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VOLUME 1

WHITEHORSE PRESENTATION

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GEOLOGY, EXPLORATION POTENTIAL

and MINERAL RESOURCES

of ANVIL DISTRICT - Yukon

Curragh Inc.

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Report WH9306

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INTRODUCTION

The orebodies of Curragh Inc.'s Faro Division occur in the Anvil Range, central Yukon. The five known deposits comprise one of the major lead-zinc-silver districts of the world. Pre-mining in-situ mineral inventory has been estimated at over 150 million tonnes. Production to date has totalled 62 million tonnes mainly from the Faro deposit.

This report summarizes the geology and exploration of the district, its mineral inventory and the potential for further discoveries. The geology of the Anvil District is described in detail by Jennings and Jilson (1986) and Pigage (1989). Exploration methods are described by Chisholm (1957), Aho (1966), Brock (1973) and an unpublished manuscript by Jennings and Simpson (1984).

LOCATION AND ACCESS

The Anvil District is located 200 air kilometres northeast of Whitehorse (Figure 1). Access is by all weather highway, approximately half of which is paved. The route runs north from Whitehorse to Carmacks on the Alaska and north Klondike highways and then east from Carmacks on the Campbell Highway. The Total highway distance from Whitehorse is 360 km.. Concentrates are hauled from the Faro concentrator to tidewater by B-train tractor trailer units a one way distance of 550 km. along the above route, plus the south Klondike Highway to Skagway, Alaska.

HISTORY

Lead-zinc-silver ore was first discovered in the Anvil Range in 1953. The Vangorda deposit was delineated by drilling between 1955 and 1956 but the reserves defined did not warrant development. Exploration resumed in the early 1960's leading to the discovery of the Swim deposit in 1964 and Faro in June of 1965. Drilling and underground work from 1964 to 1967 defined a large deposit at Faro. A feasibility study was completed in 1967 and a production decision was made in August of that year. The concentrator began production in September 1969 at 5,000 tonnes per day; the mill was expanded to 6,000 tonnes per day the following year and again to 9,000 tonnes per day in 1974. The Faro discovery lead to a major staking rush and considerable further exploration in the ensuing few years but no new finds. After considerable detailed exploration work, discoveries resumed when the Grum deposit was discovered in 1973 followed by the Dy deposit in 1976. In 1979 the operator of the Faro mine, Cyprus Anvil Mining Corporation, purchased the mineral rights to all the other deposits and began to plan for development of Grum to supplement the Faro operation.

Mining at Faro continued until 1982 when low metal prices and high operating costs compounded by the debt load of Cyprus Anvil's rapid expansion caused a mine closure

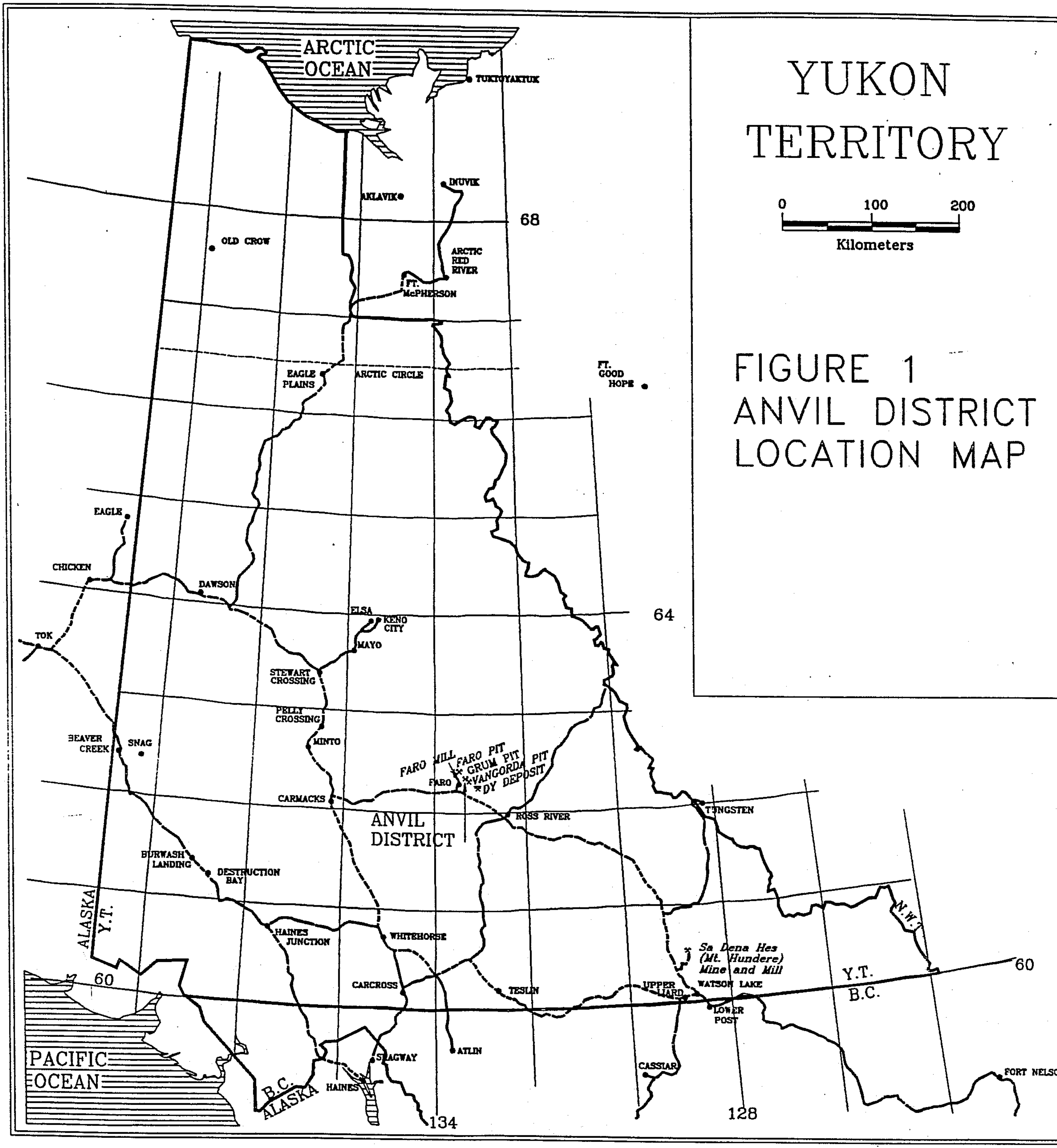


Figure 1. Location of Curragh Inc. Yukon Operations at Faro and Watson Lake. Concentrate is hauled to tidewater at Skagway, Alaska

and the failure of Cyprus Anvil. The Faro mine was re-opened in 1986 after being purchased by Curragh Resources Corporation, a predecessor to Curragh Inc., in 1985.

On re-opening mill tonnage throughput was increased to 11,600 tonnes per day and has gradually been further increased to it's current 13,500 tonne per day average. Curragh operated Faro as an open pit until mid-1992 when it was depleted. Salvage of small tonnages of high grade ore in the original pit walls continued into 1993. At the time of writing approximately 30,000 tonnes of ore remained at Faro below the main haulage ramp. An underground mine was developed in the southwest pit wall in 1990, it closed in 1992 also due to depletion of reserves. The Faro pit has now been converted into a tailings pond.

To provide further ore feed for the Faro concentrator the Vangorda and Grum deposits 14 km. to the southeast were developed as satellite pits in 1989. Mining was carried out in the Vangorda pit from 1990 to early 1993, approximately one million tonnes of ore remain. Pre production stripping has been carried out at Grum from 1990 to 1993 but only a small amount of ore has been released. Curragh's operations are currently in temporary closure due to poor market conditions.

Table 1 provides details of millfeed by year since the start of operations in 1969. Table 2 reconciles mill feed to mined quantities and Table 3 summarizes the quantities mined to date from each deposit. Production from, the Zone 3 Faro pit compares remarkably well to the reserve that formed the basis of the 1985 feasibility study for reopening the Faro mine (Table 2). To some extent this excellent comparison is an illusion since the reserve was calculated at a 4% Pb+Zn cutoff grade but Faro has mined to a 3% cutoff for several years. The extra tonnage from the change in cutoff presumably has compensated for tonnage lost when the northeast pit wall was moved to provide a greater safety berm. This reconciliation may change somewhat as various target areas in the Zone 1 pit are incorporated.

Curragh controls 2,355 claims and 67 mineral leases which form a contiguous land package encompassing 41,500 hectares. There are no other companies actively exploring in the district however several companies have various minority interests in some of the claims. A separate report (WH9301A) details the land holdings in the district at the time of writing.

REGIONAL GEOLOGY

The Anvil district is underlain by early Palaeozoic thru early Triassic sedimentary strata which, in a general sense, are part of the Selwyn Basin. The strata belong to the Cordilleran miogeocline and all but the Pennsylvanian and younger have strong affinities to ancestral North America. Chert and basalt of Pennsylvanian and Permian age may be allocthonous oceanic rocks. More certainly exotic terranes, including the arc related Yukon Tannana and oceanic Slide Mountain Terranes, both mainly Palaeozoic, abut the

TABLE 1
HISTORIC MILL PRODUCTION STATISTICS

Year	Millfeed Tonnes	%Pb+Zn	%Zn	%Pb
CYPRUS ANVIL'S OPERATING YEARS				
1969	394,629	9.10	5.60	3.50
1970	1,779,008	10.80	6.40	4.40
1971	2,424,930	11.71	6.79	4.92
1972	2,635,399	10.88	6.24	4.64
1973	2,629,956	11.25	6.37	4.88
1974	2,653,543	10.11	5.60	4.51
1975	2,925,701	9.44	5.41	4.03
1976	1,519,550	8.14	5.48	2.66
1977	3,116,212	7.62	4.88	2.74
1978	3,280,414	8.31	5.14	3.17
1979	2,823,000	8.54	5.28	3.26
1980	2,825,000	7.80	4.68	3.12
1981	2,758,603	7.70	4.80	2.90
1982	1,635,241	7.60	4.80	2.80
CURRAGH'S OPERATING YEARS				
1986	1,639,000	7.62	4.62	3.00
1987	4,539,000	8.24	4.93	3.31
1988	4,126,000	8.49	4.87	3.62
1989	4,379,084	7.62	4.69	2.93
1990	4,714,035	7.88	4.88	3.00
1991	4,126,588	7.56	4.53	3.03
1992	4,548,744	7.78	4.58	3.20
1993	1,030,483	7.62	4.54	3.08
CYPRUS ANVIL (1969–1982)	33,401,184	9.19	5.50	3.70
CURRAGH (1986–1993)	29,102,934	7.90	4.73	3.16
TOTAL MILLED (1969–1983)	62,504,118	8.59	5.14	3.45

TABLE 2
PRODUCTION RECONCILIATION, FARO OPERATION

	Tonnes	Pb+Zn %	Zn %	Pb %
TOTAL MILLLED 1969 to 1993	62,504,118	8.59	5.14	3.45
MILLED FROM VANGORDA OREBODY	4,857,650	8.51	4.66	3.85
MILLED FROM FARO OREBODY	57,646,468	8.60	5.18	3.42
REMAINING IN FARO STOCKPILES	1,762,159	4.45	2.77	1.68
TOTAL MINED FROM FARO OREBODY	59,408,627	8.47	5.11	3.36
MINED FROM FARO UNDERGROUND	1,780,241	11.47	7.01	4.46
MINED FROM FARO PIT	57,628,386	8.38	5.05	3.33
MINED BY CAMC FROM FARO PIT	34,201,184	9.08	5.43	3.65
MINED BY CI FROM FARO PIT	23,427,202	7.36	4.50	2.86
1985 FEASIBILITY STUDY RESERVE	23,763,000	7.30	4.40	2.90

TABLE 3
SUMMARY OF ORE MINED TO DATE, ANVIL DISTRICT

AREA OR DEPOSIT	Tonnes	Pb+Zn %	Zn %	Pb %
MINED BY CAMC FROM FARO PIT	34,201,184	9.08	5.43	3.65
MINED BY CI FROM FARO PIT	23,427,202	7.36	4.50	2.86
MINED FROM FARO UNDERGROUND	1,780,241	11.47	7.01	4.46
SUBTOTAL MINED FROM FARO TO DATE	59,408,627	8.47	5.11	3.36
MINED FROM VANGORDA PIT	5,662,712	8.11	4.44	3.67
MINED FROM GRUM PIT	52,000	5.48	3.65	1.83
GRAND TOTAL MINED TO DATE	65,123,339	8.44	5.05	3.39

southwest boundary of the district along a major suture, the Vangorda Creek Fault. The district is part of the Omenica Belt and shows the early Mesozoic regional metamorphic overprint, complex polyphase fold deformation and mid-Cretaceous granitic plutonism that marks the ancestral North American part of that belt in Yukon. A major northwest trending Cretaceous granitic body, the Anvil Batholith, is the central feature of the district (Figure 2). Palaeozoic metamorphic rocks dip northeast and southwest away from the batholith. The five ore bodies occur in Cambrian phyllites or schists along the southwest flank of the batholith. The deeper strata, near the batholith, are more intensely metamorphosed than those less buried or more remote from the batholith. The polyphase deformation of the district is thought to be related to collision of the exotic terranes with the North American block in early Jurassic, causing northeast verging folding, nappe emplacement (including thrust sheets of the exotic sequences), depression of the crust, partial melting of the lower crust, and uprise of granitic magmas by middle of the Cretaceous. Granite emplacement was accompanied by high heat flow, metamorphism and southeast verging folding along shallowly dipping axial planes. The final emplacement of the batholith resulted in large scale extensional displacements as the batholith and its high grade metamorphic carapace forced its way upward through the lower grade strata. The district is located just northeast of a major Cordilleran lineament which marks the locus of the Tintina Fault a transcurrent fault with 500 km. of right lateral displacement.

DISTRICT GEOGRAPHY

The are three major geographic subdivisions of the Anvil Range: the Faro Block, the Vangorda Plateau, and the Swim Basin (Figure 3). These subdivisions correspond to important geologic domains in the district.

The **Faro Block** includes that portion of the district extending from the Tie Fault trace to the northwest of the Faro minesite. The favourable stratigraphic units here have been metamorphosed to amphibolite facies in large part. Rock unit boundaries and layering are largely transposed into the second phase metamorphic foliation which is pervasive in the area and generally dips gently southwest or west. Deep drilling in this area has concentrated on the area down dip of the Faro Deposit also extending along strike a few km to the northwest; one deep hole and a number of shallow holes have been put down between Faro and Grum northwest of the Ski Hill.

The **Vangorda Plateau** is an incised plateau or bench on the south flank of the granitic highlands of Mt. Mye. There are local patches of thick glaciofluvial deposits but fair exposure can be found over much of the area. The area lies between the Tie and Blind Creek faults southwest of the granitic rocks of Anvil Batholith and northeast of Vangorda Creek Fault Zone. In general the second phase metamorphic foliation (S_2) dips shallowly southwest away from the granite as do most rock units on a large scale. In the Grum vicinity first and second phase folds (F_1 and F_2) plunge shallowly northwest; fold plunge

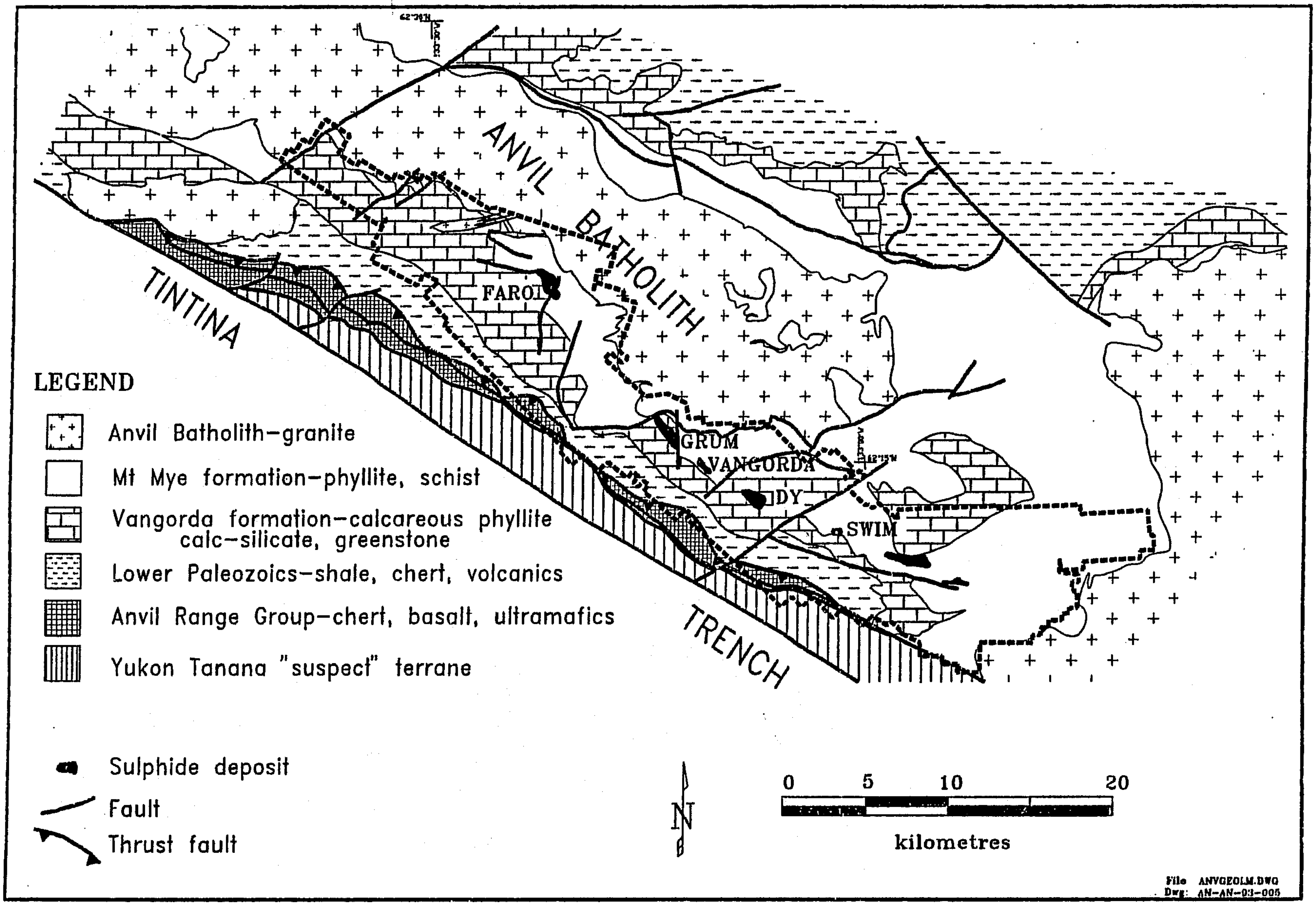


Figure 2. Geological Map of the Anvil District showing the location of known sulphide deposits. Only the named deposits are Zn-Pb-Ag bearing. The dashed-outline is the Curragh claim block.

