. | 55.00 |
| 184 | 1178. | Vangord 7 | A | Overburd | | 81. | 170. | | | | | | | | | | | |
| 185 | 1174. | Vangord 8 | A | Rock | | 2003. | 5407. | | | | | | | | | | | |
| | | | | Overburd | | 1976. | 4149. | | | | | | | | | | | |
| 186 | 1169. | Vangord 9 | A | Rock | | 8249. | 22271. | | | | | | | | | | | |
| | | | | Overburd | | 9603. | 20166. | | | | | | | | | | | |
| 187 | 1165. | Vangord 10 | A | Rock | | 15192. | 41018. | | | | | | | | | | | |
| | | | | Overburd | | 42381. | 89000. | | | | | | | | | | | |
| 188 | 1160. | Vangord 11 | A | Rock | | 23297. | 62901. | | | | | | | | | | | |
| | | | | Overburd | | 99761. | 209497. | | | | | | | | | | | |
| 189 | 1156. | Vangord 12 | A | Rock | | 31644. | 85439. | | | | | | | | | | | |
| | | | | Overburd | | 198545. | 416943. | | | | | | | | | | | |
| 190 | 1151. | Vangord 13 | A | Rock | | 38124. | 102935. | | | | | | | | | | | |
| | | | | Overburd | | 351338. | 737808. | | | | | | | | | | | |
| 191 | 1147. | Vangord 14 | A | Rock | | 50387. | 134915. | | | | | | | | | | | |
| | | | | Overburd | | 507362. | 1065459. | | | | | | | | | | | |
| | | | | Sulph Wast | | 14096. | 48923. | | | | | | | | | | | |
| | | | | Baritic | 5.0+**** | 4253. | 17802. | 7.86 | 3.02 | 4.84 | 45. | .41 | 825. | 53.00 | 636. | 3.54 | 1240. | 55.00 |
| | | | | Pyritic | 4.0- 5.0 | 4253. | 15351. | 3.85 | 1.72 | 2.13 | 25. | .28 | 359. | 49.00 | 477. | 5.33 | 396. | 51.00 |
| | | | | Pyritic | 5.0+**** | 810. | 2978. | 4.58 | 2.10 | 2.48 | 28. | .29 | 89. | 49.00 | 435. | 4.58 | 92. | 51.00 |
| 192 | 1142. | Vangord 15 | A | Rock | | 78711. | 207306. | | | | | | | | | | | |
| | | | | Overburd | | 600228. | 1260479. | | | | | | | | | | | |
| | | | | Sulph Wast | | 22010. | 75364. | | | | | | | | | | | |
| | | | | Baritic | 4.0- 5.0 | 1215. | 4630. | 4.00 | 1.92 | 2.08 | 31. | .65 | 125. | 53.00 | 688. | 8.74 | 119. | 55.00 |
| | | | | Baritic | 5.0+**** | 28350. | 116434. | 7.35 | 3.04 | 4.30 | 44. | .49 | 5453. | 53.00 | 524. | 4.18 | 7089. | 55.00 |
| | | | | Pyritic | 4.0- 5.0 | 810. | 2845. | 4.02 | 1.74 | 2.28 | 25. | .27 | 67. | 49.00 | 465. | 4.99 | 79. | 51.00 |
| | | | | Quartzitic | 4.0- 5.0 | 6885. | 25229. | 3.97 | 1.58 | 2.39 | 25. | .41 | 472. | 58.00 | 626. | 2.77 | 787. | 53.00 |
| | | | | Quartzitic | 5.0+**** | 810. | 3141. | 4.57 | 2.06 | 2.51 | 27. | .41 | 81. | 58.00 | 519. | 2.17 | 104. | 55.00 |

