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Introduction

The Anvil Range Pb-Zn-Ag District is located in the central Yukon Territory near the town of Faro (figure 2.1). The district contains one of the world's largest reserves of lead and zinc in several deposits (figure 3.1) including the recently re-opened Faro mine. This ore can support mining activities in the Anvil District for many years following the exhaustion of the Faro deposit. Most of these deposits are within economical haulage distance to the Faro concentrator (figure 2.3 and 3.2).

Regional Geology

The Anvil District is part of the Selwyn Basin (figure 3.4), a large area of central Yukon where deep water shales accumulated along the ancient North American continental margin during the Paleozoic. The shales of the Selwyn Basin host most of Canada's large stratiform lead-zinc deposits, making it a metalliferous province of world wide significance.

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

District Stratigraphy

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

The lower division ranges in age from late Precambrian to perhaps Early Silurian. It is approximately 5 km thick and divisible into three major mappable units (fig. 3.5). From the base these are non-calcareous metapelite of Mt. Mye formation, calcareous metapelite of Vangorda formation and basalt and black phyllite of Menzie Creek formation. Established formal stratigraphic nomenclature does not apply directly to this area or interval but the rocks are very similar to those of Kechika Group (Gordey, 1981) south of the district in Pelly Mountains.

The upper division includes rocks ranging in age from Devonian to Permian. In contrast to the lower division, the upper division is characteristically cherty and conspicuously coarsely clastic. All or part of the upper division may be allochthonous with respect to the lower.

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

the scope of this summary.

The Mt. Mye formation varies from non-calcareous, biotite-muscovite schist to non-calcareous, weakly carbonaceous, light to medium gray muscovite-chlorite phyllites with lesser, interlayered, black graphitic phyllite, marble, calc-silicate phyllite or schist, metabasite and psammitic schist. The unit is at least 2 kilometers thick but its base is not exposed in the district.

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

The Vangorda formation is characterized by light to medium-gray, calcareous, phyllitic rocks made up of very thin (0.1-2 cm) interlayers of a) medium grey, non-calcareous, weakly carbonaceous, muscovite-chlorite pelite and b) light grey, generally calcareous quartz \pm calcite \pm dolomite siltstone. Other rock types interbedded with the calcareous phyllite include metabasite and meta-tuffs, graphitic phyllite, and phyllitic limestone.

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

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

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

The Menzie Creek formation is a unit of basaltic metavolcanic rocks consisting of pillowed and massive flows with comparable amounts of massive, coarse, monolithic breccias and lesser, thin-bedded tuff and/or volcanic sandstone and siltstone. Whole rock major element and trace element data (Jennings et al., 1980) imply that the flows of the Menzie Creek volcanic unit are dominantly alkali basalt erupted in a within-plate setting

similar to metabasites of Vangorda formation. Carbonaceous phyllite and brown siltstone interbeds northeast of the Anvil Batholith contain graptolites of middle Ordovician or lower Silurian age (Tempelman-Kluit, 1972) suggesting correlation with the widespread Road River Formation black shale and chert to the northeast. The Menzie Creek formation varies from zero to about 1.5 kilometers in thickness in and near the district. It has been traced for 100 kilometers along strike and 30 kilometers across strike, showing that it is one of the largest of several basaltic units of its age in and around the Selwyn Basin.

Relation of Stratigraphy to ore deposits

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

Deformation, Metamorphism and Plutonism

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

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

During the second event (D_2), S_1 was strongly crenulated and ubiquitous close to tight mesoscopic folds in S_1 were produced (figure 3.6). The largest megascopic folds known to have been formed during D_2 are those at the Grum Deposit (Fig. 3.9) and comparable folds in the Swim Deposit (Fig. 3.13). Parallel to the axial planes of these D_2 folds is a crenulation cleavage (S_2) which imparts a well developed lithon structure to most rocks of the district, especially the strongly banded phyllites of the Vangorda formation. F_2 axial planes and S_2 dip shallowly, with axes subparallel F_1 axes. Three later, less intense periods of folding and associated faulting followed.

The later events (D_3 through D_5) generally produced open folds and weak crenulations in S_2 related to broad, regional structures. An important exception to this general rule is found in the vicinity of the Faro deposit where the fourth event (D_4) is quite intense, with tight mesoscopic folds developed in nearly pervasive S_2 with appreciable mica growth along S_4 (see

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

During the later stages of this deformation history a large granitic body (Anvil Batholith) was intruded into the metamorphic sequence.

Anvil Batholith ranges in composition from granodiorite to quartz monzonite and textures include equigranular massive, megacrystic massive and various strongly to weakly foliated variants. Several K/Ar ages on the granitic rocks yield ages of 85-100 ma (Tempelman-Kluit, 1972). Intrusion of the Anvil Batholith further deformed the metamorphic sequence so that the overall structure of the district is an elongate dome cored by the Batholith (Figure 3.2). In the later stages of batholith emplacement large extensional fault displacements occurred along its margins. These faults determine the present day limits of several of the deposits (Fig. 3.10, 3.12).

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

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

Ore Deposits

General Description

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

These sulphide sheets, or horizons, have since been deformed into complex fold structures. The deposits are thus elongate parallel to the fold axes and associated lineations in the host metasediments. The Faro deposit, which appears to be an

exception to this generalization, actually shows great internal complexity in the geometry of high grade and waste layers.

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

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

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

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

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

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

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

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

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

Description of Sulphide Lithofacies

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

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

Banded to homogenous, usually weakly foliated and/or lineated, massive pyrite with lesser sphalerite and galena. Total sulphide content is at least 60%, generally greater than 80% and commonly nearly 100%. Gangue consists of quartz and/or barite and/or carbonates (calcite, dolomite, ankerite). Accessory minerals include pyrrhotite, chalcopyrite, magnetite, arsenopyrite and marcasite. At amphibolite facies metamorphic grade, this rock type commonly develops a buckshot porphyroblastic texture of pyrite in a matrix of dark reddish brown to black base metal sulphides. This texture usually is restricted to rocks with economic lead-zinc grades (Unit 4F/2F).

Baritic, Massive Pyritic Sulphides: (Unit 2G/4G) strongly and thinly banded massive sulphide/sulphate rock consisting of pyrite, galena, sphalerite and commonly magnetic in a gangue of off-white barite and lesser carbonates (calcite, dolomite, ankerite and probably barytocalcite). The amount of barite may be as high as 50%; non-sulfidic, massive barite does not occur in the Anvil deposits. There is a complete gradation between this and the above facies with 10% visible barite by volume being the dividing line. This facies is usually quite high grade (10-15% combined lead-zinc). Sphalerite is characteristically honey coloured to reddish brown. Pyrrhotite is not commonly seen in the baritic facies except in the Faro deposit where overall pyrrhotite is more abundant.

Carbonate-bearing, Massive Pyritic Sulphides: (Unit 2K/4K) similar to massive pyritic sulphides but contains greater than 10% carbonate (calcite, dolomite, ankerite) either as interstitial gangue or as coarse patches and irregular blebs. This is a minor facies and is not known with certainty to always be an original composition variant. The most common occurrence of coarse pinkish beige to tan, ankerite patches may represent recrystallized original carbonate or re-worked

pre/syn-metamorphic veins.

Pyrrhotitic Massive Sulphides: (Unit 2H/4H) massive, finely crystalline, usually well foliated pyrrhotite with less than 50% pyrite porphyroblasts and highly variable amounts of shalerite and galena. Minor chalcopyrite is characteristic of this relatively copper-rich facies. Rounded to angular, rotated, foliated quartzite or quartz-vein clasts 2 cm or less in diameter are typical. This is a minor facies and is not known with certainty to be primary as some pyrite in the massive facies may invert to pyrrhotite during regional metamorphism. At Faro the pyrrhotitic facies is more volumetrically important than the other deposits. Pyrrhotite rich ores are generally much finer grained than non pyrrhotitic ores at Faro.

General Comments on Massive Sulphides

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

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

Quartzose Disseminated Lithofacies

Ribbon banded, "graphitic", pyritic quartzite: (Unit 2A/4A) dark grey to black, well banded, sulphide-bearing quartzite (metamorphic usage). Bands are: (a) dark grey, very fine grained carbonaceous phyllitic quartzite to siliceous phyllite (presumed metachert) and (b) light grey, quartz-sulphide (pyrite-sphalerite-galena) bands. These bands are usually 2 mm to 2 cm thick with a total sulphide content usually between 10 to 30% but ranging from 2% to 60%. Pyrite is usually the dominant species but higher grade examples have sub-equal pyritic and base metal sulphides. Base-metal dominant variants with little pyrite occur but are not common unless total sulphide content is low. Strong sulphide species differentiation between bands, such that barren pyrite bands are adjacent to or near sphalerite or galena rich bands, occurs but is not generally the case.

Pyritic quartzite: (Units 4B, C, D/2B, C, D) light to medium grey, generally poorly banded, moderately to weakly foliated, micaceous quartzites with highly variable base metal and pyrite contents. Pyrite contents are generally 10% to 40% ranging between 2 and 60%. Although there is a complete gradation from massive to quartzose ores there is usually little problem in separating this facies from the massive pyritic sulphides as the vast majority of examples have less than 40% total sulphides. A minor variant of this facies (unit 2B/4B) shows low pyrite (< 5%) content with base metal sulphides predominant. Barite in major amounts is uncommon in the quartzose this facies; carbonate species are not typical but locally are abundant. Chalcopyrite, pyrrhotite and magnetite-bearing varieties are common. Sphalerite in the high grade examples is characteristically a vibrant reddish brown. At Faro the more sulphide rich variants of this facies are well developed along the northeast edge of zone 3. They are spectacularly barren but contain elevated copper contents and are rich in magnetite. A similar facies is developed at Vangorda and locally at Grum where the rocks are also quite gold rich and more clearly in the deposit footwall.

General Comments on disseminated ore types

Post-metamorphic breccias are also common in the disseminated sulphide lithofacies. Pyritic quartzite breccias are often well developed in the sphalerite-rich high grade facies where again ductility contrasts between the sulphides and quartzite bands dictate ductile flow in the sulphides and brittle failure, rotation and brecciation on the quartzite. Where less intensively developed the breccias grade into examples of sulphide mobilization into D_2 or later cleavages.

Alteration

Both wallrocks and certain ore facies of the Anvil deposits are overprinted by a prominent, easily recognized, light beige, white mica dominant alteration assemblage (Unit 4L). This overprint facies is not a depositional unit and may form as a reaction product between wallrocks and deposit forming hydrothermal fluids, or as a metamorphic reaction envelope unrelated to ore forming fluids or as combination of these processes. In the multi-layered deposits, this alteration overprint appears discontinuous and often best developed in the footwall of a given lens or deposit as a whole. At Faro, a continuous envelope of this lithology encloses the entire deposit with local (especially Zone 1) best development in the hanging wall.

