GEOLOGY

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GEOLOGY, MINERAL INVENTORY

and RESERVES of the

GRUM DEPOSIT - Yukon

CURRAGH INC.
Whitehorse, Yukon

Report WH9305

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EXECUTIVE SUMMARY

Geological interpretation and modelling done from 1991-1992 has formed the basis for a new mineral inventory and minable reserve for the Grum deposit. The new work takes account of all drilling on the deposit, a grand total of 85,870m in 539 holes. From 1987-1991 Curragh has drilled 12,527m in 113 holes, to better define shallow ore in the early phases of the pit, and obtain samples for metallurgical test work.

The Grum deposit is a complexly folded and faulted, stratiform, sediment-hosted Zn-Pb-Ag-Au bearing sulphide deposit. Approximately one third of the ore is fine-grained, high grade galena and sphalerite-bearing, massive pyritic sulphides, generally with some barite. The remaining ore consists of disseminated pyrite sphalerite and galena in a quartzite host. A considerable portion of the disseminated ore is carbon bearing.

The deposit is being prepared for production by open pit methods. It will be the third deposit mined in the Anvil Range, after Faro (commenced 1969) and Vangorda (commenced 1990). To date 24 million tonnes of overburden and waste rock have been removed from the pit area from 1989-1992.

The mineral inventory for the deposit, at a 3% cutoff based on the new work, is:

		Zinc %	Lead %	Silver g/t	Gold g/t
Champ Zone 52W-61W	3,283,840	2.90	2.32	35.6	0.62
Central Portion 61W-87W	43,240,250	4.66	2.94	49.1	0.82
TOTAL	**46,524,090**	**4.54**	**2.90**	**48.2**	**0.81**

A further five (5) million tonnes may exist northwest of Section 87W.

Minable open pit reserves (fully diluted) total 24,764,220 tonnes averaging 4.54% Zn, 2.73% Pb, 46 g/t Ag and 0.70 g/t Au. Four (4) million tonnes or more of high grade in-situ mineralization is known to exist below the pit and it is likely that further drilling will eventually define additional reserves there and to the northwest of the pit. This material could be accessed by a ramp from the pit wall as was done at Faro.

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INTRODUCTION

The Grum deposit is the third lead-zinc-silver deposit to be developed in the Anvil Range, Yukon. It is 12 km southeast of the Faro mine and mill, and 2 km northwest of the Vangorda Pit (Figure 1). The Grum deposit is a multi-horizon, complexly folded and faulted, stratiform, syn-sedimentary sulphide deposit of lower Palaeozoic age. It is the second largest deposit of the Anvil District after Faro, but is followed closely by Dy. This report briefly describes the geology and mineral inventory for the Grum deposit.

HISTORY

The first discovery of mineralization in the immediate Grum area was in the early 1950's, during the delineation of the Vangorda deposit, the initial discovery in the district. The Champ and Firth showings were discovered at that time. Early drill testing of both these showings was not encouraging and little further work was done.

The Grum deposit was discovered in 1973 by AEX Minerals, in joint venture with Kerr Addison Mines. Discovery resulted from a drill test of a re-interpreted gravity anomaly in an area down fold plunge from the Vangorda deposit, in line with both the Champ and Firth showings, and the Faro deposit to the northwest.

Surface diamond drilling in 1973 and 1974 indicated a significant deposit. In 1975 and 1976 Kerr Addison Mines carried out an underground diamond drilling and sampling programme consisting of fan drilling on 61 metre sections, with some intermediate holes drilled to define the high grade core of the deposit. At the same time, further surface drilling was carried out to further define near surface horizons.

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, resampled, and reassayed most Kerr Addison drillholes.

A Curragh Inc. predecessor, Curragh Resources Corporation, acquired the assets of Cyprus Anvil Mining Corporation in late 1985 and reopened the Faro mine and milling complex in early 1986. Plans to develop the Grum and Vangorda pits in the area known as the Vangorda Plateau, were formalized in 1987. Additional surface diamond drilling programmes were carried out in 1987, 1988, 1989 and 1991, to further delineate reserves in the early phases of Grum pit and to provide samples for metallurgical testing. Several drillholes in the 1989 programme were targeted to test the lower grade "Champ" horizon in the southwest portion of the deposit. This material constituted the original Champ showing which is now known to be an upper horizon of the Grum structure.

Site preparation in the Grum pit area began in 1988. Pre-production stripping has been intermittently carried out, beginning in 1989. To date a total of 24 million tonnes of glacial till and waste rock, and 52,000 tonnes of ore, have been removed (Table 1). In 1992 a short mill test was carried out with some of the ore mined.

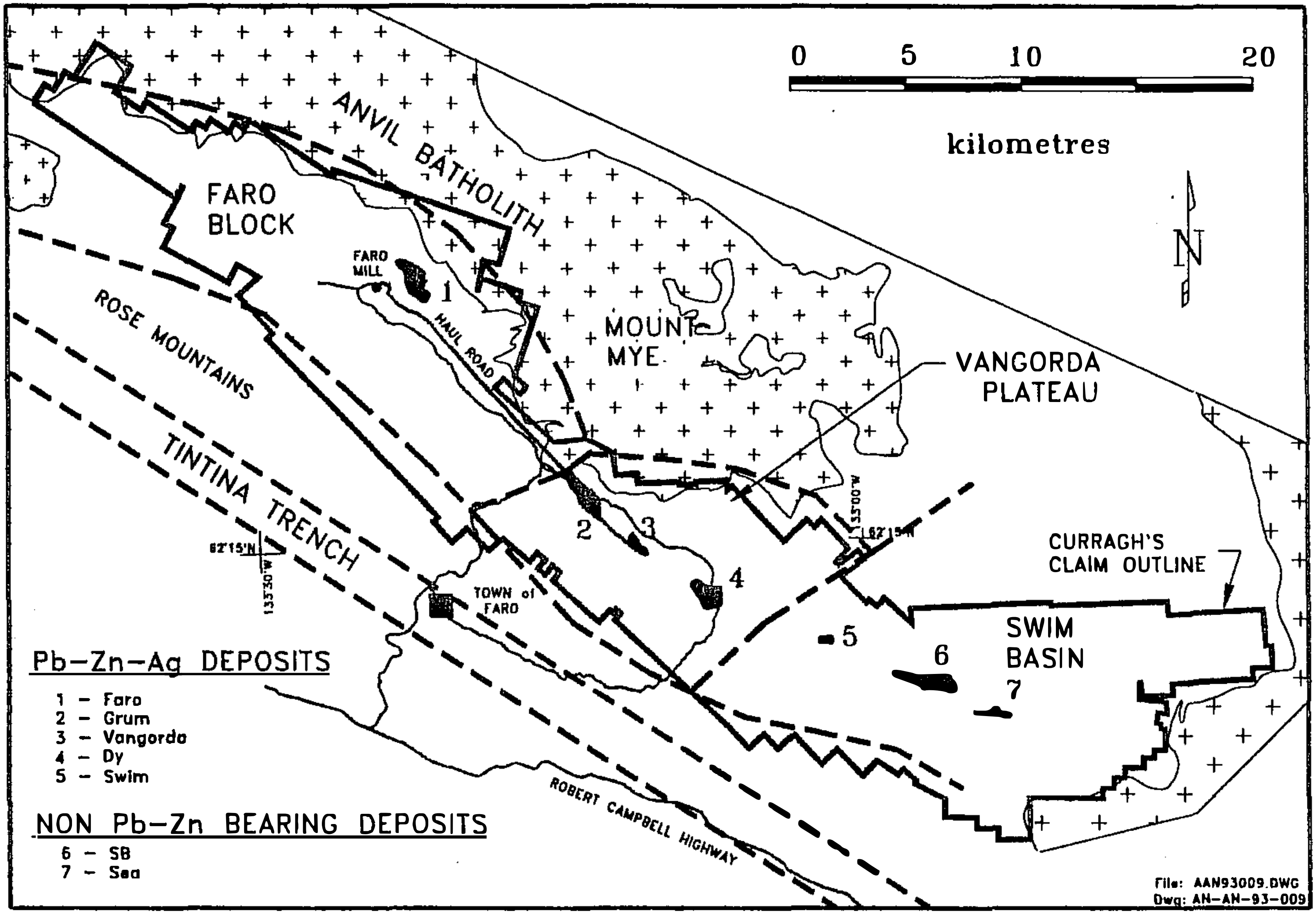


Figure 1. Location of the Grum Deposit Within the Anvil District

Table 1. Summary of Mining in Grum Pit		
Year	**Waste (tonnes)**	**Ore (tonnes)**
1992	16,097,498	52,033
1991	363,984	-
1990	4,755,356	-
1989	2,977,986	-
1988	-	-
TOTAL	**24,194,824**	**52,033**

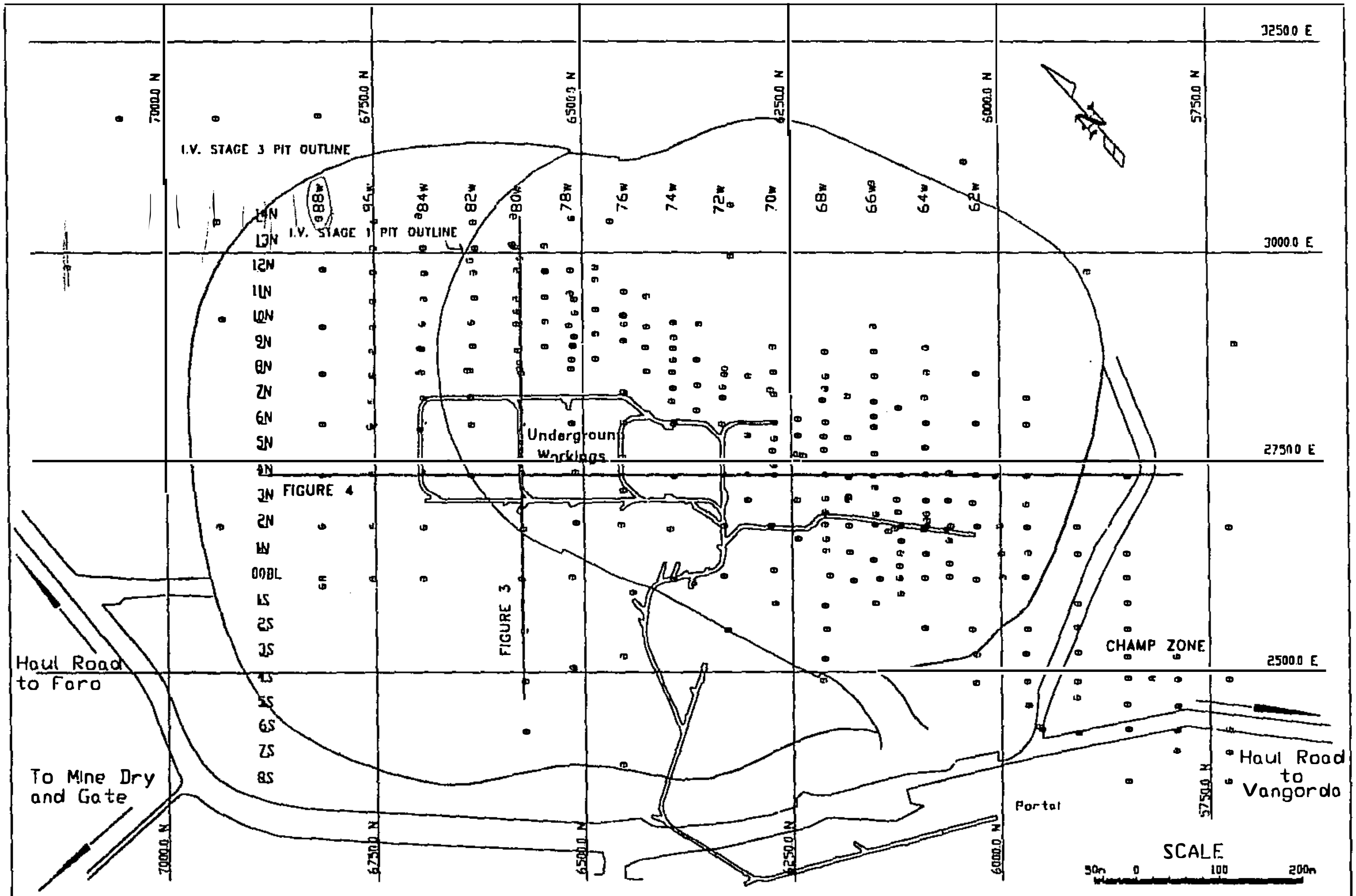
INFORMATION BASE

Since the deposit's discovery in 1973, a total of 314 surface diamond drillholes (Figure 2), and 218 underground diamond drillholes, totalling 85,870 metres of drillcore, have been drilled in the Grum deposit. The drill holes are listed by year in Table 2. Between sections 62W and 86W the deposit has also been explored by 19,000m of underground drilling in fans from a pair of parallel inclines following the deposit trend (Figure 2). The strike length of the deposit examined from underground is 700m. Underground workings, now flooded, total 2,900m. The drill fans are most complete on even numbered sections (i.e. spaced approximately 61m apart); on the odd numbered sections some fill-in drilling has also been done from underground and locally on surface (Figure 2).

In the southeast part of the deposit additional surface fill-in drilling was done by Cyprus Anvil in 1980-1982 to more closely define shallow ore for early production. Curragh's fill-in drilling was mainly in the northwest part of the Phase I Pit, with the same objective.

These holes have yielded 12,099 samples, all assayed for lead, zinc and silver. Many of the samples were also assayed for gold, total and soluble iron, copper, barium, and pulp SG. All early holes were relogged to a common standard by Cyprus Anvil in 1983. The Curragh logging has conformed to that standard, however, in 1991 a new lithologic code was used (Appendix B).

The mineralization of the Grum deposit is known to extend from Section 51W in the Champ Zone to Section 128W at the Firth showing. Only the portion from 51W to 86W has been drilled in any detail. The portion from 62W to 84W has been examined in great detail by both surface and underground drilling. The deposit was regularly but sparsely drilled on Section 88W. Between 88W and 128W there are relatively few irregularly spaced holes. Many of these holes are too short to test the Grum



LEGEND	REVISIONS	Curragh Inc.	Figure: 2
▫ DRILL HOLE COLLAR	16/01/91 31/05/93	GRUM DEPOSIT	Date 02/12/91 DRAWN BY: CVR
		SURFACE DIAMOND DRILLING	Drawing No. N-GR-93-001 File: AGR93001.DWG

structure, but those that have been drilled deep enough show that mineralization continues beyond 88W, probably all the way to the Firth showing. The following areas will be referred to in this paper:

- The Champ Zone (51W to 60W) in the southeast part of the deposit;

- The central portion 61W to 87W, and

- the Northwest Extension, which is northwest of Section 87W.