CONFIDENTIAL - NOT FOR DISCLOSURE

Reserve Listing: VP 1-1 Mining Reserves

Page: 16

| Block
No. | Bench
Desc | Dig
Rate | Material | Grade | Value
BCM | Tonnes | H E A D G R A D E | | | | C O N C E N T R A T E | | | | | | |
|--------------|---------------|-------------|----------|---------------------|--------------|----------|-------------------|------|------|--------|-----------------------|--------|-------|--------|--------|--------|-------|
| | | | | | | | 2Pb+2n | 2Pb | 2Zn | Ag g/t | Au g/t | Pb Tns | 2Pb | Ag g/t | Au g/t | Zn tns | 2Zn |
| 193 | 1138. | Vangord 16 | A | Rock | 152547. | 400341. | | | | | | | | | | | |
| | | | | Overburd | 547578. | 1149914. | | | | | | | | | | | |
| | | | | Sulph Wast | 33960. | 117734. | | | | | | | | | | | |
| | | | | Baritic 4.0- 5.0 | 203. | 912. | 3.65 | 1.59 | 2.06 | 37. | .59 | 19. | 53.00 | 989. | 9.60 | 23. | 55.00 |
| | | | | Baritic 5.0+**** | 61052. | 256406. | 7.51 | 3.24 | 4.27 | 46. | .48 | 12916. | 53.00 | 609. | 3.93 | 15464. | 55.00 |
| | | | | Pyritic 4.0- 5.0 | 3240. | 13135. | 3.92 | 1.57 | 2.35 | 34. | .00 | 276. | 49.00 | 706. | .00 | 380. | 51.00 |
| | | | | Quartzitic 4.0- 5.0 | 7929. | 29060. | 3.73 | 1.50 | 2.23 | 24. | .45 | 514. | 58.00 | 634. | 3.24 | 833. | 53.00 |
| | | | | Quartzitic 5.0+**** | 2525. | 10040. | 4.55 | 2.16 | 2.39 | 29. | .55 | 275. | 58.00 | 541. | 2.74 | 312. | 53.00 |
| 194 | 1133. | Vangord 17 | A | Rock | 179132. | 468162. | | | | | | | | | | | |
| | | | | Overburd | 511551. | 1074257. | | | | | | | | | | | |
| | | | | Sulph Wast | 38829. | 134611. | | | | | | | | | | | |
| | | | | Baritic 5.0+**** | 79785. | 331829. | 7.53 | 3.30 | 4.23 | 45. | .58 | 17115. | 53.00 | 585. | 4.66 | 19788. | 55.00 |
| | | | | Pyritic 4.0- 5.0 | 2430. | 9833. | 3.86 | 1.55 | 2.31 | 34. | .00 | 203. | 49.00 | 704. | .00 | 278. | 51.00 |
| | | | | Pyritic 5.0+**** | 8303. | 35281. | 4.49 | 1.75 | 2.74 | 39. | .00 | 843. | 49.00 | 734. | .00 | 1224. | 51.00 |
| | | | | Quartzitic 4.0- 5.0 | 5526. | 20677. | 3.89 | 1.68 | 2.21 | 27. | .52 | 420. | 58.00 | 621. | 3.32 | 588. | 53.00 |
| | | | | Quartzitic 5.0+**** | 6354. | 23246. | 5.20 | 2.15 | 3.05 | 35. | .37 | 636. | 58.00 | 635. | 1.88 | 977. | 53.00 |
| 195 | 1129. | Vangord 18 | A | Rock | 160205. | 417686. | | | | | | | | | | | |
| | | | | Overburd | 468450. | 983745. | | | | | | | | | | | |
| | | | | Sulph Wast | 74737. | 253241. | | | | | | | | | | | |
| | | | | Baritic 5.0+**** | 68054. | 287623. | 8.17 | 3.55 | 4.62 | 49. | .67 | 16161. | 53.00 | 593. | 4.96 | 19058. | 55.00 |
| | | | | Pyritic 4.0- 5.0 | 3105. | 13025. | 4.12 | 1.62 | 2.50 | 37. | .00 | 284. | 49.00 | 737. | .00 | 405. | 51.00 |