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

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

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

Idealized Anvil Deposit

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

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

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

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

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

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

Genetic Model

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

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

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

to realize that alternative interpretations of the deposits are possible. An example would be the possibility that all or part of the footwall siliceous facies are silicified and sulfidized replacements of host sediments rather than exhalative cherty sediments.

Reconnaissance studies by Kuo (1976) demonstrate the presence of chloride-rich fluid inclusions in barite, quartz, and sphalerite of several deposits perhaps implying metalliferous brines played a role in deposit formation. The ubiquitous development of generally foot wall biased envelopes further suggests these brines were relatively hot. The curvilinear deposit array may further indicate control of sedimentary facies and hence basin geometry by, a synsedimentary fault, or fault bundle, which could have provided the locus of exhalative ore fluid migration into the basin(s).

The ore deposits are thought to have formed from hot metalliferous brines discharged from submarine fumaroles localized along a synsedimentary fault or hinge line which developed in response to late Hadrynian or lower Cambrian extensional tectonism. This tectonism influenced basinal geometry resulting in reduced second order basinal facies truncating against the hinge line. Hydrothermal fluids moved up this fault zone and exhaled into a relatively deep water reduced marine basin which was receiving distal turbidite sedimentation. Sulphides may have been deposited from plumes along the hinge line or from relatively dense exhaled brines ponded in local topographic depressions near the hinge line. This model accounts for the crude associations of known deposits with apparent depositional limits of reduced sediments. The hinge line or related fault sets could have provided the channelways for the first pulse of basaltic volcanism associated temporally with the deposits (or vice versa). A regular and repetitive change in the environment of deposition or of the ore fluid composition is required to explain the origin of the Anvil cycle.

Geology of the Vangorda Plateau

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

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

The geology of the northwest part of the Plateau is outlined on maps ___ to ___ which cover the portion of the area to be affected by development of the Grum and Vangorda deposits. Maps ___ to ___ show the distribution of most drill holes in the

same area.

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

The structure of the metamorphic sequence underlying the Vangorda Plateau is indicated on a set of cross sections accompanying this report.

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

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

center could be found in an open pit environment however. Only a few holes would be required to evaluate the area between Grum and Vangorda prior to construction of waste dumps. It is essential that this be done because this area is some of the most attractive exploration ground in the Anvil Range.

REFERENCES

- Blusson, S.L., 1966, Frances Lake, Yukon Territory and District of Mackenzie; Geol. Surv. Canada, Map 8-1967.
- Carne, R.C., and Cathro, R.J., 1982, Sedimentary exhalative (sedex) zinc-lead-silver deposits, northern Canadian Cordillera; Can. Inst. Min. Metall. Bull., v.75, p. 66-78.
- Gabrielse, H., Blusson, S.L. and Roddick, J.A. 1973, Geology of Flat River, Glacier Lake and Wrigley Lake map areas; Geol. Surv. Canada, Memoir 366, 153 p.
- Gordey, S.P., 1978, Stratigraphy and structure of the Summit Lake area, Yukon and Northwest Territories; in Current Research, Part A, Geol. Surv. Canada, Paper 78-1A, p.43-48.
- Gordey, S.P., 1981 Stratigraphy, structure and tectonic evolution of southern Pelly Mountains in the Indigo Lake area, Yukon Territory; Geol. Surv. Canada, Bulletin 318, 44p.
- Gustafson, L.B., and Williams, N. 1981, Sediment-hosted stratiform deposits of copper, lead, and zinc: Econ. Geol., 75th Anniversary Volume, p. 139-178.
- Jennings, D.S., Jilson, G.A. and Pigage, L.C., 1980, Anvil Range stratigraphy, south central Yukon Territory; Programme and Abstracts, Cordilleran Section, Geol. Assoc. Canada, Jan. 1980, p.16-17.
- Kuo, S.L., 1976, Geology and geochemistry of stratabound ore deposits in south central Yukon Territory and southwestern District of Mackenzie, Northwest Territories; Ph.D. dissert., Univ. of Alberta, Edmonton.
- Large, D., 1980, Geological parameters associated with sediment-hosted submarine exhalative Pb-Zn deposits: Geologisch. J., Reihe D, Heft 40, p.49-129.
- Tempelman-Kluit, D.J., 1972, Geology and origin of the Faro, Vangorda and Swim concordant zinc-lead deposits, central Yukon Territory; Geol. Surv. Canada, Bulletin 208, 73p.

SELECTED ADDITIONAL REFERENCES ON ANVIL DISTRICT GEOLOGY

Aho, A.E., 1966, Exploration methods in Yukon with special reference to Anvil District; *Western Miner*, v. 39, p. 127-148, April issue.

Aho, A.E., 1969, Base metal province of Yukon; *C.I.M. Bulletin*, April, p. 397-409.

Brock, J.S., 1973, Geophysical exploration leading to the discovery of the Faro Deposit; *C.I.M. Bull.*, October 1973, p.97-116.

Campbell, F.A., and Ethier, V.G., 1974, Sulfur isotopes, iron content of sphalerites, and ore textures in the Anvil Ore Body, Canada; *Econ. Geol.* v. 69, p. 482-493.

Campbell, R.B., 1967, Geology of Glenlyon map-area; *Geol. Surv. Canada, Memoir 352*, 92p.

Chisholm, E.O., 1957, Geophysical exploration of a lead zinc deposit in Yukon Territory: *in* Methods and case histories in mining geophysics, Sixth Commonwealth Mining and Metallurgy Congress, p. 269-277.

Gondie, J., 1972, Geology of the Anvil Mine, *in* D.J. Glass (ed.) Major lead-zinc deposits of Western Canada: *Internat. Geol. Cong.*, 14th Session, Guidebook.

Gordey, S.F., 1983, Thrust faults in the Anvil Range and a new look at the Anvil Range Group, south central Yukon Territory; *in* Current Research, Part A, *Geol. Surv. Canada, Paper 83-1A*, p. 225-227.

Johnston, J.R., 1936, A reconnaissance of Felly River between Macmillan River and Hoole Canyon, Yukon; *Geol. Surv. Can. Mem.* 200, 19p.

Kuo, S.L. and Folinsbee, R.E., 1974, Lead isotope geology of mineral deposits spatially related to the Tintina Trench, Yukon Territory; *Econ. Geol.*, v.69, p. 806-813.

LeCouter, P., 1973, A lead isotope study of ore deposits from the East Kootenay District, B.C. and the Anvil District, Y.T., unpub. Ph.D. thesis, Univ. Brit. Col.

Roddick, J.A., 1967, Tintina Trench; *Jour. Geol.*, v.75, p.23-33.

Roddick, J.A., and Green, L.H., 1961, Tay River, Yukon Territory; *Geol. Surv. Can. Map 13-1961*.

Tempelman-Kluit, D.J., 1968, Geological setting of the Faro, Vangorda and Swim base metal deposits, Yukon Territory; *Geol. Surv. Can. Paper 68-1 pt. A*, p. 43-52.

Tempelman-Kluit, D.J., 1970, An occurrence of eclogite near
Tintina Trench, Yukon; Geol. Surv. Can. Paper 70-1B, p.
19-22.

3.2 Faro

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3.2 Faro

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

3.2.1 History

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

In November 1985, Curragh Resources bought the Faro mine and other deposits in the Anvil Range from Cyprus Anvil Mining Corporation. Waste removal from the Faro pit resumed in early 1986. The Faro concentrator resumed production in June 1986.

3.2.2 General Geology

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

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

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

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

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

central graben. Zone 3 contains the remaining reserves.

There is essentially one thick horizon at Faro although this horizon contains numerous Anvil cycles and several thin phyllite waste bands are included. Locally a thin upper horizon is differentiated from the main mass of the deposit, generally this is too thin to be mineable. Low grade sulphide interbanding with high grade ore is widespread especially in the northeast part of the deposit.

Ore type zoning is particularly strong at Faro. It follows the scheme outlined above with a massive variably baritic upper portion and a quartzose variably carbonaceous lower part (figure 7). In addition there is a prominent very low grade semi-massive zone along the northeast edge of zone 3 and unusually abundant (compared to other Anvil district deposits), but erratically distributed, pyrrhotic mineralization in the southwest part of the deposit. Grade zoning follows ore type zoning so that the base and northeast edge of the deposit contains the lower grade mineralization whereas the upper and southwest portion contains the higher grade mineralization. Zoning was also obvious in plan view at Faro. Zone 1 was rich in baritic ores thus high grade, zone 2 at the other end of the deposit was rich in carbonaceous quartzose ore types thus low grade and metallurgically undesirable. Zone 3 has intermediate characteristics.

3.2.3 Drilling Density

The Faro deposit was drilled off on 43 m (141 feet) spaced sections with holes spaced nominally at 43 m along the sections. Most holes were vertical. In some parts of the deposit fill in drilling to 43 m has not been completed. In 1986 additional fillin was added to the northeast side of the AY phase because of difficulties in making accurate projections with available data.

3.2.4 Reserve Calculation

3.2.4.1 Method and Procedure - "FI" Model

The reserves used for Faro mine planning are derived from the "FI" mine model, generated from October to December 1985.

Block geology and drillhole information

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

The geological interpretive base was derived from two sources. In the southeast part of Zone 3 (Sections 124 to 133) the geological interpretation is the most up to date possible

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

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

3.2.4.2.4 Reserve reporting

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

3.2.4.3 Results

Table 3.2 compares the FI model with three computer calculated values for phase A (for old phase A not the current AY) all based on the same geology but varying in computational methodology and in the case of FI for the assay database. Also shown is a hand calculated reserve for phase A based on a geologic interpolation done by the author in September 1985 incorporating all drilling data available and using a fault dominant as opposed to fold dominant interpretation. The FI model tends to report a lower tonnage at a higher grade than

3.2 Faro

3.2.1 History

3.2.2 General Geology

3.2.3 Drilling Density

3.2.4 Reserve Calculation

3.2.4.1 Methods and Procedure - FI Model

3.2.4.1.1 Block Geology and Drillhole Data

3.2.4.1.2 Composite Calculation

3.2.4.1.3 Interpolation

3.2.4.1.3 Reserve Reporting

3.2.4.1 Methods and Procedure - FB60B Model

3.2.4.2.1 Block Geology & drillhole information

3.2.4.2.2 Composites

3.2.4.2.3 Interpolation

3.2.4.2.4 Reserve Reporting

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(1983). In the remainder of Zone 3 (Sections 117 to 124) this new geological interpretation was not yet available thus the interpretation used was that developed in 1981. This did not take advantage of 1984 drilling. The interpretations differ in the relative importance of folds and faults which results in significant differences in bench to bench geology, but for an overall section thru the deposit the cross-section area, hence the volume, is not very different.