Table 2. Summary of Drilling History of the Grum Deposit

	No. of Holes	Total Metres
CAMC exploration holes 75-76	4	1,703
Kerr Addison Surface holes 73-77	155	41,040
Kerr Addison Underground holes 76-77	218	19,041
Sub-total discovery to 1979 sale to CAMC	377	61,783
CAMC surface holes 79-82 after purchase	46	10,662
CAMC exploration holes 79-82 after purchase	3	897
Sub-total CAMC after purchase from K-A	49	11,559
Curragh surface holes 1987	20	2,155
Curragh surface holes 1988	5	540
Curragh surface holes 1989	35	5,025
Curragh surface holes 1991	53	4,808
Sub-total holes drilled by Curragh	113	12,527
Grand Total	**539**	**85,870**

Previous work on the Grum deposit includes mineral inventory estimates by Sirola (1977) and Simpson and Adamson (1983) as well as various undocumented preliminary block models by Cyprus Anvil. Curragh completed its first block model in 1986 (Curragh Resources Inc., 1987). The results of the Kerr Addison underground programme are described in Paxton and Po (1977, and Sirola (opcit) Carson (1977) presented a mineralogical and textural investigation of the ores based on Kerr Addison's underground samplng. Jennings and Jilson (1986) and Pigage (1990) present brief descriptions of the deposit, and Modene (1982) describes a sulphur isotopic study of Grum.

GENERAL GEOLOGY

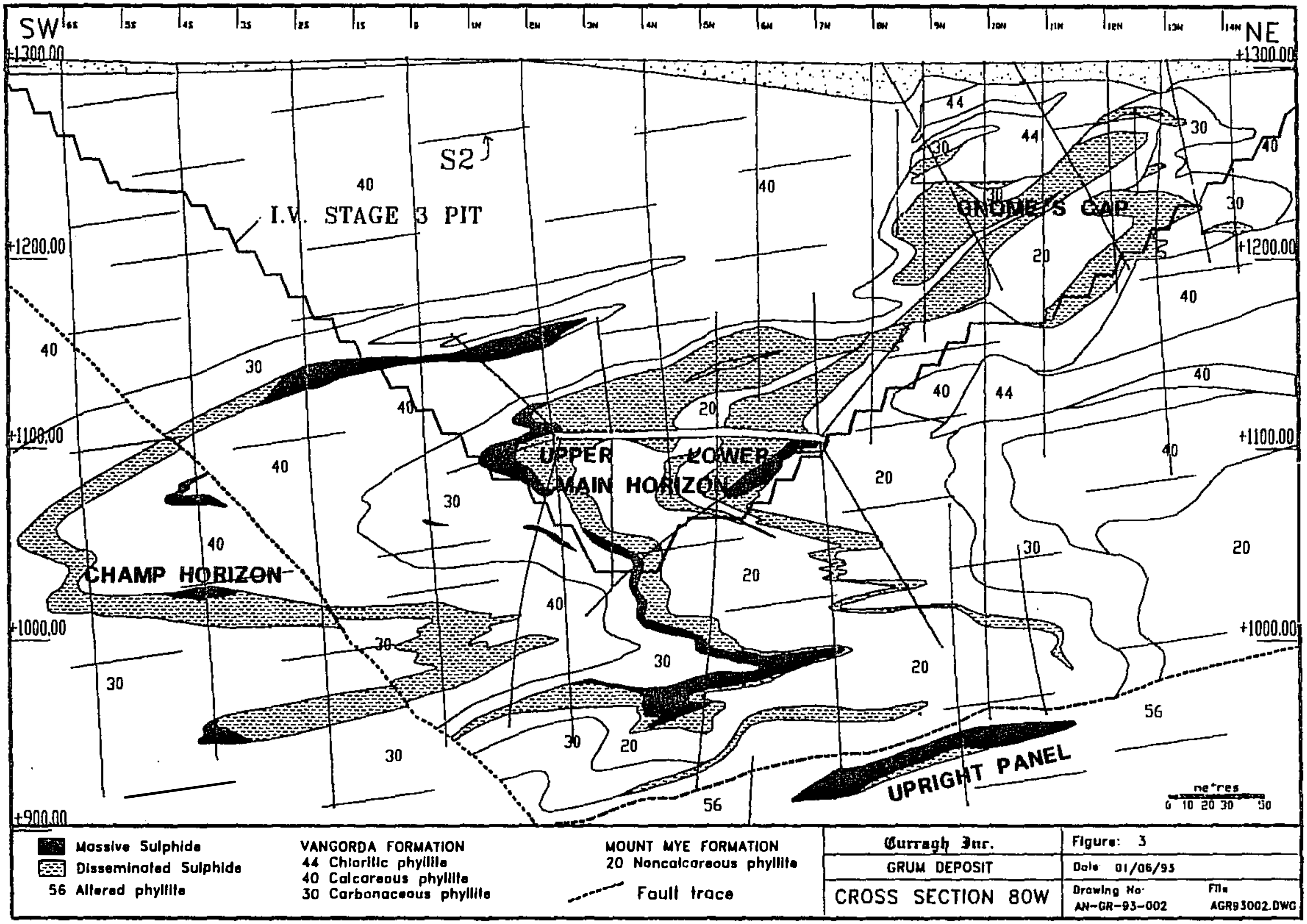
Stratigraphy and Lithology of the Grum Area

The Grum deposit is a stratiform, sediment hosted, sulphide deposit consisting of three or more distinct, highly contorted horizons of massive and disseminated pyritic sulphides. The sulphide horizons are each laterally extensive sheets parallel to original bedding, and are hosted by meta-sedimentary phyllites. The horizons are stacked above one another within a 150m thick stratigraphic interval at the transitional contact of the Vangorda and Mount Mye formations (Units 20 and 40 respectively on Figures 3 and 4).

At Grum, the Vangorda formation consists of soft, medium to light greenish grey, highly fissile, calcareous phyllite interlayered with grey non-calcareous phyllite, black carbonaceous phyllite and minor pale green chloritic phyllite. Locally the chlorite phyllite is altered to beige to cream-coloured muscovite dolomite quartz $\pm$ fuchsite (?) phyllite. The blocky, massive greenstones that typify the Vangorda formation elsewhere are absent within the Grum pit. The basal carbonaceous member of the formation is well developed at Grum (Unit 30 on Figure 3). It thickens across the deposit from about 10m in the northeast to as much as 80 or 100m southwest of the deposit. On a regional scale the sulphide horizons appear to be spatially associated with the pinchout of this unit. Immediately above the main ore horizons the carbonaceous rocks are soft, highly sheared and gouged, but to the southwest, down dip from the deposit, they are moderately hard, highly fractured, black, siliceous phyllites.

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

The most important ore horizon occurs immediately beneath the basal carbonaceous member of the Vangorda formation and has been named "**main horizon**" (Figure 3). The main horizon consists of both an upper, upright, and lower, overturned, limb of a large scale phase one northeast verging (that is, shaped like the letter 'Z' or 'N' if the viewer is facing northwest) anticline. There is another, less important, low grade ore horizon named "**Champ horizon**" of dominantly disseminated sulphide lithfacies. The Champ horizon occurs above the main horizon and is generally thin except where it thickens dramatically in highly attenuated phase two fold hinge zones. This horizon constitutes the Champ Zone where it is close to surface at the southeast, up-plunge, end of the Grum fold structure. A third important horizon, particularly in the lower phases of the Grum pit, is the "**upright panel horizon**", so named because it is stratigraphically and structurally upright. The upright panel appears to occur entirely within upper Mount Mye formation. This horizon shares many similar characteristics of the major horizon of the Vangorda deposit. Its similarities include the presence of a well developed, essentially Pb-Zn barren but Cu-Au bearing semi-massive sulphide base grading abruptly up into high grade massive


SW
NE
+1300.00
+1200.00
+1100.00
+1000.00
+900.00
S2
I.V. STAGE 3 PIT
GNOME'S CAP
UPPER
LOWER
MAIN HORIZON
CHAMP HORIZON
UPRIGHT PANEL
metres
0 10 20 30 50
Massive Sulphide
Disseminated Sulphide
56 Altered phyllite
VANGORDA FORMATION
44 Chloritic phyllite
40 Calcareous phyllite
30 Carbonaceous phyllite
MOUNT MYE FORMATION
20 Noncalcareous phyllite
Fault trace
Curragh Inc.
GRUM DEPOSIT
CROSS SECTION 80W
Figure: 3
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Drawing No: AN-GR-93-002
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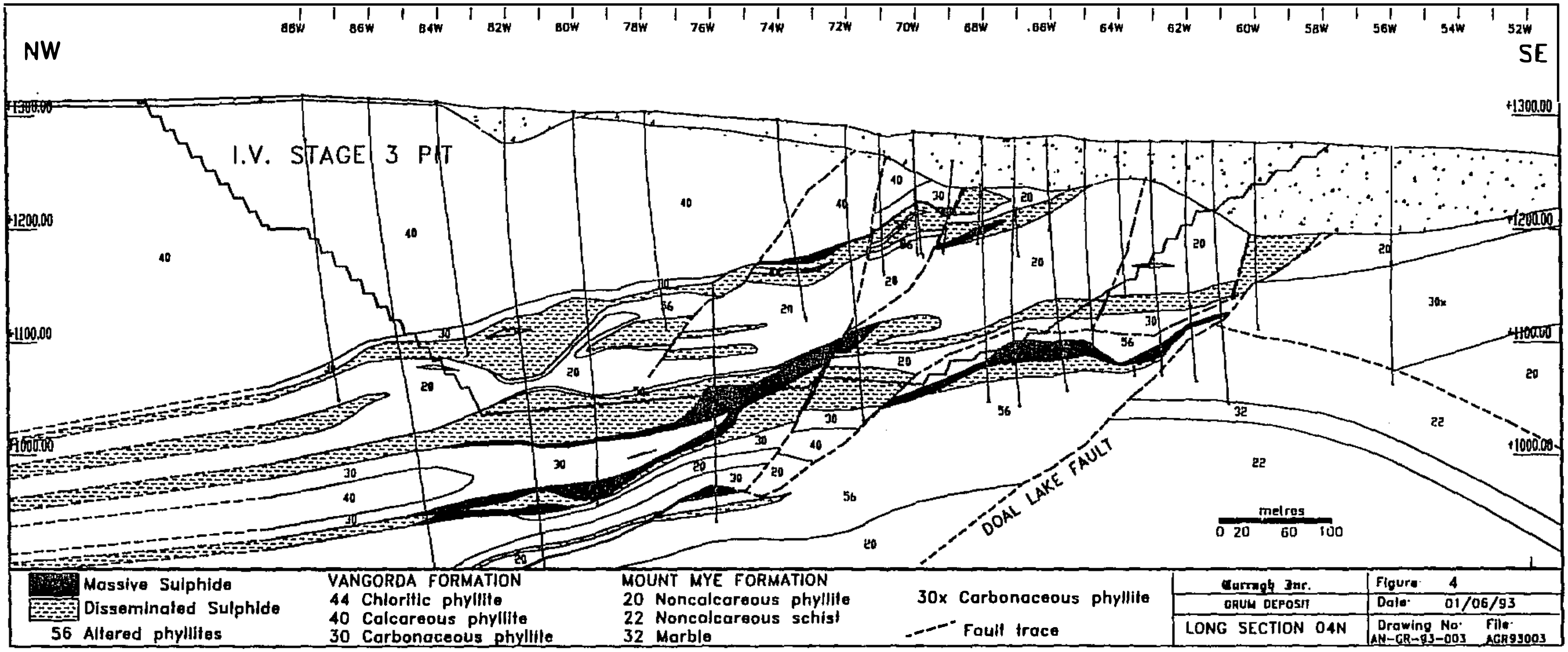
NW
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I.V. STAGE 3 PIT
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metres
0 20 60 100
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+1000.00
Massive Sulphide
Disseminated Sulphide
56 Altered phyllites
VANGORDA FORMATION
44 Chloritic phyllite
40 Calcareous phyllite
30 Carbonaceous phyllite
MOUNT MYE FORMATION
20 Noncalcareous phyllite
22 Noncalcareous schist
32 Marble
30x Carbonaceous phyllite
Fault trace
Curragh Inc.
GRUM DEPOSIT
LONG SECTION 04N
Figure 4
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pyritic/barytic sulphides at the top of the horizon. The upright panel is distinctly different from the other sulphide horizons of Grum in that it has extensive, well developed alteration, (a "bleached" muscovite-chlorite phyllite) below the semi-massive base. The intensity of alteration is similar to the alteration developed beneath the major ore horizon at Vangorda, and is unusually intense for Grum. The upright panel horizon is assumed to correlate structurally and stratigraphically with the Vangorda deposit. A sulphide horizon at the base of the basal carbonaceous member of the Vangorda formation, like the main horizon at Grum, appears to be present at Vangorda but is of insignificant grade and thickness.

There are several other less important, thin, intermediate horizons within the Mount Mye formation which generally are not continuous over significant strike intervals. These may, or may not, be stratiform occurrences.

There are no significant post metamorphic dykes at Grum. The Cretaceous 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, except for the regional scale metamorphism, recrystallization and deformation that accompanied batholith emplacement. Conversely, rocks of the Anvil Batholith do not appear to have been affected by ore formation, as expected, since they are significantly younger than the ores.

Structure

The ore layers at Grum are contorted into a complex, shallowly northwest plunging, polyphase fold structure. The prominent 'S' shaped folds (looking northwest - Figure 3), are second phase structures. They are superimposed on a large 'Z' shaped first phase anticline-syncline pair (Figure 3 -the area labelled "Gnome's Cap" is the crest of the first phase anticline). 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 west or southwest (Figure 3). 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 (Figure 4 is a section parallel to the 315° trend/11° plunge).

There are several important faults at Grum. The largest displacements occur on moderately (35° to 45°) dipping structures that truncate the deposit at both its northwest (Tie Fault) and southeast ends (Figure 4 - Doal Lake Fault). Neither of these structures would be exposed in an open pit but smaller related, subparallel faults will likely be exposed on the pit walls. A series of subparallel 060° to 080° trending faults dipping steeply to the northwest have resulted in apparent dip slip displacements ranging from 5 to 30 metres (Figure 4). Movement along these faults appears to have been oblique slip with a significant horizontal component of displacement. In the Champ Zone, faults of this set show left lateral displacement. Underground mapping has located a myriad of smaller faults and joint systems of similar orientation. Diamond drilling by itself cannot resolve displacements on these smaller fault sets and will not be suitable for predicting short term ore release. Diligent on-going pit mapping, and a short term geologic model as used at Vangorda, will be required to "fine tune" the geological interpretation to minimize surprises in short term ore production.

Predating the 060° trending faults is a steeply southwest dipping fault which cuts the high grade massive sulphide at the top of the main horizon and downdrops it to the southwest. This fault is important because it has resulted in unusual microtextures in sulphides due to high strain along the fault zone. A bulk sample collected underground by Kerr Addison Mines, within this fault zone, contained unusually fine-grained intergrowth textures and proved to be metallurgically difficult. Oxidation has penetrated more deeply along this fault and affected the adjacent sulphides and bulk samples; other samples taken by Kerr-Addison away from such faults showed acceptable metallurgy (Carson, 1977).