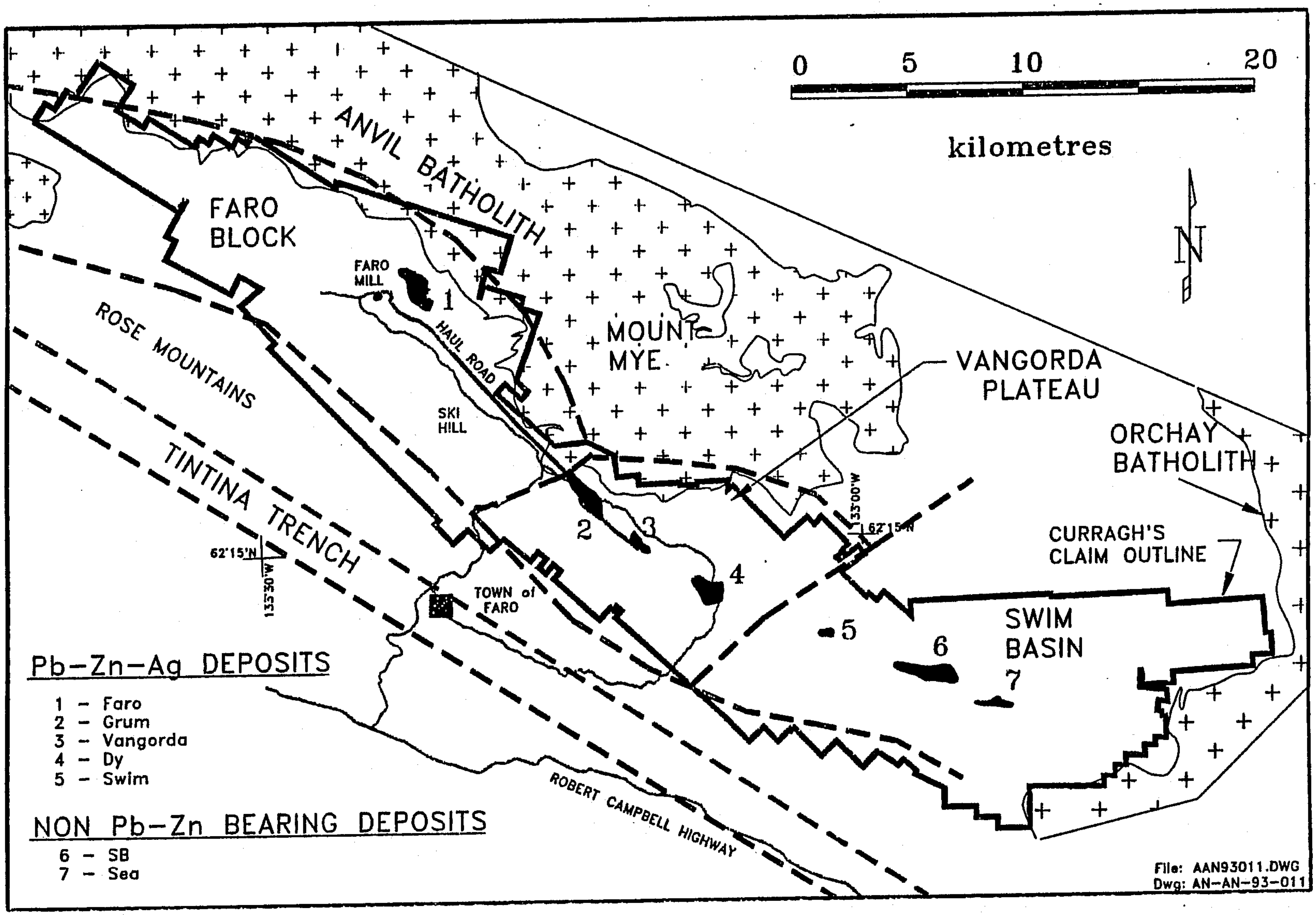


Figure 3. Geographic Regions of the Anvil District

elsewhere is probably also northwest but a reversal may occur between Vangorda and Dy. Greenschist facies metamorphism predominates and second phase transposition is less extensive than in the Faro Block.

The **Swim Basin** is a lowland between the granitic highlands of Mt. Mye on the northwest and Orchay Batholith on the southeast. To the southwest is a ridge of resistant late Palaeozoic volcanics bounded by splays of the Vangorda Creek Fault Zone. The area has few bedrock exposures and is heavily covered by thick, laterally extensive, glacial till. There are many small to moderate sized lakes and large swampy areas. Geophysical surveying and drilling are best done in much of this area in winter. The basin is essentially a structural basin whose periphery is underlain mainly by phyllites of the Mt. Mye formation. There is a large patch of Vangorda formation preserved in the centre of the basin. The basal member of the Vangorda formation is widespread as indicated by geophysical surveys. The area is mainly at greenschist facies and shows similar structural style to the Vangorda Plateau.

SURFICIAL GEOLOGY

All but the highest peaks in the area were ice covered in the last glaciation. Ice flow and transport direction was from southeast to northwest or east to west as evidenced by glacial striae and elongate land forms as well as transport of distinctive rock types such as the serpentinized pyroxenites in the upper Dixon Creek area.

Discontinuous deposits of glacial till and glacio-fluvial deposits are common in the area, however with local exceptions the deposits are not thick. Important exceptions are a buried valley filled with up to 70m of overburden over the subcrop of the Grum deposit and a ridge of highly compacted till up to 40m thick overlying the Vangorda deposit. Thick fluvial deposits also occur in the Rose Creek, the west fork of Vangorda Creek and in the Blind Creek valleys. In Swim Basin there is a widespread, thick, blanket of glacial deposits which significantly inhibits exploration.

Glaciers stripped off most of the weathered mantle, thus most geochemical anomalies are transported either hydromorphically or physically by downhill movements or ice transport.

STRATIGRAPHY

The stratigraphic sequence of Anvil District ranges in age from latest Precambrian to Permian. The lower part of the sequence (Silurian and/or earlier) is divisible into three major mappable units (Figure 4). From the base these are non-calcareous metapelite of the Mt. Mye formation, calcareous meta-pelite of the Vangorda formation, and meta-basalt of the Menzie Creek formation (Jennings and Jilson, 1986). All formational names in this sequence are informal. The aggregate thickness for this pre-Silurian sequence is approximately 5km.

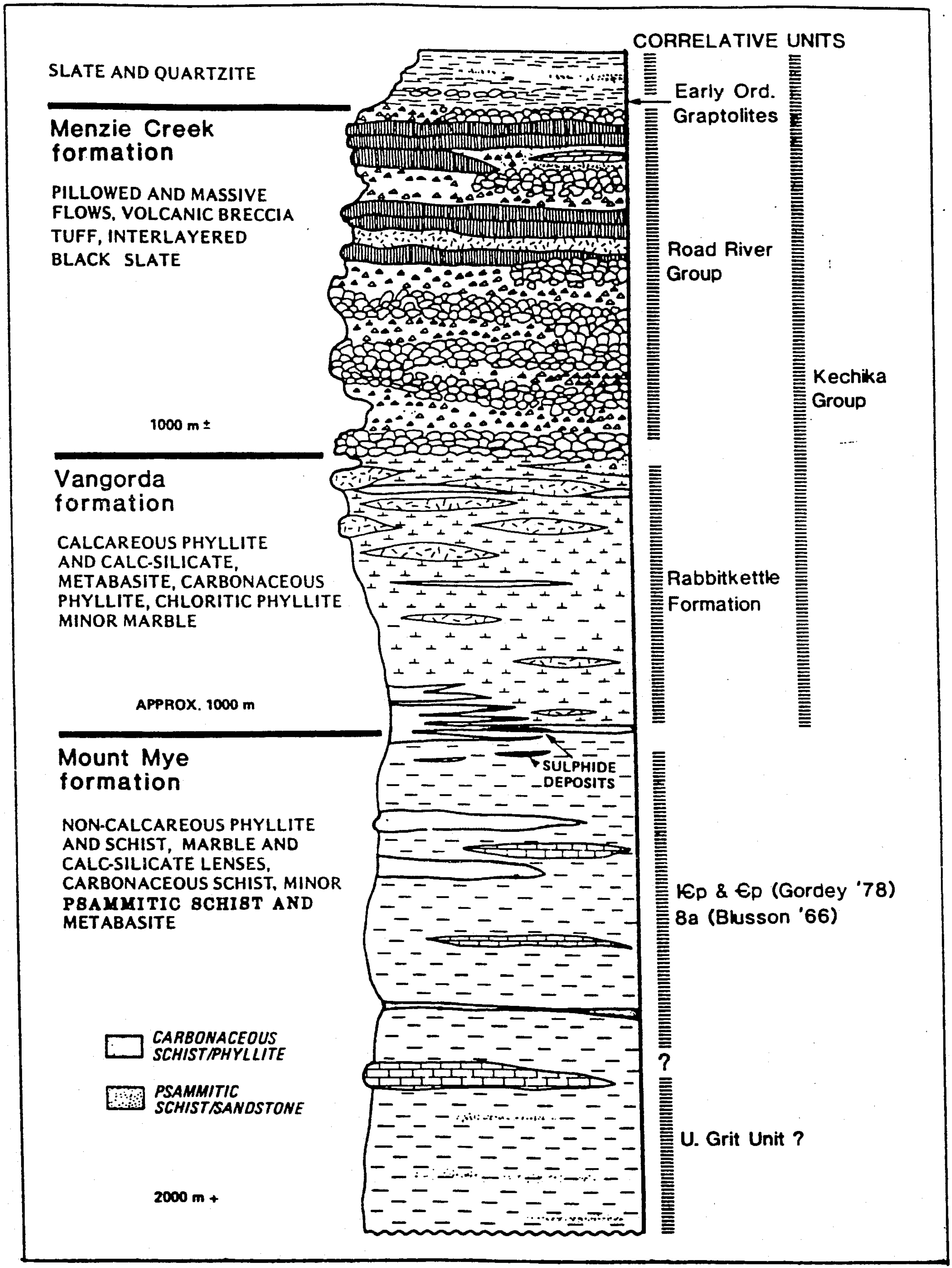


Figure 4. Stratigraphic Column of the older part of the Anvil District section

The strata overlying the above sequence are characterized by shale, chert, coarse clastics rich in chert fragments, minor limestone, and an uppermost basalt unit. Strata of the Devonian-Mississippian Earn Group (Gordey et al., 1982) and Pennsylvanian-Permian Anvil Range Group (Tempelman-Kluit, 1972) are present. All or part of this upper sequence may be allochthonous with respect to the underlying units. The boundary between allochthonous and parautochthonous strata is drawn at the base of the Anvil Range group where red cherts first become prominent in the section. The Earn group locally contains stratiform barite deposits.

The Devonian and younger rocks are not related to the ore deposits in the district and consequently are not discussed further (details can be found in Templeman-Kluit, 1972, and Jennings and Jilson, 1986). The three older units either host the ore deposits or are the environment of exploration thus are considered in more detail below.

Mt. Mye formation consists dominantly of non-calcareous, biotite-muscovite +/- andalusite +/- staurolite +/- garnet schist in areas of amphibolite facies metamorphism and non-calcareous, weakly carbonaceous, light to medium grey muscovite-chlorite phyllite in areas of greenschist facies metamorphism. It contains lesser, interlayered black carbonaceous phyllite or schist, calcitic marble, calc-silicate phyllite or schist, greenstone or amphibolite, and psammitic schist. The formation has a structural thickness of at least 2 km, the base is not exposed. The reddish brown weathering colour of the formation is characteristic and helps distinguish it from non-calcareous portions of the overlying Vangorda formation.

Dark grey to black carbonaceous phyllite or schist members comprise about 10 per cent of the formation. They are more abundant in the upper 400m of the formation. A distinctive assemblage of carbonaceous siliceous phyllite, and black carbonaceous limestone, appears to underlie, or be laterally equivalent to the lowest sulphide horizons.

Coarse-grained, white, calcite marble and calc-silicate also constitute about 10% of the Mt. Mye formation. The marble is light grey, medium crystalline calcite marble with boudins of pelite, amphibolite, and calc-silicate. Marble bodies may be up to 75m thick but are generally only a few tens of meters thick; they can be traced laterally for several kilometres. The calc-silicate lithology is a thinly interbanded sequence of purplish brown biotite pelite and pale green actinolite-epidote calc-silicates. Typically the calc-silicates are spatially associated with the marbles. The calc-silicates are identical to Vangorda formation calc-silicates, but a protolith for the associated clean marbles is not common in the Vangorda formation. The most persistent horizon of lenticular marble and calc-silicate bodies occurs about 500 to 700m below the top of the Mt. Mye formation. The marble plus calc-silicate assemblage appears to underlie the carbonaceous phyllite plus black limestone assemblage noted above. This is thought to be a lower Cambrian stratigraphic unit but alternative concepts such as a recumbent, isoclinal, synclinal infold of Vangorda formation, or thrust interleaved Vangorda and Mount Mye are possibilities which have not yet been ruled out. This problem is of great exploration significance and must be resolved to allow structural modelling of the district to advance.

Metabasite bodies in the Mt. Mye formation are generally only a few meters thick and have small lateral dimensions. Volumetrically they constitute less than 1% of the Mt. Mye formation. They are generally strongly foliated, dark green amphibolites lacking relict igneous texture. Compositions are similar to basalts of the Menzie Creek formation (Jennings and Jilson, 1986). They are interpreted as subvolcanic feeder dykes and sills of the Menzie Creek basalts.

The upper portion of the Mt. Mye 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 unit 8A of Blusson (1966). Correlation with these units would imply the top of the formation is lower Cambrian or possibly middle Cambrian. Jennings and Jilson (1986) suggested that the persistent marble and calc-silicate package may correlate with the widespread early Cambrian limestone conglomerate of Selwyn Basin. Parts of the Mt. Mye formation also resemble rocks underlying those presumed correlative units, implying that the Mt. Mye may include rocks as old as Hadrynian.

The **Vangorda formation** is characterized by light to medium-grey to greenish-grey, calcareous, phyllitic rocks made up of very thin (0.1-2cm) interlayers of medium grey, non-calcareous, weakly carbonaceous, muscovite-chlorite pelite and light grey, generally calcareous quartz-calcite +/- dolomite siltstone. At the higher metamorphic grade of amphibolite facies, the Vangorda formation phyllites are transformed to a thinly banded, pervasively foliated, green, cream, and purplish brown, calc-silicate. Major interbanded units include greenstone and carbonaceous pelite. Minor phyllitic limestone occurs locally. The Vangorda formation varies between 0.5 and 2 km. in apparent thickness. The formation becomes more calcareous up section. The light grey to tan coloured, drusy weathering of the formation is characteristic both within the district and elsewhere.

The greenstone bodies range from 1m to 100m in thickness and are up to several kilometres in length. They comprise approximately 15% of the Vangorda formation and are more prevalent near the top of the formation. Whole rock analyses show that the greenstones are compositionally similar to the overlying Menzie Creek basalts (Jennings and Jilson, 1986). Locally the greenstones contain coarsely crystalline serpentinized pyroxenite subunits, which may be pyroxene cumulates. Most greenstone bodies have medium-grained, equigranular centres with strongly foliated margins. Although marginal contacts of the bodies are superficially conformable, detailed inspection indicates the units are locally slightly crosscutting. The greenstones are thus interpreted as subvolcanic dykes and sills feeders to the Menzie Creek formation. Where the mafic units are thin, the entire body may be a foliated chloritic phyllite, commonly calcereous, with thin white bands of quartz and calcite. Some of these chloritic phyllites contain relict pyroxenes or feldspars, and develop a fine augen texture, while others have flat oviod dark chloritic spots on the foliation after vesicles or pyroxenes.

Typically the Vangorda formation adjacent to the greenstones is a thinly banded, hard, pale green, calcareous, chloritic phyllite. This lithology has been interpreted as a marginal tuff adjacent to basaltic flows (as noted in Jennings and Jilson, 1986). More extensive drill core inspection and additional outcrop exposures indicate that instead it represents a slight contact metamorphic aureole caused by intrusion of the greenstone bodies; further evidence that the greenstone bodies are intrusive. Where the mafic bodies are thin these altered phyllites can be difficult to distinguish from the sill and if the alteration is intense the mafic rock may not be noticeable at all. The greenstones are resistant and dominate outcrop in the district. Since the altered phyllites are always near the greenstones, outcrop examination gives a misleading impression of the chlorite content of Vangorda formation phyllites, compared to drill core examination. The overwhelming bulk of the formation is grey to greenish-grey, but that lithology does not crop out well.

Black, slightly calcareous to dolomitic, carbonaceous pelite members occur throughout the Vangorda formation. Dimensions and lateral continuity of these members are poorly known. The thickest and most extensive of these occurs at the base of the formation; it ranges from only a few tens of meters to 100m in thickness. This basal member becomes thicker in the immediate vicinity of the ore deposits and appears to be laterally equivalent to black, sulphide-bearing, ribbon-banded, carbonaceous, quartzite ores within some of the mineral deposits. Southwest of the Grum and Vangorda deposits the basal member is very siliceous and slightly pyritic enhancing the impression of equivalence to the carbonaceous quartzite ores.

The Vangorda formation is lithologically similar to, though more argillaceous than the 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 late Cambrian through early 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. The formation reaches a maximum structural thickness of 1.5km in the district. Whole rock major element and trace element data (Jennings and Jilson, 1986) imply that the flows of the Menzie Creek volcanic unit are dominantly alkali basalt erupted in a within-plate setting. Similar major and minor element compositions for the metabasites in the Mt. Mye and Vangorda formations suggest the metabasites are subvolcanic feeders for the Menzie Creek formation.

Carbonaceous phyllite and brown siltstone immediately overlying and interbedded with the uppermost Menzie Creek formation northeast of the Anvil Batholith contain graptolites of lower Ordovician to early Silurian age (Tempelman-Kluit, 1972; Gordey, 1983) suggesting correlation of the Menzie Creek volcanics with the widespread Road River Formation black shale and chert to the northeast. The Menzie Creek formation has been traced for 100km along strike and 30km across strike, showing that it is one of the largest of several basaltic units of its age in and around the Selwyn Basin.

FOLD DEFORMATION

The structural and metamorphic history of the Anvil District is complex and of considerable significance to present form and nature of the ore deposits, and hence exploration for them, since all of the deposits have experienced the full deformation history. Five phases of deformation have been recognized in the district. The first two are periods of intense fold deformation and concurrent metamorphism which determined the gross structure of the mineral deposits. The remaining deformations are only locally developed and do not generally form large or significant structures.

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 D_1 early folds are rarely preserved in the district; they are ubiquitously north-easterly inclined to upright, northeasterly verging (shaped like a 'Z' looking northwest) structures with shallow northwesterly or southeasterly plunging axes.

During the second deformation event (D_2), S_1 was strongly crenulated and ubiquitous close to tight mesoscopic folds (F_2) in S_1 were produced. Primary bedding (S_0) transposed into near parallelism with the S_2 foliation. Parallel to the axial planes of the 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 axial plane foliations dip shallowly to the southwest or northeast, with fold axes subparallel to F_1 fold axes. Southwest of the Anvil Batholith the S_2 surfaces dip dominantly southwest, and F_2 minor folds have southwest vergence (shaped like a 'S' looking northwest). Northeast of the batholith S_2 surfaces dip dominantly northeast, and F_2 minor folds appear, on the basis of limited evidence, to have northeast vergence. The shallow dip of F_2 axial planes, the isoclinal nature of F_2 folds, and the transposition of bedding into foliation creates some of the more important exploration characteristics of the district. Rock units are flat lying or shallowly dipping on the average (Figure 5) although there are local exceptions (e.g. Grum). Electromagnetic methods must be able to couple well with and resolve multiple flat lying conductors. The shallow dip of the area also means exploration targets tend to present their largest dimension to a vertical drill hole.