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

3.2.4.1.2 Composite Calculation

Drill hole assays were composited on a 20' bench basis. Assays, were weighted by length and specific gravity. High assay values were rolled back to the 95th percentile level before compositing (See table 1). Internal waste (10 feet thick or less) was included in the composites at zero grade. Composites were again clipped to the 95th percentile before interpolation. External waste and waste bands greater than 1/2 bench height (10 feet) were not included in the composite intervals resulting in a shorter than 20 foot composite.

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

3.2.4.1.3 Interpolation

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

3.2.4.1.4 Reserve Reporting

Reserves were computed for 6 ore types:
2A, 2BCD, 2CE, 2GE, and 2HE. Geological reserve computation was by a weighted average of all blocks in the model below

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

3.2.4.2 Methods and Procedure - F8608 Model

In August and September 1986 a new computer model of the AY and BY phases, the F8608 model, was constructed. This was largely in reaction to the poor performance of the FI model in the JB phase and concerns over the base geologic interpretation in the AY and BY phase areas as well as some of the computational methodology used for the FI model.

3.2.4.2.1 Block Geology and Drillhole Data

The F8608 model is a 3D block model made using PC Mine software. Block size was 35 feet along the deposit, 25 feet across the deposit and 20 feet high. The coordinate grid was rotated 45° so that rows of blocks are parallel to the cross sections rather than the mine survey grid. The geological control used in the model is based on new cross and longitudinal sections for the northwest part of zone 3 completed by R.S. Tolbert in early 1986. The 141 foot spaced sections were simplified and intermediate cross sections created at 70.5 foot intervals (or 35 foot where required). These sections provided geological control for block geology rather than geological bench plans as have been used for all previous Faro deposit models. This was done largely because it is a quicker process to make cross sections but also because a flat lying stratiform deposit is more logically viewed in section perpendicular to its direction of predictability. The sectional model approach also allows the use of different bench heights without changing the geology so that bench height optimization could be studied. A drawback to this approach is that if section to section geology has not been closely coordinated and a control section is not provided for each row of blocks then the bench plans of grade distribution and ore type have a patchy appearance with obviously angular contacts parallel to the sections. This is the case with the F8608 model but this problem is largely due to having rushed the model to completion without taking time to refine the base geologic interpretation. The effect on reserves is not thought to be great but the lack of "smoothly flowing" bench plans could cause problems in the mine planning stage.

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

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

Drillhole data used was that already imported into the PC Mine

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

Drillhole data used was that already imported into the PC Mine

for the FI model.

Being a sectional model the F8608 model is similar to those created for Grum and Vangorda with the exception of computational details outlined below.

3.2.4.2.2 Composites

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

3.2.4.2.3. Interpolation

Variogram analysis of Faro composites was not generally successful. A tendency of a larger along deposit than across deposit range was indicated. The search volume (and hence anisotropy) used was tailored to be a close approximation to that used for the FI Model with the exception that the search ellipsoid was tilted to follow the layering of the deposit in three domains. First pass interpolation used a search ellipsoid looking 225' along the deposit trend 150' across it and 25' vertically. This was enlarged to 300' along the deposit, 200' across and 37' vertically in 3 passes (comparable multi-pass interpolation was not available when the Grum and Vangorda models were made).

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

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

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

previous models. This is likely due to more restrictive application of geology matching during interpolation due to the greater availability of composites for interpolation.

The comparison of hand calculated reserves using new geology to FI reserves is the most critical as it deals with estimates derived from very different approaches. The FI model reports 8.5% higher tonnes than the hand model at 5% lower grade. Much of the grade reduction may be due to the comparison of a nearest neighbor to an inverse squared distance interpolation but at the worst this comparison suggests the reserves compare within 10%.

Table 3.2B compares the expanded JB phase (again not exactly the same as the current JB phase) as calculated by the FI model and with calculation done by Cyprus Anvil using the same assay data and geologic interpretation. Cyprus Anvil's approach was to compute the actual area of geologic units on the benches and make the most appropriate assay assignment to these areas by manual means. This comparison thus addresses the question of how adequate the block representation of the geology is and how the machine computations compare to manual computations. The comparison is good; the major difference being in tonnage which is probably at least in part due to the inability of 50' X 50' blocks to show every geologic unit.

The results from the FI and F8608 models the AY and BY Phases are compared in table 3.1. The F8608 composited tonnages are very close to those of the FI model or slightly higher. The grades are fairly consistently lower. The total metal for both phases is within 2.2% of that calculated by the FI model. Despite the close comparison on a large volume basis, the bench to bench variance is quite large with many benches being within only $\pm 30\%$. As might be expected the larger benches in the core of the deposit compare well.

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

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

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

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

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

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

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

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

The conclusion of these comparisons is that the FI model compared well to other calculations on the basis of total metal and for a large enough volume of material was within 10% of actual production statistics. Despite this, it was unuseable for bench by bench comparisons without an appropriate dilution factor. The grade predictions of the FI model are too high probably as a result of inappropriate methodology. Most importantly, dilution is not average through the deposit and the choice of factor must take account local deposit structure in order to provide reasonable predictions of millfeed. Furthermore without accurate drillhole data, accurate modeling is impossible regardless of calculation sophistication.

Table 3.1

6% Cutoff

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

December 5, 1985

Table 3.2 A
Phase A

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

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

FI	814,017	3.81	6.19	49.9	10.00
HAND CALC FROM F4 BENCH GEOLOGY	858,887	3.8	6.1	52.4	9.9

Table 3.3A

CURRAGH RESOURCES
Comparison of F1 and FG608 models - FOR AY PHASE ONLY

Gregg Jilson
September 24, 1986

NO ADJUSTMENTS FOR DILUTION OR MINING LOSS

*****NEW MODEL (FG608)*****											*****OLD MODEL (F1)*****										
BENCH NUMBER	TOE ELEV.	TONNES +6% ORE	Pb (%)	Zn (%)	Ag (g/l)	Pb + Zn (%)	LEAD METAL (tonnes)	ZINC METAL (tonnes)	SILVER METAL (grams)	TOTAL METAL (tonnes)	BENCH NUMBER	TOE ELEV.	TONNES +6% ORE	Pb (%)	Zn (%)	Ag (g/l)	Pb + Zn (%)	LEAD METAL (tonnes)	ZINC METAL (tonnes)	SILVER METAL (grams)	TOTAL METAL (tonnes)
17	3750	3,840	3.97	5.87	79.04	9.84	152	225	303521	378	17	3750	3,110	6.50	5.94	105.00	12.44	202	185	326550	387
18	3730	33,320	4.15	5.13	71.75	9.28	1382	1710	2390743	3092	18	3730	10,680	3.86	4.73	59.64	8.58	412	505	636923	917
19	3710	49,290	3.81	5.00	54.99	8.81	1876	2464	2710654	4340	19	3710	62,910	3.14	4.61	46.73	7.74	1972	2900	2939784	4872
20	3690	77,620	3.42	5.66	45.75	9.07	2651	4390	3551115	7041	20	3690	53,460	3.31	5.11	45.41	8.42	1770	2731	2427565	4501
21	3670	51,530	3.20	5.15	38.07	8.35	1648	2653	1961953	4302	21	3670	43,050	3.98	5.79	48.25	9.77	1714	2493	2077033	4207
22	3650	43,590	3.14	4.16	37.53	7.30	1367	1813	1636020	3180	22	3650	47,850	4.22	5.02	48.87	9.24	2021	2400	2338573	4421
23	3630	76,680	3.16	4.26	37.33	7.42	2420	3266	2862771	5686	23	3630	71,320	4.18	5.49	49.99	9.66	2980	3912	3565501	6892
24	3610	141,460	3.13	4.88	32.90	8.01	4426	6909	4654458	11335	24	3610	150,670	3.24	5.68	30.47	8.92	4885	8557	4591518	13441
25	3590	121,260	2.91	4.74	31.34	7.65	3524	5749	3799682	9273	25	3590	84,600	2.89	5.35	27.01	8.24	2441	4528	2284792	6969
26	3570	79,820	2.91	4.84	36.36	7.74	2322	3859	2902574	6181	26	3570	81,450	3.71	5.79	48.87	9.50	3019	4719	3980706	7738
27	3550	205,690	3.25	5.09	44.78	8.34	6681	10466	9211004	17146	27	3550	260,620	3.70	5.59	53.89	9.29	9640	14574	14043509	24214
28	3530	365,850	3.61	5.27	47.45	8.88	13211	19291	17358485	32502	28	3530	417,040	3.89	5.72	48.30	9.61	16240	23855	20142198	40094
29	3510	499,220	3.56	5.27	44.84	8.83	17762	26299	22386522	44061	29	3510	566,280	3.98	5.77	47.88	9.75	22555	32674	27111788	55229
30	3490	469,190	3.87	5.41	49.20	9.28	18158	25369	23082271	43527	30	3490	426,680	4.27	6.06	52.05	10.27	18211	25609	22209121	43820
31	3470	361,350	3.76	5.28	45.78	9.03	13572	19068	16541158	32641	31	3470	375,390	3.85	5.29	44.89	9.14	14441	19854	16852008	34296
32	3450	314,490	3.17	4.67	43.67	7.85	9982	14699	13734722	24681	32	3450	295,120	3.38	4.91	48.57	8.29	9975	14490	14334569	24465
33	3430	253,690	3.72	5.72	45.15	9.43	9427	14498	11454357	23926	33	3430	212,300	3.69	5.75	45.71	9.44	7828	12207	9703808	20035
34	3410	42,300	2.79	4.27	25.95	7.06	1181	1806	1097474	2987	34	3410	32,740	2.79	4.35	24.31	7.14	912	1425	795942	2337
35	3390	16,880	2.97	4.14	23.57	7.11	501	699	397929	1200	35	3390	17,630	2.85	4.23	22.71	7.08	503	745	400289	1247
TOTALS/AVERAGES:		3,207,070	3.50	5.15	44.29	8.65	112244	165235	142037415	277479			3,212,900	3.79	5.55	46.92	9.34	121,719	178,363	150,762,176	300,082