A poorly understood major shallow southwest dipping fault is interpreted above the upright panel. It is oriented nearly parallel to S_2. There is no physical evidence of the fault in drillcore. It is interpreted based on structural and lithological inconsistencies across it and is thought to be a syn-metamorphic slide, similar to those recognized at Vangorda. This low angle structure is not expected to show geotechnical behaviour any different than the phyllites.

Metamorphism in the Grum deposit area is uniformly in greenschist facies, and the structural style is typical for the Vangorda Plateau. Some of the deeper holes in the area intersected amphibolite facies schists, but these are invariably beneath extensional faults which telescope metamorphic facies. The Grum deposit sulphides have only been found in the hanging wall (lower metamorphic grade) blocks of these faults.

Surficial Geology of the Grum Area

The subcrop of the ore deposit is covered by up to 70m of clay and silt rich till and better sorted glacio-fluvial silts, sands and gravels (Figure 4). These unconsolidated sediments are water saturated and may contain pockets of permafrost. The northeast wall of the Grum pit must contend with thick sections of these unconsolidated sediments. Dewatering in advance of stripping may help increase stability substantially, as well as simplify operations in the pit. Several wells have been drilled to achieve this dewatering.

DEPOSIT GEOLOGY AND ORE TYPES

The Grum deposit is an assemblage of highly folded and faulted sulphide layers. As with other deposits in the Anvil Range a reconstructed (unfolded) pre-deformation ore horizon at Grum tends to have a massive sulphide upper and central portion and a quartzose, disseminated sulphide lower and peripheral portion. The lowest and most distal disseminated sulphide bearing quartzite tends to be more carbonaceous. The horizons can be up to 30m thick but are mostly 15m or less thick. Grade is strongly controlled by rock type. Massive, particularly baritic, sulphides, tend to be high grade, whereas disseminated ores tend to be lower grade (Figure 5). Because of this grade distribution, the tops of the horizons tend to be high grade and the bottoms lower grade (except of course where the horizons are overturned). The sulphide horizons are separated by significant thicknesses of barren phyllite. Interfaces between ore and waste tend to be sharp at the stratigraphic hanging wall contact against barren phyllite but gradational both at the footwall and laterally against sulphide waste.

The Grum deposit was discovered fairly late in the history of the district, largely because it is a completely blind deposit; there is no ore outcrop. The subcrop of the ore structure is buried beneath thick glacial overburden and the remainder of the structure is obscured by phyllite.

Ore Types

Disseminated Carbonaceous Pyritic Quartzite (Unit 2)

Unit 2 is dark grey to black, moderately hard to very hard, well banded, sulphide bearing, carbonaceous, locally micaceous quartzite. Compositional bands usually range from 1mm to 2cm thick. They are dark grey to black, very fine grained, locally micaceous quartzite interbanded with coarser grained, mottled light grey to brassy yellow (to locally red-brown) quartz-sulphide bands. Pyrite is usually the dominant sulphide species with lesser sphalerite and galena. Locally, light reddish-brown sphalerite is dominant. In some places pyrrhotite, rather than pyrite, is dominant, but pyrrhotite is only a minor constituent overall in this deposit. Carbon content is normally within the ¼ to ½% range, and generally occurs in thin coatings concentrated on cleavage surfaces (S_1 and S_2 surfaces). Chalcopyrite occurs locally in traces as small blebs within sulphide bands and fractures.

Total sulphide content varies from 10% to 30% and may locally range up to 60% (the upper boundary into pyritic massive sulphides of Unit 5).

Unit 2 rock types are more abundant than any other ore facies at Grum and constitute 53% of the millfeed in the I.V. Stage 1 Pit and 40% of the I.V. Ultimate Pit millfeed.

Disseminated Non-carbonaceous Pyritic Quartzite (Unit 3 and 4)

Unit 3 is light to medium grey, moderately hard to locally very hard, usually well banded generally well foliated, micaceous, pyritic quartzite. The unit is texturally and mineralogically similar to Unit 2, except that carbon is less abundant and it is, thus, light coloured. Banding is commonly less well developed and sulphide bands in the high grade ore are characteristically redder in colour and contain less pyrite that Unit 2. Unit 3 feed grade is slightly higher in lead and zinc, and gold content is slightly elevated over Unit 2. At Grum, contacts with Unit 2 are

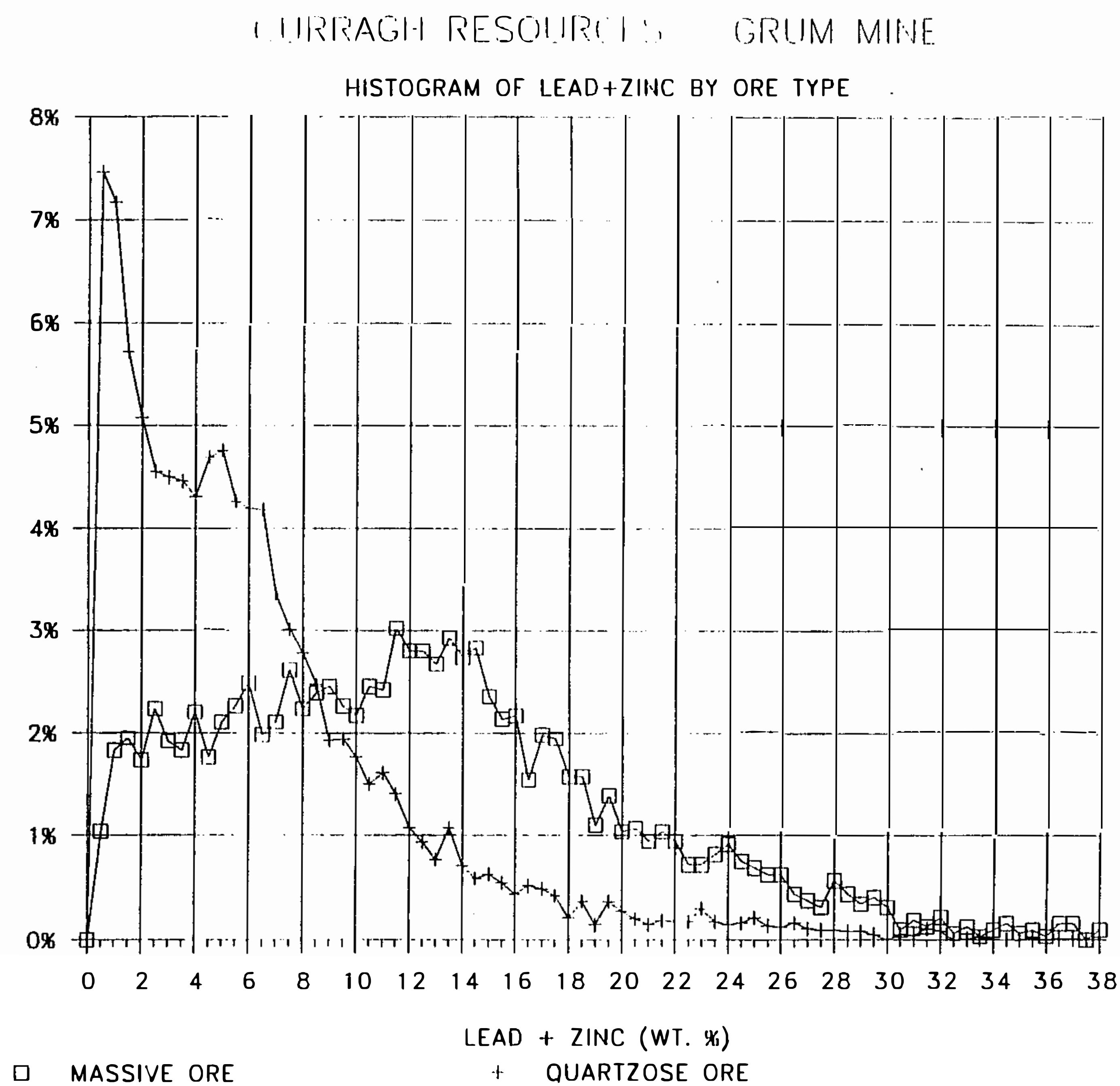

Figure 5 Histogram of Lead and Zinc Grades for Massive Sulphides and Disseminated Sulphide in Quartzite at Grum, based on Raw Core Assays

commonly gradational over a few feet making a "clean" separation of these ore types at a mining scale difficult.

Unit 3 rock represents 14% of the millfeed from both the I.V Stage 1 and the I.V. Ultimate Pit.

Unit 4 is a low grade, very pyritic variant of Unit 3 with contained lead + zinc less than 4%.

Massive Sulphide (Unit 5 and 7)

The dominant rock type of the massive sulphides is massive pyritic sulphide (Unit 5) which grades into massive sulphides with up to 10-30% barite (Unit 7). Massive sulphides consist of banded to homogeneous, usually weakly foliated, fine grained massive pyrite +/- barite with lesser sphalerite and galena. Total sulphide +/- sulphate content is by definition at least 60%, almost always greater than 80% and commonly near 100%. Gangue consists of quartz, carbonates (calcite, dolomite, ankerite, siderite)and some muscovite and chlorite. Accessory minerals include pyrrhotite, magnetite, chalcopyrite, very minor arsenopyrite and marcasite.

Ductile flow along the margins of the massive sulphides has caused grain size reduction microtextures which have not been annealed by continued heating. These textures result in fine grained ore with complex intergrowths which account for the galena in sphalerite and galena + sphalerite in pyrite middlings that characterize some Grum ore metallurgically (Carson, 1977). Ductile flow has also led to the widespread occurrence of sulphide clast in sulphide matrix breccias in the massive sulphides. Typically, less pyritic or more quartz-bearing lithologies occur as fragments in a more lead-zinc sulphide rich matrix. The clasts commonly show some matching margins along which the clasts can be fit back together indicating these are flow breccias, rather than primary slump or explosion breccias. Ductile flow causes abrupt local thickness change, particularly of massive sulphides in the lower second phase fold hinge. In extreme cases of ductile flow massive sulphides have been observed to intrude phyllites (Paxton and Po, 1977)

Near fault zones, or near the bedrock surface massive sulphides, are slightly oxidized and porous or vuggy. In extreme cases the sulphides become disaggregated sulphide sand. As noted below, some allowance has been made for potential metallurgical impact of this material within 6m of the bedrock surface.

Massive sulphide ore types are the highest grade ore type at Grum and represent 33% of the millfeed from the I.V. Stage 1 Pit and 37% of the millfeed from the I.V. Stage 3 Pit.

Comparison of Grum to Faro and Vangorda

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 respective deposits have reached. The most outstanding difference between Grum, along with all other Vangorda Plateau deposits, and Faro, is the form of the deposit. The Vangorda Plateau deposits consist of several distinct, highly contorted horizons separated by barren phyllite waste. Faro, on the other hand, is essentially one thick horizon in overall outline with lesser internal phyllitic waste. This has been interpreted to imply that dilution could be significantly higher at Grum than at Faro. Faro, however, contained considerable internal sulphide waste, thus its dilution was much higher and grade control much more challenging than might appear at first glance. It is, nonetheless, inescapable that Grum has more varied, and potentially more complex, mining situations than Faro, which will add to the challenge of dilution control. On the positive side, the dilutant at Grum will be more commonly easily identifiable phyllite, rather than the lower grade sulphides as at Faro. Experience at Faro shows that phyllite dilution is much easier to control than low grade sulphides. The Vangorda pit has provided an opportunity to work with problems similar to those posed by Grum, and as noted below, methods have already been developed to deal successfully with selectivity in the pit.

The next most obvious difference between Faro and Grum is that the Grum massive sulphides have a finer grain size and more complex mineral intergrowth, necessitating finer grinding than Faro ores. Another feature unique to Grum among the Vangorda Plateau deposits, is the relative abundance of quartzose ore types, particularly carbonaceous pyritic quarzites, which compromise about 50% of total projected millfeed. This ore type is harder than massive sulphide and will, presumably, place greater demands on the grinding circuit to achieve liberation of sphalerite and galena grains. Interestingly, Grum disseminated ores do not appear to be as fine-grained as the massive ores, and in hand specimen are very difficult to distinguish from Faro disseminated ores.

The lower metamorphic grade deposits, like Grum, tend to have higher gold values than Faro (Grum ranges from 0.7 to 1.0 g/t Au, compared to Faro at 0.1 to 0.2 g/t) This results in significantly more gold in the lead concentrate. Along with gold, the other members of the gold geochemical association, such as arsenic and mercury, are enriched in Grum ores compared to Faro.

The Grum deposit is similar in structural style and grade of metamorphism to the Vangorda deposit. Techniques for grade control and mining with minimum dilution that have been developed at Vangorda will be directly applicable at Grum. Grum differs from Vangorda, however, in that although the Grum massive sulphides and disseminated sulphides of the main horizon are in relatively sharp contact, as at Vangorda, a greater proportion of the disseminated sulphides are ore grade at Grum and there will be a footwall assay cutoff within the disseminated sulphides. The hanging wall contact will be sharp between massive sulphide and phyllite, as at Vangorda.

Grum is unique in the Anvil District in that disseminated quartzite hosted sulphide ore, particularly carbon bearing quartzite (Rock Type 2) is the predominant ore type. These carbon bearing ores have been extensively tested to check for metallurgical difficulties caused by the carbon, but none have been encountered. In the northwest area of the pit, in an area known as the "Gnome's Cap" (Figure 3), which is the hinge of the crest phase anticline, these carbonaceous disseminated quartzite ores are particularly abundant, and are interlayered with carbonaceous phyllite. This is one area where dilution may be difficult to control. Since the ore is low grade this area has been avoided by the early phases of the I.V. Pit design.

The extent of near surface oxidation like that experienced at Vangorda is uncertain. Most ores do not outcrop at Grum, as they did at Vangorda, thus there has been less opportunity for post-glacial weathering. Because of this and limited testing, it is assumed that Grum will not have an oxide cap like Vangorda. Despite this assumption, some allowance has been made for adverse metallurgy in the upper part of Grum.

MINERAL INVENTORY

The Grum mineral inventory and reserve is derived from a 3-D block model known as the G9110 model. The model was begun in October, 1991, and completed in March, 1992. It is based on all available drilling and underground information for the deposit. A new geological interpretation, starting from base principles, was completed for the model. There are actually two G9110 models created, one using bench assay composites, and the other using equal length 2m geological interval assay composites. The geologically composited model has been selected for mine planning. A diluted version of the G9110 geological composited model has been created for mine reserve determination and medium-to-long range mine planning.

The G9110 model is the latest in a series of Curragh block models of the deposit which began with the G8606 model developed in 1986, based on the data and geological interpretation available at the time Curragh purchased the Anvil District reserves. Others in the series include G8705 (essentially the same as G8606 but with increased densities), G8911, G9008, G9009, and G9108. The latter 4 models incorporated various preliminary interpretations of fill-in drilling done by Curragh from 1987 to 1991, as well as the assay information from the new drilling. The latter four models were also used to experiment with various bench heights, grade interpolation methods, and assay compositing methods.