| | | | | Pyritic 5.0+**** | 10733. | 45947. | 4.51 | 1.75 | 2.76 | 40. | .00 | 1098. | 49.00 | 743. | .00 | 1604. | 51.00 |
| | | | | Quartzitic 4.0- 5.0 | 10355. | 37816. | 3.88 | 1.80 | 2.08 | 29. | .56 | 835. | 58.00 | 622. | 3.33 | 1002. | 53.00 |
| | | | | Quartzitic 5.0+**** | 2637. | 9481. | 4.64 | 2.03 | 2.61 | 32. | .50 | 242. | 58.00 | 626. | 2.66 | 329. | 53.00 |
| 196 | 1124. | Vangord 19 | A | Rock | 202229. | 538251. | | | | | | | | | | | |
| | | | | Overburd | 386838. | 812360. | | | | | | | | | | | |
| | | | | Sulph Wast | 83507. | 278358. | | | | | | | | | | | |
| | | | | Baritic 5.0+**** | 36936. | 160728. | 9.21 | 3.89 | 5.32 | 62. | .72 | 10065. | 53.00 | 691. | 4.95 | 12533. | 55.00 |
| | | | | Pyritic 4.0- 5.0 | 405. | 1465. | 3.92 | 1.73 | 2.00 | 26. | .96 | 39. | 49.00 | 443. | 16.27 | 35. | 51.00 |
| | | | | Quartzitic 4.0- 5.0 | 13824. | 46684. | 3.84 | 1.57 | 2.27 | 28. | .51 | 874. | 58.00 | 685. | 3.45 | 1371. | 53.00 |
| | | | | Quartzitic 5.0+**** | 6624. | 21157. | 6.72 | 2.44 | 4.28 | 34. | .60 | 682. | 58.00 | 542. | 2.70 | 1321. | 53.00 |
| 197 | 1120. | Vangord 20 | A | Rock | 258136. | 686928. | | | | | | | | | | | |
| | | | | Overburd | 290151. | 609317. | | | | | | | | | | | |
| | | | | Sulph Wast | 70535. | 235584. | | | | | | | | | | | |
| | | | | Baritic 5.0+**** | 52020. | 222716. | 9.81 | 4.14 | 5.66 | 60. | .61 | 15009. | 53.00 | 628. | 3.87 | 18653. | 55.00 |
| | | | | Quartzitic 4.0- 5.0 | 6849. | 22162. | 3.92 | 1.48 | 2.44 | 26. | .48 | 385. | 58.00 | 694. | 3.46 | 708. | 53.00 |
| | | | | Quartzitic 5.0+**** | 18567. | 60528. | 6.18 | 2.41 | 3.78 | 29. | .34 | 1896. | 58.00 | 476. | 1.54 | 3251. | 53.00 |
| 198 | 1115. | Vangord 21 | A | Rock | 270934. | 721981. | | | | | | | | | | | |
| | | | | Overburd | 173763. | 364902. | | | | | | | | | | | |
| | | | | Sulph Wast | 65092. | 210268. | | | | | | | | | | | |
| | | | | Baritic 4.0- 5.0 | 338. | 1514. | 4.02 | 1.91 | 2.11 | 7. | .52 | 41. | 53.00 | 144. | 7.04 | 40. | 55.00 |
| | | | | Baritic 5.0+**** | 43920. | 189277. | 9.42 | 3.86 | 5.56 | 53. | .56 | 11764. | 53.00 | 593. | 3.79 | 15525. | 55.00 |
| | | | | Pyritic 4.0- 5.0 | 203. | 735. | 3.81 | 1.48 | 2.33 | 21. | .25 | 14. | 49.00 | 457. | 5.51 | 21. | 51.00 |
| | | | | Pyritic 5.0+**** | 7290. | 27071. | 10.21 | 4.98 | 5.23 | 71. | .16 | 2241. | 49.00 | 480. | 1.08 | 2000. | 51.00 |
| | | | | Quartzitic 4.0- 5.0 | 6807. | 27825. | 3.99 | 1.42 | 2.56 | 19. | .32 | 460. | 58.00 | 514. | 2.36 | 943. | 53.00 |
| | | | | Quartzitic 5.0+**** | 23360. | 74269. | 6.39 | 2.44 | 3.95 | 29. | .46 | 2382. | 58.00 | 463. | 2.07 | 4241. | 53.00 |