Table 3.3B

-----PERCENT VARIANCES (NEW-OLD)/OLD-----											-----ACTUAL VARIANCES (NEW-OLD)-----										
TONNES	Pb	Zn	Ag	Pb + Zn	LEAD	ZINC	SILVER	TOTAL BENCH	TDE		TONNES	Pb	Zn	Ag	Pb + Zn	LEAD	ZINC	SILVER	TOTAL		
+6% ORE	(%)	(%)	(%)	(%)	METAL	METAL	METAL	METAL NUMBER	ELEV.		+6% ORE	(%)	(%)	(%)	(%)	METAL	METAL	METAL	METAL		
					(%)	(%)	(%)	(%)								(tonnes)	(tonnes)	(ounces)	(tonnes)		
23.5%	-38.9%	-1.2%	-24.7%	-20.9%	-24.6%	22.0%	-7.1%	-2.3%	17	3750	730	-2.53	-0.07	-25.9%	-2.6%	-50	41	-672	-9		
212.0%	7.6%	8.5%	20.3%	8.1%	235.8%	238.6%	275.4%	237.4%	18	3730	22,640	0.29	0.4%	12.11	0.70	971	1205	51212	2176		
-21.6%	21.4%	8.5%	17.7%	13.7%	-4.9%	-15.0%	-7.8%	-10.9%	19	3710	(13,620)	0.67	0.39	8.26	1.06	-96	-436	-6691	-531		
45.2%	3.1%	10.7%	0.8%	7.7%	49.8%	60.7%	46.3%	56.4%	20	3690	24,160	0.10	0.55	0.34	0.65	881	1659	32608	2540		
19.7%	-19.7%	-11.1%	-21.1%	-14.6%	-3.8%	6.4%	-5.5%	2.2%	21	3670	8,480	-0.78	-0.64	-10.17	-1.43	-66	160	-3360	94		
-8.9%	-25.8%	-17.0%	-23.2%	-21.0%	-32.4%	-24.4%	-30.0%	-28.1%	22	3650	14,260	-1.09	-0.85	-11.34	-1.94	-654	-586	-20515	-1241		
7.5%	-24.5%	-22.4%	-25.3%	-25.3%	-18.8%	-16.5%	-19.7%	-17.5%	23	3630	5,360	-1.02	-1.23	-12.66	-2.25	-560	-646	-20520	-1206		
-6.1%	-3.5%	-14.0%	8.0%	-10.2%	-9.4%	-19.3%	1.4%	-15.7%	24	3610	(9,210)	-0.11	-0.79	2.43	-0.91	-458	-1648	1838	-2106		
43.3%	0.7%	-11.4%	16.0%	-7.2%	44.4%	27.0%	66.3%	33.1%	25	3590	36,660	0.02	-0.61	4.33	-0.59	1083	1221	44235	2304		
-2.0%	-21.5%	-16.6%	-25.6%	-18.5%	-23.1%	-18.2%	-27.1%	-20.1%	26	3570	(1,630)	-0.80	-0.96	-12.51	-1.76	-697	-860	-31481	-1556		
-21.1%	-12.2%	-9.0%	-16.9%	-10.3%	-30.7%	-28.2%	-34.4%	-29.2%	27	3550	(54,930)	-0.45	-0.50	-9.10	-0.96	-2960	-4108	-141109	-7068		
-12.3%	-7.3%	-7.8%	-1.8%	-7.6%	-18.7%	-19.1%	-13.8%	-18.9%	28	3530	(51,190)	-0.28	-0.45	-0.95	-0.73	-3029	-4563	-81284	-7592		
-11.8%	-10.7%	-8.7%	-6.3%	-9.5%	-21.2%	-19.5%	-17.4%	-20.2%	29	3510	(67,060)	-0.43	-0.50	-3.03	-0.93	-4793	-6375	-137976	-11168		
10.0%	-9.3%	-9.9%	-5.5%	-9.7%	-0.3%	-0.9%	3.9%	-0.7%	30	3490	42,510	-0.40	-0.59	-2.86	-0.99	-53	-240	25496	-293		
-3.7%	-2.4%	-0.2%	2.0%	-1.1%	-6.0%	-4.0%	-1.8%	-4.8%	31	3470	(14,040)	-0.09	-0.01	0.88	-0.10	-869	-786	-9077	-1655		
6.6%	-6.1%	-4.8%	-10.1%	-5.3%	0.1%	1.4%	-4.2%	0.9%	32	3450	19,370	-0.21	-0.24	-4.90	-0.44	7	209	-17516	216		
19.5%	0.8%	-0.6%	-1.2%	-0.1%	20.4%	18.8%	18.0%	19.4%	33	3430	41,390	0.03	-0.04	-0.56	-0.01	1600	2291	51116	3891		
29.2%	0.2%	-1.9%	6.7%	-1.1%	29.5%	26.8%	37.9%	27.8%	34	3410	9,560	0.01	-0.08	1.63	-0.08	249	381	8805	650		
-4.3%	4.0%	-2.0%	3.8%	0.4%	-0.4%	-6.2%	-0.6%	-3.8%	35	3390	(750)	0.12	-0.08	0.87	0.03	-2	-46	-69	-48		
-0.2%	-7.6%	-7.2%	-5.6%	-7.4%	-7.8%	-7.4%	-5.8%	-7.5%	AVERAGE VARIANCE		(5,830)	-0.29	-0.40	-2.64	-0.69	-9475	-13127	-254763	-22603		

Table 3.4A

CURRAGH RESOURCES
Comparison of F1 and FB608 models - FOR DIL PHASE ONLY

Gregg Jilson
September 30, 1986

NO ADJUSTMENTS FOR DILUTION OR MINING LOSS

*****NEW MODEL (FB608)*****											*****OLD MODEL (F1)*****										
BENCH NUMBER	TOE ELEV.	TONNES +6Z ORE	Pb (%)	Zn (%)	Ag (g/t)	Pb + Zn (%)	LEAD METAL (tonnes)	ZINC METAL (tonnes)	SILVER METAL (grams)	TOTAL METAL (tonnes)	BENCH NUMBER	TOE ELEV.	TONNES +6Z ORE	Pb (%)	Zn (%)	Ag (g/t)	Pb + Zn (%)	LEAD METAL (tonnes)	ZINC METAL (tonnes)	SILVER METAL (grams)	TOTAL METAL (tonnes)
20	3690	2,770	2.55	6.17	25.05	8.72	71	171	69383	242	20	3690	0	0.00	0.00	0.00	0.00	0	0	0	0
21	3670	11,930	3.14	7.75	27.81	10.89	374	925	331797	1299	21	3670	0	0.00	0.00	0.00	0.00	0	0	0	0
22	3650	0	0.00	0.00	0.00	0.00	0	0	0	0	22	3650	2,360	6.18	7.81	61.00	13.99	146	184	143960	330
23	3630	0	0.00	0.00	0.00	0.00	0	0	0	0	23	3630	0	0.00	0.00	0.00	0.00	0	0	0	0
24	3610	3,650	3.78	5.42	47.62	9.19	138	198	173809	336	24	3610	0	0.00	0.00	0.00	0.00	0	0	0	0
25	3590	51,360	3.79	5.76	51.14	9.55	1948	2958	2626756	4906	25	3590	1,420	3.59	4.44	50.00	8.03	51	63	71000	114
26	3570	120,300	3.74	4.83	54.03	8.57	4499	5809	6499328	10309	26	3570	105,810	4.32	5.41	54.45	9.73	4574	5725	5761566	10300
27	3550	176,490	3.93	4.76	56.83	8.69	6931	8403	10029927	15333	27	3550	181,470	4.31	4.73	63.08	9.04	7816	8584	11446220	16399
28	3530	252,270	3.52	4.62	54.13	8.14	8877	11660	13655880	20537	28	3530	267,200	4.40	5.18	69.03	9.58	11757	13833	18444549	25590
29	3510	319,950	4.26	5.60	63.46	9.86	13636	17908	20304987	31544	29	3510	272,490	4.18	5.47	66.52	9.65	11390	14905	18126307	26295
30	3490	471,560	4.03	5.55	55.40	9.59	19018	26181	26125839	45199	30	3490	471,420	4.32	5.77	61.81	10.09	20342	27220	29137999	47562
31	3470	745,720	4.24	5.83	56.23	10.07	31633	43461	41929598	75094	31	3470	751,690	4.13	5.70	56.51	9.83	31045	42846	42474243	73891
32	3450	560,290	4.06	5.54	49.38	9.59	22737	31012	27666000	53749	32	3450	489,640	4.13	5.71	55.03	9.84	20207	27973	26945379	48181
33	3430	529,670	3.61	5.22	40.87	8.83	19116	27638	21648143	46754	33	3430	503,770	3.68	5.34	39.35	9.02	18534	26896	19824861	45430
34	3410	679,850	3.13	4.84	31.59	7.97	21273	32884	21476462	54157	34	3410	706,020	3.30	5.06	34.35	8.36	23299	35746	24252493	59044
35	3390	428,320	3.02	4.65	29.36	7.68	12948	19934	12576332	32882	35	3390	450,190	3.07	4.38	36.45	7.45	13816	19736	16408075	33553
36	3370	305,530	2.96	4.89	26.96	7.85	9053	14943	8236172	23996	36	3370	243,000	2.96	4.79	31.70	7.76	7203	11642	7702128	18845
37	3350	109,550	2.78	5.26	29.86	8.03	3040	5761	3270725	8801	37	3350	148,630	2.88	5.28	34.04	8.17	4286	7849	5059811	12136
38	3330	14,410	2.62	5.20	43.80	7.82	377	750	631086	1127	38	3330	28,830	2.61	4.92	39.32	7.52	751	1417	1133624	2168
TOTALS/AVERAGES:		4,783,620	3.67	5.24	45.42	8.91	175668	250596	217252222	426264			4,623,940	3.79	5.29	49.08	9.08	175,217	244,620	226,932,216	419,837