Co-ordinate System

The first surveys for the Vangorda Plateau were completed in the 1950's to provide survey control for exploration activities in the areas of the Vangorda and Swim Lake mineral deposits. The Vangorda coordinate system was oriented N-S, E-W with units being measured in feet.

In 1966, Sutcliffe and Co. Ltd surveyed an exploration base line on Grum running approximately northwest-southeast relative to the original Vangorda grid. Cross lines were established at 200 foot intervals (68W, 70W, etc.), and tie lines parallel to the base line were established at 1000 foot intervals (10S, 10N, etc). These cut lines were used to locate and guide all exploration geochemical sampling, geophysics, and diamond drilling in the Grum area. The south end of the base line was located at (30,000 E, 30,000 N) using the Vangorda grid coordinate system.

With the beginning of underground exploration on Grum by Kerr-Addison in 1975, it was imperative that surface and underground activities be completed using a common and accurate survey control. Hosford, Impey, and Welter established a new metric survey control which was parallel to the original Vangorda survey grid system. The south end of the Grum base line therefore was located at (9144.0 E, 9144.0 N) relative to the HIW survey control. Resurveying of the Sutcliffe base line established that although the direction of the baseline was accurate as represented (N46 12' 02" W), there was a systematic chaining error along its length. If this chaining error was evenly distributed among the cross lines, the distance between the cross sections was actually 199.123 feet (60.693 metres) rather than 200 feet (60.96 metres). All drill hole collars and underground development were surveyed and reported using the 1975 HIW survey control.

In 1979 Cyprus Anvil Mining Corporation acquired mineral rights to all the deposits in the Anvil District. Northwest Survey Corporation (Yukon) Ltd. completed a new survey control for the entire district. All survey control points used in earlier local surveys were incorporated into the 1979 District survey. This control survey was then tied into the international UTM (Universal Transverse Mercator) grid system using Canada Geodetic Survey control points in the Faro-Ross River areas. The locations of these CGS control points were adjusted in mid-1979 using satellite control; these adjusted coordinates were incorporated into the Anvil District control survey. Any subsequent adjustments to these control points have not been incorporated into this district wide survey control. These UTM coordinates and elevations have become the "common denominator" coordinates for exploration and development activities in the Anvil District. All regional surveys completed subsequently to 1979 have used the 1979 UTM survey control as the basis for their activities.

In the Grum area a local grid has been utilized for the generation of computer block models and ore reserve estimates. The model coordinate systems have generally been established parallel to the blocks to facilitate the computer intensive model calculations.

Beginning in 1987, the various ore reserve models generated by Curragh utilized a coordinate system with model North oriented parallel to the exploration grid base line. Models G8705, G9008, G9108, G9009, and G9110 all use this same coordinate system.

The G9110 coordinate system is tied to survey control station 1404 (earlier named VG4) located on the Blind Creek road between the Grum and Vangorda areas. Table 3 lists the UTM and Grum G9110 model coordinates for this survey station.

Table 3. Co-ordinates for Survey Station 1404 (VG4)		
UTM Grid Co-ordinates	Northing	6,904,623.172
	Easting	593,847.979
	elevation	1,300.062
Grum Local and Co-ordinates	Northing	5,000.000
	Easting	3,500.000
	elevation	1,300.062

The Grum G9110 coordinate grid has been adopted by mine operations as the Grum local mine grid. It is an orthogonal grid oriented parallel to the exploration cross and long sections. It is also parallel to the current PC-MINE model blocks. Local north (for this grid) is rotated 47.7741667 degrees (0.833816 radians) counterclockwise from UTM north.

Conversion between Grum Local and Anvil District UTM coordinate systems can be completed using the following equations:

$$N_{utm} = N_o + S_h * (N_{local} * \cos(x) + E_{local} * \sin(x))$$
$$E_{utm} = E_o + S_h * (E_{local} * \cos(x) - N_{local} * \sin(x))$$

where N_o = 6,898,674.069
E_o = 595,197.633
S_h = 0.99950853
x = 47.7741667 degrees (0.833816 radians).

The S_h scaling factor is averaged for the elevations typically encountered on the Vangorda Plateau.

In general, survey control for drillhole collars is good at Grum since Kerr Addison and, later, Cyprus Anvil, were careful about hole locations in potential mining areas after difficulties at Faro. Downhole surveys were not as carefully done and many holes had no surveys at all or questionable surveys. In cases of missing surveys, the averages of surrounding holes were used. Some questionable downhole survey data was discarded and nearby averages substituted.

Grum Model Property Definition

The plan size, plan location, and block size of the Grum G9110 model have been slightly modified from the earlier Grum models. These changes were made to allow the G9110 rows and columns to correlate exactly with the exploration drill section grid pattern. Table 4 lists the new overall coordinate information for the G9110 locations.

Table 4. Grum G9110 Model Corner Co-ordinates

Lower Left Corner	UTM	Northing	6,903,968.48
		Easting	592,687.06
	LOCAL	Northing	5,419.88
		Easting	2,234.37
Lower Right Corner	UTM	Northing	6,904,684.74
		Easting	593,337.11
	LOCAL	Northing	5,419.88
		Easting	3,202.11
Upper Left Corner	UTM	Northing	6,905,083.55
		Easting	591,458.41
	LOCAL	Northing	7,079.90
		Easting	2,234.37
Upper Right Corner	UTM	Northing	6,905,799.81
		Easting	592,108.47
	LOCAL	Northing	7,079.90
		Easting	3,202.11

Block Length is 7.5866m in the longitudinal section direction (135°)and 7.62m in the cross section direction (045°). Blocks are 6m high. Blocks volume is 346.9m^3; thus blocks represent 1,000 to 1,400 tonnes of ore, or 940 tonnes of waste.

The cross section lines pass through the centres of a particular row of blocks. There are 219 rows in the model. Eight rows occur between each even-numbered cross section. Column centres for the G9110 model correspond exactly to the longitudinal section lines. There are 127 columns in the model. Eight columns occur between each even-numbered longitudinal section. The correspondence between the model block column and row numbers, and the longitudinal and cross sections, is listed in Tables 5 and 6, respectively.

Table 5. Location of Grum Longitudinal Sections		
Section	Local Easting	Model Column
12 S	2,245.8	2
10 S	2,306.8	10
08 S	2,367.8	18
06 S	2,428.7	26
04 S	2,489.7	34
02 S	2,550.6	42
00 B/L	2,611.6	50
02 N	2,672.6	58
04 N	2,733.5	66
06 N	2,794.5	74
08 N	2,855.4	82
10 N	2,916.4	90
12 N	2,977.3	98
14 N	3,038.3	106
16 N	3,099.3	114
18 N	3,160.2	122

Even numbered long sections are spaced every 60.96 metres (200 feet)

Table 6. Location of Grum Cross Sections		
Section	**Local Northing**	**Model Rows**
42 W	5,423.7	219
44 W	5,484.3	211
46 W	5,545.0	203
48 W	5,605.7	195
50 W	5,666.4	187
52 W	5,727.1	179
54 W	5,787.8	171
56 W	5,848.5	163
58 W	5,910.0	155
60 W	5,969.9	147
62 W	6,030.6	139
64 W	6,091.3	131
66 W	6,152.0	123
68 W	6,212.7 6182.33	115
70 W	6,273.4	107
72 W	6,334.1	99
74 W	6,394.7	91
76 W	6,455.4	83
78 W	6,516.1	75
80 W	6,576.8	67
82 W	6,637.5	59
84 W	6,698.2	51
86 W	6,758.9	43
88 W	6,819.6	35
90 W	6,880.3	27
92 W	6,941.0	19
94 W	7,001.7	11
96 W	7,062.4	3

Even cross sections are spaced every 60.693 metres (199.1 feet).

The elevation at the top of the model is 1336.0m. Bench height is 6.0m. There are 70 benches in the model. A printout of the property definition is included in Appendix A.

Surface Grids

In 1979 the surface topography for the Grum area was mapped by Northwest Survey (Yukon) Ltd. from low level aerial photography completed in August 1979. The resultant base map was at 1:2,000 scale and contained 2.5m elevation contours. In October 1990 the Grum area was re-mapped by Orthophoto Shop using new low level aerial photographs completed in September 1990. This map was at 1:2,000 scale with a 2m elevation contour interval.

The two maps are not strictly comparable because portions of the Grum pit area had been excavated before the second (1990) set of aerial photographs. In addition the haul road between the Vangorda deposit and the Faro mill had been constructed through the area. In scattered areas which have not been disturbed the two maps show inconsistencies with elevation differences of up to +/- 2 metres. Spot checks by Mine Survey crews indicate that the 1990 surface topography is reasonably accurate.

All of the surface grids in the G9110 model were created in GEOMODEL from digitized surface elevation contours. The surfaces were calculated using the average of row and column elevation calculations. Each row and column was further subdivided into four subrows and subcolumns for the resultant average value. A list of surface grids to March, 1992, is provided in Appendix A.

The 1979 surface grid was digitized from the 1979 Northwest Survey Ltd. of the Grum area completed for Cyprus Anvil Mining Corporation (1:2000 scale). The 1990 surface grid was imported into GEOMODEL from a DXF file of the Autocad drawings for Grum sheets 1 and 2 prepared by Orthoshop using the 1990 aerial photography (1:2000 scale). Before the DXF file was imported all co-ordinates for the contour lines were changed from UTM grid to Grum Mine grid. This conversion was accomplished using a small BASIC program written specifically for the conversion.

The "**best initial surface**" was prepared by using the digitized lines from the 1979 Grum map and re-assigning elevations to these contour lines using the 1990 map as a guideline. These changes were made only for the contour lines north (Mine Grid) of Doal Lake (not shown on Figure 2, but from 68W to 74W, and 9N to 20N). In addition spot elevations from the 1990 map corresponding to the exposed bottom of Doal Lake were digitized as spot traverses (this is surface #3 in Appendix A).

The Ion Vintila design pits (the I.V. Pit Stages I to III) were digitized in Autocad and imported into GEOMODEL from the Autocad DXF files (the ultimate pit surface is surface II in Appendix A). The Quick Sink pits were digitized directly into GEOMODEL and created from the resulting surface topography files.

Care must be used when dealing with surfaces, in order not to confuse reserve reporting. As mining proceeds it is important to always merge the latest surface with the previous one so that any fill placed in the pit is removed. Merging surfaces in PCMINE takes the lowest elevation in a grid cell, thus giving the deepest excavation, or what is termed the cut surface in Appendix A.

Rock Type Model

In 1991 a new interpretation of the cross and longitudinal sections was completed using all drill hole information through the 1991 drilling program. Drill holes on cross section were projected using a plunge correction of -11 degrees in the North direction (Grum Mine grid). This plunge correction accounts for the overall structural plunge of the deposit along the F2 fold axial hinge line. Drill hole information on long section was projected horizontally perpendicular to the long section lines.

The initial geological interpretation was completed on the cross sections, and the resulting geology was digitized into GEOMODEL. Long sections were then cut from the cross sections and compared to the projected drill hole information. The long sections were interpreted from this information and also digitized into GEOMODEL.

Plan maps corresponding to mid-bench elevations were then cut from the cross and long section interpretations. The mid-bench geology was then re-interpreted on plan and digitized in GEOMODEL. From this geological interpretation non-overlapping polygons outlining each rock type on a bench were created. Rock-type polygons were transferred to PCMINE from the GEOMODEL plan map geological interpretations. The resulting PCMINE block model geology is defined by these polygons. PCMINE assigns a block rock code if the block centre is within a rock type polygon.

Geology was interpreted from Section 51W to 88W. Block codes between these sections are assigned according to the bench polygons. Beyond these sections, to the northwest and southeast, blocks are coded as undifferentiated rock (Rock Type 77), overburden, or air (see below). The creation of the PCMINE model from the plan geology results in a more realistic resolution of fault and fold geometries than previous models which were based only on cross-sections.

To ensure internal compatibility between the section and plan interpretations, the long and cross sections were then re-interpreted from the mid-bench plan geology. The final Grum cross and long sections have been adjusted where necessary to be consistent with the plans. In general only minor changes, mainly new faults, were needed.

All drill holes before 1991 were logged according to a Cyprus Anvil alpha-numeric lithostratigraphic code. A new numeric lithologic code was introduced to Grum in 1991. The earlier alphanumeric rock codes have been modified to correspond to the new lithologic code. A list of the new rock codes, along with the older rock code in use from 1976 to 1990, is provided in Appendix B. This model differentiates among non-calcareous phyllites (Rock Codes 20, 22), carcareous phyllites (Rock Code 40), carbonaceous phyllites (Rock Code 30), greenstones (Rock Code 48), and altered (bleached) phyllites (Rock Code 56). Mineralization lithologies incorporated in the model include carbonaceous quartzite (Rock Code 2), non-carbonaceous quartzite (Rock Code 3), pyritic massive sulphides (Rock Code 5), baritic massive sulphides (Rock Code 7), mixed carbonaceous and non-carbonaceous quartzite (Rock Code 12), mixed quartzite and massive sulphides (Rock Code 15), and mixed pyritic and baritic massive sulphides (Rock Code 18). In addition, the geological interpretation differentiated between different ore horizons within the Grum deposit. Table 7 lists the different ore horizons incorporated into the current geological interpretation and block coding. For the bench composite interpolation, the horizon codes were deleted from the rock type model.

Table 7. Sulphide Horizon Codes

Horizon Code	Description
100	Lower main horizon (overturned limb on F1)
200	Upper main horizon (upright limb on F1)
300	Upper horizon (upright limb on F1)
400	Champ horizon
500	Upright panel horizon
600	Gnome's cap horizon
700	Upper horizon (overturned limb on F1)

Overburden was also entered into the model as Rock Type 82 using a surface grid generated in GEO-MODEL. The overburden surface grid was prepared by hand contouring the triconed elevations in plan at five metre intervals. This surface was digitized into GEOMODEL and converted to a surface grid using the average from both rows and columns. Mid-bench elevation contour lines of this surface grid were then incorporated into the plan geological interpretation. Air (rock type 500) was entered into the rock type model using the contoured bench toe elevations in the plan geological interpretation. The "best initial surface", described previously, was used to generate the toe elevation contour lines.

Assay Composites

All drill hole data is stored in a PC-XPLOR database (Database A). A copy of the database (Database B), containing only lithology and assay data, is used to generate assay composites. Database B contains header information in database table 1, downhole deviations in database table 2, lithologies in database table 3, assays in database table 4, and various composites in database table 5, and later database tables.

The Anvil District alpha-numeric rock codes, in database tables 3 and 4, were converted to numeric values using a SYMPHONY spreadsheet with extensive lookup tables.

Assays for the quartzose ores (Rock Codes 2, 3) and massive sulphide ores (Rock Codes 4, 5, 7, 8) were grouped and analyzed using univariate histograms. Pb, Zn, Ag, Au, and pulp SG assays for each of these two groups were then cut to the 95th percentile. Table 8 contains the 95th percentile values used to cut the assays.