CURRAGH RESOURCES

Reserve Listing: VP 1-1 Mining Reserves

Page: 17

| Block | Bench | Block | Dig | Material | Grade | Volume | Tonnes | H E A D G R A D E | | | | C O N C E N T R A T E | | | | | | | |
|-------|-------|------------|-----|------------|----------|---------|---------|-------------------|------|------|---------|-----------------------|---------|-------|------|--------|--------|--------|-----|
| | | | | | | | | No. | Desc | Rate | 2Pb+2Zn | BCM | 2Pb+2Zn | 2Pb | 2Zn | Ag g/t | Au g/t | Pb tns | 2Pb |
| 199 | 1111. | Vangord 22 | A | Rock | | 268142. | 714917. | | | | | | | | | | | | |
| | | | | Overburd | | 88911. | 186713. | | | | | | | | | | | | |
| | | | | Sulph Mast | | 73494. | 232794. | | | | | | | | | | | | |
| | | | | Baritic | 5.0+**** | 45270. | 193735. | 9.90 | 4.18 | 5.72 | 55. | .47 | 13188. | 53.00 | 567. | 2.95 | 16406. | 55.00 | |
| | | | | Pyritic | 4.0- 5.0 | 608. | 2307. | 3.52 | 1.59 | 1.93 | 24. | .35 | 49. | 49.00 | 487. | 7.08 | 53. | 51.00 | |
| | | | | Quartzitic | 4.0- 5.0 | 12150. | 39702. | 3.87 | 1.60 | 2.27 | 25. | .46 | 760. | 58.00 | 602. | 3.10 | 1164. | 53.00 | |
| | | | | Quartzitic | 5.0+**** | 39393. | 121022. | 5.90 | 2.36 | 3.54 | 32. | .58 | 3728. | 58.00 | 526. | 2.64 | 6049. | 53.00 | |
| 200 | 1106. | Vangord 23 | A | Rock | | 268514. | 715234. | | | | | | | | | | | | |
| | | | | Overburd | | 17751. | 41476. | | | | | | | | | | | | |
| | | | | Sulph Mast | | 66468. | 208166. | | | | | | | | | | | | |
| | | | | Baritic | 5.0+**** | 49001. | 212001. | 10.10 | 4.25 | 5.85 | 57. | .55 | 14741. | 53.00 | 575. | 3.37 | 18422. | 55.00 | |
| | | | | Pyritic | 4.0- 5.0 | 1013. | 3964. | 3.62 | 1.71 | 1.91 | 24. | .35 | 92. | 49.00 | 458. | 6.72 | 90. | 51.00 | |
| | | | | Quartzitic | 4.0- 5.0 | 15179. | 48179. | 3.93 | 1.57 | 2.37 | 25. | .50 | 895. | 58.00 | 614. | 3.42 | 1484. | 53.00 | |
| | | | | Quartzitic | 5.0+**** | 48272. | 145340. | 5.82 | 2.40 | 3.42 | 32. | .59 | 4536. | 58.00 | 521. | 2.69 | 6946. | 53.00 | |
| 201 | 1102. | Vangord 24 | A | Rock | | 237073. | 630851. | | | | | | | | | | | | |
| | | | | Overburd | | 14409. | 30259. | | | | | | | | | | | | |
| | | | | Sulph Mast | | 74475. | 230551. | | | | | | | | | | | | |
| | | | | Baritic | 4.0- 5.0 | 810. | 3543. | 3.86 | 1.88 | 1.98 | 33. | 1.30 | 93. | 53.00 | 736. | 17.85 | 86. | 55.00 | |
| | | | | Baritic | 5.0+**** | 47768. | 210012. | 9.33 | 4.03 | 5.30 | 58. | .78 | 13706. | 53.00 | 618. | 5.07 | 16274. | 55.00 | |
| | | | | Quartzitic | 4.0- 5.0 | 12996. | 38550. | 3.93 | 1.52 | 2.41 | 22. | .41 | 690. | 58.00 | 571. | 2.87 | 1215. | 53.00 | |