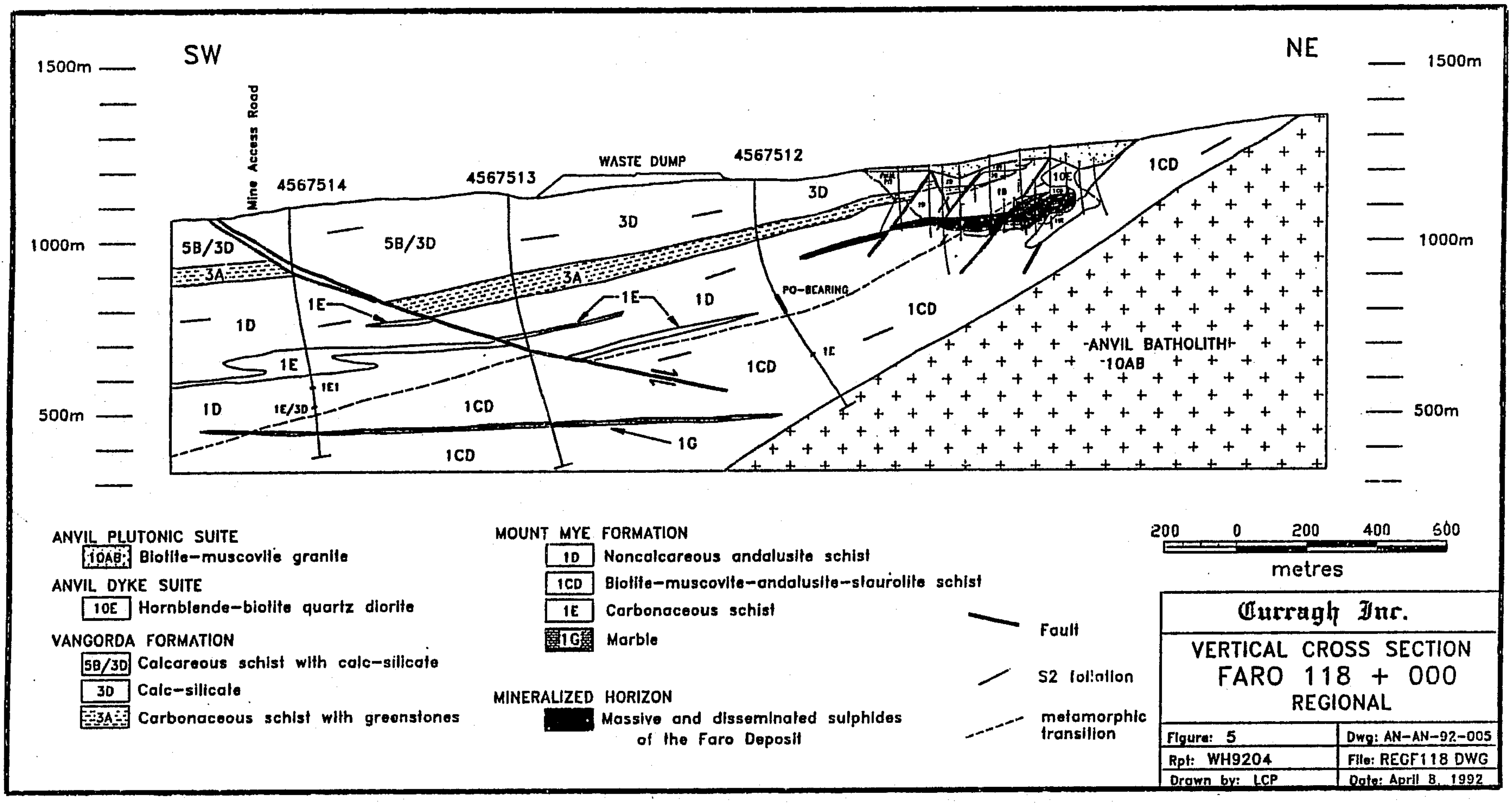


Figure 5. Cross Section through the centre of the Faro Deposit showing large scale transposition of layering into S_2 as is characteristic of the Faro Block

The largest megascopic folds known to have been formed during D_2 are those at the Grum Deposit (Figure 6 and 7) and comparable folds in the Swim Deposit (Figure 8). 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 intense with tight mesoscopic folds developed in nearly pervasive S_2. D_4 minor folds have appreciable mica growth along S_4 axial plane crenulation cleavages.

METAMORPHISM

Metamorphism was concurrent with deformation and was most intense during the major D_1 and D_2 folding deformations. D_1 metamorphism has been largely overprinted by the later D_2 metamorphism. Metamorphic grades during these two events appear to be comparable since mica mineral assemblages between microlithons (i.e. S_1 foliations) are similar to those defining the S_2 foliation surfaces. The rest of the discussion will focus on the D_2 metamorphism.

Metamorphic grade ranges from upper amphibolite facies (sillimanite-muscovite zone) to lower greenschist facies (muscovite-chlorite zone) in a low pressure Buchan type facies series. In pelites adjacent to the intrusions the typical assemblage is andalusite-staurolite-garnet-biotite-muscovite-quartz-plagioclase with local fibrolite and cordierite. Lower greenschist facies pelites contain the assemblage muscovite-chlorite-quartz-plagioclase.

Metamorphic isograds are roughly concentric about the Anvil Batholith. Locally isograds are truncated and juxtaposed by the late D_2 extensional faults. The Faro deposit (closer to the Batholith) is metamorphosed to amphibolite facies. All other deposits are metamorphosed to lower greenschist facies. This difference in intensity of metamorphism is reflected in decreased grain size and increased degree of mineral intergrowth in the less metamorphosed deposits (Tempelman-Kluit, 1970). This has a significant impact on metallurgical response of Anvil district ores.

IGNEOUS INTRUSIVES

During the later stages of the deformation history a large granitic body (Anvil Batholith) was intruded into the metamorphic sequence. Anvil Batholith ranges in composition from a biotite-muscovite peraluminous granite to a metaluminous to peraluminous hornblende-biotite granodiorite (Pigage and Anderson, 1985). Textures include equigranular massive, megacrystic massive, and various strongly to weakly foliated variants. Foliation within the intrusive rocks is concordant with S_2 surfaces in the surrounding metasediments. Several K-Ar ages on the granitic rocks yielded ages of 85-100 Ma (Tempelman-Kluit, 1972). Rb-Sr isochron ages of 99-100 Ma (Pigage and Anderson, 1985) and unpublished zircon model ages (Mortenson, pers comm.) are

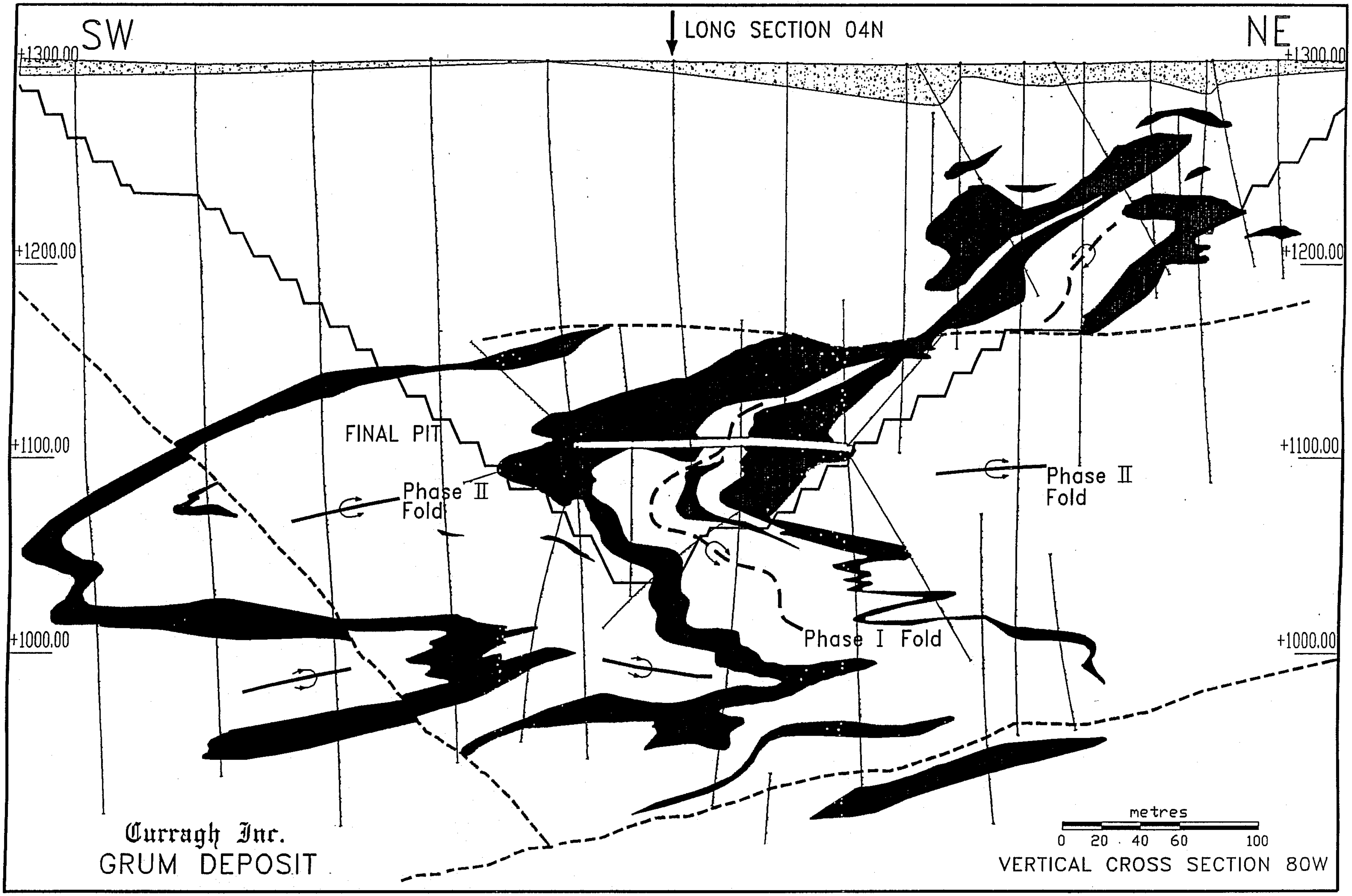


Figure 6. Cross Section through the Grum Deposit showing second phase S-shaped folds superimposed on a first phase Z-shaped fold

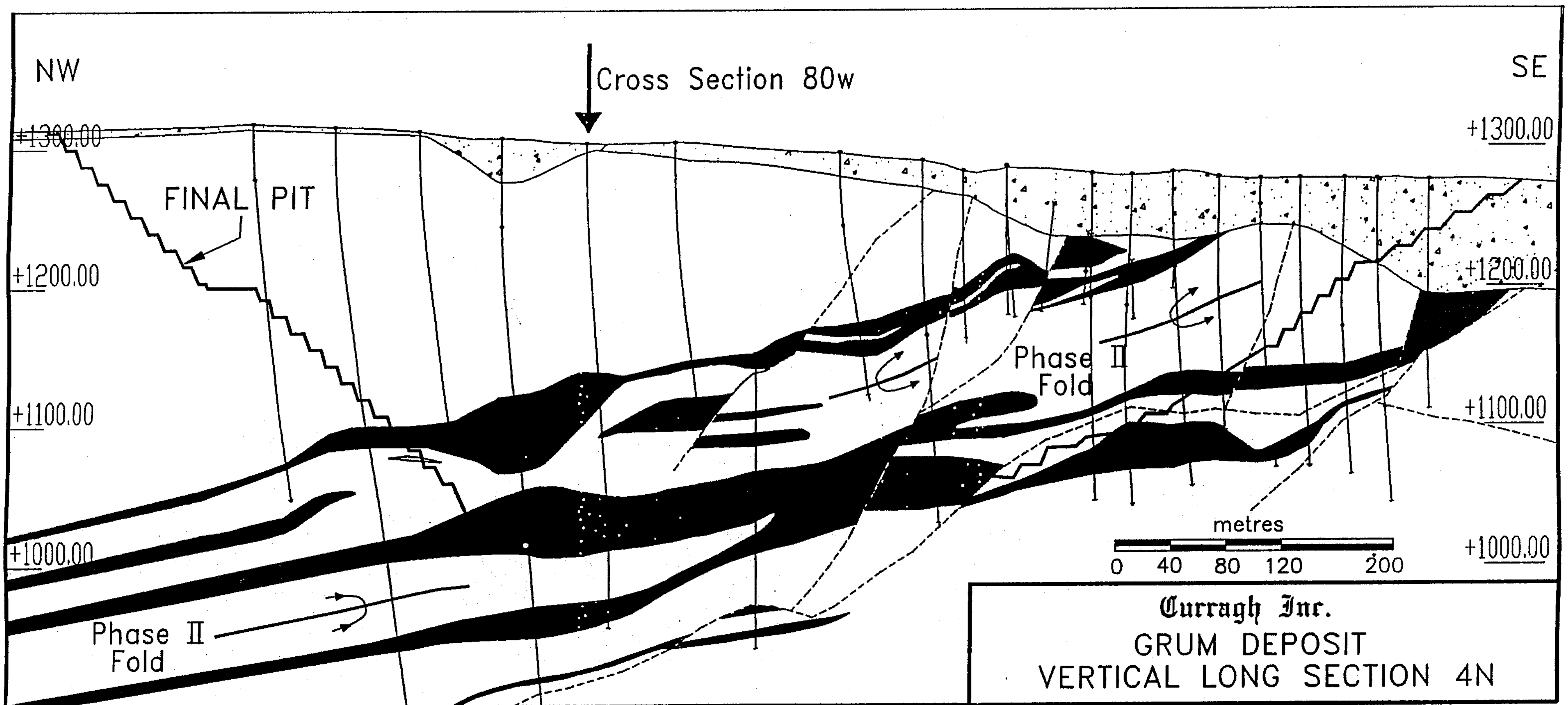
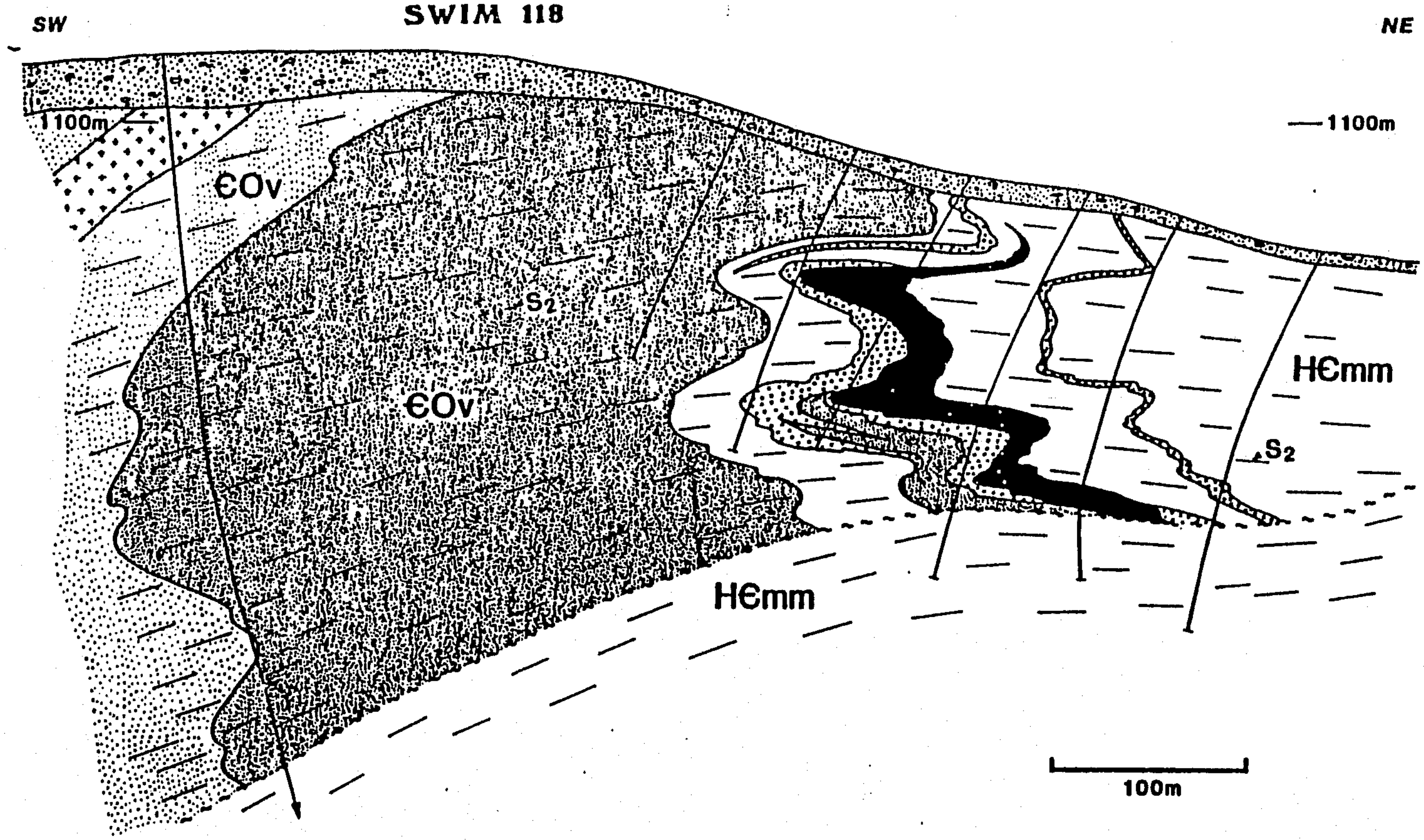


Figure 7. Longitudinal Section parallel to the second phase fold hinge at Grum, showing the 11° northwest plunge



massive ore
quartzose ore
graphitic phyll.
metabasite

Figure 8. Cross-section 118 through the centre of the Swim Deposit

concordant with the K-Ar ages and indicate rapid cooling after high-level emplacement.

Anvil Batholith and surrounding metasedimentary rocks are crosscut by two families of post-tectonic dykes. The majority of the dykes are northeast-trending, medium to dark green, porphyritic, unfoliated, hornblende-biotite quartz diorite. These quartz diorite dykes appear to be associated with late extensional faults. Unfoliated, pale tan, smoky quartz-feldspar porphyry also occurs as late crosscutting dykes. The dyke suites have not been extensively isotopically dated; their absolute ages are thus uncertain. One important date has been obtained on a unsheared quartz-feldspar porphyry intruding the Tie fault zone (see below). This zircon age indicates that the dyke cooled at 100ma, (Mortenson, pers. comm., 1991) essentially the same age as the Batholith. This leaves little doubt that the Tie Fault is coeval with late stage high level emplacement and rapid cooling of the Batholith.

FAULTING

During the first and second phases of deformation low angle faulting appears to have occurred but the details are as yet poorly understood. Thrust faulting during the first phase of folding is likely and several candidates have been found. The most obvious of these are along the north flank of the district (Gordey, 1983) but additional, smaller, thrusts may occur in the vicinity of the ore deposits. Gordey and Irwin (1987) have indicated a major thrust northwest of the Faro mine (the Faro Thrust). It is not likely that such a structure crops out in that area but a similar thrust fault could exist if it is below the level of exposure in the core of the district. Brown and McClay (1992 and in preparation) have hypothesized the existence of such a thrust which would reach the surface along the north flank of the district and presumably would correspond to the Two Pete Thrust recognized there by Gordey and Irwin (op.cit.).