Table 8. 95th Percentile Cutoff Values

Rock Code	Pb %	Zn %	Ag g/t	Au g/t	SG
2 and 3	6.24	10.77	102.0	1.85	3.86
4 thru 8	9.72	17.19	157.0	2.36	4.85

Two types of composites were created. One type corresponds to the bench elevations that will be mined. The other is approximately equal length composites that correspond to geologic units.

Bench Composites

Drill holes were composited in 6m bench intervals starting at 1336m elevation. The composite location is given by the mid bench elevation. The holes with shallower inclination (mostly underground holes) were composited using 6 metre equal length intervals starting at the drill hole collar. The composite location is given by the mid elevation of the interval. G9110 composites are length weighted only. No S.G. weighting was used.

In the case where assays were missing for ore rock types the missing intervals were not included in the composite. If these missing assays comprised more than 25% of the composite interval, then that composite was discarded. This mainly affects gold as the Grum data set is otherwise nearly complete. Waste rock type intervals within a composite were assigned a 0 assay value.

PC-XPLOR assigns a rock code to the composited interval based on the lithology at the centre of the interval. It does not take into account whether that particular rock type is the dominant lithology in the interval. To overcome this limitation, the Fortran program RKCOMP1 (L. Pigage, 1990, modified in 1991) was written to incorporate a weighting factor when assigning the rock code to a particular composite interval. With this programme the rock type was assigned to the composites using a length weighting algorithm. The weighted length of each different rock code in the composite interval was calculated separately. The rock code assigned to the composite was the dominant rock type based on the cumulative lengths. These composite rock codes were calculated from the assay Table 4 for each drill hole. Because the different non-sulphide rock types are all assigned the same rock code in the assay database, the non-sulphide types were lumped as a single unit when determining the dominant rock type.

Geological Composites

PC-XPLOR database B was used to calculate geological composites for intervals based on the plan, cross and long geological interpretation. For each drill hole the "from" and "to" intervals for the different lithologies to be composited were specified based on the cross section geological interpretation. Only those intervals containing assays were defined for compositing. Internal waste intervals, which were not separated out in the geological interpretation, were also not differentiated in the compositing interval. These waste intervals were included in the interval with 0.0 grade assays for all elements. Rock type codes assigned to the compositing intervals included the full horizon coding described for the ore types (see Table 5). There is no minimum width for compositing since cross section units were defined, keeping in mind the potential mining of the deposit (the sections were interpreted by an experienced pit geologist, C. V. Reed).

A small BASIC programme (COMP.BAS) was written to further subdivide these geological intervals into 2m long composite intervals for use in interpolation. The equal length intervals were calculated from the start of the geological interval. The final composite interval, using this algorithm, consisted of a partial length, less than the selected composite length. The assays within the resulting calculated composite intervals were length weighted.

Grade Interpolation

Several interpolation tests were completed for Bench 34 in the G8911 model using varied search parameters. These test runs looked at varying the search distance in the model north direction, east direction, and elevation, using different powers with inverse distance weighting. They also examined tilting the search ellipsoid to account for the structural grain. The parameters selected based on these tests are middle-of-the-road in their effect upon the grades of the blocks.

Models for SG, Pb, Zn, Ag, and Au were then interpolated. The Pb+Zn model was calculated by adding the interpolated values for Pb and Zn for each block. The SG value interpolated is used for ore density in reporting reserves. The value used is the experimentally determined pulp specific gravity reduced by 2% to account for porosity. This reduction factor is based on experience at the Faro mill with Faro ore. Various reductions from 0% to 5% have been used over the years for Grum; this is a factor which must be monitored over time and, as experience develops, it may need modification.

The models were interpolated in four passes. Blocks with interpolated grades in any particular pass were skipped in the subsequent passes. Table 9 contains the pertinent search parameters for each of the passes. The pass four interpolation parameters are similar to those used for the G8608 and G8705 models, upon which the previous official reserve for Grum was based. After the fourth pass certain sulphide blocks, remote from drill holes, do not have interpolated grades. These blocks are assigned zero grade. The geological composite model contained 1.6 million tonnes of un-interpolated material and the bench composite model, 2.3 million tonnes.

Table 9. Interpolation parameters for G9110 Model

	Pass 1	Pass 2	Pass 3	Pass 4
North search	50 m	75 m	75 m	75 m
East search	50 m	50 m	50 m	50 m
Elev search	11 m	11 m	20 m	50 m
Tilt	-11	-11	-11	-11
Weighting	inverse distance	inverse distance	inverse distance	inverse distance
Power	1	1	1	1
Minimum #	2	2	2	2
Maximum. #	10	10	10	10

Grade interpolation for the bench composite model.

Loose rock matching was used during the interpolation. All massive sulphides (Rock Codes 5 and 7) were considered equivalent, and all quartzites with disseminated sulphides were considered equivalent (Rock Codes 2 and 3). Rock Type 15 was equated with all ore types during interpolation. Horizon coding was not used during the interpolation.

After the interpolation was completed the specific gravity model was edited using program RKDENS (Pigage 1987) to put in the missing SG values. All waste phyllite blocks were assigned an SG of 2.7. All overburden blocks were assigned an SG value of 2.2. The uninterpolated ore blocks were assigned the average SG values (SG assays reduced by 2%) of the interpolated blocks for each ore type, as listed in Table 10.

Table 10. Average SG of Interpolated Blocks

Rock Type Code	Description	Average SG
2	Carbonaceous Pyritic Quartzite	3.06
3	Non-carbonaceous Pyritic Quartzite	3.29
5	Pyritic Massive Sulphide	4.00
7	Barytic Massive Sulphide	3.89
12	Mixed Types 2 and 3	3.17
15	Mixed Types 2, 3 and 5	3.65
18	Mixed Types 5 and 7	3.96

Grade Interpolation for the Geological Composite Model

Models for SG, Pb, Zn, Ag, and Au were interpolated using the same parameters as used in the bench composite interpolation. Pb + Zn grades were calculated by adding the interpolated grades for Pb and Zn. The interpolation was completed in 4 passes. Table 9 contains the parameters used in the interpolation for each pass.

Horizon coding was incorporated into the rock type matching during the grade interpolation. All quartzites (Rock Codes 2, 3, 12), within a mineralized horizon, were equivalent during interpolation. Similarly all massive sulphides (Rock Codes 5 and 7), within a mineralized horizon, were equivalent during interpolation. Rock Type 15 was correlated with both massive sulphides and quartzites for that horizon during the interpolation. After interpolation was completed, the horizon code was dropped from the rock model to simplify reporting. A copy of the horizon coded model has been retained for future interpolation.

Programme RKDENS was used to assign average SG values to uninterpolated blocks for each ore type, as described above and listed in Table 10. Waste model blocks were assigned an SG value of 2.7 (phyllite), 2.2 (overburden), or 2.85 (altered phyllite).

Mineral Inventory Results

Mineral inventory results are derived by totalling the tonnage and grade of all blocks above a given block Pb + Zn cutoff grade, which are within relevant geographic limits, and that fall between certain surface elevations that correspond to topographic or pit design surfaces. Blocks partly within a surface are weighted by the proportion of the block height at the block centre between the bounding surfaces. Blocks are not similarly subdivided in plan view; a block is either within or outside of the constraining limits, based on the location of its centre. The overall mineral inventory sums all blocks below starting topography, down to the bottom of the model. Mine reserves are derived by summing only the blocks between topography and various pit design surfaces, then applying relevant corrections for dilution or recovery (see below for further detail). The surfaces referred to are the surface grids listed in Appendix A and discussed previously. The in-situ mineral inventory is reported below surface 3, the best initial surface.

The results are provided for various Pb + Zn cutoff grades in Table 11. Table 12 only presents data from Sections 61W to 87W, since only this portion of the deposit was quantified in previous studies. The bench model is 14% lower grade than the geologically composited model at the 3% cutoff to be used for the mine. This is thought to be largely due to the bench composites averaging in waste so that dilution is built into the bench model. The tonnage differential above cutoff is high at high cutoff grades, but bench model tonnage is only about 5% lower than the geological model tonnage at a 3% Pb + Zn cutoff grade. There are probably many reasons for this, but the primary one is due to the lower overall block grades resulting in fewer blocks falling above cutoff grade. The combination of lower tonnage and lower cutoff grade results in a significant difference between the bench and geological models (21%) in total contained lead and zinc metal above a 3% cutoff grade.

Previous experience at the Faro and Vangorda pits has shown that bench composited models seriously understate the tonnage, but can come reasonably close to mined grade. In general, bench composited models are conservative compared to reality, and geologically composited models better represent the resource <u>if the dilution and mining recovery factors are appropriate</u>. Since it was felt best to deal with the geological model and dilute in a controlled fashion, rather than dilute it during interpolation, the bench model will not be considered further.

Table 12 compares the G9110 geological composite model to the G8705 model. It can be seen that total contained metal is relatively similar at various cutoff grades, but that the newer G9110 model is lower grade, while having a larger tonnage above cutoff grade. The major reason for the lower overall grade in the newer model is thought to be due to three factors. The first factor is the much more restrictive composite rock type to block rock type matching scheme used during interpolation of the older model. Experience at Faro has shown that matching by rock type is important but too many different rock types seems to

Table 12

GRUM DEPOSIT – COMPARISON OF G8705 AND G9110 IN–SITU MINERAL INVENTORY

in–place mineralization / no dilution / no mining recovery consideration

	MINERAL INVENTORY									CONTAINED METAL			
	Cutoff (wt%) Pb+Zn	Volume (cu. m.) *1000	Density (tn/m3)	Tonnage (tonnes) *1000	Pb+Zn (wt%)	Pb (wt %)	Zn (wt %)	Ag (g/t)	Au (g/t)	lead tonnes *1000	zinc tonnes *1000	silver kg	gold kg
Geological Model	6	7,044.82	3.595	25,326.32	9.75	3.73	6.01	61.8	0.93	945.178	1,522.872	1,565,243	23,503
G9110–GEOL(2m)	5	9,061.44	3.508	31,785.05	8.88	3.40	5.48	56.5	0.88	1,081.645	1,741.185	1,797,190	27,812
Central Portion (61W–87W)	4	10,914.13	3.455	37,704.48	8.19	3.15	5.05	52.5	0.84	1,186.183	1,903.322	1,978,618	31,521
all horizons	3	12,597.71	3.432	43,240.25	7.59	2.94	4.66	49.1	0.82	1,269.101	2,014.996	2,124,653	35,284
Geological Model	6	6,558.30	3.650	23,963.52	10.36	3.90	6.46	66.0	1.00	934.577	1,548.043	1,581,592	23,964
G8705–GEOL(4.5m)	5	7,711.20	3.610	27,855.47	9.68	3.66	6.02	61.0	0.98	1,019.510	1,676.899	1,699,184	27,298
Central Portion (61W–87W)	4	9,023.94	3.570	32,181.62	8.98	3.40	5.58	57.0	0.95	1,094.175	1,795.734	1,834,352	30,573
all horizons	3	10,087.20	3.540	35,723.39	8.45	3.22	5.23	54.0	0.93	1,150.293	1,868.334	1,929,063	33,223
Absolute Difference	6	486.52	−0.055	1,362.80	−0.61	−0.17	−0.45	−4.2	−0.07	10.601	(25.172)	(16,350)	(461)
G9110 compared to G8705	5	1,350.24	−0.102	3,929.58	−0.80	−0.26	−0.54	−4.5	−0.11	62.135	64.286	98,007	514
	4	1,890.19	−0.115	5,522.86	−0.79	−0.25	−0.53	−4.5	−0.11	92.008	107.588	144,266	948
	3	2,510.51	−0.108	7,516.86	−0.86	−0.29	−0.57	−4.9	−0.11	118.808	146.662	195,590	2,061
Relative Percent Difference	6	6.9%	−1.5%	5.4%	−6.3%	−4.5%	−7.4%	−6.8%	−7.8%	1.1%	−1.7%	−1.0%	−2.0%
G9110 compared to G8705	5	14.9%	−2.9%	12.4%	−9.0%	−7.6%	−9.9%	−7.9%	−12.0%	5.7%	3.7%	5.5%	1.8%
	4	17.3%	−3.3%	14.6%	−9.6%	−8.1%	−10.5%	−8.6%	−13.6%	7.8%	5.7%	7.3%	3.0%
	3	19.9%	−3.1%	17.4%	−11.3%	−9.7%	−12.2%	−9.9%	−14.0%	9.4%	7.3%	9.2%	5.8%