TABLE 3.4B

-----PERCENT VARIANCES ((NEW-OLD)/OLD)-----											-----ACTUAL VARIANCES ((NEW-OLD)/OLD)-----										
TONNES +6% ORE	Pb (%)	Zn (%)	Ag (%)	Pb + Zn (%)	LEAD METAL (%)	ZINC METAL (%)	SILVER METAL (%)	TOTAL BENCH METAL NUMBER (%)	ICE ELEV.		TONNES +6% ORE	Pb (%)	Zn (%)	Ag (%)	Pb + Zn (%)	LEAD METAL (tonnes)	ZINC METAL (tonnes)	SILVER METAL (tonnes)	TOTAL BENCH METAL NUMBER (tonnes)	ICE ELEV.	
ERR	ERR	ERR	ERR	ERR	ERR	ERR	ERR	ERR	20	3690	2,770	2.55	6.17	25.05	8.72	71	171	2026	242	20	3690
ERR	ERR	ERR	ERR	ERR	ERR	ERR	ERR	ERR	21	3670	11,930	3.14	7.75	27.81	10.89	374	925	9688	1299	21	3670
-100.0%	-100.0%	-100.0%	-100.0%	-100.0%	-100.0%	-100.0%	-100.0%	-100.0%	22	3650	(2,360)	-6.18	-7.81	-61.00	-13.99	-146	-184	-4204	-330	22	3650
ERR	ERR	ERR	ERR	ERR	ERR	ERR	ERR	ERR	23	3630	0	0.00	0.00	0.00	0.00	0	0	0	0	23	3630
ERR	ERR	ERR	ERR	ERR	ERR	ERR	ERR	ERR	24	3610	3,650	3.78	5.42	47.62	9.19	138	198	5075	336	24	3610
3516.9%	5.6%	29.7%	2.3%	19.0%	3720.4%	4592.2%	3599.7%	4202.4%	25	3590	49,940	0.20	1.32	1.14	1.52	1897	2895	74628	4792	25	3590
13.7%	-13.5%	-10.8%	-0.8%	-12.0%	-1.6%	1.5%	12.8%	0.1%	26	3570	14,490	-0.58	-0.58	-0.43	-1.16	-75	84	21543	9	26	3570
-2.7%	-8.8%	0.7%	-9.9%	-3.9%	-11.3%	-2.1%	-12.4%	-6.5%	27	3550	(4,980)	-0.38	-0.03	-6.25	-0.35	-885	-181	-41356	-1066	27	3550
-5.6%	-20.0%	-10.7%	-21.6%	-15.0%	-24.5%	-15.7%	-26.0%	-19.7%	28	3530	(14,930)	-0.88	-0.55	-14.90	-1.44	-2879	-2173	-139829	-5052	28	3530
17.4%	2.0%	2.3%	-4.6%	2.2%	19.7%	20.1%	12.0%	20.0%	29	3510	47,460	0.08	0.13	-3.06	0.21	2246	3002	63617	5249	29	3510
0.0%	-6.5%	-3.8%	-10.4%	-5.0%	-6.5%	-3.8%	-10.3%	-5.0%	30	3490	140	-0.28	-0.22	-6.41	-0.50	-1324	-1039	-87955	-2363	30	3490
-0.8%	2.7%	2.2%	-0.5%	2.4%	1.9%	1.4%	-1.3%	1.6%	31	3470	(5,970)	0.11	0.13	-0.28	0.24	589	614	-15904	1203	31	3470
14.4%	-1.7%	-3.1%	-10.3%	-2.5%	12.5%	10.9%	2.7%	11.6%	32	3450	70,650	-0.07	-0.18	-5.65	-0.25	2529	3039	21042	5568	32	3450
5.1%	-1.9%	-2.3%	3.9%	-2.1%	3.1%	2.8%	9.2%	2.9%	33	3430	25,900	-0.07	-0.12	1.52	-0.19	582	742	53240	1324	33	3430
-3.7%	-5.2%	-4.5%	-8.0%	-4.7%	-8.7%	-8.0%	-11.4%	-8.3%	34	3410	(26,170)	-0.17	-0.23	-2.76	-0.40	-2026	-2861	-81060	-4888	34	3410
-4.9%	-1.5%	6.2%	-19.4%	3.0%	-6.3%	1.0%	-23.4%	-2.0%	35	3390	(21,870)	-0.05	0.27	-7.09	0.22	-868	198	-111887	-671	35	3390
25.7%	-0.0%	2.1%	-15.0%	1.3%	25.7%	28.4%	6.9%	27.3%	36	3370	62,530	-0.00	0.10	-4.74	0.10	1850	3301	15594	5152	36	3370
-26.3%	-3.8%	-0.4%	-12.3%	-1.6%	-29.1%	-26.6%	-35.4%	-27.5%	37	3350	(39,080)	-0.11	-0.02	-4.19	-0.13	-1246	-2088	-52241	-3334	37	3350
-50.0%	0.4%	5.9%	11.4%	4.0%	-49.8%	-47.1%	-44.3%	-48.0%	38	3330	(14,420)	0.01	0.29	4.47	0.30	-374	-667	-14674	-1041	38	3330
3.5%	-3.1%	-1.0%	-7.5%	-1.9%	0.3%	2.4%	-4.3%	1.5%	AVERAGE VARIANCE		159,680	-0.12	-0.05	-3.66	-0.17	452	5975	-282656	6427	AVERAGE VARIANCE	

Table 3.5 A

CURRAGH RESOURCES
 FARO DEPOSIT - JB PHASE
 COMPARISON OF FI AND FB608 MODELING METHODS IN THE JB PHASE

Gregg Jilson
 September 26, 1986

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

"FB608" hybrid model (the new calculation) is based on interpolation of grade into the FI model blocks using the same geology and composite database as the FI model (the old calculation) but techniques as close as possible to those used in the AY phase for the actual FB608 model which uses smaller blocks and revised geology.

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

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

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

Table 3.5B

RESULTS OF OLD CALCULATION WITH MINTEC, LENGTH WEIGHTING OF COMPOSITES

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

Table 3.5C

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

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

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

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

Table 3.6A

BLASTHOLE RESULTS (FROM MINE GEOLOGY DEPARTMENT PRINTOUT)

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

Table 3.6B

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

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

ACTUAL VARIANCE BLASTHOLES TO NEW CALCULATION (BH-NEW)

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

Table 3.6c

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

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

ACTUAL VARIANCE BLASTHOLES TO OLD CALCULATION (BH-OLD)

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

Table 3.6 D

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

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

BENCH NUMBER	TOE ELEV.	TONNES ORE **	Pb (%)	Zn (%)	Ag (g/t)	Au (g/t)	Pb + Zn (%)	LEAD METAL (tonnes)	ZINC METAL (tonnes)	SILVER METAL (grams)	GOLD METAL (grams)	TOTAL METAL (tonnes)	
10	3890	18.6%	-4.7%	1.1%	-6.1%		-1.2%	14.7%	19.5%	13.6%		17.6%	
11	3870	70.3%	0.1%	1.4%			0.9%	70.3%	70.7%			70.6%	
12	3850	-89.3%	-43.2%	-42.2%			-42.6%	-171.1%	-169.2%			-169.9%	
13	3830	17.9%	-8.8%	-8.6%			-8.7%	10.7%	10.9%			10.8%	
14	3810	21.3%	-1.9%	-27.4%			-16.7%	19.8%	-0.3%			8.1%	
15	3790	30.2%	-3.1%	-21.5%			-13.9%	28.1%	15.2%			20.5%	
16	3770	19.3%	-5.6%	-14.9%	-11.7%		-11.4%	14.8%	7.3%	9.9%		10.1%	
17	3750	-10.8%	4.9%	1.6%	10.8%		2.9%	-5.4%	-9.0%	1.2%		-7.6%	
18	3730	-3.8%	9.1%	9.3%	14.7%		9.2%	5.6%	5.8%	11.5%		5.7%	
19	3710	ERR	ERR	ERR	ERR		ERR	ERR	ERR	ERR		ERR	
20	3690	ERR	ERR	ERR	ERR		ERR	ERR	ERR	ERR		ERR	
AVERAGES: 3890 to 3770			19.1%	-5.6%	-16.8%	-203.4%	ERR	-12.4%	14.6%	5.5%	-145.4%	ERR	9.1%

ACTUAL VARIANCE BLASTHOLES TO NEW CALCULATION (BH-NEW)

BENCH NUMBER	TOE ELEV.	TONNES ORE **	Pb (%)	Zn (%)	Ag (g/t)	Au (g/t)	Pb + Zn (%)	LEAD METAL (tonnes)	ZINC METAL (tonnes)	SILVER METAL (grams)	GOLD METAL (grams)	TOTAL METAL (tonnes)
10	3890	8,210	-0.15	0.05	-2.7		-0.09	202	405	264056		607
11	3870	14,328	0.00	0.07			0.08	473	764			1237
12	3850	(26,386)	-1.04	-1.62			-2.66	-1218	-1915			-3133
13	3830	14,655	-0.25	-0.39			-0.63	244	401			645
14	3810	21,223	-0.06	-1.21			-1.27	630	-15			615
15	3790	55,664	-0.11	-1.06			-1.17	1779	1386			3165
16	3770	41,985	-0.15	-0.66	-4.1		-0.81	880	699	751050		1579
17	3750	(22,465)	0.15	0.07	4.6		0.23	-346	-866	103679		-1212
18	3730	(9,984)	0.27	0.43	6.2		0.71	438	709	1256532		1148
19	3710											
20	3690											
AVERAGES: 3890 to 3770		129,679	-0.17	-0.77	-28.7	-0.22	-0.94	2989	1725	*****	-120065	4714

Table 3.6 E

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

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

BENCH NUMBER	TDE ELEV.	TONNES DRE **	Pb (%)	Zn (%)	Ag (g/t)	Au (g/t)	Pb + Zn (%)	LEAD METAL (tonnes)	ZINC METAL (tonnes)	SILVER METAL (grams)	GOLD METAL (grams)	TOTAL METAL (tonnes)	
10	3890	3.8%	-4.2%	9.5%	-2.9%		4.0%	-0.3%	12.9%	1.0%		7.7%	
11	3870	64.9%	0.8%	10.4%			6.7%	65.2%	68.6%			67.3%	
12	3850	-123.7%	-41.3%	-28.8%			-33.6%	-216.0%	-188.1%			-198.9%	
13	3830	3.0%	-9.4%	1.0%			-3.0%	-6.2%	3.9%			0.0%	
14	3810	6.9%	-1.4%	-15.2%			-9.4%	5.6%	-7.2%			-1.8%	
15	3790	17.5%	-1.3%	-9.3%			-6.0%	16.4%	9.8%			12.5%	
16	3770	4.6%	-7.1%	-4.5%	-11.0%		-5.5%	-2.1%	0.3%	-5.9%		-0.6%	
17	3750	-30.9%	3.9%	9.7%	11.1%		7.4%	-25.8%	-18.2%	-16.4%		-21.3%	
18	3730	-22.7%	6.9%	16.3%	14.1%		12.6%	-14.3%	-2.7%	-5.4%		-7.2%	
19	3710	ERR	ERR	ERR	ERR		ERR	ERR	ERR	ERR		ERR	
20	3690	ERR	ERR	ERR	ERR		ERR	ERR	ERR	ERR		ERR	
AVERAGES:													
3890 to 3770			4.4%	-5.4%	-5.9%	-197.6%	ERR	-5.7%	-0.7%	-1.2%	-184.4%	ERR	-1.0%

ACTUAL VARIANCE BLASTHOLES TO NEW CALCULATION (BH-NEW)