During the second phase of deformation, S_2 parallel displacement occurred along a attenuated second phase fold limbs. These S_2 parallel faults (tectonic slides) have been recognized in the Grum deposit, and are thought to exist at Vangorda. The role of these S_2 parallel structures is not well understood but they are thought to have displacements of no more than few 100's of metres.

Post folding and post metamorphism faulting is widespread and of great significance for exploration in the district. 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. In the later stages of emplacement large extensional fault displacement occurred along the margins of the Batholith (Pigage and Jilson, 1985). S-C mylonitic banding within these fault zones, and in the granitic footwall of some, is consistent with development of the faults during late D_2 deformation. These faults determine the present day limits of several of the deposits. The faults with known offset appear to result from extension along the trend of the District and the Tintina Fault. The Tie Fault is one of the best examples of such a structure. As noted above, its age is well constrained at 100 ma. and it is coeval with the final stage of emplacement of the Batholith. These relationships suggest that the Anvil Batholith may have been intruded during the strike slip regime of the Tintina. Figure 9 is a longitudinal section through Grum and the Tie Fault. The slip line of the fault is in the plane of the section. The section shows the relationship of Firth to Grum and the relatively large displacements of this family of faults. Other similar structures occur at the southeast end of Grum, between Dy and Vangorda, and below Swim. These structures typically juxtapose a less intensely metamorphosed hanging wall against a more metamorphosal footwall. The identification of these structures is significant for exploration as they have the potential to open up large gaps in the stratigraphy because of their extensional nature and large displacement.

The youngest faults of the district are steeply dipping and diversely oriented. One of the most prominent sets strikes northeast. A second important set strikes approximately north-south. The northeast striking set is subvertical and commonly shows left lateral strike slip offset. This set may be second order structures to the Tintina Fault. The best example of such a fault is the Blind Creek Fault which offsets the favourable trend of deposits by 1.3 km. A group of faults of this set, northwest of the Faro mine, may have similarly offset the favourable trend there, however, this concept has not yet been drill tested. Many late faults show subhorizontal slickenslides, suggesting the last displacements were strike slip This is true even of structures in the Faro Pit which are well constrained to have small horizontal displacement compared to the vertical component of movement. Jennings (personal comment 1972) and Brown and McClay, (1992, in preparation) have noted late stage northeast directed small displacement, post metamorphic, thrust faults in the Faro and Vangorda pits, respectively.

ORE DEPOSITS

The lead, zinc, silver deposits of the Anvil District belong to the sediment hosted, stratiform, massive pyritic sulphide class (Gustafson and Williams, 1981; Large, 1980) also referred to as sedimentary exhalative (sedex) deposits (Carne and Cathro, 1982). They occur either as a thick sulphide lens with little or no interbanded metasedimentary rocks (e.g. Faro) or as several thinner lenses stacked approximately one above the other with substantial metasedimentary interlayers (e.g. Grum and Dy). The deposits and their ore types are described in more detail in Jennings and Jilson (1986) and Pigage (1990).

All deposits are composed of a small number of different ore types. The ore types are broadly divisible into massive sulphides and disseminated sulphides in quartzite. There are pyritic, baritic, pyrrhotitic and carbonate bearing variants of the massive sulphide ore types and carbonaceous and non-carbonaceous variants of the disseminated ore types. Ore type zoning is pronounced in the deposits. Stratigraphically lower and distal ore types are disseminated carbonaceous quartzites, upper and proximal types are baritic massive sulphides. An idealized and vertically exaggerated section through a model ore horizon is shown in Figure 10.

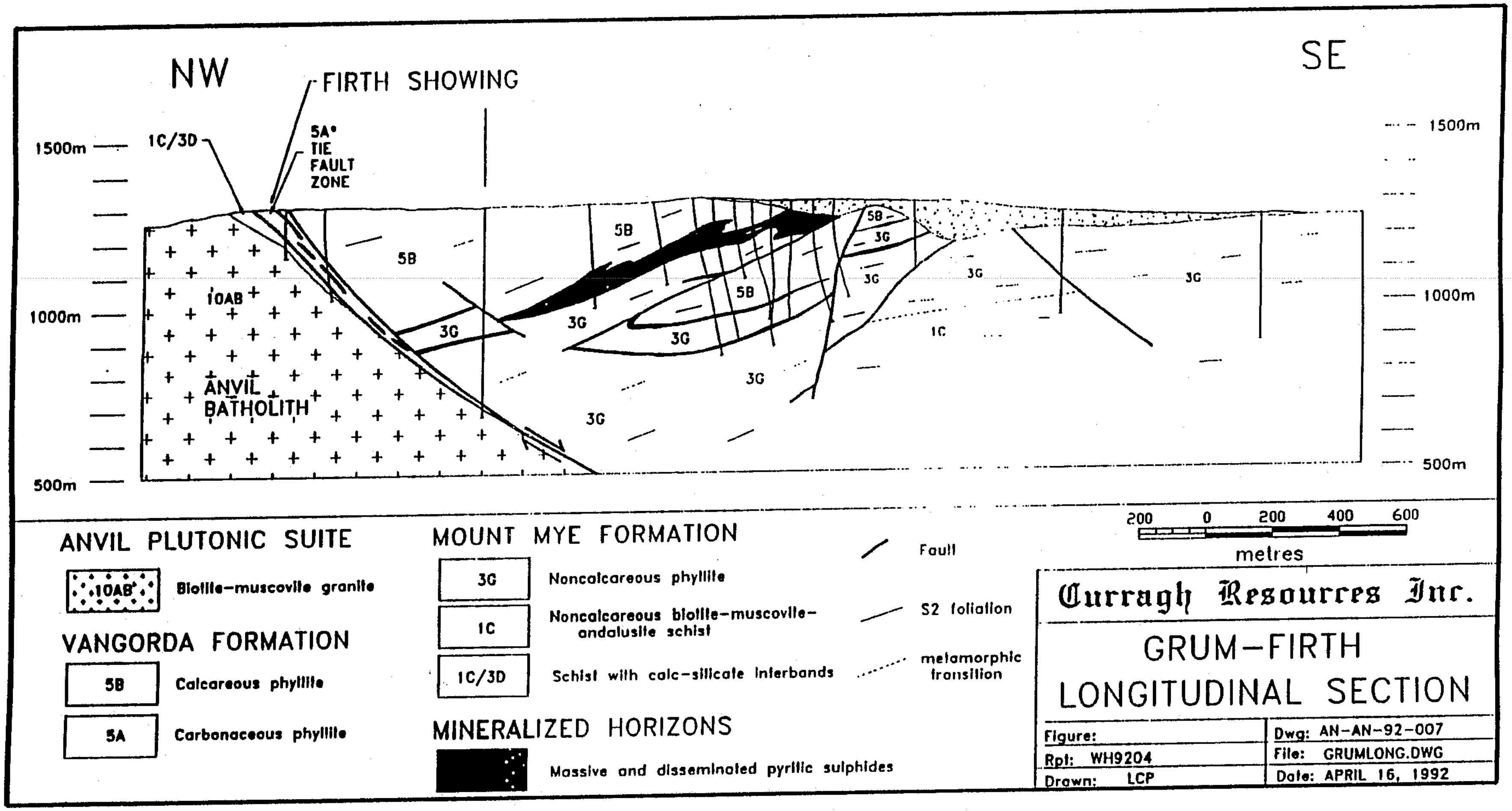


Figure 9. A longitudinal section along the plunge of the Grum fold structure. The section shows the interaction of the Grum fold with the extensional Tie fault. The Firth showing may represent mobilized or detached mineralization related to Grum. Such an interpretation implies the Tie fault slip line is directly down the dip. This direction is confirmed by S and C bands in the mylonitic margin of the Batholith.

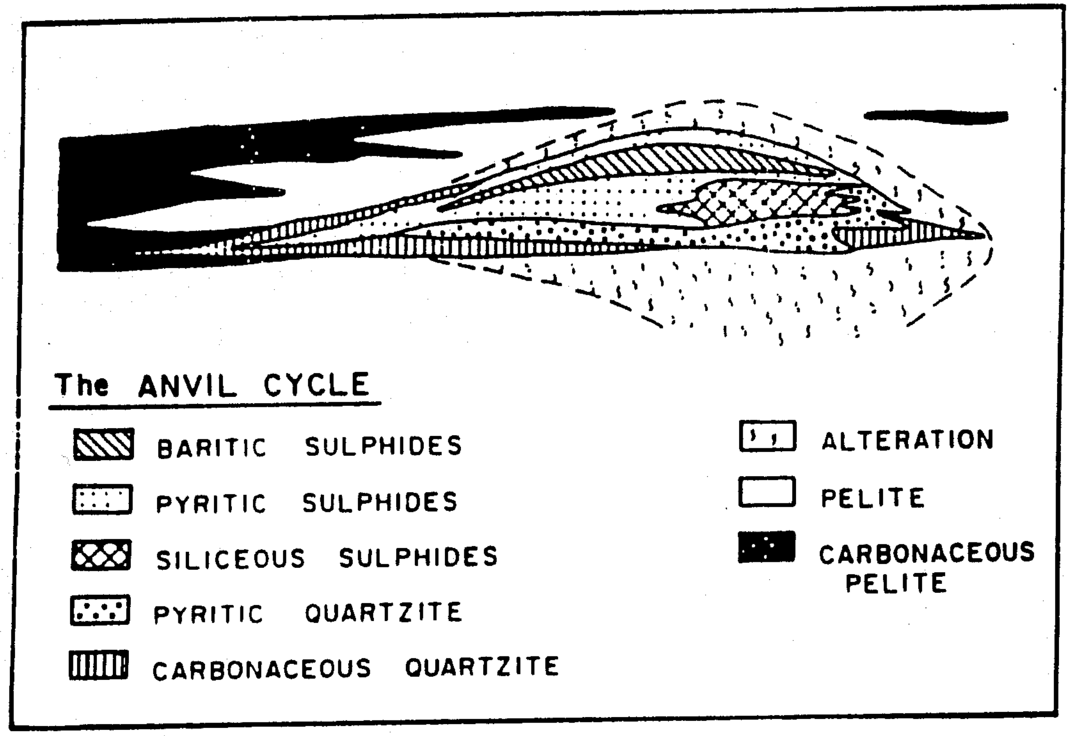


Figure 10. Idealized Anvil cycle of ore type facies variations based largely on the Faro and Vangorda deposit. The section is greatly vertically exaggerated.

The mineralization occurs in the thin, laterally extensive, sulphide sheets or horizons are deformed into complex fold structures. The deposits are elongate parallel to the D_2 fold axes and associated lineations in the host metasediments. The Faro deposit, which superficially does not appear to be complexly folded, actually showed great internal complexity in the geometry of high grade and waste layers.

Present deposit lengths are generally two to three times widths; unfolded, the deposits are interpreted to have had an amoeboid shape with diameters up to 4,000m. Individual sulphide horizons commonly are 10 to 40m in thickness. The upper contact and generally the lower contacts of sulphide horizons are sharp while lateral extensions grade into the enclosing host rocks. Parts of some deposits, particularly Vangorda (Figure 11), show a footwall rich in quartz and iron sulphides/oxides and enriched in copper and gold relative to zinc. This may be a footwall silicified and sulphide impregnated feeder zone.

All deposits show a variably developed, white mica-dominant, alteration overprint in the wall rocks. This results in the phyllites having a bleached appearance. Less intensely altered chlorite-muscovite ± pyrrhotite ± carbonate variants of the alteration are also found widely. At lower metamorphic grade this alteration tends to be found in the footwall of the ore horizons. At the Faro deposit, this bleaching/alteration halo is particulary intense and encloses the entire mineralized sulphide lens. The halo at Faro may be a fundamentally different sort of alteration related to the metamorphism of the deposit.

In general there is little signature in a drillhole that is a "near miss" to a deposit. Alteration is restricted to the vicinity of the ore bearing structure and is typically most pronounced below the sulphides, thus it provides little help. There has been little study of more cryptic chemical, mineralogical or isotopic signatures around the deposits or, for example, in the basal carbonaceous member of the Vangorda formation, thus no such guide to ore is currently available.

Relation of Stratigraphy to Ore Deposits

The ore deposits of Anvil District are stratiform and confined to an approximately 150m to 200m thick stratigraphic interval which includes the contact of the Mt. Mye and Vangorda formations. It is presumed that the deposits are syn-sedimentary, however, direct evidence of this is no longer preserved. This stratigraphic position and the nature of the deposits suggests the host rocks for the mineralization and the age of mineralization are Cambrian. The deposits consist of one to five layers of sulphide mineralization interbanded with barren metasedimentary rocks. For those deposits with more than one sulphide horizon, the mineralized horizons are generally stacked one above the other or

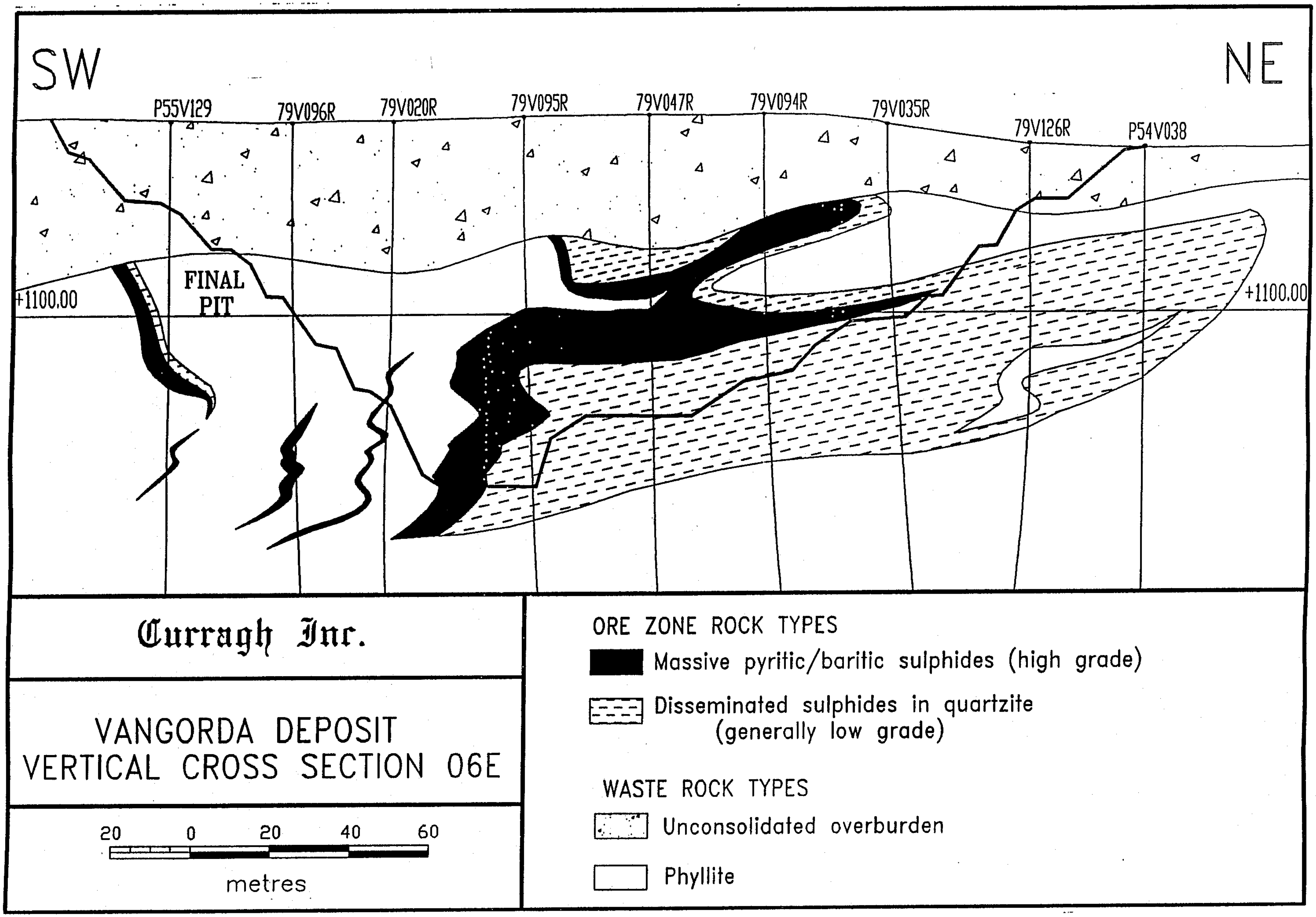


Figure 11. Cross Section through the Vangorda Deposit. The dashed unit lowest in holes 79V-095R-047R-094R and 035R is massive barren pyrite grinding down into pyrite + pyrrhotite + magnetite-bearing quartzites grinding further down into siliceous pyritic altered phyllite, and eventually altered phyllite. This unit has a higher Cu or/and Au ratio relative to Zn than the massive sulphides. It may represent a silicified feeder zone.

in en-echelon fashion. At least three of these mineralized horizons appear to be laterally equivalent to part of the basal carbonaceous member of the Vangorda formation.

The known deposits occur in a 25 km long curving trend following the prominent fold axial trends of the district. Southwest of this trend there is a tendency for the basal carbonaceous member of the Vangorda formation to thicken. The ore horizons tend to occur at the base of thick carbonaceous units suggesting the exhalative ore forming event was an initial stage in the formation of a anoxic sub-basin.

Unlike other sedimentary exhalative deposits of Selwyn Basin, the Anvil deposits are not characterized by a host stratigraphic section dominated by black carbonaceous rocks. Instead the carbonaceous rocks in the district are thin and subordinate or locally not even present near the sulphide deposits.

Mapping and drill results suggest the linearly distributed deposits lie close to a northeasterly "pinch out" of the basal carbonaceous member of Vangorda formation. To date, no sulphide deposit lithofacies have been encountered in a small number of drill holes through the ore-bearing horizon southwest or northeast of the deposit line. These observations and the relationships to carbonaceous rocks noted previously, suggest some genetic link between sulphide deposits and facies changes at a anoxic sub-basin margin. The linear trend suggests the possibility of fault controlled hinge lines of sub-basins. The faults may have channelled ore fluids leading to sea floor exhalation followed by sulphide deposition in the sub-basin where reduced sulphur was available.

EXPLORATION HISTORY AND EXPLORATION TECHNIQUES

The thirty-five year exploration history of the Anvil District has seen techniques evolve gradually through the following stages:

- conventional prospecting, resulting in the discovery of Vangorda in 1953,
- saturation geophysical and geochemical prospecting, resulting in the discovery of Swim in 1964 and Faro in 1965,
- geological extrapolation aided by detailed geophysics, resulting in the discovery of Grum in 1973,
- deep drilling guided by geological projections, which resulted in the discovery of Dy in 1976.

Each stage of exploration has detected deposits at greater depths of burial.

Successful techniques have included airborne magnetics, electromagnetic and gravity surveys, lead and zinc soil geochemistry, geology and prospecting. These techniques have been applied throughout the district from the late 1960's to early 1980's. Since the early 1980's there have been no substantive geophysical or geochemical surveys.

Conventional prospecting and local but highly effective use of geophysics and geochemistry (Chisholm, 1957) led to the discovery of the Vangorda as well as the shallow or outcropping Firth and Champ occurrences at either end of the Grum structure.

