Grum Reserves 68608 model

020741

3.3 Grum Geology and Reserves

3.3.1 History

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

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

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

3.3.2 General Geology

3.3.2.1 Stratigraphy and lithology

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

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

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

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

3.3.2.2 Structure

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

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

3.3.2.3 Surfical Geology

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

3.3.2.4 Ore Deposit Geology

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

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

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

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

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

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

3.3.3 Drill Definition & Information Base

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

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

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

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

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

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

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

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

3.3.4 Methods and Procedure of Reserve Calculation

3.3.4.1 <u>Introduction</u>

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

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

3.3.4.2 Block Geology and Drillhole information

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

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

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

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

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

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

Table 3.15

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

3.3.4.3 Composite Calculation

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

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

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

Table 3.16

Maximum Permitted Assay Values and SG Values

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

Table 3.17

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

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

Table 3.18

Maximum Permitted Assay and SG
For DDH Composites

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

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

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

3.3.4.4 Variogram analysis:

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

3.3.4.5 Interpolation:

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

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

TWO DIMENSIONAL VARIOGRAM

EXTRACTION DATA USED :

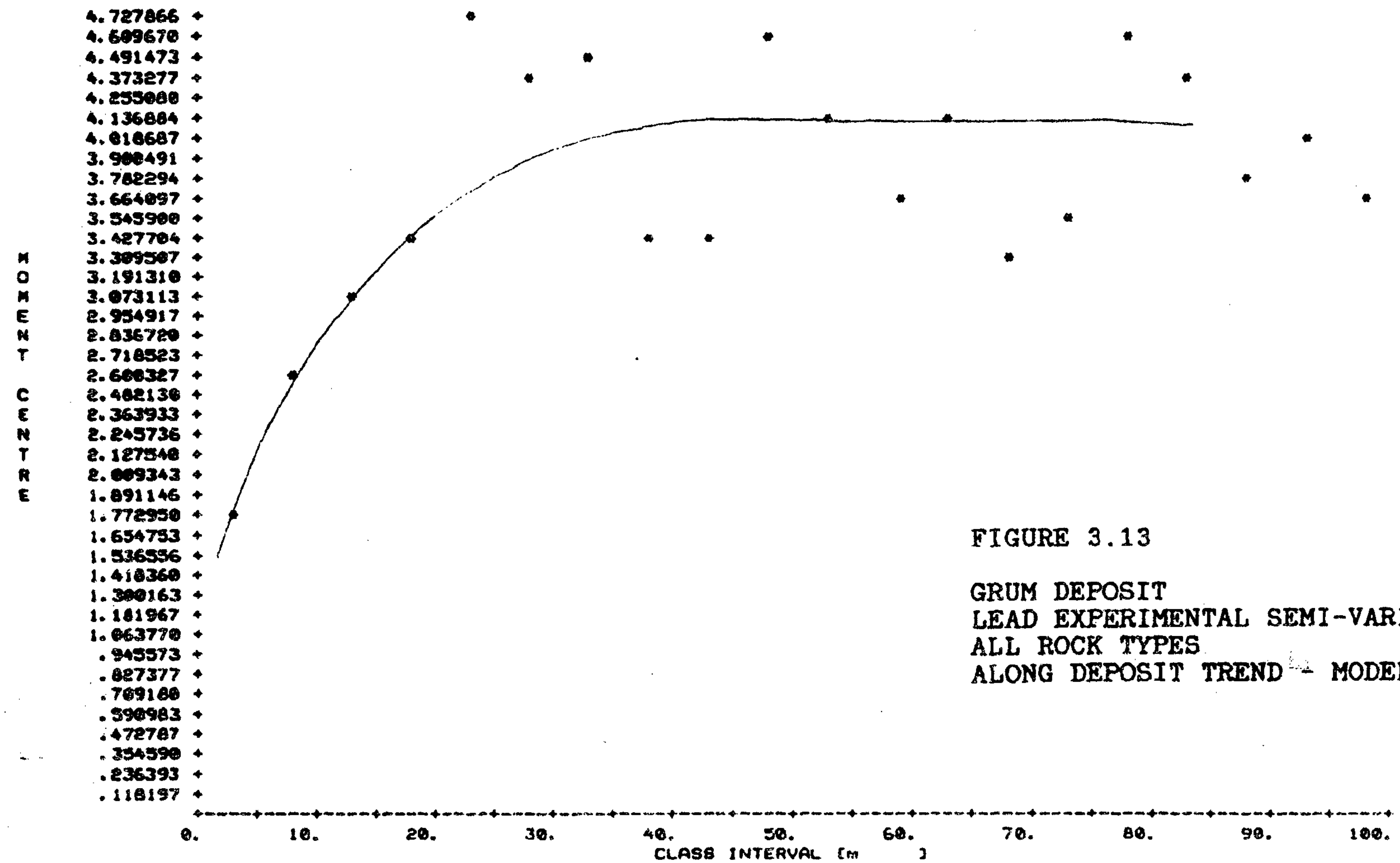

FIGURE 3.13

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

TWO DIMENSIONAL VARIOGRAM

EXTRACTION DATA USED :