| | | | | Quartzitic | 5.0+**** | 42728. | 126472. | 5.50 | 2.26 | 3.25 | 30. | .54 | 3661. | 58.00 | 526. | 2.60 | 5665. | 53.00 | |
| 202 | 1097. | Vangord 25 | A | Rock | | 216915. | 575567. | | | | | | | | | | | | |
| | | | | Overburd | | 6075. | 12758. | | | | | | | | | | | | |
| | | | | Sulph Mast | | 74532. | 236383. | | | | | | | | | | | | |
| | | | | Baritic | 5.0+**** | 54068. | 237495. | 9.25 | 4.09 | 5.16 | 58. | .70 | 15795. | 53.00 | 614. | 4.47 | 17816. | 55.00 | |
| | | | | Quartzitic | 4.0- 5.0 | 10112. | 29092. | 3.94 | 1.66 | 2.29 | 21. | .39 | 578. | 58.00 | 507. | 2.51 | 861. | 53.00 | |
| | | | | Quartzitic | 5.0+**** | 52814. | 97003. | 5.81 | 2.44 | 3.37 | 31. | .62 | 3085. | 58.00 | 509. | 2.75 | 4543. | 53.00 | |
| 203 | 1093. | Vangord 26 | A | Rock | | 196682. | 517901. | | | | | | | | | | | | |
| | | | | Sulph Mast | | 65731. | 210471. | | | | | | | | | | | | |
| | | | | Baritic | 5.0+**** | 69053. | 308925. | 10.37 | 4.73 | 5.64 | 69. | .64 | 24403. | 53.00 | 626. | 3.59 | 25704. | 55.00 | |
| | | | | Quartzitic | 4.0- 5.0 | 5747. | 16962. | 3.92 | 1.57 | 2.35 | 24. | .45 | 315. | 58.00 | 587. | 3.07 | 518. | 53.00 | |
| | | | | Quartzitic | 5.0+**** | 15998. | 46846. | 5.44 | 2.32 | 3.13 | 29. | .59 | 1397. | 58.00 | 503. | 2.75 | 2009. | 53.00 | |
| 204 | 1088. | Vangord 27 | A | Rock | | 167292. | 438655. | | | | | | | | | | | | |
| | | | | Sulph Mast | | 60548. | 203115. | | | | | | | | | | | | |
| | | | | Baritic | 5.0+**** | 64683. | 288971. | 10.61 | 4.85 | 5.76 | 69. | .68 | 23639. | 53.00 | 618. | 3.70 | 24708. | 55.00 | |
| | | | | Quartzitic | 4.0- 5.0 | 6885. | 20010. | 3.96 | 1.64 | 2.32 | 23. | .46 | 394. | 58.00 | 553. | 3.02 | 602. | 53.00 | |
| | | | | Quartzitic | 5.0+**** | 18270. | 55953. | 5.64 | 2.26 | 3.38 | 31. | .44 | 1617. | 58.00 | 553. | 2.11 | 2630. | 53.00 | |
| 205 | 1084. | Vangord 28 | A | Rock | | 163737. | 432877. | | | | | | | | | | | | |
| | | | | Sulph Mast | | 73310. | 257826. | | | | | | | | | | | | |
| | | | | Baritic | 4.0- 5.0 | 203. | 866. | 4.23 | 2.18 | 2.05 | 35. | .47 | 27. | 53.00 | 677. | 5.63 | 22. | 55.00 | |
| | | | | Baritic | 5.0+**** | 46436. | 205616. | 10.69 | 4.89 | 5.80 | 71. | .59 | 16938. | 53.00 | 629. | 3.18 | 17707. | 55.00 | |
| | | | | Pyritic | 4.0- 5.0 | 203. | 765. | 2.51 | 1.66 | 1.85 | 27. | .46 | 17. | 49.00 | 528. | 9.01 | 17. | 51.00 | |
| | | | | Quartzitic | 4.0- 5.0 | 5045. | 16410. | 3.99 | 1.76 | 2.23 | 23. | .49 | 353. | 58.00 | 510. | 3.01 | 471. | 53.00 | |
| | | | | Quartzitic | 5.0+**** | 2835. | 8408. | 5.32 | 2.18 | 3.14 | 31. | .53 | 233. | 58.00 | 559. | 2.64 | 361. | 53.00 | |