Table 11

GRUM DEPOSIT – IN-SITU MINERAL INVENTORY

in-place mineralization / no dilution / no mining recovery consideration

	MINERAL INVENTORY									CONTAINED METAL			
	Cutoff (wt%) Pb+Zn	Volume (cu. m.) *1000	Density (tn/m3)	Tonnage (tonnes) *1000	Pb+Zn (wt %)	Pb (wt %)	Zn (wt %)	Ag (g/t)	Au (g/t)	lead tonnes *1000	zinc tonnes *1000	silver kg	gold kg
GEOLOGICAL COMPOSITES	8	4,159.73	3.82	15,882.26	11.44	4.39	7.05	72.6	1.05	696.437	1,120.335	1,153,211	16,708
G9110 MODEL–GEOL (2m)	6	7,044.82	3.60	25,326.32	9.75	3.73	6.01	61.8	0.93	945.178	1,522.872	1,565,243	23,503
Central Portion (61W–87W)	5	9,061.44	3.51	31,785.05	8.88	3.40	5.48	56.5	0.88	1,081.645	1,741.185	1,797,190	27,812
all horizons	4	10,914.13	3.46	37,704.48	8.19	3.15	5.05	52.5	0.84	1,186.183	1,903.322	1,978,618	31,521
	3	12,597.71	3.43	43,240.25	7.59	2.94	4.66	49.1	0.82	1,269.101	2,014.996	2,124,653	35,284
	0	16,298.25	3.41	55,500.12	6.32	2.46	3.87	42.0	0.77	1,363.083	2,145.080	2,331,560	42,957
	uninterp	487.60	3.32	1,618.19									
BENCH COMPOSITES	8	2,718.40	3.88	10,546.63	10.92	4.10	6.82	68.7	1.03	432.623	718.753	724,617	10,905
G9110 MODEL–BENCH (6m)	6	5,410.80	3.61	19,552.17	9.07	3.43	5.65	57.4	0.91	669.662	1,104.111	1,121,864	17,734
Central Portion (61W–87W)	5	7,419.10	3.52	26,076.73	8.18	3.10	5.08	51.9	0.85	807.596	1,324.959	1,354,451	22,191
all horizons	4	9,703.26	3.44	33,380.75	7.37	2.80	4.57	47.1	0.79	935.329	1,525.166	1,572,767	26,504
	3	12,111.14	3.40	41,193.07	6.64	2.54	4.10	42.8	0.74	1,047.540	1,687.680	1,763,063	30,648
	0	16,096.90	3.38	54,381.41	5.50	2.13	3.37	36.2	0.70	1,157.780	1,833.197	1,970,184	37,904
	uninterp	688.96	3.40	2,345.50									
Absolute Difference	8	(1,441)	0.06	(5,336)	−0.52	−0.28	−0.24	−3.9	−0.02	(263.814)	(401.582)	(428,594)	(5,803)
Bench compared	6	(1,634)	0.02	(5,774)	−0.67	−0.31	−0.37	−4.4	−0.02	(275.516)	(418.761)	(443,378)	(5,769)
to Geological model	5	(1,642)	0.01	(5,708)	−0.70	−0.31	−0.40	−4.6	−0.02	(274.049)	(416.226)	(442,739)	(5,621)
	4	(1,211)	−0.01	(4,324)	−0.82	−0.34	−0.48	−5.4	−0.04	(250.854)	(378.156)	(405,851)	(5,017)
	3	(487)	−0.03	(2,047)	−0.95	−0.39	−0.56	−6.3	−0.07	(221.562)	(327.316)	(361,590)	(4,636)
	0	(201)	−0.03	(1,119)	−0.82	−0.33	−0.49	−5.8	−0.08	(205.303)	(311.882)	(361,376)	(5,053)
	uninterp	201	0.09	727									
Relative Percent Difference	8	−53.0%	1.6%	−50.6%	−4.8%	−6.9%	−3.5%	−5.7%	−1.7%	−61.0%	−55.9%	−59.1%	−53.2%
Bench compared	6	−30.2%	0.5%	−29.5%	−7.4%	−9.0%	−6.5%	−7.7%	−2.3%	−41.1%	−37.9%	−39.5%	−32.5%
to geological model	5	−22.1%	0.2%	−21.9%	−8.6%	−9.9%	−7.8%	−8.9%	−2.8%	−33.9%	−31.4%	−32.7%	−25.3%
	4	−12.5%	−0.4%	−13.0%	−11.2%	−12.3%	−10.5%	−11.4%	−5.3%	−26.8%	−24.8%	−25.8%	−18.9%
	3	−4.0%	−0.9%	−5.0%	−14.4%	−15.4%	−13.7%	−14.8%	−9.7%	−21.2%	−19.4%	−20.5%	−15.1%
	0	−1.3%	−0.8%	−2.1%	−14.9%	−15.4%	−14.7%	−16.0%	−11.0%	−17.7%	−17.0%	−18.3%	−13.3%
	uninterp	29.2%	2.5%	31.0%									

result in blocks having extreme high and low grades, which are meaningless on a mining scale (i.e. the ore cannot be sorted in reality into more than a few types). The second factor is that the newer model only used length weighting for assay intervals during compositing, rather than length and specific gravity as done on the older model. Since the massive ores tend to be higher grade, length and specific gravity weighting tends to increase grade above cutoff. The last factor is that the newer model uses inverse distance weighting, while the older one used inverse distance squared. Inverse distance squared tends to overweight high grade assays and result in higher grades above a cutoff. M. Dagbert (personal communication) indicated that inverse distance is more appropriate for deposits with high nugget effect, such as the Anvil deposits.

The increase in tonnage is thought to be due to the increase in volume of sulphide rock types that results from a proper interpretation of cross and longitudinal sections. Interpreting both sets of sections resulted in longer extrapolations of ore horizons from hole to hole in the less densely drilled part of the deposit (deeper and peripheral parts) than was the case working with cross sections only (as was the case for the G8705 model). The tonnage difference between the models within the pit is relatively small (less than 2% -see Table 14), which tends to confirm this hypothesis since the entire pit is relatively well drilled and not subject to these longer extrapolations.

Table 13 compares the central (61W - 87W) portion of the deposit as estimated by a number of different methods over the years. The known extent of the deposit has increased since 1977, but by 1983 the limits were basically known. This table shows that if the data set and the geologic interpretation is comparable, then 3-D block models tend to give lower grades than sectional hand calculations. The current mineral inventory calculation has not resulted in a tonnage, and more importantly, a grade which is outside of previous experience with the Grum deposit.

Table 14 compares the in-situ mineral inventory for the I.V. Pit design for a number of Curragh block models. Since the pit envelope restricts the comparison to well drill defined sulphides, the variance due to geologic interpretation is smaller than for the entire deposit. The comparison shows that the in-situ tonnage is closer to the other models. The G9110 model gives lower grade than the G8705, but higher grade than the two bench composited models, as would be expected.

Additional mineralization in the Champ Zone (Sections 52W to 61W) above a 3% Pb + Zn cutoff grade amounts to 3,283,840 tonnes averaging 2.90% Zn, 2.32% Pb, 35.6 g/t Ag and 0.62 g/t Au.

Classification of Mineral Inventory

The portion of the mineral inventory within the I.V. Pit is very well drilled and has been intersected by the underground workings. This part of the deposit is so

Table 13

HISTORICAL SUMMARY OF GRUM MINERAL INVENTORY ABOVE A 4% Pb+Zn CUTOFF GRADE

Total resource for the 61W to 87W portion of the deposit – no adjustments for dilution or mining recovery

NAME	METHOD	DATE	TONNAGE (tonnes*1,000,000)	Pb+Zn %	Pb %	Zn %	Ag g/t	Total Pb+Zn Metal (tonnes)	notes
KA hand	hand sectional	1977	26.1	10.5	4.1	6.4	62	2,739,000	1
KA–NORANDA	computer block	1978	27.7	8.0	3.1	4.9	48	2,212,000	1
CA–G1	computer block	1981	30.8	8.0	3.1	4.9	49	2,462,000	2
CA / DOME	hand sectional	1983	32.6	9.2	3.5	5.7	59	3,000,000	3
CA / DOME	hand sectional	1983/86	31.6	9.2	3.5	5.7	59	2,909,000	4
G8606	computer block	1986	30.6	9.0	3.4	5.6	57	2,758,000	5
G8705	computer block	1987	32.2	9.0	3.4	5.6	57	2,890,000	6
G9110	computer block	1991	37.7	8.2	3.1	5.0	52	3,090,000	

NOTES:

1 based on drilling up to end of underground program
2 similar to KA–NORANDA model with same geological interpretation and assays but minor additional drilling
3 based on new drilling, re–assay of core and new interpretation adding approx 3 Mt of new ore in northwest
4 same as above but corrected for math errors and unrealistic density assumption for the new ore
5 same geological interpretation as above, all CAMC drilling but no Curragh drilling
6 same as G8606 but 5% reduction of density removed to better reflect tonnage factor experience at Faro

Table 14

Summary of in–situ mineralization within the Grum I.V. Stage III Pit

total mineralization without dilution or mining recovery adjustments, at a 4% Pb+Zn cutoff grade

NAME	METHOD	DATE	TONNAGE (tonnes*1,000,000)	Pb+Zn %	Pb %	Zn %	Ag g/t	Total Pb+Zn Metal (tonnes)	notes
G8705	computer block	1987	23.0	9.2	3.4	5.8	58	2,111,000	
G8911	computer block	1989	22.4	8.0	3.0	5.1	50	1,805,000	1
G9009	computer block	1990	22.4	8.1	3.0	5.1	49	1,807,000	2
G9110	computer block	1991	23.4	8.6	3.2	5.3	54	2,007,000	

NOTES:

1 similar to G8705 but some revision of geology and new CI drilling (up to 1989) 7m. bench model and composites

2 identical to G8911 but 6m. bench model and composites

well drilled that relatively little variance of interpreted ore volume is possible. Continuity is well established from section to section by drifting in ore. Away from the underground openings continuity is also well established and there is still a strong sample base, but this mineralization is less well defined. The mineralization, near the underground workings, is thus considered proven; the remainder is probable. The proportion of the two classes has not been worked out, thus, the inventory, and the reserve derived from it, is considered undifferentiated proven plus probable. The Champ Zone is considered probable as there is no exposure of sulphides there. The deeper parts of the model, below the pit, are a combination of probable and possible mineralization. Since no reserve is established for this area, no classification is needed. Mineralization inferred outside of the model limits is considered potential only and would be the weakest of possible mineralization.

MINABLE RESERVE

The minable reserve is derived from the in-situ resource between topographic and pit design surfaces, as noted above. The upper surface used for the minable reserve is number 3, and the lower surface is number 11 on the surface list in Appendix A. Table 15 provides the in-situ mineralization with the I.V. Stage III ultimate pit limit for various cutoff grades from 0% to 8% Pb and Zn.

Rather than simply apply blanket dilution and recovery assumptions to the total reserve or even to each bench, it was deemed to be appropriate to calculate a new block model based on the G9110 2m geological composite block model. This model was designed to adjust each block according to its surroundings. This was done by checking each block to see if it was above cutoff grade and adjoined a block below cutoff grade. If so the block grade was reduced by 20%. If the block adjoined only blocks above cutoff, then its grade was not changed. Block tonnage was unchanged. This is equivalent to exchanging 20% of peripheral ore blocks with waste of the same specific gravity at 0% Pb + Zn, so that dilution is equivalent to mining loss. Figure 6 and 7 show the result of this adjustment for a portion of the 1126 bench. ON PLAN ONLY

Table 16 provides the results of this diluted model for all three stages of the Grum I.V. pit. Comparing the total on Table 16 to the total in Table 15 implies that overall dilution is approximately 11% if by material at 0% Pb + Zn content, and that mining recovery of diluted in-situ ore is 88%. This overall dilution is in reasonable accord with the unadjusted grade of the bench composited model, as well as experience in the Faro and Vangorda pit. As noted previously, the minable reserve in the I.V. Pit is considered undifferentiated proven + probable ore.

A larger pit, termed the A.B. pit, was laid out in 1992 using a Lerch Grossman pit optimization routine. This pit extracted the Champ zone, as well as deeper parts of the rest of the Grum deposit. The A.B. pit contained diluted reserves of 30.54 million tonnes averaging 4.36% Zn, 2.62% Pb, 44 g/t Ag and 0.7 g/t Au, above a 3% cutoff. The increment between the I.V. Pit and the A.B. Pit thus contained 5.78 million diluted tonnes averaging 3.59% Zn, 2.16% Pb, 35.4 g/t Ag and 0.79 g/t Au, above a 3% cutoff.

GRADE MODEL FOR LABEL 3 [%Zn]

DESCRIPTION : zinc - pass 1 CREATED ON : 11/ 2/1992

LEVEL : 35 1126 UPPER ELEVATION : 1132.0 LOWER ELEVATION : 1126.0

SELECTED ROWS FROM [55] TO [97] SELECTED COLUMNS FROM [55] TO [79]

	55	56	57	58	59	60	61	62	63	64	65	66	67	68	69	70	71	72	73	74	75	76	77	78	79
55	.00	.00	.00	3.76	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00
56	.00	.00	.00	3.75	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	3.51	3.35	3.34	2.81	2.97	3.03	.00
57	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	2.86	2.86	2.92	2.88	3.05	2.62	2.77	2.67
58	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	1.81	2.21	2.18	2.54	2.58	1.94	1.23	1.22	3.54
59	.00	.00	4.38	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	2.49	2.19	1.99	1.63	1.20	1.20	1.20	1.20	1.20	1.20	.00
60	.00	.00	3.69	3.45	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	2.51	2.32	2.05	1.72	1.15	1.13	1.18	1.23	1.15	1.17	.00
61	.00	3.75	3.59	3.48	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	2.34	2.06	1.88	1.59	1.20	.98	1.09	1.18	1.24	1.27	.00
62	3.84	3.75	3.67	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	2.30	2.12	1.97	1.81	1.47	1.19	1.13	1.20	1.25	1.28	.00
63	3.76	3.79	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	3.47	2.97	3.75	2.54	2.08	1.87	1.56	1.57	1.62	1.49	1.28	.00	.00
64	4.30	.00	.00	.00	.00	.00	.00	.00	.00	.00	3.98	3.75	3.57	3.27	2.75	2.56	2.28	2.16	1.68	1.65	1.70	.00	.00	4.20	4.42
65	.00	.00	.00	.00	.00	.00	.00	.00	.00	4.17	4.10	3.67	3.45	3.23	2.74	2.38	2.31	2.11	1.84	.00	.00	3.66	3.80	3.89	3.89
66	.00	.00	.00	.00	.00	.00	.00	4.98	4.82	4.67	4.06	3.74	3.30	3.20	2.87	2.43	2.38	.00	.00	.00	3.45	3.36	3.28	3.20	3.39
67	.00	.00	.00	.00	.00	.00	.00	5.09	5.03	4.66	4.05	3.64	3.33	3.09	2.79	2.36	.00	.00	.00	.00	2.98	3.19	2.90	2.50	.00
68	.00	.00	.00	.00	.00	.00	.00	4.98	4.73	4.37	3.95	3.35	2.67	2.71	.00	2.18	.00	.00	.00	3.04	3.00	2.84	2.68	2.56	.00
69	.00	.00	.00	.00	.00	.00	.00	8.94	4.78	4.31	3.85	3.28	2.75	.00	.00	2.53	2.50	.00	.00	2.60	2.82	2.74	2.74	2.71	.00
70	.00	.00	.00	.00	.00	.00	.00	9.00	9.23	4.31	3.84	3.28	.00	.00	.00	2.44	2.41	.00	.00	2.62	2.70	2.75	2.77	2.74	.00
71	.00	.00	.00	.00	.00	.00	.00	.00	9.63	4.70	.00	.00	.00	.00	.00	.00	2.49	.00	.00	2.66	2.65	2.69	2.67	2.73	.00
72	.00	.00	.00	.00	.00	.00	.00	9.55	9.81	5.46	.00	.00	.00	.00	.00	2.46	2.69	.00	.00	2.77	2.76	2.75	2.82	2.80	.00
73	.00	.00	.00	.00	.00	9.52	9.76	9.89	6.92	.00	.00	.00	.00	.00	2.44	2.54	.00	.00	.00	2.69	2.70	2.77	2.79	2.91	.00
74	.00	.00	.00	.00	10.33	10.24	6.93	6.80	6.70	.00	.00	.00	.00	2.29	2.04	.00	.00	.00	.00	2.72	2.80	2.85	2.87	2.91	.00
75	.00	.00	.00	.00	10.36	7.34	7.69	7.15	6.57	.00	.00	.00	.00	2.27	.00	.00	.00	.00	.00	.00	2.88	2.96	3.02	3.26	.00
76	.00	.00	.00	.00	11.05	7.00	7.12	6.72	6.42	.00	.00	.00	.00	2.25	.00	.00	.00	.00	.00	.00	3.05	3.18	3.23	.00	.00
77	.00	.00	.00	.00	.00	11.82	6.99	6.59	6.15	.00	.00	.00	.00	1.90	.00	.00	.00	.00	.00	3.17	3.30	3.35	3.40	.00	.00
78	.00	.00	12.96	12.24	6.49	.00	6.62	6.48	6.09	.00	.00	.00	.00	1.96	.00	.00	.00	.00	.00	3.07	3.44	3.46	3.56	.00	.00
79	.00	.00	12.71	12.37	12.60	5.32	5.54	5.39	5.07	.00	.00	.00	2.05	2.00	.00	.00	.00	.00	.00	3.07	3.35	3.49	3.73	.00	.00
80	.00	.00	.00	11.28	11.48	4.78	4.93	.00	6.14	.00	.00	.00	2.18	.00	.00	.00	.00	.00	2.83	2.91	3.24	3.70	.00	.00	.00
81	.00	.00	.00	10.72	11.23	11.25	5.41	.00	.00	.00	.00	2.27	2.26	.00	.00	.00	.00	.00	3.10	3.33	3.71	.00	.00	.00	.00
82	.00	.00	.00	.00	11.10	10.96	6.37	.00	.00	.00	.00	2.64	.00	.00	.00	.00	.00	3.32	3.22	3.57	3.86	.00	.00	5.39	.00
83	.00	.00	.00	.00	9.52	10.51	5.18	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	3.24	3.46	3.89	4.38	.00	5.93	5.73	.00
84	3.29	.00	.00	.00	9.15	9.37	4.68	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	3.51	3.63	4.09	4.46	.00	6.19	.00	.00
85	.00	.00	.00	.00	8.57	4.40	4.29	4.15	.00	.00	.00	.00	.00	.00	.00	.00	.00	3.75	3.84	4.30	4.55	.00	5.96	.00	.00
86	.00	.00	.00	8.18	8.28	4.00	3.75	3.54	.00	.00	.00	.00	.00	.00	.00	4.30	.00	4.00	4.28	4.70	4.77	.00	5.82	.00	.00
87	.00	.00	.00	8.22	8.45	3.13	2.89	2.72	.00	.00	.00	.00	.00	.00	.00	4.77	.00	4.87	6.80	5.16	4.94	.00	5.99	.00	.00
88	.00	.00	.00	8.30	2.13	1.92	1.98	2.03	.00	.00	.00	.00	.00	.00	.00	.00	.00	5.44	5.36	5.67	5.45	.00	.00	.00	.00
89	.00	.00	.00	8.20	1.80	1.80	1.80	1.80	.00	.00	.00	.00	.00	.00	.00	.00	4.88	4.91	5.15	5.60	5.65	5.64	.00	.00	.00
90	.00	.00	6.26	1.64	1.60	1.58	1.56	1.50	.00	.00	.00	.00	.00	.00	.00	3.86	4.26	6.42	6.98	5.30	5.63	.00	.00	.00	.00
91	.00	.00	.00	1.57	1.56	1.51	1.50	1.53	.00	.00	.00	.00	.00	.00	.00	3.37	3.39	3.20	4.27	5.13	.00	.00	.00	.00	.00
92	.00	.00	.00	5.09	1.47	1.52	1.50	1.52	.00	.00	.00	.00	.00	.00	2.86	3.06	3.52	3.71	4.08	4.55	.00	.00	.00	.00	.00
93	.00	.00	.00	5.33	1.56	1.55	1.53	1.55	.00	.00	.00	.00	.00	2.51	2.81	3.21	3.69	4.24	5.40	.00	.00	.00	.00	.00	.00
94	.00	.00	.00	6.91	1.83	1.79	1.76	1.76	.00	.00	.00	.00	.00	2.51	3.18	.00	4.89	5.69	5.99	6.21	.00	.00	.00	.00	.00
95	.00	.00	8.65	2.02	1.85	1.70	1.68	1.75	.00	.00	.00	.00	2.69	2.93	.00	4.55	5.33	6.18	6.50	6.83	.00	6.88	.00	.00	.00
96	.00	.00	10.09	2.83	2.63	1.89	1.74	.00	.00	.00	.00	.00	.00	.00	3.85	6.48	.00	6.13	6.92	.00	6.89	.00	.00	.00	.00
97	.00	.00	10.67	1.99	2.09	2.07	1.94	.00	.00	.00	.00	.00	.00	.00	6.21	.00	5.57	6.11	6.44	6.27	6.31	.00	.00	.00	.00