In the mid-sixties these methods were replaced with more widespread saturation airborne geophysical (magnetic and later EM) and less widely used geochemical methods (Aho, 1966; Brock, 1973). These rapid regional surveys were followed up by ground geophysics and rotary or diamond drilling. The second phase located two covered but near surface deposits with relatively strong but not always unambiguous geophysical and geochemical signatures: Swim in 1963 and Faro in 1964.

Followup of the second phase continued until the early seventies producing a patchwork of disconnected surveys, many of them conducted in haste and with poor control during the hectic years following the Anvil staking rush.

In the early seventies, a third phase started when a commitment was made by Cyprus Anvil (then Anvil Mining Corporation) and its parent corporations (Cyprus Mines Corporation and Dynasty Explorations) to initiate district-wide geologic mapping and more systematic ground surveys. A major rotary drilling program, designed to sample overburden, was also carried out in 1971.

Over the years a district wide Turam EM survey coverage was built up. By the mid to late 1970's EM took on a passive role than in the past, being intended not just to search directly for ore but more so to help trace units indicative of ore potential and to aid geologic mapping in areas of poor exposure. Many conductors were screened by gravity surveys and anomalous situations drill tested with generally unencouraging results.

By the mid-seventies geological, electromagnetic and drilling information were combined to produce a geological map of the district with common scaled compilations of other exploration information.This compilation and ongoing regional geologic mapping allowed the establishment of a tentative district stratigraphy which, in turn, led to a structural model for the main part of the belt. Figure 12 shows the outline of three of the most important sheets of this compilation, E6 (with Faro), F6 (the Vangorda Plateau) and G6 (the western Swim Basin).

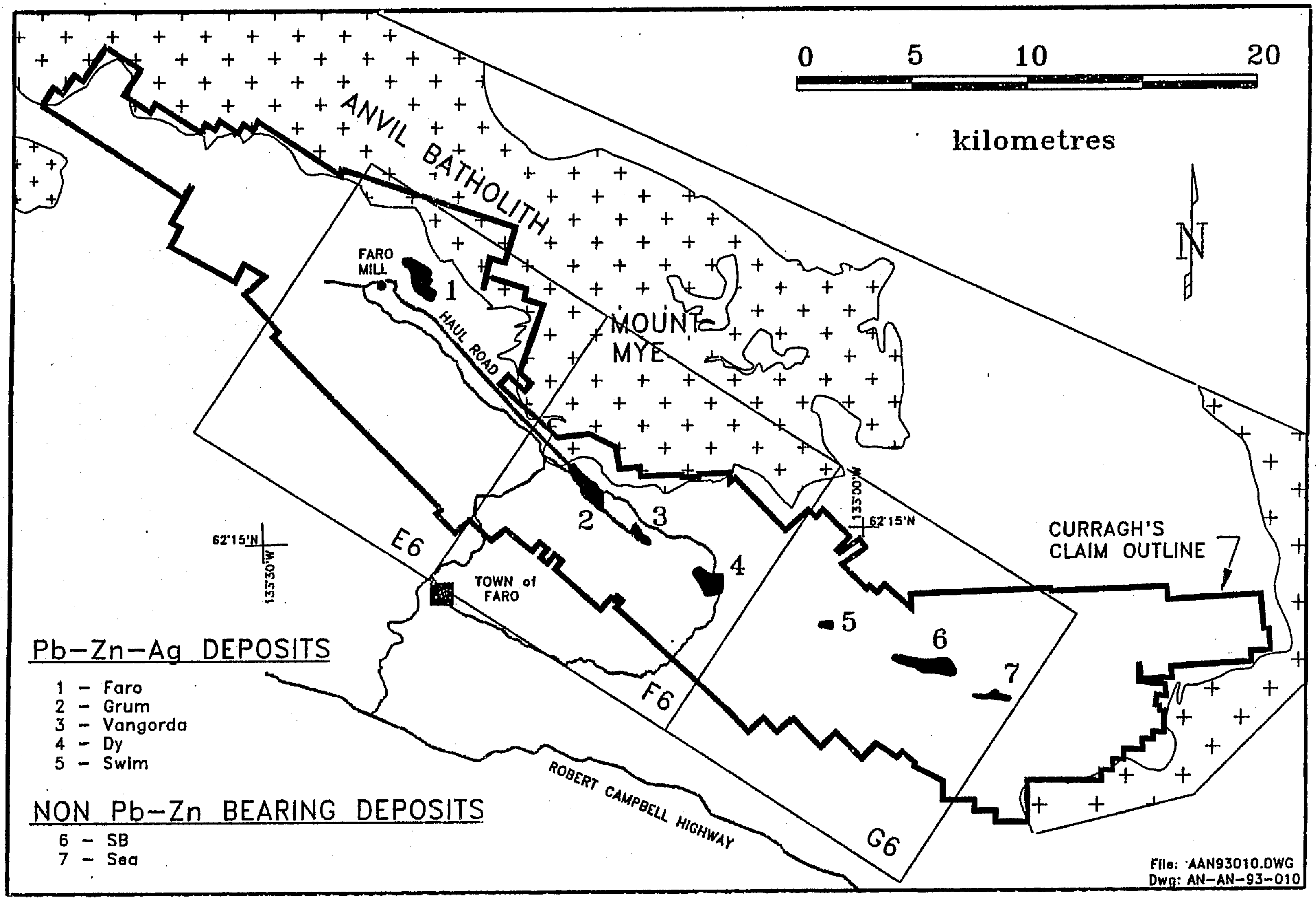


Figure 12. Location of three of the main sheets of the 1976-1981 Exploration Data compilation. Additional sheets in Row 6 are H6 to the southeast and C6 and D6 to the northwest.

The continuing negative results of anomaly testing and dwindling inventory of attractive targets began to indicate that the near surface, open pittable, potential of the district was becoming limited on C.A.M.C.'s ground. As target depths increased, gravity rapidly became in ineffective screen due to interference by bedrock relief, poor terrain corrections and instruments subject to high drift. Attention turned toward subtle anomalies supported by geology and to blind drilling beyond the limits of geophysical penetration following predictions based on geology.

A re-examination of Kerr Addison's land by A.E.X. Minerals led to the re-evaluation of gravity work following geological projections of favourable structure. The Grum discovery resulted in 1973.

The Dy discovery in 1976 was the result of drill testing the structural and stratigraphic model noted above. The Dy drilling was the beginning of a program of deep drilling laid out with spacing sufficient to make detection of various size targets and various depths likely (basically a Swim sized target to 330m (1,000 ft) and a Faro sized target at greater depths). The discovery hole was specifically drilled to test the favourable trend where the geologic model predicted favourable stratigraphy at 600m depth. This can be regarded as the fourth phase of exploration.

The deep drilling program was essentially suspended by the massive drilling requirements at Dy but resumed between 1979 and 1981 when several deep holes were put down particularly to test the down dip extensions of the Faro, Vangorda and Swim deposits with scattered holes elsewhere. Unfortunately since the stratigraphic model was incompletely developed at that time, some of these holes were not deep enough. The stratigraphic sequence, as now understood, provides a good "shut down unit", the marble and calc-silicate assemblage about 500 to 700m below the top of the Mt. Mye formation. Future drill holes should attempt to reach this unit if at all possible.

EXPLORATION MODEL

The geologic model of the deposits used to guide exploration is a variation of the sedex model. It is assumed that brine exhalation was fault controlled and that the favoured depositional setting was the edge of anoxic sub-basins (Shanks et. al., 1987) within which black shales were preserved. Empirical evidence shows that a broad stratigraphic interval, the 150 m. to 200 m. thick transition from Mt. Mye formation to Vangorda formation, is conducive to ore deposition. This interval will be termed the "Favourable Sequence". Secondly we observe that there is a "Favourable Trend", an alignment of the identified deposits. To date there are five known Pb-Zn bearing deposits along this 35 km. long trend and no important deposits off of it.

Exploration in the district is thus conceptually simple in that the Favourable Sequence must be traced through the area and extrapolated into the sub-surface where it can be tested by drilling with a priority on areas on the Favourable Trend. In practice exploration is more complex to do efficiently because of the structural complexity and the relatively poorly known detailed stratigraphy. The holes must be deep enough and avoid places where the Favourable Sequence is faulted out.

As noted previously at length, the structure of the district is highly complex involving isoclinal recumbent folds. In many areas the metamorphic foliation dips at a low angle and the stratigraphy is so strongly transposed that one can generalize that bedding has the same orientation on a large scale. On most of the claim block foliation dips to the southwest thus the Favourable Sequence southwest of the Favourable Trend is mostly buried but to the northeast it is mostly eroded and mineral potential is limited. Because of the low dips the target sequence is readily tested by vertical drill holes provided that they are deep enough. As the stratigraphy has been pieced together over the years it has become apparent that many holes were not.

Exploration has tended to focus on the Favourable Trend to date however there is no known reason that ore could not occur off this trend. The Favourable Trend heavily biases exploration thinking because of the simple empirical observation that success has occurred "on trend" but not yet "off trend" despite considerable effort. Areas on trend are thus considered to have significantly higher potential for ore discovery. It must be constantly realized that the off trend favourable stratigraphic horizon is only exposed at two places in the district and is fully tested by only a few drill holes thus there is considerable scope for new discoveries which could lead to revision of this restrictive exploration model.

There are currently no chemical mineralogic or isotopic zoning patterns recognised to provide a guide to ore. One feature that may be useful is a weak chloritic alteration associated with chloritic metamorphosed mafic igneous rocks. This has been recognized along the northeast edge of the Dy deposit and seems to also be present at Grum. The distribution of alteration suggests it may be a "hood" developed over the centre of exhalation and could be an important guide to deep ore. It is also important to begin carrying out analytical work on samples of the basal member of the Vangorda as they are collected. Unfortunately the existing sample set is very limited.

Exploration should focus on the edges of regional airborne EM conductors since these may indicate the margins of reduced sub-basins. One such area is thought to occur along the south edge of Swim Lake.

GEOPHYSICAL METHODS

The Faro deposit has generally been considered a geophysical discovery and geophysics played a part in the discovery of Swim and Grum, consequently geophysics has been widely used in the district. Vangorda, while not a geophysical discovery gives strong responses. Table 4 ranks anomalies associated with the various deposits and two minor

Table 4
Geophysical and Geochemical Response of Anvil District Deposits and Showings
(after Roth, 1990)

Year Discovered	Deposit Name	Zn-Pb-Ag Mineral Inventory (tonnes)	Tonnage of sulphide rock (tonnes)	Airborne EM	Ground EM	Airborne Magnetics	Ground Magnetics	Gravity	Induced Polarization/ Resistivity	Geochemistry
1953	Vangorda	7,000,000	22,000,000	3 Graphitic	3? Graphitic	3	(2)	3	3	2
1964	Swim	5,000,000	8,000,000	2? Graphitic	2? Graphitic	2	2	2	(2)	2
1965	Faro #2	4,500,000	Total 70,000,000	2	3	N	N	2	1	2
	Faro #1	29,000,000		?	1?	2 Mt & Po on Dyke contact	1 Mt & Po on Dyke contact	3	(2)	1
	Faro #3	24,000,000		N	1? Shallower Graphitic?	?	N	1+		N
1973	Grum	45,000,000	60,000,000 to 80,000,000	1? Graphitic	1? Graphitic	2	(1)	2	?	?
1976	Dy	20,000,000	80,000,000 to 100,000,000	N	??	?	(N?)	N	?? Shallower Graphitic ?	N
1964	S.B.	NA	NA	?	1?	1	1	?	(1)	N
1964	Sea	NA	NA	1?	2	1	1?	1?	(1)	1

3	Strong response
2	Moderate response
1	Weak but distinguishable
?	Weak uncertain
??	Very questionable
N	No response
( )	No survey, estimated response

Sulphide tonnages are from Cyprus Anvil "In-house" Tonnage and Grade Compilation by D.S. Jennings, Nov. 20, 1981 showings. This table shows that the deposits when close to surface, respond to a variety of methods. What is not so clear from the table is that there are significant sources of geologic noise that interfere with interpretation of survey results and that as the targets become deeper, and responses more subtle, the surveys are subject to severe limitations as signal is overcome by noise. This limitation of the existing data set will be dealt with further below.

A prerequisite of sound geophysical practice is a good understanding of the physical characteristics of the target and its host environment. Some quantitative work has been done over the years but it was not well documented and is now largely lost. The following discussion is thus in general and in qualitative terms.

Geophysical Characteristics of the Formations

The phyllites and schists of the Anvil District have a density of approximately 2.6 to 2.7 gm/cc. They are soft and easily eroded but have few other distinctive characteristics other than the schists have a high background chargeability due to their micaceous nature. The carbonaceous layers are black and highly conductive along carbon smeared S_2 folia. These rocks are the most conductive lithologies of the district and do cause strong electromagnetic anomalies. They form distinct layers which can be traced to help map the distribution of units in areas of poor exposure since the enclosing phyllites are non-conductive.

Greenstones and amphibolites are relatively dense (3.0 gm/cc), resistant, and commonly magnetic thus tend to form bedrock knobs which when buried by till create a positive residual gravity anomaly. Greenstones in the upper Vangorda formation are locally bounded by carbonaceous lithologies. This lithologic association can create coincident gravity, magnetic and electromagnetic anomalies which are of no economic interest. The combination of gravity and magnetic high is even more common and is often viewed with some scepticism in this district. In general, greenstones create difficult interference for gravity surveys.

Calc-silicates are also a dense rock type, approximating the density of greenstones, however the calc-silicates are more widespread and flaggy, thus less likely to form bedrock ridges. Where calc-silicates do form such ridges the high contrast between the rock and till densities can create a misleading positive residual gravity anomaly.

The Menzie Creek formation on the southwest side of Anvil Batholith is interlayered with carbonaceous phyllites. The Menzie Creek formation and overlying units thus have a very "active" EM signature on airborne and Turam EM surveys and make it easy to delineate the top of the Vangorda formation.

Granitic rocks are homogenous and restive, they create very flat EM response and have very low magnetic relief.

Geophysical Characteristics of the Ores

The sulphides have a number of physical characteristics which are important for geophysical exploration. The massive sulphides have densities in the range of 4.0 to 4.5 grams/cc thus form excellent density contrasts and against all rock types and strong positive gravity anomalies. Because of this density contrast gravity surveys have been an important and definitive exploration tool in the district. As the search depth increases however gravity surveys rapidly become ineffective because of the numerous corrections and spurious influences they are prone to. Disseminated sulphide bearing quartzites can be high grade but have densities only slightly greater than greenstones or calc-silicates (3.0 gm/cc), a further complicating factor; this surely limited the effectiveness of gravity over the F_1 fold closure at Grum.

The massive sulphides are conductive but are actually less conductive than associated carbonaceous phyllites or graphitic quartzites and will not necessarily stand out compared to the carbonaceous rocks.

Several sulphide lithologies are pyrrhotitic and/or magnetite bearing and are strongly to weakly magnetic. Of particular interest is the low grade copper-gold footwall sequence at Vangorda which is rich in pyrrhotite with lesser magnetite. There is a less well developed footwall sequence at Swim. Similar magnetite ± pyrrhotite lithologies occur throughout the upper (Champ) horizon at Grum as well as in the footwall of one of the lower structural panels. At Faro, barren massive pyrite is commonly magnetite bearing and slightly magnetic. In all the greenschist facies deposits the baritic ores are slightly magnetic due to fine, disseminated magnetite. Adjacent to dykes the pyritic sulphides may be altered to pyrrhotitic assemblages or, in extreme cases, to massive pyrrhotite or even massive magnetite. This alteration is particularly pronounced adjacent to the large dyke at the northwest end of the Faro Pit and to a lesser extent along the dyke separating Zone I and Zone II (near section 118). Clearly while coincident positive magnetic and gravity anomalies may be the hallmark of greenstone they can be, and are, also caused by sulphides.

Airborne and Ground Electromagnetic Surveys

The entire area has been covered by reconnaissance airborne electromagnetic surveys which have proved to be very useful. The survey was flown in June 1965 (Brock, 1973) using a helicopter transported Lockwood AEM system operating at 4000 hz. Line spacing was 330 m. and mean terrain clearance was 50 m. (15 m. to 60 m.). The survey was not flown at the same time as the magnetic survey. The Lockwood survey has been of immense help in sorting out the geology of the district however the line spacing and uncertainty of location creates complications in some areas. There have been only a handful of additional conductors discovered that the Lockwood system failed to detect. There was a strong response over Vangorda and Swim although in both cases this may be mainly due to graphitic mete-sediments rather than the orebody. A weak and not very distinctive response was obtained over the Faro #2 but the response over the other zones was poor.

Much of the district has been covered by detailed Turam electromagnetic surveys and more limited Crone JEM and CEM surveys. These later surveys are useful but they are difficult to interpret in areas of overburden and only a small part of the area has been done in a systematic fashion. The JEM survey at Faro assisted in discovery as the airborne responses did not stand out well. The Turam survey was selected in the early 1970's as the method to be used for systematic district wide coverage as it was felt to give good depth penetration; anomaly resolution and interpretation was difficult with both systems except on areas of a few discrete conductors. The Turam surveys did not help discover any mineralization not already detected by the airborne system but locations were considerably refined and conductor correlation was improved.

Considering the value the old survey has been in unravelling the geology in the past even with its limitations, it is felt that a new district wide airborne EM survey would be beneficial to assist with geological interpretation prior to undertaking further deep drilling.

Further surveying is recommended (Roth, 1990) however state of the art time domain systems are preferred over further Turam work.

Airborne and Ground Magnetic Surveys

Airborne magnetic surveying was one of the first reconnaissance exploration tools used in the district and it was applied with some success. The Swim discovery is attributed to magnetics, Vangorda also has a strong response but the response over Faro is limited and local. Despite the local variations, every deposit but Dy has some known magnetic response. Even on the high level Geological Survey of Canada survey (1968) this is apparent. The older systems in use in the district may not have been sensitive enough to detect broad subtle anomalies from deep sources. In all cases it is not clear what causes the anomaly over known deposits. In retrospect it is clear that the strongest anomaly at Faro is due to magnetite and pyrrhotite derived from the massive sulphides where contact metamorphosed by a dyke at the northwest end of the Pit. Anomalies at Vangorda and Swim are likely due to low grade footwall lithologies rather than the ore. The anomaly at Grum may be due to the slight magnetite content of the Champ Horizon above the actual ore. These relationships show the importance of testing the anomaly setting thoroughly.

Ground magnetic surveys have been of more limited extent and were neither carried out with sensitive instruments nor were they well controlled. Some useful detail has been gleaned from these surveys (such as clarification of the Faro anomaly) but they are not helpful in attempting to outline subtle low magnitude features.

Gravity Surveys

Early in the history of the district both the utility and the limitations of the gravity method were realized. The method was attempted as a primary exploration tool but proved too slow and subject to drift and various interferences. Gravity surveys were then used as a follow-up tool to screen anomalies detected by other methods, particularly EM.

Many electromagnetic conductors near the Favourable Trend have been tested with gravity surveys but the coverage by detailed, well controlled surveys is far from complete. More work is needed in unsurveyed areas and while this is done the existing high quality and well documented surveys should be tied together better.