BENCH NUMBER	TDE ELEV.	TONNES DRE **	Pb (%)	Zn (%)	Ag (g/t)	Au (g/t)	Pb + Zn (%)	LEAD METAL (tonnes)	ZINC METAL (tonnes)	SILVER METAL (grams)	GOLD METAL (grams)	TOTAL METAL (tonnes)	
10	3890	1,670	-0.13	0.45	-1.3		0.31	-4	268	18806		264	
11	3870	13,228	0.03	0.55			0.58	438	741			1179	
12	3850	(36,556)	-0.99	-1.10			-2.10	-1538	-2129			-3667	
13	3830	2,425	-0.26	0.04			-0.22	-141	144			3	
14	3810	6,929	-0.04	-0.67			-0.71	180	-316			-136	
15	3790	32,272	-0.04	-0.46			-0.50	1042	895			1937	
16	3770	10,055	-0.19	-0.20	-3.8		-0.39	-126	29	-446325		-97	
17	3750	(64,413)	0.12	0.45	4.8		0.57	-1668	-1747	-1469371		-3414	
18	3730	(59,274)	0.21	0.76	5.9		0.97	-1114	-326	-591843		-1440	
19	3710												
20	3690												
AVERAGES:													
3890 to 3770			30,023	-0.16	-0.27	-27.9	-0.22	-0.43	-150	-368	*****	-141895	-518

Table 3.6F

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

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

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

ACTUAL VARIANCE BLASTHOLES TO OLD CALCULATION (BH-OLD)

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

Table 3.6G

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

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

BENCH NUMBER	TOE ELEV.	TONNES ORE **	Pb (%)	Zn (%)	Ag (g/t)	Au (g/t)	Pb + Zn (%)	LEAD METAL (tonnes)	ZINC METAL (tonnes)	SILVER METAL (grams)	GOLD METAL (grams)	TOTAL METAL (tonnes)
10	3890	22.4%	0.2%	11.1%	-2.6%		6.8%	22.6%	31.0%	20.4%		27.7%
11	3870	40.8%	19.8%	24.8%			22.9%	52.5%	55.5%			54.4%
12	3850	-84.8%	-49.9%	-66.3%			-60.0%	-177.0%	-207.4%			-195.6%
13	3830	-11.3%	-18.9%	-21.1%			-20.3%	-32.4%	-34.8%			-33.9%
14	3810	-0.2%	-2.6%	-19.5%			-12.4%	-2.8%	-19.8%			-12.7%
15	3790	25.0%	-6.1%	-23.0%			-16.1%	20.5%	7.8%			13.0%
16	3770	-2.0%	-4.4%	-5.5%	-16.7%		-5.1%	-6.4%	-7.6%	-19.0%		-7.2%
17	3750	-28.2%	15.5%	9.8%	13.8%		12.1%	-8.3%	-15.6%	-10.5%		-12.7%
18	3730	-7.2%	15.0%	9.5%	26.0%		11.7%	8.9%	3.0%	20.6%		5.3%
19	3710	ERR	ERR	ERR	ERR		ERR	ERR	ERR	ERR		ERR
20	3690	ERR	ERR	ERR	ERR		ERR	ERR	ERR	ERR		ERR
AVERAGES:												
3890 to 3770		3.8%	-6.5%	-14.9%	-204.4%	ERR	-11.6%	-2.5%	-10.6%	-193.0%	ERR	-7.4%

ACTUAL VARIANCE BLASTHOLES TO OLD CALCULATION (BH-OLD)

BENCH NUMBER	TOE ELEV.	TONNES ORE **	Pb (%)	Zn (%)	Ag (g/t)	Au (g/t)	Pb + Zn (%)	LEAD METAL (tonnes)	ZINC METAL (tonnes)	SILVER METAL (grams)	GOLD METAL (grams)	TOTAL METAL (tonnes)
10	3890	9,908	0.01	0.52	-1.2		0.53	310	644	396511		954
11	3870	8,318	0.65	1.31			1.97	353	599			953
12	3850	(25,051)	-1.20	-2.54			-3.74	-1260	-2347			-3607
13	3830	(9,280)	-0.53	-0.95			-1.48	-740	-1287			-2027
14	3810	(228)	-0.08	-0.86			-0.94	-90	-872			-962
15	3790	46,140	-0.21	-1.13			-1.34	1297	711			2008
16	3770	(4,280)	-0.12	-0.25	-5.8		-0.37	-383	-733	-1444100		-1116
17	3750	(58,631)	0.48	0.45	5.9		0.93	-535	-1496	-942093		-2032
18	3730	(18,873)	0.45	0.44	10.9		0.89	691	365	2263826		1056
19	3710											
20	3690											
AVERAGES:												
3890 to 3770		25,527	-0.20	-0.68	-28.8	-17.7%	-0.88	-513	-3285	*****		-3798

3.3 Grum

3.3.1 History

3.3.2 General Geology

3.3.2.1 Lithology

3.3.2.2 Structure

3.3.2.3 Surficial geology

3.3.2.4 Ore deposit geology

3.3.3 Drill definition and information base

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3.3.4.1 Introduction

3.3.4.2 Block geology and drillhole information

3.3.4.3 Composite calculation

3.3.4.4 Variogram analysis

3.3.4.5 Interpolation

3.3.4.6 Geological reserve reporting

3.3.5 Results

3.3.6 Comparison of geological reserves

3.3.7 Reliability of geological reserves

3.3.8 Additional work needed

3.3 Grum

3.3.1 History

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

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

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

3.3.2 General Geology

3.3.2.1 Stratigraphy and lithology

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

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

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

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

3.3.2.2 Structure

The ore layers at Grum are contorted into a complex, shallowly northwest plunging, polyphase fold structure. The prominent S shaped folds (fig 3.9) are second phase structures. They are superimposed on a larger Z shaped first phase fold. The dominant plane of fissility (S_2) in the phyllites at Grum is axial planar to the second phase folds and dips shallowly (10° to 30°) generally to the southwest. This fissility is a major factor in assessing slope stability for a Grum pit. The overall deposit elongation parallels the axial direction of the second phase folds (315° trend 11° plunge).

There are several important faults at Grum. The largest displacements occur on moderately (35° - 45°) dipping structures that truncate the deposit at both its northwest and southeast ends (fig.3.10). Neither of these structures would crop out in an open pit but smaller subparallel faults will be found in the pit. A steeply northwest dipping fault trending about 060° , passes between sections 70W and 72W and downdrops the deposit about 60 m to the northwest. A myriad of smaller faults were mapped underground by Kerr Addison trending on the average 080° and dipping steeply. Joints mapped underground and on surface tend to strike 060° and dip subvertically.

3.3.2.3 Surficial Geology

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

3.3.2.4 Ore Deposit Geology

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

Grum, like Vangorda and Dy, has several characteristics that distinguish it from Faro. In large part this is due to the lower metamorphic grade the deposit has reached. The most outstanding difference between Grum, and all the other Vangorda Plateau deposits, as opposed to Faro is the form of the deposit. All deposits other than Faro consist of several distinct, highly

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

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

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

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

3.3.3 Drill Definition & Information Base

The Grum deposit extends from section 52W in the southeast to section 112W in the northwest (figure 3.14). The deposit has been most densely drilled between 62W and 86W and it is this portion of the deposit for which proven geologic reserves are reported.

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

Between sections 62W and 86W the deposit has also been explored by 15,000 m of underground drilling in fans from a pair of

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

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

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

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

There are 9000 samples in the Grum deposit assay database. Assay intervals generally average 1.5 m in length and are keyed as closely as possible to sulphide rock types. 90% of these were determined for Cyprus Anvil by Kamloops Research and Assay Labs between 1980 and 1983. For most of these samples Pb, Zn, Cu, and Ag assays are available. For 2/3 of these samples there is also insoluble Fe, soluble (in hot concentrated HCL) Fe, Au, and pulp SG. All assays were determined using a set of Anvil District ore type standards for control. Rejects and N₂ purged pulps (by now somewhat oxidized) have been retained for additional analytical work. There are no BaO or Mn assays available nor is there systematic data available for Hg, As, Cd or any other elements.

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

3.3.4 Methods and Procedure of Reserve Calculation

3.3.4.1 Introduction

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

The reserves are calculated from a computer based 3D block model based on a set of cross-sections produced by Cyprus Anvil

geologists in 1982. The PC Mine software package was used for grade interpolation and reserve calculation. The block geology and composites had been previously calculated using Mintec's Medsystem release 10. The results of the calculation are outlined in Table - .

3.3.4.2 Block Geology and Drillhole information

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

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

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

The rock types used in the Grum model are summarized in table 3.7. Since there is a large amount of low grade quartzose mineralization in the footwall of mineralized horizons at Grum it was judged desirable to carry a separate code for base metal poor and base metal rich lithologies. This is inherent in the lithologic codes of 4C and 4D however exactly the same distinction is made by the modifiers 4A0 and 4A4 (the latter being the base metal rich variant). A similar distinction was

made between 4E0 and 4E4. The purpose of this distinction is to avoid undue averaging of grade into the footwall (and vice versa) and to improve modeling of the relatively sharp grade separation within mineralized horizons.

Sectionally assigned codes were applied to 2 rows of blocks on either side of the section. Since the Grum deposit plunges to the northwest this assignment would create a stairstep appearance in long section. By coincidence the diagonal of two blocks in long section is parallel to the deposit plunge thus a "plunge correction" was made by raising the first row of blocks one level (southeast of the section) and lowering the 4th row of blocks one level (northwest of the section). The second and third rows of blocks are kept the same.

Table 3.7

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

This has no affect on deposit reserves. All this block coding was done outside of PC Mine either using the Mintec Medsystem release 10 software package or by manual means. The block codes were reformatted to suit PC Mine then imported.

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

3.3.4.3 Composite Calculation

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

Drill hole assay data were clipped to the 95th percentile levels to avoid assigning unusually high assays to large blocks. These levels are listed in table 3.8. Intervals with no measured S.G. were assigned an SG depending on rock type as listed in table 3.9. These are based on statistical analysis of the measured data.

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

Table 3.8

Maximum Permitted Assay Values and SG Values

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

Table 3.9

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

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

Table 3.10

Maximum Permitted Assay and SG
For DDH Composites

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

The final composites can be as short as 0.1 m if only a small part of a mineralized band is within a composite interval. When the composites were imported to PC Mine the length data could no longer be carried. After calculation, the composites were also clipped to the 95th percentile level as outlined in Table 3.10. Every composite was manually checked against the cross-sections to insure that the codes applied to sectional units were consistent with the composite codes.

3.3.4.4 Variogram analysis:

Experimental variograms were calculated for the Grum composites in vertical, across deposit (model 000°) and along deposit (model 090°) directions. As was expected the variograms are generally ambiguous. Those for lead are shown in figure 3.15 to 3.17. The variogram analysis shows the range along the deposit is greater than either vertically or across the deposit. The across deposit range is uncertain but it can be argued that the along deposit range is about twice the across deposit. This conforms to what was expected on geologic grounds.

3.3.4.5 Interpolation:

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

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

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

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

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

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

3.3.5 Results

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

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

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

Unfortunately the calculations listed in table 3.13 are not exactly comparable. The first 3 (A-C) are all based on Kerr-Addisons geological interpretation. The Cyprus Anvil computer models differs in that some new assay data was used but the bulk of the data was the same as A and B.