Figure 6 A portion of bench 1126 showing the zinc grade for each block, as calculated by the G9110 2m geologic composite model. Compare to Figure 7 for the diluted model block grades.

GRADE MODEL FOR LABEL 3 [%Zn dil]

DESCRIPTION :

CREATED ON : 20/ 4/1992

LEVEL : 35 1126 UPPER ELEVATION : 1132.0 LOWER ELEVATION : 1126.0

SELECTED ROWS FROM [55] TO [97] SELECTED COLUMNS FROM [55] TO [79]

	55	56	57	58	59	60	61	62	63	64	65	66	67	68	69	70	71	72	73	74	75	76	77	78	79
55	.00	.00	.00	3.01	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00
56	.00	.00	.00	3.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	2.81	2.68	2.67	2.25	2.38	2.42	.00
57	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	2.29	2.86	2.92	2.88	3.05	2.62	2.77	2.14
58	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	1.45	2.21	2.18	2.54	2.58	1.94	1.23	1.22	2.83
59	.00	.00	3.51	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	1.99	1.75	1.99	1.63	1.20	1.20	1.20	1.20	1.20	.96	.00
60	.00	.00	2.95	2.76	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	2.01	2.32	2.05	1.72	1.15	1.13	1.18	1.23	1.15	.93	.00
61	.00	3.00	2.87	2.79	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	1.87	2.06	1.88	1.59	1.20	.98	1.09	1.18	1.24	1.02	.00
62	3.07	3.00	2.94	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	1.84	2.12	1.97	1.81	1.67	1.19	1.13	1.20	1.25	1.02	.00
63	3.01	3.03	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	2.78	2.38	2.73	2.54	2.08	1.87	1.59	1.57	1.62	1.19	1.03	.00	.00
64	3.44	.00	.00	.00	.00	.00	.00	.00	.00	.00	3.18	3.00	3.57	3.27	2.75	2.56	2.28	2.16	1.68	1.32	1.36	.00	.00	3.36	3.52
65	.00	.00	.00	.00	.00	.00	.00	.00	.00	3.34	4.10	3.67	3.45	3.23	2.74	2.38	2.31	1.69	1.47	.00	.00	2.93	3.04	3.89	3.12
66	.00	.00	.00	.00	.00	.00	.00	3.98	3.85	4.47	4.06	3.74	3.30	3.20	2.87	1.95	1.90	.00	.00	.00	2.76	3.34	3.28	3.20	2.7[illegible]
67	.00	.00	.00	.00	.00	.00	.00	4.07	5.03	4.66	4.05	3.64	3.33	3.09	2.79	1.89	.00	.00	.00	.00	2.38	3.19	2.93	2.00	.00
68	.00	.00	.00	.00	.00	.00	.00	3.98	4.73	4.37	3.95	3.35	2.67	2.17	.00	1.75	.00	.00	.00	2.43	3.00	2.86	2.68	2.05	.00
69	.00	.00	.00	.00	.00	.00	.00	7.15	4.78	4.31	3.85	3.28	2.20	.00	.00	2.02	2.00	.00	.00	2.08	2.82	2.74	2.74	2.16	.00
70	.00	.00	.00	.00	.00	.00	.00	7.20	9.23	4.31	3.07	2.62	.00	.00	.00	1.96	1.93	.00	.00	2.10	2.70	2.73	2.77	2.19	.00
71	.00	.00	.00	.00	.00	.00	.00	.00	7.70	3.76	.00	.00	.00	.00	.00	.00	1.99	.00	.00	2.13	2.65	2.69	2.67	2.18	.00
72	.00	.00	.00	.00	.00	.00	.00	7.64	9.81	4.37	.00	.00	.00	.00	.00	1.97	2.15	.00	.00	2.21	2.76	2.75	2.80	2.24	.00
73	.00	.00	.00	.00	.00	7.62	7.81	9.89	5.53	.00	.00	.00	.00	.00	1.96	2.03	.00	.00	.00	2.15	2.70	2.77	2.79	2.33	.00
74	.00	.00	.00	.00	8.26	10.24	6.93	6.80	5.36	.00	.00	.00	.00	1.83	1.63	.00	.00	.00	.00	2.18	2.80	2.85	2.87	2.33	.00
75	.00	.00	.00	.00	8.27	7.34	7.69	7.15	5.26	.00	.00	.00	.00	1.82	.00	.00	.00	.00	.00	.00	2.30	2.96	3.02	2.61	.00
76	.00	.00	.00	.00	8.84	7.00	7.12	6.72	5.13	.00	.00	.00	.00	1.80	.00	.00	.00	.00	.00	.00	2.44	3.18	2.59	.00	.00
77	.00	.00	.00	.00	.00	9.46	6.99	6.59	4.92	.00	.00	.00	.00	1.52	.00	.00	.00	.00	.00	2.53	3.30	3.35	2.72	.00	.00
78	.00	.00	10.36	9.79	5.19	.00	6.82	6.48	4.87	.00	.00	.00	.00	1.57	.00	.00	.00	.00	.00	2.45	3.44	3.46	2.85	.00	.00
79	.00	.00	10.17	12.27	12.60	5.32	5.54	4.31	4.05	.00	.00	.00	1.64	1.60	.00	.00	.00	.00	.00	2.46	3.35	3.49	2.99	.00	.00
80	.00	.00	.00	9.02	11.48	4.78	3.94	.00	3.31	.00	.00	.00	1.74	.00	.00	.00	.00	.00	2.27	2.91	3.24	2.96	.00	.00	.00
81	.00	.00	.00	8.58	11.23	11.25	4.33	.00	.00	.00	.00	1.81	1.80	.00	.00	.00	.00	.00	2.48	3.33	2.97	.00	.00	.00	.00
82	.00	.00	.00	.00	8.88	10.96	5.25	.00	.00	.00	.00	2.11	.00	.00	.00	.00	.00	2.65	3.22	3.37	3.09	.00	.00	4.32	.00
83	.00	.00	.00	.00	7.61	10.51	4.14	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	2.60	3.46	3.89	3.51	.00	6.74	4.58	.00
84	2.63	.00	.00	.00	7.32	9.37	3.74	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	2.81	3.63	4.09	3.57	.00	4.95	.00	.00
85	.00	.00	.00	.00	6.86	4.40	4.29	3.32	.00	.00	.00	.00	.00	.00	.00	.00	.00	3.00	3.86	4.30	3.64	.00	6.77	.00	.00
86	.00	.00	.00	6.54	8.28	4.00	3.75	2.83	.00	.00	.00	.00	.00	.00	.00	3.44	.00	3.20	4.28	4.70	3.82	.00	4.66	.00	.00
87	.00	.00	.00	6.57	8.45	3.13	2.89	2.17	.00	.00	.00	.00	.00	.00	.00	3.81	.00	3.90	6.89	5.16	3.95	.00	4.79	.00	.00
88	.00	.00	.00	6.64	2.13	1.92	1.98	1.63	.00	.00	.00	.00	.00	.00	.00	.00	.00	6.36	5.36	5.67	4.36	.00	.00	.00	.00
89	.00	.00	.00	6.56	1.80	1.80	1.80	1.64	.00	.00	.00	.00	.00	.00	.00	.00	3.90	6.91	5.15	5.60	4.32	4.51	.00	.00	.00
90	.00	.00	5.00	1.31	1.60	1.58	1.34	1.20	.00	.00	.00	.00	.00	.00	.00	3.08	4.26	4.62	4.98	5.30	4.50	.00	.00	.00	.00
91	.00	.00	.00	1.25	1.96	1.51	1.50	1.22	.00	.00	.00	.00	.00	.00	.00	2.70	3.39	3.20	4.27	4.12	.00	.00	.00	.00	.00
92	.00	.00	.00	6.07	1.47	1.52	1.50	1.22	.00	.00	.00	.00	.00	.00	2.27	3.06	3.52	3.71	4.08	3.64	.00	.00	.00	.00	.00
93	.00	.00	.00	4.27	1.54	1.55	1.55	1.24	.00	.00	.00	.00	.00	2.01	2.25	2.57	3.69	4.24	4.32	.00	.00	.00	.00	.00	.00
94	.00	.00	.00	5.53	1.83	1.79	1.74	1.41	.00	.00	.00	.00	.00	2.01	2.94	.00	3.91	5.69	5.99	4.97	.00	.00	.00	.00	.00
95	.00	.00	6.76	2.02	1.85	1.70	1.68	1.40	.00	.00	.00	.00	2.16	2.35	.00	3.64	4.26	6.18	6.50	5.47	.00	5.51	.00	.00	.00
96	.00	.00	8.07	2.05	2.03	1.89	1.39	.00	.00	.00	.00	.00	.00	.00	3.09	3.38	.00	4.91	6.52	.00	5.51	.00	.00	.00	.00
97	.00	.00	8.54	1.99	3.09	2.07	1.55	.00	.00	.00	.00	.00	.00	.00	3.37	.00	4.46	6.11	6.66	6.27	5.05	.00	.00	.00	.00

Figure 7 A portion of bench 1126 showing the result of the calculation of the diluted G9110 block model. See Figure 6 for the unadjusted Zinc grades.

Table 15

GRUM DEPOSIT – IN-SITU MINERAL INVENTORY WITHIN IV STAGE III PIT LIMIT, G9110 MODEL

in-place mineralization / no dilution / no mining recovery consideration

	MINERAL INVENTORY									CONTAINED METAL			
	Cutoff (wt%) Pb+Zn	Volume (cu. m.) *1000	Density (t/m3)	Tonnage (tonnes) *1000	Pb+Zn (wt %)	Pb (wt %)	Zn (wt %)	Ag (g/t)	Au (g/t)	lead tonnes *1000	zinc tonnes *1000	silver kg	gold kg
GEOLOGICAL COMPOSITES	8	2,791.59	3.799	10,604.09	11.77	4.49	7.28	74.2	1.06	475.593	771.978	786,930	11,187
G9110 MODEL–GEOL (2m)	6	4,652.67	3.541	16,476.35	10.04	3.80	6.24	63.1	0.94	625.772	1,028.783	1,039,822	15,537
Total Model (52W–88W)	5	5,947.05	3.430	20,401.23	9.17	3.45	5.72	57.6	0.88	703.842	1,166.134	1,175,927	17,973
all horizons	4	6,939.07	3.376	23,423.28	8.57	3.22	5.35	54.0	0.85	754.698	1,252.209	1,265,560	19,840
	3	7,595.00	3.356	25,488.40	8.16	3.07	5.09	51.7	0.83	783.513	1,296.595	1,317,240	21,206
	0	8,494.97	3.338	28,358.74	7.52	2.83	4.69	48.0	0.81	802.552	1,330.592	1,362,354	22,999

Table 16

Minable Reserve Grum Deposit I.V. Pit
From diluted G9110 geological composite model

	Total Material (tonnes*1000)	Waste (tonnes*1000)	High Grade +5% (tonnes*1000)	Zn (%)	Pb (%)	Ag (g/t)	Low Grade 3–5% (tonnes*1000)	Zn (%)	Pb (%)	Ag (g/t)	Total Ore +3% (tonnes*1000)	Zn (%)	Pb (%)	Ag (g/t)
Phase I	69,778	61,923	6,388	5.47	3.27	53.3	1,468	2.69	1.48	27.2	7,856	4.95	2.94	48.4
Phase II	84,499	73,086	8,095	5.28	3.24	54.6	3,318	2.56	1.53	27.0	11,413	4.49	2.74	46.6
Phase III	38,903	33,408	3,433	5.05	2.95	50.2	2,062	2.45	1.54	26.2	5,495	4.07	2.42	41.2
Total I.V. Pit Design	**193,180**	**168,417**	17,916	5.30	3.20	53.3	6,848	2.55	1.52	26.8	24,764	4.54	2.73	46.0

7.27

86.4%

Economic analysis shows that the mineralization recovered from the increment is not economic if the time value of the waste stripping is included, thus mine planning has continued to follow the I.V. Pit.