Other Methods

Induced polarization and resistivity surveys have been used locally in the district but these methods saw limited acceptance partly due to expense. Dynasty experimented with resistivity profiling to check bedrock relief at the site of gravity anomalies and found it accurate but not practical. Self potential surveys were tested by Kerr Addison (Chisholm, 1957) however this method also did not see widespread use. Hammer seismic was also tested to determine its use in conjunction with gravity to detect bedrock highs but concern over false bedrock indications from permafrost limited its use. Some engineering seismic investigation was also completed with without encouraging results. Borehole geophysics was tested in 1989 but this method proved to be difficult in Anvil District ground conditions; furthermore the abundance of carbonaceous phyllite suggests this technique will likely give misleading indications.

GEOCHEMICAL METHODS

Geochemistry has also been used widely in the district. Had greater emphasis been placed on geochemistry then some discoveries might be considered geochemical discoveries aided by geophysics rather than the reverse. The major media are soil and glacial sediments and these are discussed below in more detail.

Silt sampling has also been used as a reconnaissance tool with some success. Experimental heavy mineral sampling in Vangorda creek detected anomalous gold and barium in -60 mesh heavy, non-magnetic fraction of stream sediments 12 km. downstream from the deposit. Lead and zinc were not detected at that distance.

Bedrock geochemistry studies in the district are limited. Morton has shown that Pb and Zn are anomalous up to a hundred metres above and below the Faro deposit and further that these anomalous values have contributed to the metal content of surficial material where the orebody is buried. Barium is anomalous for a shorter distance into the hanging wall but not the footwall. Mo is anomalous in the white mica alteration that surrounds Faro.

Soil Geochemistry

Soil Geochemistry is an effective exploration tool for sub-cropping ore in areas of discontinuous till cover but is hampered by till in excess of 10 m. thick. This method is of limited value in areas with a thick relatively continuous till blanket such as the Swim Basin (Figure 3). Soil surveys have tended to exploit the poorly developed "B" horizon as sample media and analyses have generally been limited to Cu,Pb and Zn. Both total and cold extractable analyses have been used and there is no agreement on the optimum method. Much of the district was sampled in the early to mid 1970's on lines spaced 400 m. apart. The compilation of results shows clear dispersion trains extending down ice from Vangorda, Grum (Champ and Firth) and particularly Faro. In addition there are more local downhill anomalies, at several sites. Much of this more local dispersion was hydromorphic in the case of zinc.

Consideration was given to the use of mercury in soil for exploration as the ores are mercury bearing and the possibility of upward gaseous dispersion was appealing. In practice it was found that the strong coupling of organic carbon and mercury made the results too difficult to interpret thus be of little use. In 1988 test was done over Grum and Dy of Boliden's GEOGAS method; the results were uninterpretable apparently due to contamination of the proprietary collectors. This indicated the method to be impractical as the collectors were carefully placed.

Overburden Geochemistry

To help explore areas covered by thick glacial till a portion of the district was covered in 1971 by wide spaced overburden drilling with geochemical analysis of the recovered till. This work was coupled with coring of the first few feet of bedrock where it was reached. The bedrock data from this drilling has been of considerable use and there are very interesting anomalies that still have to be explained in Swim Basin but in general the overburden component was of limited effectiveness. Part of the reason for this may be that there was inadequate geologic support for the programme thus there was no surficial geological background within which to interpret the results. Sample descriptions were rudimentary and this may have added to the difficulties of interpretation as Morton (1973) showed that the background and anomaly threshold for the two types of overburden encountered at Faro, till and outwash, were very different. This method warrants further use as there is still no coverage for the bulk of the Swim Basin and methods have advanced considerably in the last 20 years.

DRILLING

The only diagnostic and reliable exploration tool in the district is the diamond drill guided by a reliable geologic model. Unfortunately there is limited drilling in the district away from the deposits and the narrow corridor they fall in. Much of the drilling that exists was done without a reliable stratigraphic model and it appears now that a considerable number of the deeper holes are too short; many are not diagnostic as they stopped short of the Favourable Sequence or did not completely test it. This is true of the holes down dip from Vangorda and particularly of the several holes down dip from Faro. It is an unfortunate fact of life that the poor exposure of the district means that little detail comes from surface mapping and only the drillholes allow significant advance in understanding of the district. In all there are approximately 40 deep holes in the district both on and off the favourable trend (but not near a known deposit) of which less than half are deep enough to be considered diagnostic tests.

Much more drilling needs to be done but the next phase of deep drilling should be preceded by a complete structural re-evaluation of the district using new information sources such as the proposed airborne MAG/EM survey. As noted above further overburden drilling in the Swim Basin is warranted.

EXPLORATION POTENTIAL

As indicated above the Anvil District has a long exploration history with many techniques having been applied. Most of the exploration data for the district up to 1980 has been compiled on a series of common scaled maps at 1:12,000. The three major sheets are indicated on figure 12. In the core areas of the district, such as the Vangorda Plateau, geology and drilling compilations have been updated on a new series of 1:5,000 scale maps although coverage is not yet complete. Geological mapping of most of the district still dates back to the early 1970's. About half the Vangorda Plateau and most of the claim block northwest of the Faro Mine have been re-mapped at 1:5,000 using the new stratigraphic concepts worked out in 1983.

Figure 13 shows the exploration potential of the district inferred from application of the above model in light of the known stratigraphy and structure. There has been very little diagnostic sampling of the area southwest of the "favourable trend" thus exploration off that trend should not be ignored.

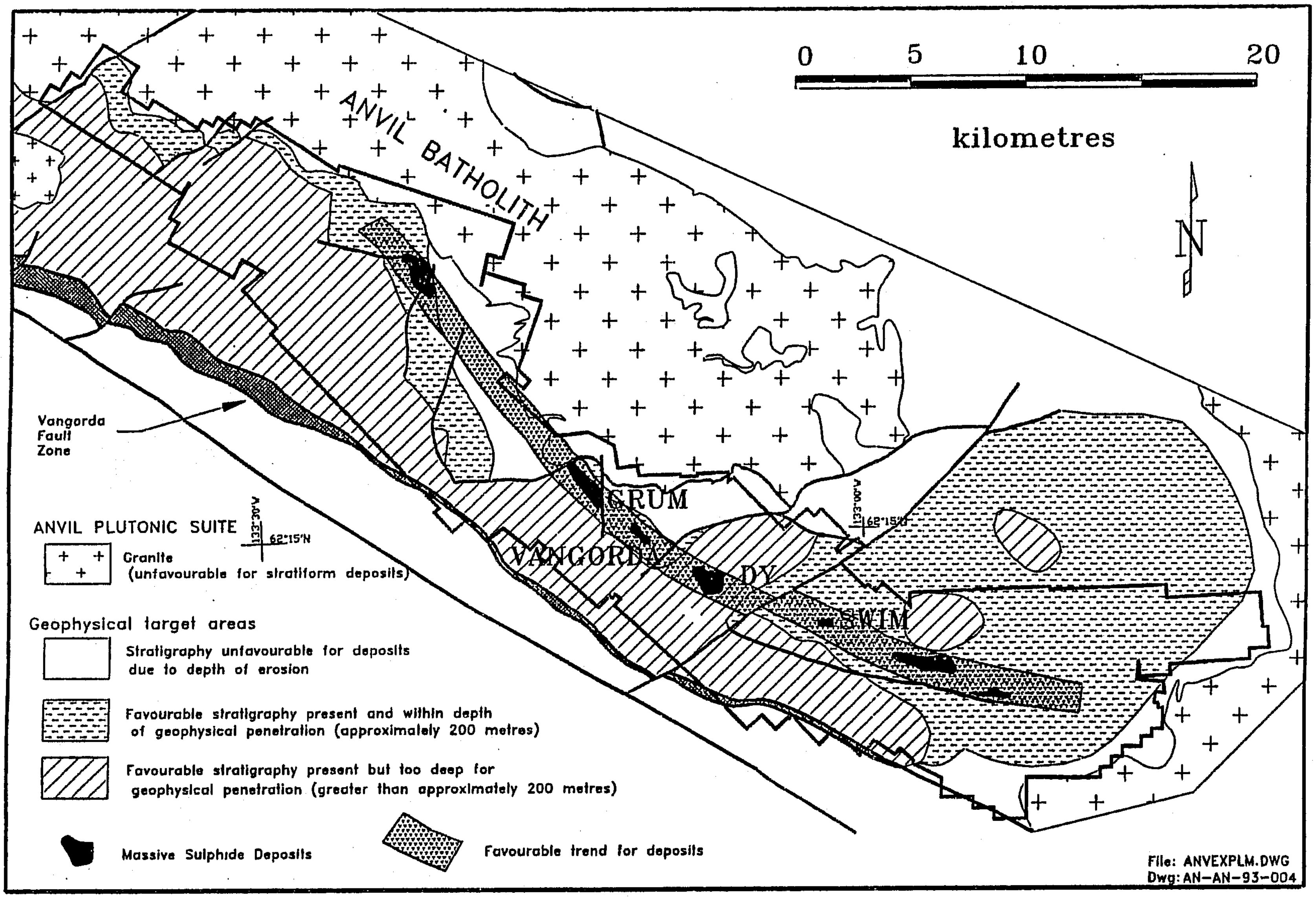


Figure 13. Exploration potential of the south flank of the Anvil Batholith, Anvil District, Yukon. Potential is highest along the favourable trend. It can be seen that there is little exposive in the favourable stratigraphy southeast of the trend; it is eroded to northeast.

Table 5

Geophysical Survey Coverage over the Favorable Phyllite Belt

Map Sheet	Airborne EM Coverage	Airborne Mag Coverage	Ground Mag Coverage	Turam Coverage	Gravity Coverage
D6	100 %	100 %	25 %	70 %	30 %
E6	100 %	100 %	70 %	70 %	50 %
F6	100 %	100 %	20 %	70 %	60%
G6	100 %	100 %	40 %	75 %	50 %

In essence there is little known of the district in the subsurface and practically nothing of the areas off the favourable trend. It is important that the district see a resurgence of exploration activity particularly deep drilling both on and off trend coupled with renewed surface geological mapping. Detailed, modern, airborne electromagnetic and magnetic surveys should also be completed to assist with the geology and to cover unsurveyed or inadequately surveyed areas as outlined in Table 5.

It is believed that this exploration effort will be rewarded with additional ore discoveries. Particular targets are the area northwest of Faro where there has been little deep drilling and possible left lateral, northeast trending faults may have offset the Favourable Trend away from the batholith. There are several areas requiring testing on Vangorda Plateau particularly in the vicinity of the Dy deposit. In general there is virtually nothing known of the deep potential of the area southwest of the Favourable Trend on Vangorda Plateau. The Swim Basin offers a number of targets, of special interest is the south shore of Swim Lake and an area at the east end of the Favourable Trend where overburden geochemistry suggests that there may be additional subcropping sulphides up ice flow direction from the last occurrence.

Closer to the deposits there is excellent potential to extend the Grum deposit down plunge to the northwest into an area where scattered high grade intersections have been encountered but most holes are too short to fully test the potential. Similarly the Dy deposit has not been closed off by drilling to the south, southwest or southeast and further deep drilling is likely to encounter additional mineralization. Both the Grum and Dy areas have been estimated to have the potential to host an additional 5 million tonnes each of mineralization with similar tenor to that already known.

In summary although the Anvil District has had a relatively long history of exploration, only the near surface has been well explored and the potential for additional discoveries at depth and in overburden covered areas remains good.

MINERAL RESOURCES AND MINERAL INVENTORY

There are five deposits with delineated zinc-lead-silver mineralization in the district, one of these, Faro, is now essentially depleted and, Vangorda is close to depletion as well. Despite this there is still a substantial inventory of mineralization in the district contained within the Grum, Dy and Swim deposits. As noted above there is excellent potential to increase this inventory further at the currently interpreted deposit periphery. Exploration is also likely to add to the inventory by the discovery of new deposits.

Table 6 presents the mineral inventory for the district at the end of May 1993 in terms of the tonnage and grade of defined in-situ mineralization at a variety of cutoff grades. Also presented in Table 6 is the potential mineralization speculated to be present near Grum And Dy. The tonnage at Dy is quite sensitive to cutoff grade emphasizing the

Table 6

Anvil District Geological Inventory as of start of Second Quarter 1993

Total in-situ mineralization, no adjustments for mining recovery or dilution

Deposit Name	Cutoff % Pb+Zn	Classification	Tonnage (tonnes)	Zn (%)	Pb (%)	Ag (g/t)	Au (g/t)
Faro	3	Prov.+ Prob.	30,000	5.60	3.30	35.0	0.10
Vangorda	3&4	Prov.+ Prob.	2,017,143	4.26	3.34	42.8	0.81
Grum 61W to 87W	3	Prov.+ Prob.	43,240,250	4.66	2.94	49.1	0.08
Grum 61W to 87W	8	Prov.+ Prob.	15,882,260	7.05	4.39	72.6	1.05
Grum northwest of 87W	NA	Potential	5,000,000	6.00	4.00	65.0	0.75
Dy AB and Extn., Above and Below AB	6	Prob. + Poss.	41,555,000	5.72	4.12	61.9	0.65
Dy AB and Extn., Above and Below AB	8	Prob. + Poss.	24,947,000	7.01	5.21	77.4	0.85
Dy AB and Extn., Above and Below AB	9	Prob. + Poss.	21,356,000	7.33	5.54	81.1	0.87
South, SW and SE of AB Zone	NA	Potential	5,000,000	7.33	5.54	81.1	0.87
Swim	4	Possible	5,130,000	4.40	3.50	47.0	0.65
Stockpiles	3	Proven	2,598,456	2.89	1.96	17.9	0.19
Total (Grum @3%, Dy at 9%)			**84,371,849**	**5.49**	**3.83**	**58.8**	**0.42**

importance of reducing miningand transportation cost to not only improve profits but also to add tonnage to the reserve base in areas already delineated.

MINABLE RESERVES

Table 7a presents the minable reserve for the Anvil District as of the end of March 1993. The reserve is based only on mineralization classified as proven or probable and for which the economics of extraction has been established. This material is fully diluted and expressed as recoverable ore. The details of the derivation of dilution are explained in separate reports on the specific deposits.

The inventory of mineralization that can reasonably be expected to be mined is greater than the reserve base however this material requires further drilling or improved economics to be added to the reserve base. This material is expressed as diluted recoverable material as with the ore reserve in Table 7b. Included in this category is incremental mineralization within a larger pit design for Grum, mineralization below the Grum Pit but not the northwest extension. Also included is possible mineralization in the AB Zone and AB extension Zone at Dy and the Swim Deposit.

The current status of high and low grade stockpiles at the Faro and Vangorda Plateau sites is indicated in Table 7c.

Table 7a

Anvil District

Start of 2nd Quarter Minable Reserves

Remaining at End of First Quarter 1993, cutoff (c/o) as shown below	Classification	(metric tonnes)	Zn (%)	Pb (%)	Ag (g/t)	Au (g/t)	total zinc (million lbs.)	total lead (million lbs.)	total silver (thousand oz.)	total gold (thousand oz.)
Faro Pit – Zone I Ramp area	Prov. + Prob.	30,000	5.60	3.30	35.0	0.10	3.704	2.183	33.756	0.096
Vangorda Pit (3% Pb + Zn c/o)	Proven	1,004,736	4.40	3.60	46.9	0.92	97.467	79.796	1,516.000	29.703
Grum Pit (IV Pit, 3% Pb+Zn c/o)	Prov. + Prob.	24,760,000	4.54	2.74	46.0	0.70	2,478.222	1,495.667	36,616.425	557.206
Dy (9% Pb+Zn C/o)	Prob.	9,390,095	6.62	5.50	80.3	0.82	1,370.445	1,138.588	24,241.138	247.543
Stockpile	Proven	2,598,456	2.89	1.96	17.9	0.19	165.366	112.061	1,492.916	15.543
Total Minable Reserve	**Prov. + Prob.**	**37,783,287**	**4.94**	**3.40**	**52.6**	**0.70**	**4,115.204**	**2,828.294**	**63,900.235**	**850.092**

Table 7b

Anvil District

Start of 2nd Quarter Minable Inventory

Remaining at End of First Quarter 1993, cutoff (c/o) as shown below	Classification	(metric tonnes)	Zn (%)	Pb (%)	Ag (g/t)	Au (g/t)	total zinc (million lbs.)	total lead (million lbs.)	total silver (thousand oz.)	total gold (thousand oz.)
Increment from IV to AB Pit (incl.Champ, 3%c/o)	Possible	5,782,000	3.59	2.16	35.4	0.70	457.621	275.338	6,580.347	130.120
Grum below AB Pit (52W to 88W, 8% Pb+Zn c/o)	Possible	2,832,000	6.38	4.17	68.6	1.00	398.334	260.353	6,241.201	91.046
Dy (9% c/o)	Possible	5,879,683	7.70	5.10	75.6	0.83	998.110	661.086	14,297.921	156.892
Swim (4% c/o)	Possible	3,910,000	3.91	3.22	42.0	0.65	337.045	277.566	5,279.509	81.707
Total Minable Inventory	**Prov. + Prob.**	**18,403,683**	**5.40**	**3.63**	**54.8**	**0.78**	**2,191.110**	**1,474.343**	**32,398.979**	**459.764**
Total Defined Minable Mineralization	**Prov.,Prob.&Poss**	**56,186,970**	**5.09**	**3.47**	**53.3**	**0.73**	**6,306.314**	**4,302.637**	**96,299.214**	**1,309.856**

Table 7c

Anvil District

Status of Stockpiles at Start of 2nd Quarter

Stockpiles at end of First Quarter 1993	Classification	(metric tonnes)	Zn (%)	Pb (%)	Ag (g/t)	Au (g/t)	total zinc (million lbs.)	total lead (million lbs.)	total silver (thousand oz.)	total gold (thousand oz.)
Grum HG +5%, Plateau		7,414	4.28	2.02	20.0	0.70	0.700	0.330	4.767	0.167
Grum LG 3–5%, Plateau		23,821	3.11	1.51	15.0	0.30	1.633	0.793	11.487	0.230
Faro HG +5%, Faro		4,000	5.46	3.19	30.0	0.10	0.481	0.281	3.858	0.013
Faro LG 3–5%= A.C,LL S/Ps, Faro		1,758,159	2.76	1.68	15.0	0.10	106.980	65.118	847.846	5.652
Vangorda HG +6% G1 baritic, Plateau		13,954	4.82	3.76	45.0	0.70	1.483	1.157	20.187	0.314
Vangorda HG +6% Oxide Cap, Plateau		57,197	4.93	4.09	45.0	0.70	6.217	5.157	82.747	1.287
Vangorda HG +6% G1 baritic, Faro		32,321	4.56	3.55	45.0	0.70	3.249	2.530	46.759	0.727
Vangorda HG +6% G2 carbon, Faro		30,055	4.29	3.23	45.0	0.70	2.843	2.140	43.481	0.676
Vangorda LG 3–5%, Plateau		510,154	2.75	2.31	20.0	0.30	30.929	25.980	328.019	4.920
Vangorda LG 3–5%, Faro		161,381	3.05	2.41	20.0	0.30	10.851	8.574	103.765	1.556
Total Stockpiles		**2,598,456**	**2.89**	**1.96**	**17.9**	**0.19**	**165.366**	**112.061**	**1,492.916**	**15.543**
Total HG Stockpiles		144,941	4.69	3.63	43.3	0.68	14.972	11.595	201.799	3.185
Total LG Stockpiles		2,453,515	2.78	1.86	16.4	0.16	150.393	100.466	1,291.117	12.359
Total HG Stockpiles at Faro Site		66,376	4.49	3.38	44.1	0.66	6.573	4.951	94.098	1.417
Total LG Stockpiles at Faro site		1,919,540	2.78	1.74	15.4	0.12	117.831	73.692	951.611	7.209
Total Stockpiles from Faro Deposit		1,762,159	2.77	1.68	15.0	0.10	107.461	65.399	851.704	5.665
Total Stockpiles from Grum Deposit		31,235	3.39	1.63	16.2	0.39	2.333	1.123	16.254	0.397
Total Stockpiles from Vangorda Deposit		805,062	3.13	2.57	24.1	0.37	55.572	45.539	624.958	9.482

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2

SUMMARY of the

GEOLOGY, MINERAL INVENTORY

and RESERVES of the DY DEPOSIT

YUKON

Curragh Inc.