CURRAGH RESOURCES

Reserve Listing: VP 1-1 Mining Reserves

Page: 18

| Block Bench No. | Block Desc | Dig Material Rate | Grade TPb+Zn | Volume BCM | Tonnes | HEAD GRADE | | | | | CONCENTRATE | | | | |
|-----------------|--------------------|-------------------|---------------------|------------|---------|------------|------|------|--------|--------|-------------|-------|--------|--------|--------|
| | | | | | | TPb+Zn | TPb | ZZn | Ag g/t | Au g/t | Pb Tns | TPb | Ag g/t | Au g/t | ZZn |
| 206 | 1079. Vangord 29 A | Rock | | 109966. | 286118. | | | | | | | | | | |
| | | | | | 227021. | | | | | | | | | | |
| | | | Sulph Wast | | 67593. | | | | | | | | | | |
| | | | Baritic 5.0+**** | 53559. | 235081. | 9.47 | 4.09 | 5.37 | 60. | .62 | 15639. | 53.00 | 629. | 3.99 | 18516. |
| | | | Pyritic 4.0- 5.0 | 5265. | 19787. | 3.68 | 1.79 | 1.89 | 24. | .46 | 486. | 49.00 | 432. | 8.40 | 441. |
| | | | Quartzitic 4.0- 5.0 | 7902. | 24362. | 4.06 | 1.71 | 2.35 | 24. | .47 | 503. | 58.00 | 553. | 2.93 | 747. |
| 207 | 1075. Vangord 30 A | Rock | | 83505. | 209104. | | | | | | | | | | |
| | | | | | 180121. | | | | | | | | | | |
| | | | Sulph Wast | | 52761. | | | | | | | | | | |
| | | | Baritic 5.0+**** | 83993. | 367679. | 9.27 | 4.00 | 5.27 | 59. | .66 | 23808. | 53.00 | 635. | 4.32 | 28307. |
| | | | Pyritic 4.0- 5.0 | 6278. | 22917. | 3.78 | 2.41 | 1.37 | 26. | 1.04 | 806. | 49.00 | 350. | 14.28 | 353. |
| | | | Pyritic 5.0+**** | 810. | 2960. | 4.44 | 2.73 | 1.71 | 28. | .90 | 120. | 49.00 | 342. | 10.85 | 59. |
| 208 | 1070. Vangord 31 A | Rock | | 49407. | 114287. | | | | | | | | | | |
| | | | | | 165988. | | | | | | | | | | |
| | | | Sulph Wast | | 47399. | | | | | | | | | | |
| | | | Baritic 4.0- 5.0 | 203. | 828. | 3.59 | 1.72 | 1.87 | 29. | 1.13 | 19. | 53.00 | 703. | 17.04 | 19. |
| | | | Baritic 5.0+**** | 92984. | 407872. | 8.64 | 3.80 | 4.84 | 55. | .69 | 24837. | 53.00 | 626. | 4.72 | 28533. |
| | | | Pyritic 4.0- 5.0 | 5670. | 20389. | 3.86 | 2.24 | 1.62 | 25. | .87 | 658. | 49.00 | 365. | 12.84 | 380. |
| 209 | 1066. Vangord 32 A | Rock | | 25479. | 57076. | | | | | | | | | | |
| | | | | | 101652. | | | | | | | | | | |
| | | | Sulph Wast | | 28272. | | | | | | | | | | |
| | | | Baritic 5.0+**** | 52704. | 231189. | 8.49 | 3.76 | 4.72 | 55. | .65 | 13924. | 53.00 | 634. | 4.48 | 15687. |
| | | | Pyritic 4.0- 5.0 | 2201. | 8185. | 3.90 | 2.25 | 1.66 | 26. | .82 | 265. | 49.00 | 377. | 12.03 | 156. |
| | | | Pyritic 5.0+**** | 3618. | 15194. | 7.57 | 4.33 | 3.24 | 53. | .36 | 1109. | 49.00 | 403. | 2.61 | 644. |
| 210 | 1061. Vangord 33 A | Rock | | 8422. | 14976. | | | | | | | | | | |
| | | | | | 98378. | | | | | | | | | | |
| | | | Sulph Wast | | 27453. | | | | | | | | | | |
| | | | Baritic 4.0- 5.0 | 203. | 888. | 4.23 | 2.40 | 1.83 | 27. | 1.47 | 31. | 53.00 | 477. | 15.99 | 20. |
| | | | Baritic 5.0+**** | 35402. | 154679. | 7.95 | 3.55 | 4.40 | 52. | .69 | 5691. | 53.00 | 629. | 5.07 | 9668. |
| | | | Pyritic 4.0- 5.0 | 909. | 3388. | 3.73 | 1.89 | 1.84 | 30. | .62 | 89. | 49.00 | 511. | 10.82 | 73. |
| 211 | 1057. Vangord 34 A | Rock | | -1607. | -9483. | | | | | | | | | | |
| | | | | | 117828. | | | | | | | | | | |
| | | | Sulph Wast | | 53064. | | | | | | | | | | |
| | | | Baritic 5.0+**** | 26325. | 114478. | 7.77 | 3.49 | 4.29 | 50. | .72 | 6286. | 53.00 | 618. | 5.36 | 6956. |
| | | | Quartzitic 4.0- 5.0 | 1247. | 5006. | 3.83 | 1.79 | 2.04 | 31. | .54 | 109. | 58.00 | 686. | 5.06 | 129. |
| | | | Quartzitic 5.0+**** | 1454. | 5572. | 5.31 | 2.22 | 3.09 | 40. | .75 | 157. | 58.00 | 708. | 3.55 | 236. |