The last 2 (D & E on table 3.13) are based on the current geological interpretation and the current assay information which nearly completely replaces Kerr-Addisons data. These two calculations reflect some ore in the northwest part of the deposit that was drilled off in 1982, this amounts to about 3,000,000 tonnes largely of low grade material on sections 78W to 86W.

The early computer models did not use geologic control on interpolations which tends to average grade between disseminated and massive mineralization resulting in a larger tonnage of lower grade material than a comparable sectional calculation with sharp grade distinctions (compare A to B and C on table 3.13).

The current model has attempted to counter this averaging effect by restricting the choice of composites to the same rock type as the block being estimated. The result is a closer comparison in grade to the corresponding sectional calculation (compare D to E on table 3.13). The current model however was not able to assign a grade to every block due to the stringent matching requirements and local paucity of assay information. Thus there are 1023 uninterpolated blocks representing about 2,000,000 tonnes of sulphides distributed through the deposit. Only a portion of this is likely to be over 4% Pb + Zn in reality but for the purposes of the current model it is assigned a zero grade and included as sulphide waste. Much of this material (about 700,000 tonnes in 423 blocks) is in the lower part of section 86W partly accounting for the poor comparison on that section (table 3.14B). The distribution of uninterpolated blocks by section in the two constituent models is given in table 3.15.

A further complication in comparing the current model to the results obtained from the same interpolation is that two problems were encountered with the Cyprus Anvil/Dome calculation: a) there were some calculation errors involving volumes and to a lesser extent tonnes on sections 78W to 86W and b) the SG used for sections 82W-86W was assumed to be 3.6 rather than the measured SG as used for the rest of the deposit, in retrospect this value was too high as much of the mineralization on those sections is considerably lighter. These points have been adjusted for in Table 3.14A. Table 3.14B shows the section by section variance between calculations. Between the Cyprus Anvil/Dome calculation and the current one the grade is almost always slightly down. In large part of this is probably due to the clipping of the extreme high end of the assay and composite population which was not done in the Cyprus Anvil/Dome calculation. The large variance in volume on sections 80W-86W is partly due to uninterpolated blocks but must be mainly due to differences in calculation method. The computer model interpolated grade into each block whereas the hand calculation averaged a large number of intersections to arrive at the grade for large panels of ore. In light of the relative paucity of drillhole information for some panels such large variances are not surprising. The interpolation method would be expected to give a more realistic picture of grade distribution.

Table 3.11

CURRAGH RESOURCES
GRUM DEPOSIT
68606 MODEL FOR TOTAL DEPOSIT

ABOVE GRUM (1336.0 m to 1088.5 m)

GRADE CATEGORY	VOLUME (bcm)	S.G.	DRE (tonnes)	LEAD (%)	ZINC (%)	Pb + Zn (%)	SILVER (g/t)	GDLD (g/t)
+6 %	4,677,480	3.37	15,765,300	3.84	6.50	10.34	64.0	0.92
4 - 6%	1,942,920	3.12	6,065,990	1.90	3.10	4.99	33.0	0.77
+4 %	6,620,400	3.30	21,831,290	3.30	5.56	8.85	55.4	0.88

UNDER GRUM (1088.5 m to 868.0 m)

GRADE CATEGORY	VOLUME (bcm)	S.G.	DRE (tonnes)	LEAD (%)	ZINC (%)	Pb + Zn (%)	SILVER (g/t)	GDLD (g/t)
+6 %	1,880,820	3.75	7,057,100	4.04	6.36	10.40	68.5	1.18
4 - 6%	522,720	3.37	1,760,770	2.12	2.76	4.88	35.5	0.99
+4 %	2,403,540	3.67	8,817,870	3.66	5.64	9.30	61.9	1.14

TOTAL DEPOSIT (1336.0 m to 868.0 m)

GRADE CATEGORY	VOLUME (bcm)	S.G.	DRE (tonnes)	LEAD (%)	ZINC (%)	Pb + Zn (%)	SILVER (g/t)	GDLD (g/t)
+6 %	6,558,300	3.48	22,822,400	3.90	6.46	10.36	65.4	1.00
4 - 6%	2,465,640	3.17	7,826,760	1.95	3.02	4.97	33.6	0.82
+4 %	9,023,940	3.40	30,649,160	3.40	5.58	8.98	57.2	0.95

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

+4 %	8,225,911	3.96	32,611,000	3.48	5.72	9.20	59	
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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 a ~~similar~~ ³¹⁵ number in above grum not all of this material will be ore however. Each block represents about 1900 tonnes of material thus there is about 2,000,000 tonnes of unaccounted for sulphides.

* The large variance ~~between~~ ⁱⁿ volume and specific gravity between the two calculations is ~~presently unexplained.~~ ^{explained in table -}

Table 3R#

88606 COMPUTER MODEL, 1986
TOTAL DEPOSIT

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

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

Table 3.13

CURRAGH RESOURCES

GRUM DEPOSIT

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

Calculation	ore (tonnes)	Pb (%)	Zn (%)	Pb+Zn (%)	Ag (g/t)	total metal (tonnes)	<i>total variance</i> from avg. (tonnes)	<i>total percent variance</i> from avg.
<i>A</i> Kerr Addison - 1977 sectional hand calculation	26,083,000	4.1	6.4	10.5	62	2,738,715	104,352	4.0%
<i>B</i> Kerr Addison/Noranda - 1978 3D block computer model	27,650,000	3.1	4.9	8.0	48	2,212,000	(422,363)	-16.0%
<i>C</i> Cyprus Anvil - 1981 "61" 3D block computer model	30,781,000	3.1	4.9	8.0	49	2,462,480	(171,883)	-6.5%
<i>D</i> ^{Cyprus-Anvil} (Simpson-Adamson/Dome - 1983) * sectional hand calculation (uncorrected see table -)	32,611,000	3.5	5.7	9.2	59	3,000,212	365,849	13.9%
<i>E</i> Curragh Resources - 1986 * "88606" 3D block computer model	30,649,000	3.4	5.6	9.0	57	2,758,410	124,047	4.7%
Average of all the models	29,554,800					2,634,363		

NOTES:

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

Table 3.14A

KERR ADDISON SECTIONAL CALCULATION, 1977
TOTAL DEPOSIT

SECTION	VOLUME (bcm)	SPECIFIC GRAVITY	TONNAGE (tonnes)	LEAD (%)	ZINC (%)	SILVER (g/t)	GOLD (g/t)	Pb+Zn (%)	TOTAL METAL (tonnes)
B6 W	n/a	n/a	1,678,746	3.59	5.80	55.0	n/a	9.39	157,634
B4 W	n/a	n/a	1,375,120	3.33	5.60	53.0	n/a	8.93	122,798
B2 W	n/a	n/a	2,185,447	4.06	7.30	69.0	n/a	11.36	248,267
B0 W	n/a	n/a	3,039,373	3.67	5.71	56.0	n/a	9.38	285,093
7B 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
6B 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 B6W	n/a	n/a	26,067,842	4.07	6.43	61.50	n/a	10.51	2,739,161

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

SECTION	VOLUME (bcm)	SPECIFIC GRAVITY	TONNAGE (tonnes)	LEAD (%)	ZINC (%)	SILVER (g/t)	GOLD (g/t)	Pb+Zn (%)	TOTAL METAL (tonnes)
B6 W*	997,045	3.23	3,221,532	2.88	5.21	49.0	n/a	8.09	260,622
B4 W*	776,317	3.20	2,486,855	2.86	4.83	48.0	n/a	7.69	191,239
B2 W*	989,573*	3.29	3,256,446	3.00	5.10	52.0	n/a	8.10	263,772
B0 W	1,118,740*	3.38	3,783,293	3.31	5.41	56.0	n/a	8.72	329,903
7B 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
6B W	386,618	3.73	1,440,349	4.54	6.88	73.0	n/a	11.42	164,488
66 W	446,215	3.62	1,613,702	4.41	5.95	71.0	n/a	10.36	167,180
64 W	282,918	3.64	1,028,545	3.84	5.75	62.0	n/a	9.59	98,637
62 W	170,190	3.72	632,340	3.32	5.66	63.0	n/a	8.98	56,784
62W to B6W	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 ⁹⁹36 was used on these sections. ~~99~~ 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 previous calculation

Table 3.1 ~~4B~~ 4B

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

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

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

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

Table 3.15

UNINTERPOLATED BLOCKS IN 68608
GRUM COMPUTER MODEL

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

3.3.6 Reliability of Reserves

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

The density of drilling is sufficient to limit the possibility of major changes in deposit volume due to variance in interpretation. Volume ranges of $\pm 10\%$ would be possible. Variance possible due to calculation methods is also not quantified but changes of a few percent would be possible through adjustment of interpolation parameters such as anisotropy and geology matching.

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

The earliest production comes from a long troughlike pit following the subcrop of the ore horizons. This part of the deposit is the most densely 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 100' on 200' spaced sections, close to the current density of Vangorda drilling. There is room for improvement especially in light of the complex structure involving steeply dipping ore layers. In general the earliest production is from an area that is only moderately reliable but it is underlain by highly reliable deeper reserves.

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

The need to use a higher dilution than the historic 5% used for Anvil District deposits in the past is brought on largely by the restrictive geology matching during interpolation. This matching was not done previously thus the models tended to dilute themselves in unpredictable ways during interpolation.

In order to present a more realistic portrayal of grade distribution especially with respect to grade averaging into otherwise barren sulphides, particularly footwall sulphides, the modeling technique was changed. It is however necessary to change dilution practice at the same time. This was not done at Faro when using the FI model with the result that predicted tonnages were far too low and grade far too high compared to blasthole indicated tonnage and grade.

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

3.3.8 Additional Work needed

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

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

One of the major inadequacies of geologic data at Grum is geotechnical data. There is reason to believe that the S_2 foliation under Doal Lake (in the northeast part of the proposed pit) may be dipping northeast rather than southwest. This could have significant implications for slope stability and the size of the area required to be pre-stripped. A program of oriented core drilling is required to evaluate this question. These holes should also sample the overburden to evaluate its clay content and hydrogeology. If the overburden is 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. The northwest half of the ore in the first years pit will be almost entirely 4A.

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

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3.4 Vangorda

3.4.1 History

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

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

3.4.2 General Geology

3.4.2.1 Stratigraphy and lithology

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

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

3.4.2.2 Structure

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

The northwest half of the deposit plunges about 10° towards the northwest but the southeast half is sub-horizontal. The S_2

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

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

3.4.2.3 Deposit Geology

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

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

The massive sulphides that overlie the pyritic quartzite are commonly baritic and rich in lead and zinc. The unit is actually a mixture of about 50% 4E and 50% 4G ore types but separate treatment of pure types at Vangorda is not realistic from either the point of view of mining or the level of detail carried in this model. Metallurgical predictions must take into

account the mixed nature of this dominant ore type compared to the relatively pure samples on which test work was done. 90% of the mineralization exceeding 6% Pb & Zn is barite bearing massive sulphides (4EG). Most pyritic quartzite (4C and 4EC) is sulphide waste on the basis of lead zinc content.