May 1993

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INTRODUCTION

One of five known lead-zinc deposits in the Anvil District, the Dy deposit is located 22 kilometres southeast of the Faro concentrator, 480 to 900 metres below the surface. The Dy deposit was discovered in 1976 by Cyprus Anvil Mining Corporation. The discovery hole (76X-21) was targeted to intersect favourable stratigraphy interpreted to exist at least 500 m below the surface. The hole was successful in intersecting several thick sulphide horizons over an interval from 513.6 m to 622.8 m. In the five years that followed, Cyprus Anvil drilled 52 holes in the vicinity of the deposit and produced two versions of a preliminary reserve calculation.

After acquiring the assets of Cyprus Anvil in 1985, Curragh Inc. (then Curragh Resources Inc.) completed an additional 21 drillholes at Dy. The holes were drilled between 1989 and 1991. The majority of the drillholes were drilled to test geotechnical conditions near proposed underground development. Nine drillholes were targeted to test and delineate parts of the Dy mineralized zone.

This document is an excerpt from Report WH9103 "Dy Deposit Mineral Inventory", dated December 1991. Some new information is included in this document as well. Report WH9103 presented a new structural interpretation for the deposit, based partly on the results of the above drilling, and presented a polygonal calculation of the mineral inventory based on all drilling completed to date. Detailed calculation sheets, maps, vertical sections, and drillhole assays were included in the Appendices at the end of that report. The assumptions used to derive the minable reserves have been added to this document and it has been updated to reflect the proposed central shaft access.

LOCATION AND ACCESS

The Dy property is located in the Anvil Range of central Yukon near the town of Faro, approximately 200 km northeast of Whitehorse. The Dy property is 6 km southeast of the Grum deposit on the southeast limit of the Vangorda Plateau (Figure 1). Ground elevations on the property range from 800 to 1175 m.

Access to the property can be gained by all weather roads from two directions. A secondary road from Faro southeast along Pelly River and northeast along Blind Creek can be used as can a road extending southeast from the Vangorda deposit. Access to Faro is via all weather highway or scheduled air service.

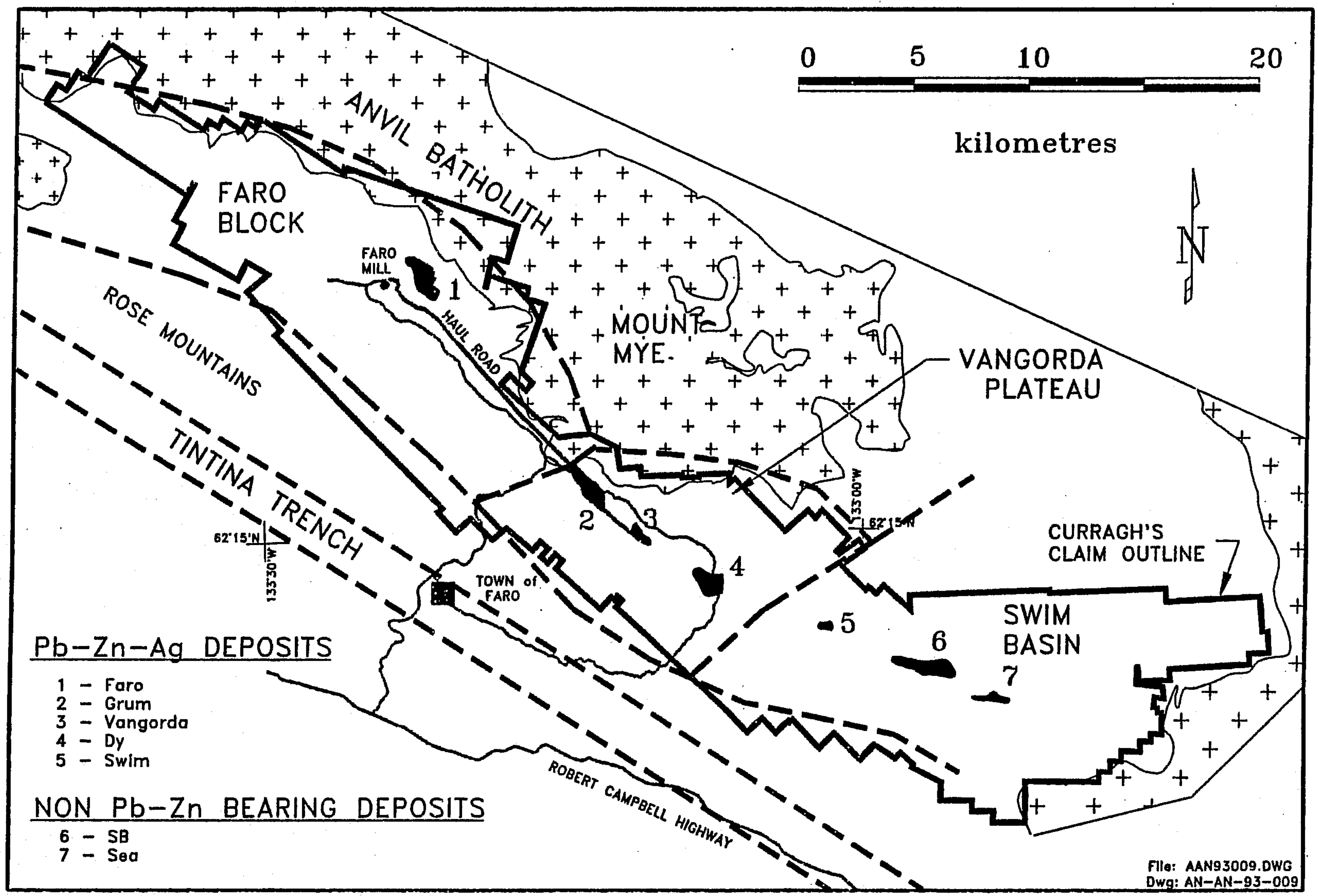


Figure 1. Location of the Dy Deposit (No. 4) Relative to the Other Zn-Pb-Ag Deposits of the Anvil District

GENERAL GEOLOGY

The Dy deposit is hosted by the same phyllites, and is at the same general stratigraphic position as all other Anvil District deposits. The Dy drillholes provide some of the best examples of **Vangorda formation**, and the Blind Creek road exposures above the Dy deposit are the type locality of the formation. The dominant lithology of the Vangorda formation at Dy is the usual medium to light greenish grey, strongly banded, calcareous metasedimentary phyllite which is so characteristic of that formation. At Dy there are also excellent examples of blocky, medium grained greenstones interlayered with the phyllites. Where the greenstones are thin they are strongly foliated medium olive green, chloritic phyllite, commonly calcareous, and contain quartz and calcite bands that appear to be foliation parallel veins. The greenstones are interpreted to be intrusive sills and show considerable alteration along both their margins. Adjacent to these greenstones, the grey Vangorda formation metasedimentary phyllite is altered to a banded, pale green, chlorite-muscovite-quartz-calcite phyllite, which, in advanced stages of alteration, can be very difficult to distinguish from the foliated sill margins, or, if the sills are thin, from the sill itself. Along the northeast margin of the deposit greenstone, chloritic phyllite, and the pale green chlorite-altered Vangorda formation metasedimentary phyllites are particularly abundant.

The **Mt. Mye formation** at Dy is also typical of its phyllitic variant elsewhere in the district. It is a medium grey, homogenous, strongly foliated, muscovite-chlorite phyllite which is typically non-calcarceous and less distinctly banded than the Vangorda formation phyllite. In deeper holes biotite-muscovite schists of the Mount Mye formation have been intersected. In the deepest holes typical Mount Mye formation coarser grained calcite marble and calc-silicate have been intersected; this is "shut down" rock for the Dy deposit.

Carbonaceous phyllites are present at Dy in both the Vangorda and Mount Mye formations. Carbonaceous phyllite is more abundantly represented in the stratigraphic section toward the southwest edge of the deposit. This is similar to the situation at Vangorda and Grum.

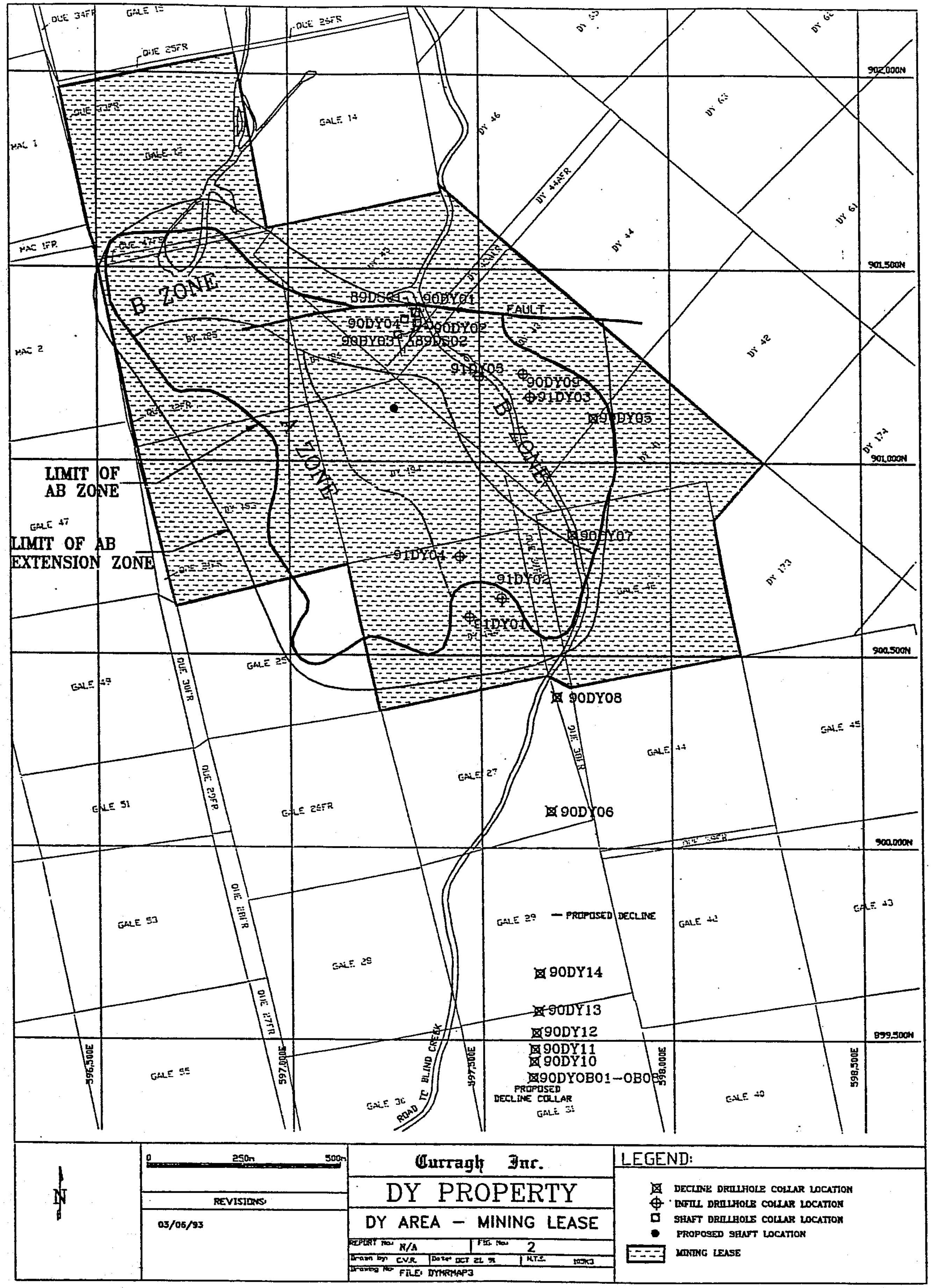
The internal structure of Dy is relatively poorly understood because of lack of data. It is reasonable, however, to expect that the structural complexity of the other more densely drilled Vangorda Plateau deposits (Vangorda, Grum) also exists at Dy. There is evidence of at least five phases of deformation in the district. On Vangorda Plateau the first two are generally most significant in that they are penetrative and affect the overall shape and geotechnical characteristics of the mineralized zone and its host rocks. A well developed, moderately southwest dipping metamorphic cleavage (S_2) is generally subparallel to the sulphide layering. In the phyllite host rocks, S_2 is a well developed micaceous cleavage axial planar to second phase folds in layering. S_2 is generally the most important parting or plane of fissility in the rocks. This cleavage is an important geotechnical consideration for underground development, particularly where S_2 is cross cut by faults and joints. Within massive and disseminated sulphide horizons, S_2 (or S_1) is present as thin compositional bands. The sulphide rock types are generally competent and the foliation does not represent a significant geotechnical concern. A possible exception to this generalization is due to local, carbonaceous partings along S_2 which impart a fissility to some lower grade disseminated sulphide bearing quartzites.

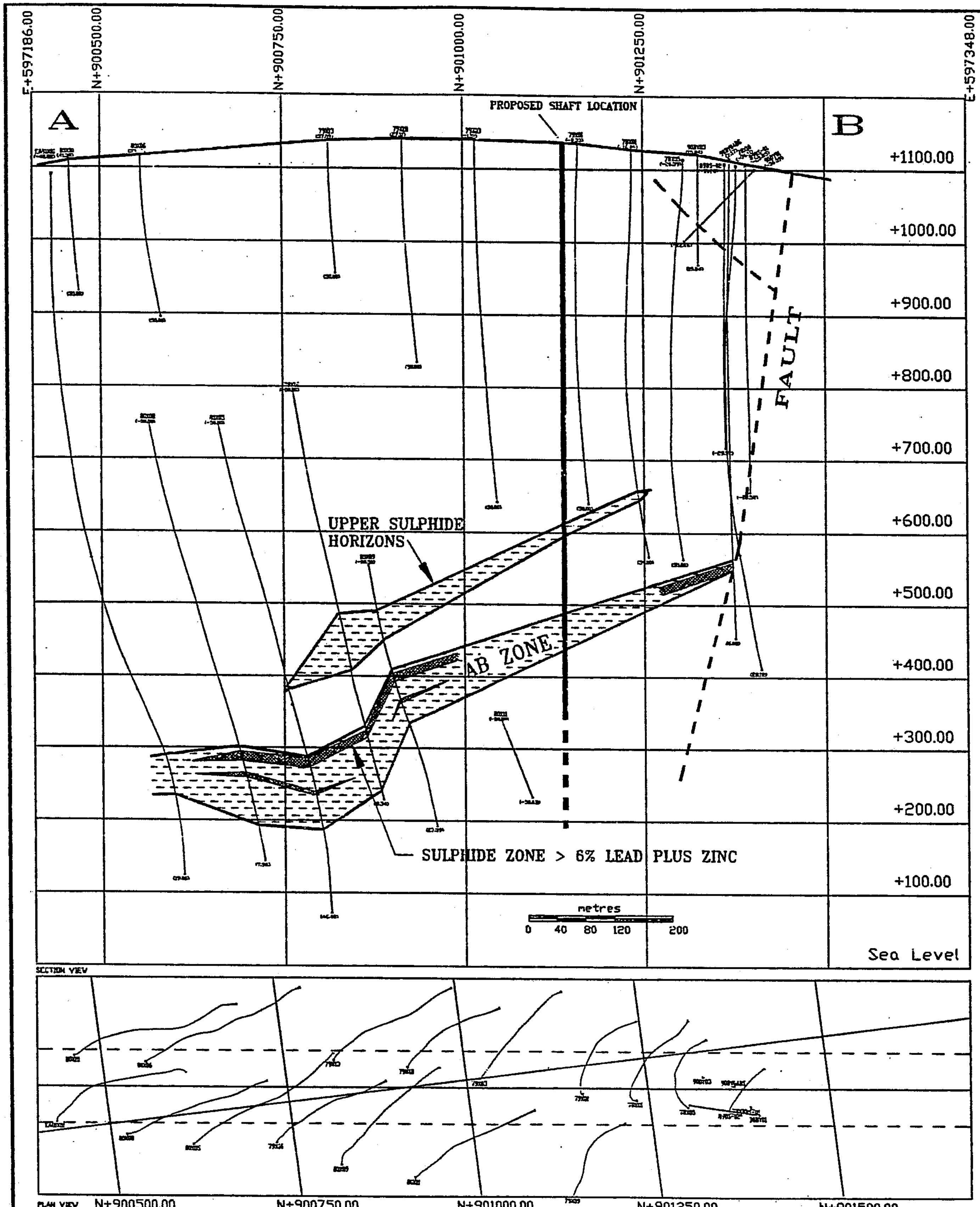
Metamorphic facies at Dy is mainly greenschist and all host rocks in the vicinity of the deposits are phyllites. The ores are the typical fine-grained sulphide assemblages of the other Vangorda Plateau deposits. As noted above, deeper holes have intersected amphilobite facies schists, but these are not generally associated with the sulphides.

Several important post-metamorphic dykes are present at Dy Most of the dikes are hornblende-bearing quartz diorite. Quartz-feldspar porphyry dykes are also present. The Dy dykes seem to combine features of both members of the Anvil Dyke Suite. These dykes are generally porphyritic and on the order of 10m to 30m thick. Most dykes with known orientations dip moderately toward the southeast. One set of hornblende diorite dykes, intersected by most holes above the sulphide deposit, has been interpreted to intersect the ore horizon at the east edge of the known Dy deposit. Because of this possible dike interference, drilling was not continued in that direction by Cyprus Anvil after one disappointing hole. This dyke rock is very competent and is a logical choice for a location of an upper shaft pump station.

There are numerous steep faults which cut the deposit and there are important, shallowly dipping faults present beneath the deposit. Many of these faults, especially the steeply dipping ones, contain significant clay/mud gouge, and the fractured zone adjacent to the fault is water bearing. The current drilling density precludes the possibility of resolving frequency of occurrence or orientation and displacement on most of these faults. Two important faults were detected in the 1990 hole drilled to test the shaft site proposed at that time. The upper one strikes northeast-southwest and dips moderately northwest. The lower fault is perhaps more significant in that it may truncate the ore zone along its northeast boundary (the straight portion of the outline northeast of DDH 90DY04 in Figure 2). It is suspected to strike east-northeast/west-southwest and dip steeply south (Figure 3).

Several shallowly dipping extensional faults are thought to exist beneath the deposit. These are represented by a strongly foliated carbonaceous phyllite with an augen texture and clasts of vein quartz and carbonated greenstone. Quartz porphyry dykes are associated with some of these low angle structures. The most important of these structures crops out in the headwaters of Dixon Creek, between Dy and Vangorda, and appears to have downdropped Dy from the up-plunge extension of Vangorda (now eroded), a separation of nearly 2,000m. The shallowly dipping faults are generally marked by intact fault rock and may not pose a significant geotechnical concern.