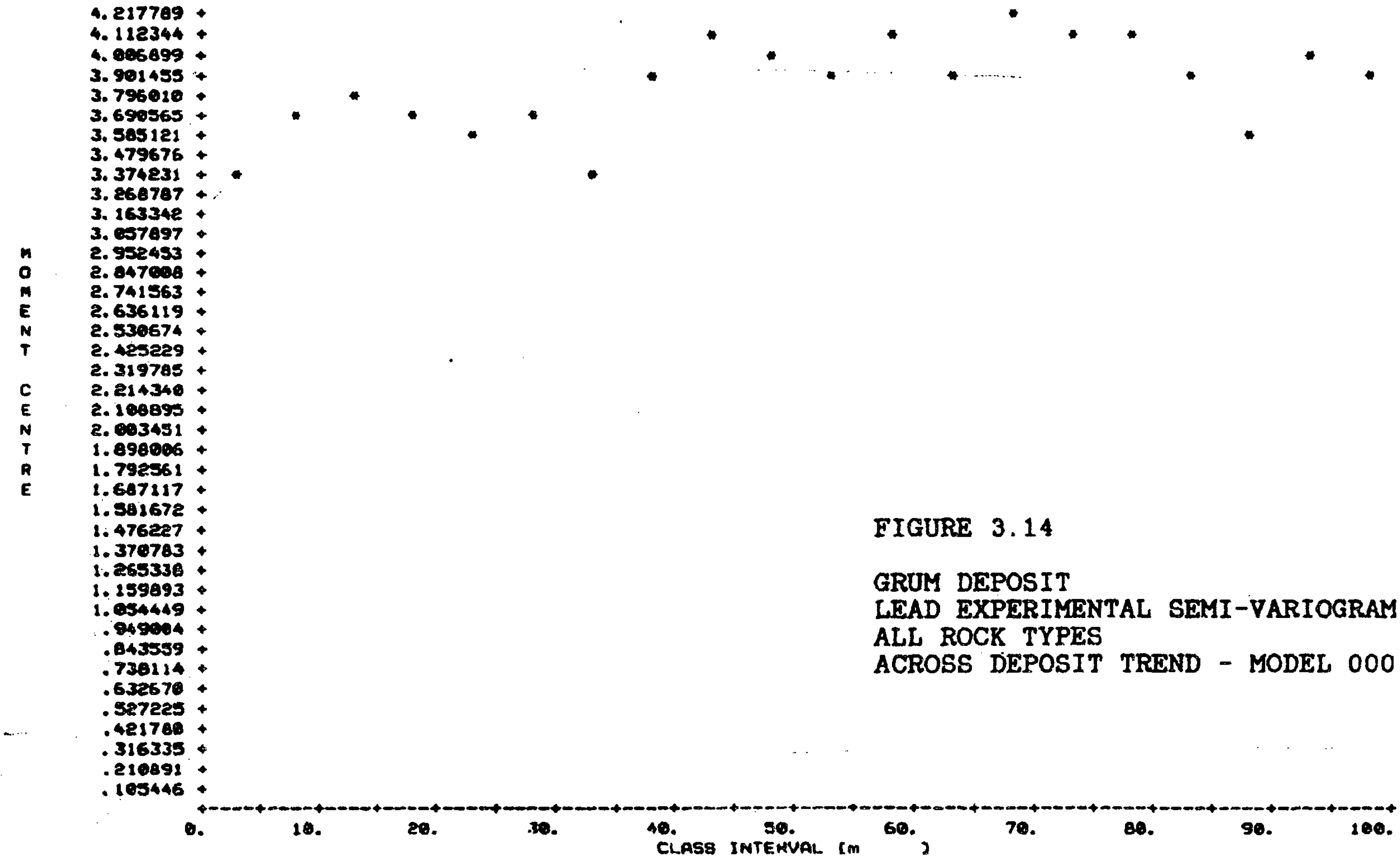

FIGURE 3.14

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

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

DOWN-THE-HOLE VARIOGRAM - SAMPLE DATA

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

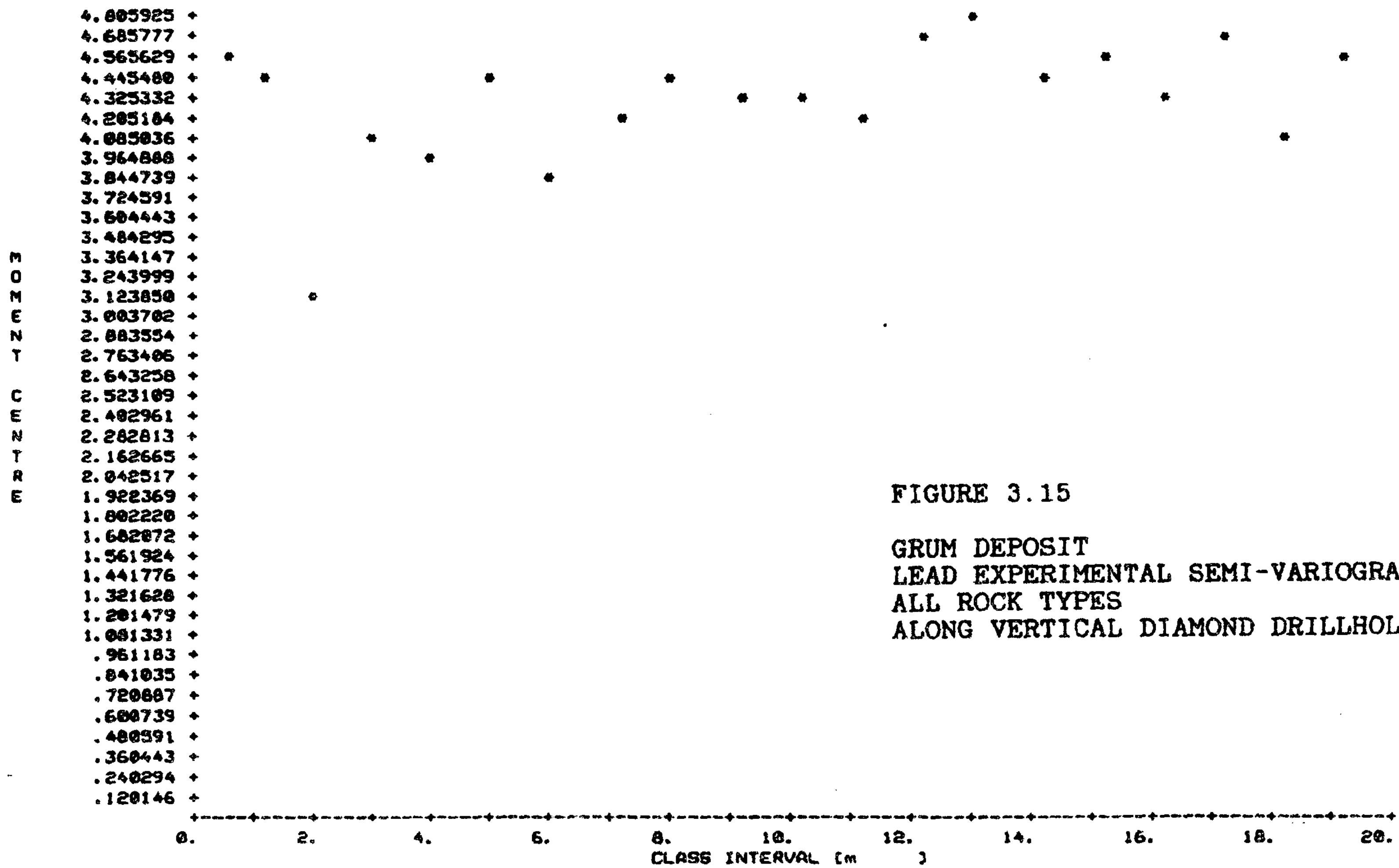

FIGURE 3.15

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

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

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

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

3.3.4.6 Geological Reserve Reporting

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

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

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

3.3.5 Results

3.3.5.1 Geological Reserves

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

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

3.3.5.2 Model to Model Comparisons

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

TABLE 3.19

GG606 MODEL GEOLOGICAL RESERVES FOR THE TWO CONSTITUENT MODELS
AND FOR THE ENTIRE DEPOSIT Sects 62W - 86W (i.e. 61W-87W)

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

NOTES:
* Part of this difference in tonnage is due to uninterpolated blocks. There are 707 such blocks in under grum and 316 in above grum, not all of this material will be ore however. Each block represents about 1900 tonnes of material thus there is about 2,000,000 tonnes of un accounted for sulphides.
x The large variance in volume and specific gravity between the two calculations is explained in the text and the notes for Tables 3.22 B and D.