ADDITIONAL POTENTIAL

Additional potential open pit minable mineralization which could be found in the Champ zone and deeper in the Grum structure has already been quantified in the A.B. pit.

Below the A.B. pit, in the Champ zone, there is little high grade ore, however, below the A.B. pit in the area between 61W and 87W, there is considerable high grade mineralization; approximately 3.961 million tonnes of in-situ mineralization, averaging 6.61% Zn, 4.25% Pb, 70.1 g/t Au and 1.1 g/t Au, above an 8% Pb + Zn cutoff grade.

This material has not been assessed for continuity and may not be minable, however, using the recovery and dilution assumptions used for the Dy deposit (65% recovery, 10% dilution, by material at 7.5% Pb + Zn) and experienced in the Faro underground, it is concluded that 2.8 million tonnes averaging 6.4% Zn, 4.1% Pb, 67 g/t Ag and 1 g/t Au, may be minable. More drilling will be needed to define this possible mineralization as the pit is developed.

The G9110 model does not quantify the entire Grum mineral deposit, only that portion most densely drilled from section 51 W to 88 W. The Grum mineralized structure is known from reconnaissance drilling to extend much further northwest than section 88 W; it almost certainly connects to the Firth zone at 124 W to 128 W. There is little drilling in this area, referred to as the Northwest Extension, but one definitive section has been completed at 108 W. The four holes on 108 W indicate that ore grade and thickness mineralization may not extend that far northwest. There are only eight holes in the 600m interval from 88 W to 108 W. Three of these holes are northeast of the extension of the mineralized structure. Four of the holes are on trend with the structure and have intersected some ore grade sulphides but at least three of them are too shallow to have hit the best grade massive sulphides. The remaining hole, and the only one deep enough to intersect the main ore structure , is on section 92W (hole A18). This hole intersected several high grade bands from 311 m. to 426 m. including 6.6m of 15.4% Pb+Zn, 2.7m of 13.6% Pb+Zn, 3.0m of 14.5% Pb+Zn and 5.3m of 10.0% Pb+Zn. These results suggest that there is considerable potential for the ore structure to extend at least as far as 102W. An area of approximately 200,000 sq.m. is likely to be underlain by sulphides which if 6 m. thick on average could amount to approximately five million tonnes additional high grade ore. It is reasonable to assume that the grade will be approximately the same as the material below the A.B. Pit or 6% Zn, 4% Pb, 65 g/t Ag and 1 g/t Au.

Additional drilling is needed in this area to evaluate the feasibility of detailed underground exploration and eventual production. Fifteen holes each 475m deep would be needed to evaluate this area; this would amount to 7,125m (23,400 ft) of drilling. Access could be gained by ramp from the later stages of the Grum pit, in similar fashion to the Faro underground mine.

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APPENDIX A

G9110 MODEL

PROPERTY DEFINITION,

ROCK CODE LISTING

AND

SURFACE GRID LISTING

Curragh Resources Inc. : Whitehorse Office : Property Definition Listing:
93/05/31 : 10:58:54 Page
GEMCOM Services MOD11: DB=E:\PMOBG4 :
: = 1

Model Description (max 64 characters) : GRUM 9110A - GEOLOGY COMPOSITES - (Pigage / Reed Interp.)
Easting coordinate of model bottom left hand corner : 2234.37
Northing coordinate of model bottom left hand corner : 5419.88
Easting coordinate of model top right hand corner : 3202.11
Northing coordinate of model top right hand corner : 7079.90
Model rotation : 0.00
Datum elevation of top of model : 1336.00
Number of columns in model (max 128) : 127
Number of rows in model (max 128) : 219
Width of columns : 7.62
Height of rows : 7.58

Number of labels : 5 ; %Pb+Zn ; %Pb ; %Zn ; g/tnAG ; g/tnAu ;

Current Units are :

Linear : m
Area : m**2
Volumetric : bcm
Density : tn/bcm
Monetary : Cdn$

BENCH	HEIGHT [m]	CREST ELEVATION [m]	TOE ELEVATION [m]	CREST DEPTH [m]	TOE DEPTH [m]	Description	Bottom Grid #
1	6.00	1336.00	1330.00	0.00	6.00	1330	0
2	6.00	1330.00	1324.00	6.00	12.00	1324	0
3	6.00	1324.00	1318.00	12.00	18.00	1318	0
4	6.00	1318.00	1312.00	18.00	24.00	1312	0
5	6.00	1312.00	1306.00	24.00	30.00	1306	0
6	6.00	1306.00	1300.00	30.00	36.00	1300	0
7	6.00	1300.00	1294.00	36.00	42.00	1294	0
8	6.00	1294.00	1288.00	42.00	48.00	1288	0
9	6.00	1288.00	1282.00	48.00	54.00	1282	0
10	6.00	1282.00	1276.00	54.00	60.00	1276	0
11	6.00	1276.00	1270.00	60.00	66.00	1270	0
12	6.00	1270.00	1264.00	66.00	72.00	1264	0
13	6.00	1264.00	1258.00	72.00	78.00	1258	0
14	6.00	1258.00	1252.00	78.00	84.00	1252	0
15	6.00	1252.00	1246.00	84.00	90.00	1246	0
16	6.00	1246.00	1240.00	90.00	96.00	1240	0
17	6.00	1240.00	1234.00	96.00	102.00	1234	0
18	6.00	1234.00	1228.00	102.00	108.00	1228	0
19	6.00	1228.00	1222.00	108.00	114.00	1222	0
20	6.00	1222.00	1216.00	114.00	120.00	1216	0
21	6.00	1216.00	1210.00	120.00	126.00	1210	0
22	6.00	1210.00	1204.00	126.00	132.00	1204	0
23	6.00	1204.00	1198.00	132.00	138.00	1198	0
24	6.00	1198.00	1192.00	138.00	144.00	1192	0
25	6.00	1192.00	1186.00	144.00	150.00	1186	0
26	6.00	1186.00	1180.00	150.00	156.00	1180	0

27	6.00	1180.00	1174.00	156.00	162.00	1174	0
28	6.00	1174.00	1168.00	162.00	168.00	1168	0
29	6.00	1168.00	1162.00	168.00	174.00	1162	0
30	6.00	1162.00	1156.00	174.00	180.00	1156	0
31	6.00	1156.00	1150.00	180.00	186.00	1150	0
32	6.00	1150.00	1144.00	186.00	192.00	1144	0
33	6.00	1144.00	1138.00	192.00	198.00	1138	0
34	6.00	1138.00	1132.00	198.00	204.00	1132	0
35	6.00	1132.00	1126.00	204.00	210.00	1126	0
36	6.00	1126.00	1120.00	210.00	216.00	1120	0
37	6.00	1120.00	1114.00	216.00	222.00	1114	0
38	6.00	1114.00	1108.00	222.00	228.00	1108	0
39	6.00	1108.00	1102.00	228.00	234.00	1102	0
40	6.00	1102.00	1096.00	234.00	240.00	1096	0
41	6.00	1096.00	1090.00	240.00	246.00	1090	0
42	6.00	1090.00	1084.00	246.00	252.00	1084	0
43	6.00	1084.00	1078.00	252.00	258.00	1078	0
44	6.00	1078.00	1072.00	258.00	264.00	1072	0
45	6.00	1072.00	1066.00	264.00	270.00	1066	0
46	6.00	1066.00	1060.00	270.00	276.00	1060	0
47	6.00	1060.00	1054.00	276.00	282.00	1054	0
48	6.00	1054.00	1048.00	282.00	288.00	1048	0
49	6.00	1048.00	1042.00	288.00	294.00	1042	0
50	6.00	1042.00	1036.00	294.00	300.00	1036	0
51	6.00	1036.00	1030.00	300.00	306.00	1030	0
52	6.00	1030.00	1024.00	306.00	312.00	1024	0
53	6.00	1024.00	1018.00	312.00	318.00	1018	0
54	6.00	1018.00	1012.00	318.00	324.00	1012	0
55	6.00	1012.00	1006.00	324.00	330.00	1006	0
56	6.00	1006.00	1000.00	330.00	336.00	1000	0
57	6.00	1000.00	994.00	336.00	342.00	994	0
58	6.00	994.00	988.00	342.00	348.00	988	0
59	6.00	988.00	982.00	348.00	354.00	982	0
60	6.00	982.00	976.00	354.00	360.00	976	0
61	6.00	976.00	970.00	360.00	366.00	970	0
62	6.00	970.00	964.00	366.00	372.00	964	0
63	6.00	964.00	958.00	372.00	378.00	958	0
64	6.00	958.00	952.00	378.00	384.00	952	0
65	6.00	952.00	946.00	384.00	390.00	946	0
66	6.00	946.00	940.00	390.00	396.00	940	0
67	6.00	940.00	934.00	396.00	402.00	936	0
68	6.00	934.00	928.00	402.00	408.00	928	0
69	6.00	928.00	922.00	408.00	414.00	922	0
70	6.00	922.00	916.00	414.00	420.00	916	0

APPENDIX B

CURRENT LITHOLOGIC CODE IN USE AT GRUM WITH CROSS REFERENCE TO PREVIOUS CYPRUS ANVIL LITHOSTRATIGRAPHIC CODE

CURRAGH RESOURCES INC. – NUMERIC ANVIL LITHOSTRATIGRAPHIC CODE

ROCK CODES (OLD CODES INCLUDED FOR COMPARISON)

DISSEMINATED QUARTZITES		
2	4A	Ribbon banded carbonaceous quartzite
3	4C/4D	Pyritic quartzite (<30% pyrite)
SEMI MASSIVE SULPHIDE (Generally low grade)		
4	4EC/4E1/ 4C3	Siliceous pyritic sulphides (30–60% pyrite, generally <4% Pb+Zn)
MASSIVE SULPHIDES		
5	4E/4F	Massive pyritic sulphides (60–100% pyrite)
6	4K	Massive pyritic sulphide with clasts of dolomite/ankerite
7	4G	Baritic pyrite sulphides (> 10% barite)
8	4H	Pyrrhotitic sulphides
9	4J	Nonpyritic sulphides & oxides – pyrite poor
METASEDIMENTS		
20	3G	Noncalcareous, muscovite–chlorite, medium grey phyllite
22	1C/1CD/1D	Noncalcareous, bio–musc–qtz staurolite+andalusite+garnet+fibrolite schist
30	5A/5G/3E/1E	Carbonaceous phyllite/schist
32	5E/3F/1G	Marble + calc–silicate bands
33	1B	Skarn and "silicated" marble
36	3D	Calc–silicate
40	5B	Calcareous, silvery grey, muscovite chlorite phyllite
METAIGNEOUS		
44	5C/3C/1F	Metabasite, poorly foliated greenstone (relict igneous texture)
45	5C/3C/1F	Pyroxenite – commonly serpentinized (relict bastites)
46	5C/3C/1F	Amphibolite – blue–green hornblende + plagioclase + quartz
47	5D/3B/1H	Chloritic phyllite/schist – pale green, homogenous
ALTERED ROCKS		
52	4LD	Muscovite>chlorite quartz phyllite/schist – light cream to white
54	4L6	Chlorite>muscovite quartz phyllite/schist – pale green
CRETACEOUS INTRUSIVES		
60	10Q	Quartz vein – white bull quartz
61	10AB	Anvil Batholith – Mt Mye phase of Anvil plutonic suite. Musc–bio granite
65	10C	Pegmatite
66	–	Aplite
68	10E	Hornblende–biotite quartz diorite – massive and unfoliated
69	10F	Smokey quartz–feldspar porphyry – massive and unfoliated
FAULT ROCK (use only if parent not recognized)		
72		Gouge
74		Tectonic breccia
76		Mylonite
OVERBURDEN		
82		Unclassified – general
84		Triconed – no recovery
86		Till – silt – sand
88		Ferricrete
99		Air
OTHER		
0		No Recovery
&		+/–

USAGE: ROCK CODE MINERAL IDENTIFIERS TEXTURE GRADE

(ie) 2ZGRXH (20#) (60ZG) 80:10:10 = 4A47 BXA (3G4) (10Q9) 80:10:10

(1) The most abundant rock type comes first.
(2) Mineral identifiers are used in order of abundance.
(3) The grade descriptor for 0 grade (N) may be omitted
(4) In general, characteristics which are normally found in a rock type should not be indicated by a mineral or textural identifier.
(5) Parentheses are used to separate subordinate rock types.
(6) Ratio of main and subordinate rock types follows entire code.
(7) Not all four parts of the rock code construction shown above are necessarily used. Rock number is mandatory. Grade modifier is mandatory for ore types.

GRADE MODIFIERS

N	no visible grade
W	1–3% Pb+Zn
L	3–5% Pb+Zn
H	5–10% Pb+Zn
V	>10% Pb+Zn

MINERAL IDENTIFIERS

Carbonates
- c calcite
- k ankerite
- v carbonate–non specific
- w dolomite

Micas
- b biotite
- j "fuchsite"
- l chlorite
- m muscovite
- s sericite
- t talc

Felspars – Quartz
- f feldspar
- q quartz (fine grained)
- y kaolinite (clay minerals)
- p potash feldspar
- Q quartz (vein)

Calc Silicates
- a actinolite
- e epidote
- h hornblende
- i diopside

Alumino–Silicates/Pelites
- d andalusite
- n garnet
- r fibrolite
- u staurolite
- z chloritoid

Oxide/Sulphide/Sulphates
- A Arsenopyrite
- B Barite
- C Chalcopyrite
- G Galena
- L Limonite (iron oxides)
- M Magnetite
- P Pyrite
- R Pyrrhotite
- Z Sphalerite
- F Marcasite

Other
- g carbon
- x noncalcareous

ROCK TEXTURES

+	equigranular
!	foliated
=	laminated/ribbon banded
>	coarse–grained
^	medium grained
<	fine grained
\	clotted
:	porphyroblastic
%	porphyritic
'	interstitial
@	porous
*	weathered
~	fault gouge
X	fault breccia (tectonic)
?	mylonite
#	altered
$	"stringered"
o	spotted