CURRAGH RESOURCES

Reserve Listing: VP 1-1 Mining Reserves

Page: 19

| Block | Bench | Block Desc | Dig Material | Grade | Volume | Tonnes | HEAD GRADE | | | | | CONCENTRATE | | | | |
|-------|-------|------------|--------------|----------|-----------|------------|------------|------|------|--------|--------|-------------|-------|--------|--------|---------|
| | | | | | | | 2Pb+2n | 2Pb | 2Zn | Ag g/t | Au g/t | Pb tns | 2Pb | Ag g/t | Au g/t | Zn tns |
| 0 | 0. | Grum 1 | Rock | | 14574390. | 39207270. | | | | | | | | | | |
| | | | Overburd | | 7175088. | 15067690. | | | | | | | | | | |
| | | | Sulph Wast | | 415898. | 1254449. | | | | | | | | | | |
| | | | Ore | 4.0- 5.0 | 382284. | 1244585. | 3.91 | 1.28 | 2.63 | 24. | .47 | 17255. | 60.00 | 898. | 8.90 | 43998. |
| | | | Ore | 5.0+**** | 2128392. | 7262196. | 7.25 | 2.59 | 4.66 | 43. | .66 | 239102. | 60.00 | 806. | 6.23 | 506335. |
| 0 | 0. | Grum 25 | Rock | | 17576960. | 47445630. | | | | | | | | | | |
| | | | Overburd | | 1321848. | 2776652. | | | | | | | | | | |
| | | | Sulph Wast | | 29791. | 99787. | | | | | | | | | | |
| | | | Ore | 4.0- 5.0 | 16920. | 69419. | 4.05 | 1.96 | 2.09 | 35. | .98 | 1618. | 60.00 | 856. | 12.30 | 1870. |
| | | | Ore | 5.0+**** | 49452. | 219664. | 11.21 | 4.85 | 6.36 | 77. | .94 | 15011. | 60.00 | 788. | 4.90 | 21880. |
| 0 | 0. | Grum 2M | Rock | | 4999196. | 13486850. | | | | | | | | | | |
| | | | Overburd | | 4498632. | 9447329. | | | | | | | | | | |
| | | | Sulph Wast | | 9075. | 29778. | | | | | | | | | | |
| | | | Ore | 4.0- 5.0 | 14352. | 47084. | 4.13 | 1.42 | 2.71 | 27. | .63 | 741. | 60.00 | 919. | 10.69 | 1720. |
| | | | Ore | 5.0+**** | 93468. | 355281. | 11.12 | 4.34 | 6.78 | 73. | .84 | 21711. | 60.00 | 842. | 4.70 | 38423. |
| 0 | 0. | Grum 2C | Rock | | 8797051. | 23499650. | | | | | | | | | | |
| | | | Overburd | | 12864. | 34733. | | | | | | | | | | |
| | | | Sulph Wast | | 555661. | 1704585. | | | | | | | | | | |
| | | | Ore | 4.0- 5.0 | 393912. | 1292265. | 3.87 | 1.54 | 2.33 | 26. | .64 | 22498. | 60.00 | 822. | 10.08 | 39645. |
| | | | Ore | 5.0+**** | 2219952. | 8352860. | 9.34 | 3.48 | 5.86 | 59. | .93 | 390987. | 60.00 | 836. | 6.55 | 764822. |
| 0 | 0. | Grum 3 | Rock | | 26480290. | 71302610. | | | | | | | | | | |
| | | | Overburd | | 2025788. | 4254012. | | | | | | | | | | |
| | | | Sulph Wast | | 326388. | 1050922. | | | | | | | | | | |
| | | | Ore | 4.0- 5.0 | 173304. | 599910. | 3.86 | 1.55 | 2.31 | 28. | .78 | 10531. | 60.00 | 870. | 12.25 | 18212. |
| | | | Ore | 5.0+**** | 1406856. | 5550125. | 8.80 | 3.34 | 5.46 | 58. | .94 | 245743. | 60.00 | 856. | 7.00 | 465848. |
| 0 | 0. | Grum | Rock | | 72427920. | 194942000. | | | | | | | | | | |
| | | | Overburd | | 15634220. | 31580410. | | | | | | | | | | |
| | | | Sulph Wast | | 1336812. | 4139521. | | | | | | | | | | |
| | | | Ore | 4.0- 5.0 | 980772. | 3253262. | 3.89 | 1.45 | 2.44 | 26. | .61 | 52643. | 60.00 | 859. | 10.20 | 105446. |
| | | | Ore | 5.0+**** | 5898120. | 21740130. | 8.55 | 3.18 | 5.38 | 54. | .84 | 912555. | 60.00 | 833. | 6.51 | ***** |
| 0 | 0. | Vangord | Rock | | 3264514. | 8593620. | | | | | | | | | | |
| | | | Overburd | | 4318749. | 9069372. | | | | | | | | | | |
| | | | Sulph Wast | | 1145866. | 3824375. | | | | | | | | | | |
| | | | Baritic | 4.0- 5.0 | 3173. | 13181. | 3.95 | 1.92 | 2.02 | 29. | .88 | 356. | 53.00 | 636. | 11.83 | 329. |
| | | | Baritic | 5.0+**** | 1095570. | 4750546. | 9.08 | 3.96 | 5.12 | 57. | .63 | 304904. | 53.00 | 617. | 4.16 | 354034. |
| | | | Pyritic | 4.0- 5.0 | 36590. | 138091. | 3.84 | 1.92 | 1.92 | 28. | .50 | 3705. | 49.00 | 466. | 8.71 | 3157. |
| | | | Pyritic | 5.0+**** | 39866. | 163821. | 7.39 | 3.94 | 3.44 | 57. | .20 | 10748. | 49.00 | 467. | 1.70 | 7493. |
| | | | Quartzitic | 4.0- 5.0 | 147024. | 482793. | 3.92 | 1.61 | 2.31 | 25. | .43 | 9284. | 58.00 | 602. | 3.15 | 14450. |
| | | | Quartzitic | 5.0+**** | 293999. | 910336. | 5.69 | 2.36 | 3.38 | 31. | .53 | 27085. | 58.00 | 527. | 2.50 | 42978. |