TABLE 3.20

GEOLOGICAL RESERVES BY CROSS SECTION FROM 68606 COMPUTER MODEL, 1986
TOTAL DEPOSIT, 4% Pb + Zn CUTOFF GRADE

SECTION	VOLUME (bcm)	SPECIFIC GRAVITY	TONNAGE (tonnes)	LEAD (%)	ZINC (%)	SILVER (g/t)	GOLD (g/t)	Pb+Zn (%)	TOTAL METAL (tonnes)
86 W	684,180	3.09	2,116,210	2.60	4.99	46.0	0.44	7.59	160,592
84 W	584,820	3.16	1,849,610	2.80	4.90	47.6	0.60	7.70	142,432
82 W	787,860	3.40	2,680,540	3.19	5.55	56.5	0.84	8.73	234,069
80 W	1,297,080	3.35	4,340,270	3.10	4.85	51.8	1.05	7.95	345,081
78 W	975,780	3.27	3,195,500	3.16	5.50	54.7	1.04	8.66	276,762
76 W	907,200	3.36	3,048,860	3.49	5.80	58.6	1.07	9.29	283,135
74 W	1,013,040	3.45	3,498,640	3.72	6.20	62.3	1.05	9.92	347,148
72 W	748,440	3.45	2,584,690	3.64	5.77	61.2	1.02	9.41	243,256
70 W	694,980	3.52	2,449,940	3.86	6.57	63.6	1.04	10.43	255,415
68 W	355,320	3.70	1,314,260	4.10	6.27	66.7	1.00	10.37	136,297
66 W	442,260	3.69	1,632,190	4.14	5.75	66.7	0.99	9.90	161,507
64 W	301,860	3.61	1,091,070	3.79	5.52	61.2	0.99	9.30	101,466
62 W	178,200	3.69	657,640	3.39	5.03	54.8	1.10	8.42	55,347
	8,971,020	3.40	30,459,420	3.41	5.60	57.3	0.95	9.00	2,742,527

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

deposit and sectional basis respectively.

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

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

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

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

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

TABLE 3.21

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

	CALCULATION		ORE (tonnes)	Pb (%)	Zn (%)	Pb+Zn (%)	Ag (g/t)	TOTAL METAL (tonnes)
A	KERR ADDISON - 1977 sectional hand calculation		26,083,000	4.1	6.4	10.5	62	2,738,715
B	KERR ADDISON / NORANDA - 1978 computer block model		27,650,000	3.1	4.9	8.0	48	2,212,000
C	CYPRUS ANVIL - 1981 "81" computer block model		30,781,000	3.1	4.9	8.0	49	2,462,480
D	CYPRUS ANVIL / DOME -1983 sectional hand calculation (uncorrected, see table 3.22)	*	32,611,000	3.5	5.7	9.2	59	3,000,212
D	CYPRUS ANVIL / DOME -1983 sectional hand calculation (corrected, see table 3.22)	*	31,615,000	3.5	5.7	9.2	59	2,918,065
E	CURRAGH RESOURCES - 1986 "BB608" computer block model	*	30,649,000	3.4	5.6	9.0	57	2,758,410

ρ = 3.40

9000 → 9650 N
6510 → 7290 E

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

8705 (61w-87w) 32.18

8.98

ρ = 3.57

TABLE 3.22 A

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

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

TABLE 3.22 B

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

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

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

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

TABLE 3.22 C

PERCENT VARIANCE OF G8608 (TABLE 3.20) TO KERR ADDISON (TABLE 3.22 A) [(NEW-OLD)/OLD]
TOTAL DEPOSIT, 4% Pb + Zn CUTOFF GRADE

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

TABLE 3.22 D

PERCENT VARIANCE OF G8608 (TABLE 3.20) TO SIMPSON -ADAMSON/DOME (TABLE 3.22 B) [(NEW-OLD)/OLD]
TOTAL DEPOSIT, 4% Pb + Zn CUTOFF GRADE

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

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

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

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

Table 3.15

UNINTERPOLATED BLOCKS IN G8608
GRUM COMPUTER MODEL

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

3.3.5.3 Reliability of Reserves

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

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

Based on the reproducability of geological reserve calculations for the Grum deposit and the central position of the current model in the range of values it is concluded that the geological reserves presented herein for the entire deposit are reliable within ± 10%, as such can be considered proven. Local reserves or bench reserves on the other hand can not be as reliable. In the vicinity of the dense drill pattern around the underground workings the local reserves are probably reasonably accurate however towards the periphery there is considerably more possible variance.

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

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

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

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

3.3.6 Additional Work needed

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

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

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

One of the major inadequacies of geologic data at Grum is geotechnical data. There is reason to believe that the S_2 foliation under Doal Lake (in the northeast part of the proposed pit) may be dipping northeast rather than southwest. This could have significant implications for slope stability and the size of the area required to be pre-stripped. A program of oriented core drilling is required to evaluate this question. These holes should also sample the overburden to evaluate its clay content and hydogeology. If the overburden is amendable to dewatering and can be kept dry then steeper slopes will be possible than in the current water saturated state.

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

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

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

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