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

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

3.4.3 Drilling Density and Information Base

In the portion of the deposit from 2W to 12E (figure 3.18) there are 53 diamond drill holes in a 60 m X 30 m pattern. All holes are vertical and all are NQ diameter. Collar locations have been surveyed, checked and appear accurate. Assay intervals are 1.5 to 2 m long and are where possible are confined to one rock type.

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

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

3.4.4 Methods and procedure of reserve calculation

3.4.4.1 Introduction.

The Vangorda deposit reserves were generated using a computer based 3D block model. The PC MINE software package was used.

3.4.4.2 Block geology and drillhole information

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

- 1.) The deposit must fit the overall structural setting of this part of the Vangorda Plateau as defined by surface mapping and deep drilling nearby. This work shows that the deposit is in the hinge region of a large recumbent second phase fold.
- 2.) Stratigraphic facings implied by Anvil cycles were followed wherever possible. Of particular importance was the barren pyritic quartzite (units 4C and 4EC) which shows strong indications of being a footwall facies.
- 3.) Symmetric sulphide intersections were taken to imply folds.
- 4.) The dominant deformation episode would be the second phase implying a deposit shape similar to Grum but different from Faro.

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

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

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

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

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

3.4.4.3 Composite calculation

The assays were composited on the basis of geology with an attempt to conform to a 4 to 6 m length. This composite coding ensures that an interpreted geologic unit will only be assigned an assay from a length of drill core that was actually used to define the unit. In some cases these defined units will actually contain two or more different geological types but this simplification was necessary in order to produce units that would be reasonable from a mining point of view with the minimum number of necessary ore types. In general these mixed types are related for example: mixed 4E and 4G or mixed 4C and 4A. The minimum composite length was about 1 m. Internal waste was included in the composites at a 0% assay value.

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

3.4.4.4 Interpolation

Experimental variograms calculated from these composites showed only nugget effect due to the relatively large drill hole spacing. For this reason the anisotropy developed for Grum was used at Vangorda with a slightly larger search volume. Search volume parameters were:

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

Composites were weighted by the inverse square of the distance from the composite center to the block center.

These interpolation parameters allowed most of the blocks to be assigned a value.

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

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

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

3.4.5 Results

Geological reserves at a 4% and 6% Pb+Zn cutoff grade are summarized in Table 3.16. The results quoted are for all mineralization regardless of potential pit outlines. There has been no allowance made for dilution or mining recovery in the geological reserves.

3.4.5 Comparison of Geological Reserves

Two hand calculations have been done for the entire Vangorda deposit. Both are based on the old drilling and assays for the deposit. Table 3.17 compares these results to the present model.

The oldest one by Prospector Airways is based on the triangular method. The comparison to the present model is quite close. The large variance in lead grade is expected as it is known that the older lead assays were low probably because a correction for barium was not made. The areal extent of this calculation is not known but it may extend as far as 40E.

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

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

Previous computer models of Vangorda have only covered the northwest end of the deposit (2W-13E). For purposes of comparison to these models a separate reserve calculation was made for the area covered by these models. The results are given in Table 3.18 and table 3.19.

As expected the current model reports higher grades but somewhat unexpectedly the tonnage is higher or comparable. This may be due to the fold interpretation used since zones that might be thought of as disconnected pods are connected into continuous horizons.

3.4.7 Reliability of Geological Reserves

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

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

On a small gold deposit modeled with PC Mine (P. Clarke, personal communication, 1986) changing only the anisotropy parameters from isotropic to about the same anisotropy used for Vangorda and Grum increased the tonnage above cutoff by about 5% and the average grade above cutoff by 1 to 2%. This may have no relevance to Vangorda but it does give some idea of the effect of changing only computational parameters.

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

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

3.4.8 Additional Work Required

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

inferred to exist.

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

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

Considerable metallurgical work on the fill-in drill core should be carried out in order to answer the question of the extent of oxidation of Vangorda massive sulphides, especially in the southeast part of the deposit.

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

Table 3.76

CURRAGH RESOURCES

JULY 10, 1986

VANGORDA DEPOSIT - GEOLOGICAL RESERVES - 86-07 MODEL

in-situ reserves - no adjustments

entire deposit, irrespective of pit outlines

plus 6 %

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

4 % to 6 %

rock type	% of ore type	tonnes (x1000)	density (tn/bcm)	Pb (%)	Zn (%)	Pb + Zn (%)	Ag (g/t)	Au (g/t)	
4A	1	58.6	1,194.40	2.83	1.86	3.02	4.88	26.48	0.46
4C	2	9.5	194.12	3.27	2.34	2.08	4.42	25.82	0.79
4EC	3	14.1	286.99	3.68	2.28	2.40	4.68	33.18	0.85
4E	4	10.2	207.58	3.76	1.90	2.87	4.77	37.94	0.14
4EG	5	6.7	137.28	3.75	2.47	2.76	5.23	36.96	0.76
4EH	6	0.9	19.13	3.78	2.43	2.95	5.38	34.31	0.41
TOTAL	100.0	2,039.51	3.16	2.01	2.81	4.83	29.30	0.53	

minus 4 %

rock type	% of ore type	tonnes (x1000)	density (tn/bcm)	Pb (%)	Zn (%)	Pb + Zn (%)	Ag (g/t)	Au (g/t)	
4A	1	36.2	3,721.08	2.92	0.90	1.44	2.34	12.48	0.33
4C	2	42.8	4,393.21	3.27	0.79	1.00	1.79	14.50	0.65
4EC	3	17.9	1,842.51	3.67	1.33	1.34	2.67	21.57	0.94
4E	4	2.9	299.17	3.65	1.50	1.40	2.90	17.48	0.28
4EG	5	0.1	5.88	3.23	0.67	0.86	1.53	11.66	0.65
4EH	6	0.1	9.09	2.99	0.39	0.49	0.88	5.95	0.08
TOTAL	100.0	10,270.94	3.23	0.95	1.23	2.18	15.12	0.57	

TOTAL DEPOSIT (all grades)

rock type	% of ore type	tonnes (x1000)	density (tn/bcm)	Pb (%)	Zn (%)	Pb + Zn (%)	Ag (g/t)	Au (g/t)	
4A	1	29.9	5,297.81	2.89	1.28	2.03	3.31	17.60	0.38
4C	2	25.9	4,588.75	3.27	0.86	1.04	1.90	15.00	0.66
4EC	3	12.1	2,150.90	3.67	1.47	1.51	2.98	23.37	0.93
4E	4	2.9	510.92	3.69	1.67	2.02	3.69	26.03	0.23
4EG	5	28.5	5,061.05	3.96	4.45	5.74	10.19	63.24	0.75
4EH	6	0.7	118.46	3.62	5.62	4.36	9.98	69.33	0.48
TOTAL	100.0	17,727.88	3.42	2.14	2.78	4.92	31.24	0.62	

plus 4 %

rock type	% of ore type	tonnes (x1000)	density (tn/bcm)	Pb (%)	Zn (%)	Pb + Zn (%)	Ag (g/t)	Au (g/t)	
4A	1	21.1	1,576.72	2.83	2.18	3.42	5.60	29.69	0.51
4C	2	2.6	195.54	3.27	2.37	2.07	4.44	26.05	0.80
4EC	3	4.1	308.40	3.68	2.32	2.50	4.82	34.11	0.84
4E	4	2.8	211.75	3.75	1.91	2.90	4.81	38.11	0.15
4EG	5	67.8	5,055.17	3.96	4.45	5.74	10.20	63.30	0.75
4EH	6	1.5	109.36	3.67	6.05	4.68	10.74	74.60	0.52
TOTAL	100.0	7,456.94	3.68	3.78	4.92	8.71	53.46	0.69	

OTHER GEOLOGICAL RESERVE ESTIMATES FOR THE ENTIRE DEPOSIT *Table 3.17*

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

	tonnes (x1000)	density (tn/bcm)	Pb (%)	Zn (%)	Pb+Zn (%)	Ag (g/t)	Au (g/t)	total metal (tonnes)	% variance (new-old)/old
high grade (+4%)	8,528	4.0 ?	3.16	4.96	8.12	60.33	0.69	692,474	-6.2
low grade	11,431				not determined				
total deposit	19,959								

KERR-ADDISON / PAXTON (recalculated to 3W to 29E)

high grade (+4%)	6,942	4.0 ?	nd		8.67	nd		601,871	7.9
low grade	9,139	3.5 ?			not determined				
total deposit	16,081								

THIS CALCULATION (3W to 29E)

high grade (+4%)	7,457	3.68	3.78	4.92	8.71	53.46	0.69	649,224	
low grade (-4%)	10,271	3.23	0.95	1.23	2.18	15.12	0.57		
total deposit	17,728	3.42	2.14	2.78	4.92	31.24	0.62		

Table 3.8

 OTHER GEOLOGICAL RESERVE ESTIMATES FOR THE NORTHWEST PART OF THE DEPOSIT

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

	tonnes (x1000)	density (tn/bcm)	Pb (%)	Zn (%)	Pb+Zn (%)	Ag (g/t)	Au (g/t)	total metal (tonnes)	% variance (new-old)/old
high grade (+4%)	6,751	4.20	3.5	4.6	8.1	50.7		546,831	-11.4
low grade (-4%)									
total deposit									

80-10 MODEL (i.e. the V1 model)

high grade (+4%)	5,209	3.83	3.3	5.2	8.5	54.5	0.65	441,608	9.8
low grade (-4%)	4,165								
total deposit	9,375	3.70							

KERR ADDISON / PAXTON (recalculated to 3W to 13E)

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

THIS CALCULATION (3W TO 13E)

high grade (+4%)	5,471	3.59	3.9	5.0	8.9	55.6	0.72	484,718	
low grade (-4%)	5,695								
total deposit	11,166	3.39	2.4	3.1	5.5	36.7	0.74		

Table 3.19

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

 86-07 TO 79-12 MODEL

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

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

 86-07 TO 81-10 MODEL

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

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

NOTE: in both of these comparisons there is some doubt as to the limit of the volume modeled to the NW. The 86-07 model is definitely quoted for 3W to 13E, the older models may only cover 2W to 13E or possibly 2W to 12E. In the former case the 86-07 tonnage would be about 35,000

tonnes too high, in the latter they would be about 160,000 tonnes too high. In the latter case the deposit volume and tonnage would be within 1%.