APPENDIX F
ENVIRONMENTAL

DRAFT

ENVIRONMENTAL ISSUES

Regulatory

- 1.) Prior to production a surface lease will be obtained from the Department of Indian and Northern Affairs to ensure control and exclusive use of the mine sites. In order to obtain the surface lease the project will be subjected to a comprehensive environmental impact review. This will include the impact on the water resources (ground water and surface water) fisheries resource, wildlife, land use conflicts and air quality.
- 2.) The diversion, impoundment and use of more than 10,000 gallons per day of water require the issuance of a water licence from the Yukon Territory Water Board. The major water licence issue is the diversion of Vangorda Creek around the Vangorda Pit. It is anticipated that the regulatory agencies will be concerned about its impact on water quality. (suspended solids metal content and ph) and the fisheries resources as well as its ability to handle flood flows.

Design Considerations

- 1.) Generally, water supply and tailings disposal are the major environmental concerns. However, these have already been

addressed as the existing facilities at the Faro mine site will be used for the entire Vangorda Plateau development.

- 2.) Location and design of waste dumps should consider the impact on the water resources of the area. Regulatory agencies will be concerned about the potential blockage or diversion of streams, and the potential for acid leaching from sulphide waste rock and from ore stockpiles. In addition, no waste dumps should be placed on top of potential ore reserves.
- 3.) The diversion of the Vangorda Creek should address the impact on downstream water quality by ensuring that it has the ability to control spring flood conditions and does not expose the water flows to sulphide ore or sulphide waste rock.

Existing Information

Kerr-AEX carried out extensive baseline environmental data gathering and impact assessment work during the period 1975-77. The major concerns at that time were water supply and tailings disposal which are not concerns for this project.

Kerr-AEX did not find any other significant fish and wildlife resources or potential land use conflicts.

Additional Study Required

Three years of baseline environmental data has already been gathered

(1975-77) and the major environmental concerns have been identified by Kerr-AEX. However, it will be necessary to update the baseline data and address the new issues of: the Vangorda Creek diversion, new location of the waste dumps, and the drainage of Doal Lake.

A field program is proposed for the summers of 1987-88.