B ZONE
A ZONE
B ZONE
FAULT
LIMIT OF AB ZONE
LIMIT OF AB EXTENSION ZONE
89DS01
90DY01
90DY04
90DY02
90DY03
89DS02
91DY05
90DY09
91DY03
90DY05
90DY07
91DY04
91DY02
91DY01
90DY08
90DY06
PROPOSED DECLINE
90DY14
90DY13
90DY12
90DY11
90DY10
90DYOB01--OB08
PROPOSED DECLINE COLLAR
ROAD TO BLIND CREEK
902,000N
901,500N
901,000N
900,500N
900,000N
899,500N
596,500E
597,000E
597,500E
598,000E
598,500E
GALE 14
GALE 27
GALE 29
GALE 44
GALE 45
GALE 51
GALE 53
GALE 55
DY 42
DY 44
DY 46
N
0
250m
500m
REVISIONS
03/06/93
Curragh Inc.
DY PROPERTY
DY AREA — MINING LEASE
REPORT No: N/A
FIG. No: 2
Drawn by: C.V.R.
Date: OCT 21 91
N.T.S. 105K3
Drawing No: FILE: DYMRMAP3
LEGEND:
DECLINE DRILLHOLE COLLAR LOCATION
INFILL DRILLHOLE COLLAR LOCATION
SHAFT DRILLHOLE COLLAR LOCATION
PROPOSED SHAFT LOCATION
MINING LEASE



Curragh Inc. FIGURE 3	DY DEPOSIT VERTICAL SECTION SHAFT LOCATION STUDY

DEPOSIT GEOLOGY

The Dy deposit is similar to the other Anvil District deposits in that it is a multi-layered, polydeformed, sediment hosted sequence of exhalative, massive and disseminated pyritic sulphides. Sulphide layers are variably mineralized and commonly interbanded with metasedimentary and lesser metavolcanic or meta-intrusive phyllites. The enclosing rocks are muscovite-chlorite-quartz ± calcite phyllites which are locally altered near the deposit, to pale coloured bleached muscovite >> chlorite ± quartz ± pyrite phyllites.

The known mineralized zone ranges up to 200 m thick in aggregate, has a strike length of approximately 2200 m, and a width up to 1800 m. The horizons of the Dy deposit span a poorly defined transition zone from the Mt. Mye formation to the younger, calcareous Vangorda formation. The deposit is amoeboid shaped in plan view and is unusual for the Anvil District in that it has two distinct zones (Figure 2) of varying lead to zinc ratio; in the southwest, the A Zone which is relatively lead rich and, in the northeast, the B Zone which is relatively zinc rich. Between the A and B zones is a central area characterized by nearly barren massive pyrite, pyritic quartzite and semi-massive quartz bearing pyrite. This unit is significant as it is the target for shaft development into the centre of gravity of the ore deposit.

There are several sulphide lithofacies which comprise all of the Anvil District deposits. Two principal subdivisions exist; massive and disseminated pyritic sulphides. The proportion of each type varies from deposit to deposit. The distribution and proportion of each is not well known at Dy. Drilling to date indicates that the bulk of the higher grade material is massive sulphide, but some disseminated sulphide-bearing quartzites are also high grade. All rock code references in the following sections are to the old Cyprus Anvil lithostratigraphic code since all Dy drillholes are logged in this format.

Massive Sulphides

The dominant rock type in the massive sulphide lithofacies is massive pyritic sulphide (4E) which is gradational into barite bearing massive sulphide (4G). Massive pyritic sulphide consists of homogeneous to finely banded, usually weakly foliated, fine grained massive pyrite with lesser sphalerite and galena. Total sulphide content is at least 60%, generally greater than 80%, and commonly near 100%. Gangue consists of quartz ± barite (less than 10%) ± carbonates (calcite, dolomite, ankerite, siderite). Accessory minerals include pyrrhotite, magnetite, chalcopyrite, arsenopyrite and marcasite.

The baritic massive sulphides (4G) are a well banded rock consisting of alternating barite poor and barite rich bands on a scale of a few millimetres. Barite content is at least 10% and generally near 30%; rarely is there more than 50% barite by volume in this rock type. The baritic massive sulphides are usually high grade. They tend to be slightly more lead and silver rich than other rock types. The barite lithofacies commonly contains fine magnetite and less commonly is carbonate bearing.

Other less important massive sulphide lithofacies at Dy contain up to 70% pyrrhotite (4H), or up to 50% carbonate as flesh-coloured blebs or rounded clasts in barren pyrite (4K).

Disseminated Sulphides in Quartzite

The dominant rock type in the quartzose, disseminated sulphide lithofacies is ribbon banded graphitic quartzite (4A). This unit is dark grey to black, moderately hard to very hard, well banded, fine grained, sulphide bearing, carbonaceous, locally micaceous quartzite. Compositional bands usually range from 1 mm to 2 cm thick. The bands are alternating dark grey to black, very fine grained, locally micaceous quartzite interbanded with light grey to locally red-brown, fine grained, quartz-sulphide bands. Pyrite is usually the dominant sulphide species with lesser sphalerite and galena. Locally, lead-zinc sulphides, particularly light red-brown sphalerite, are dominant. Locally, pyrrhotite is present rather than pyrite but is only a minor constituent overall. Carbon content is normally within the ¼ to ½% range and generally occurs in thin coatings concentrated on cleavage surfaces.

Chalcopyrite occurs locally in traces as small blebs and infills of hairline fractures. Total sulphide content varies from 15% to 30% and may locally range up to 60%.

An important variant of the disseminated sulphides (4D where > 4% Pb+Zn, or 4C where < 4% Pb+Zn)is lighter coloured, deficient in carbon, less well banded and more sulphide (particularly pyrite) rich than the ribbon banded quartzites. Major sulphide minerals are pyrite, galena and sphalerite. Total sulphide content is generally in the range 30 to 60%. Gangue is quartz with lesser carbonate. Accessory minerals are magnetite, chalcopyrite and/or pyrrhotite. 4A is completely gradational to 4D/4C and some pyritic quartzites appear to be related to 4A by alteration involving decarbonation adjacent to greenstone sills/dykes.

Several types of altered phyllite (4L) are present at the Dy deposit. The most common are light greenish-grey chlorite muscovite phyllites with blebs of pyrrhotite and/or pyrite. More intensely bleached muscovite > > chlorite phyllites are also common; locally these are siliceous and pyrite bearing. Intensely altered phyllites are particularly abundant toward the southwest edge of the Dy deposit.

MINERAL INVENTORY CALCULATION

Drillhole Database

The current mineral inventory calculation uses all previous drillhole information some of which was corrected prior to quantification.

In 1989 and 1990, Curragh Resources Inc. (CRI) drilled sixteen holes to test ground conditions at locations proposed for shaft and ramp access to the deposit. Four of these holes intersected the Dy deposit and returned high grade intersections from the B2 horizon.

Five additional delineation holes were drilled in 1991. Hole locations are given in Figure 2. Three holes, located to test the southeastern part of an upper horizon, failed to intersect high grade mineralization (i.e. 9% lead + zinc over 3.5 m), although the holes were not continued to test deeper horizons. Two holes targeted to intersect the B2 horizon were successful.

The new mineral inventory incorporates the 1990-1991 drill results.

All drillhole data in the vicinity of the Dy deposit was entered into a computer database using GEMCOM PCXPLOR database software. All data was visually verified and corrected as necessary including field and office checks on selected surveyed collar locations. Data includes all collar survey information, downhole survey information, downhole lithologics, assays (Pb, Zn, Ag, Au, Cu, $BaSO_4$, Fe_{sol}, Fe_{insol}, Pulp SG), and structural observations.

The locational information for Dy drill holes is in particularly good order since all collars have been surveyed and most downhole surveys have been done by gyroscopic methods to avoid magnetic interference.

Table 1. Summary of Drill Holes		
Year	**Number**	**Metres**
Cyprus Anvil 1976	1	1,192
1977	11	12,175
1978	11	12,039
1979	17	18,999
1980	13	14,613
1981	3	3,222
Sub-Total	56	62,240
Curragh 1989	2	2,214
1990	22	20,549
1991	5	5,426
Sub-Total	29	28,189
GRAND TOTAL	**85**	**90,428**

Calculation Method

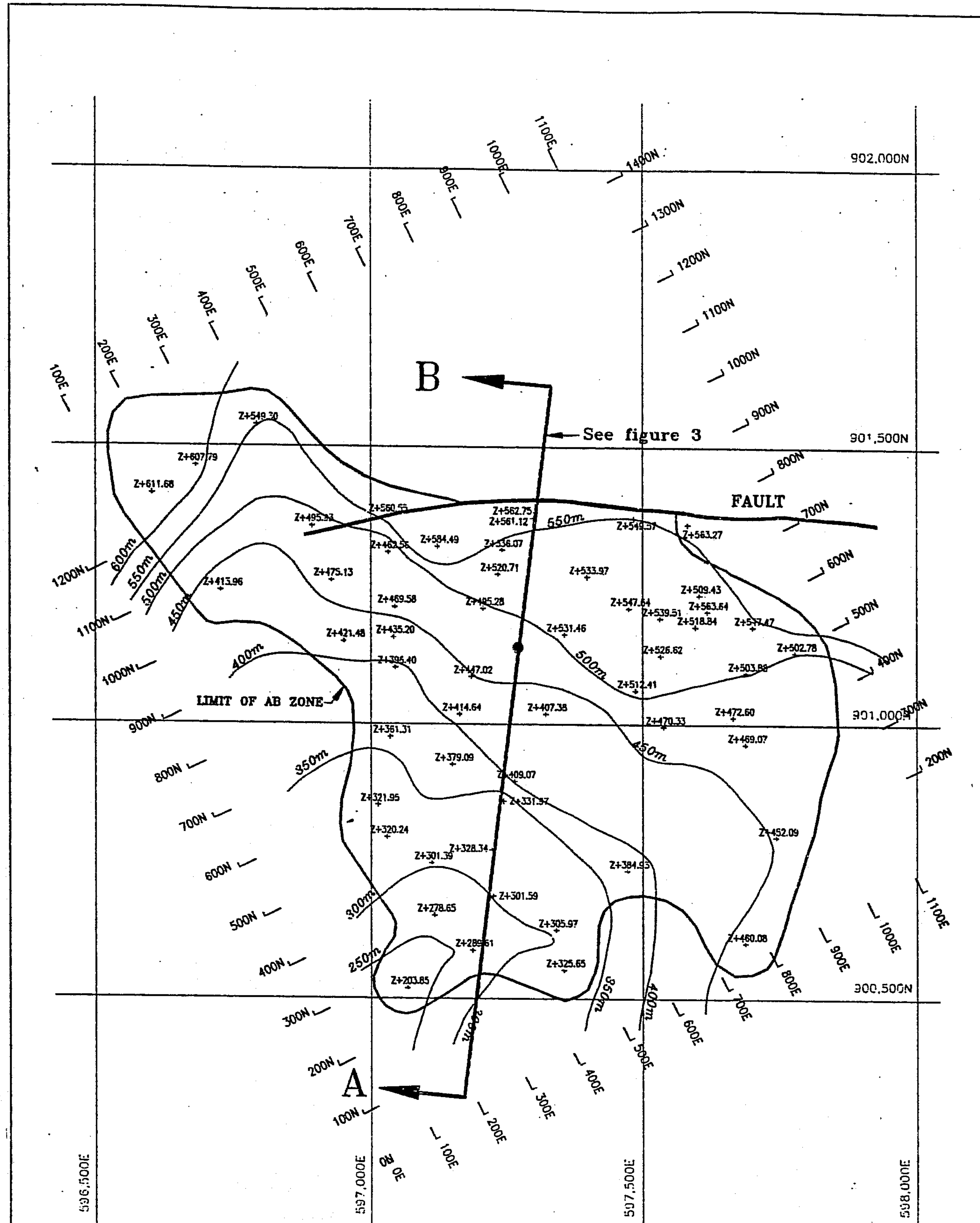
Using the above database, vertical cross and longitudinal section drawings were plotted at 50 metre intervals at 1:1250 scale. Cross and long section grids are at azimuth 63° and azimuth 153° respectively. The orientation of the cross sections was chosen to be at right angles to the long dimension of the A Zone portion of deposit. The orientation of the section grid differs greatly from the earlier Cyprus Anvil exploration grid which was more closely perpendicular to the strike of the deposit and of the second phase lineation. The new section orientation reduces drillhole offset and projection problems; as a result the deposit appears to have better continuity.

The bulk of high grade mineralization was observed to occur largely in one thick layer termed the AB Zone. The AB Zone includes both the A2 and B2 horizons identified by Rollings (1982). The inventory calculated in this study are focused on the AB Zone. The AB Zone is not fully delineated by diamond drilling; an attempt has been made to quantify possible lateral extensions to the AB Zone (termed the AB Extension Zone). Mineralization intersected above and below the AB zone possibly represents fold repeats, fault dislocations or lateral extensions of the layer, or additional separate layers. This material is considered as additional potential and is also separately quantified (as the above AB Zone and below AB Zone). The northeast portion of the AB Zone corresponds to the A-Zone mentioned previously; the west portion corresponds to the B-Zone.

The AB Zone

The AB Zone is a broad vertical interval consisting of sulphide rock types and/or altered rock, which collectively may be exhalite and/or chemical sediments. Some may simply be altered phyllitic wall rock. Lesser amounts of unmineralized, unaltered wall rock and intrusive are locally included. In a very general way the rocks above the AB Zone belong to Vangorda formation, and those below it to Mount Mye formation. The AB Zone thus corresponds approximately to the transition zone between the two formations, however, this is an oversimplification. One or more sulphide horizons variably enriched in lead and zinc are usually present. It is important to realize that the AB Zone is a broad zone that hosts subordinate sulphide lenses; most of it is phyllite. In some places the lenses may be at the AB Zone hanging wall, and elsewhere at the footwall. Contours of the AB Zone hanging wall and footwall elevation are given in Figures 4 and 5, respectively. In general the zone dips southwest from 20 to 35°. The zone is up to 160 metres thick but is mostly 40 to 75 metres or less thick. The inventory within the AB Zone was calculated at Pb+Zn cutoffs of 6%, 8% and 9% over a minimum core length of 3.5 metres. Due to the angle of intersections between the drill holes and the mineralized horizons the core length closely approximates the true thickness of the mineralization.

Assay composites were calculated over a minimum core length of 3.5 metres. If a drillhole intersected more than one qualifying intersection separated by a waste zone greater than 3.5 metres thick, the waste zone was excluded from the composite. If thin internal waste intervals were present (generally less than 3.5 metres in length) they were included in the composite. Low grade or waste was included in some composites to establish a minimum 3.5 metre core length provided that the composited grade for the 3.5 metres was still greater than the cutoff. Composites were calculated by weighting each individual assay interval by its length. There was no consideration given for lost core recovery; however core recovery is generally very good. Due to software limitations there is only



Curragh Inc.

DY PROPERTY

AB ZONE HANGINGWALL CONTOURS

0m 150m 300m

REVISIONS:

04/06/93

N/A 4

CVR OCT 21, 91 105K3

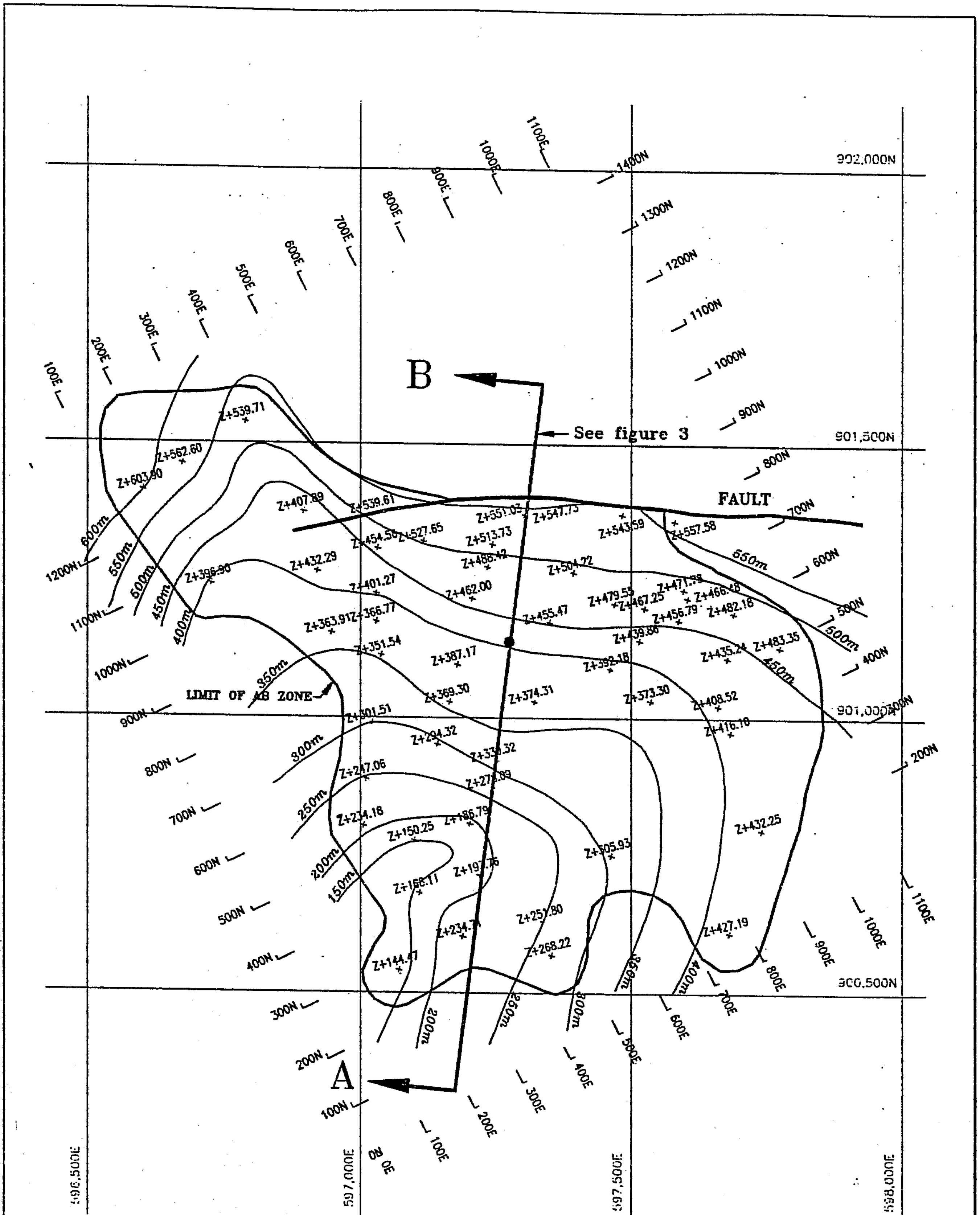
ABCONTSB.DWG

LEGEND:

Contour interval = 50m

Contours are metres above sea level

● PROPOSED SHAFT LOCATION



0m 150m 300m	**Curragh Inc.**
REVISIONS:	DY PROPERTY
04/06/93	AB ZONE FOOTWALL CONTOUR
	N/A 5
	CVR OCT 21, 91 105K3
	ABCONT5A.DWG

LEGEND:

Contour interval = 50m

Contours are metres above sea level

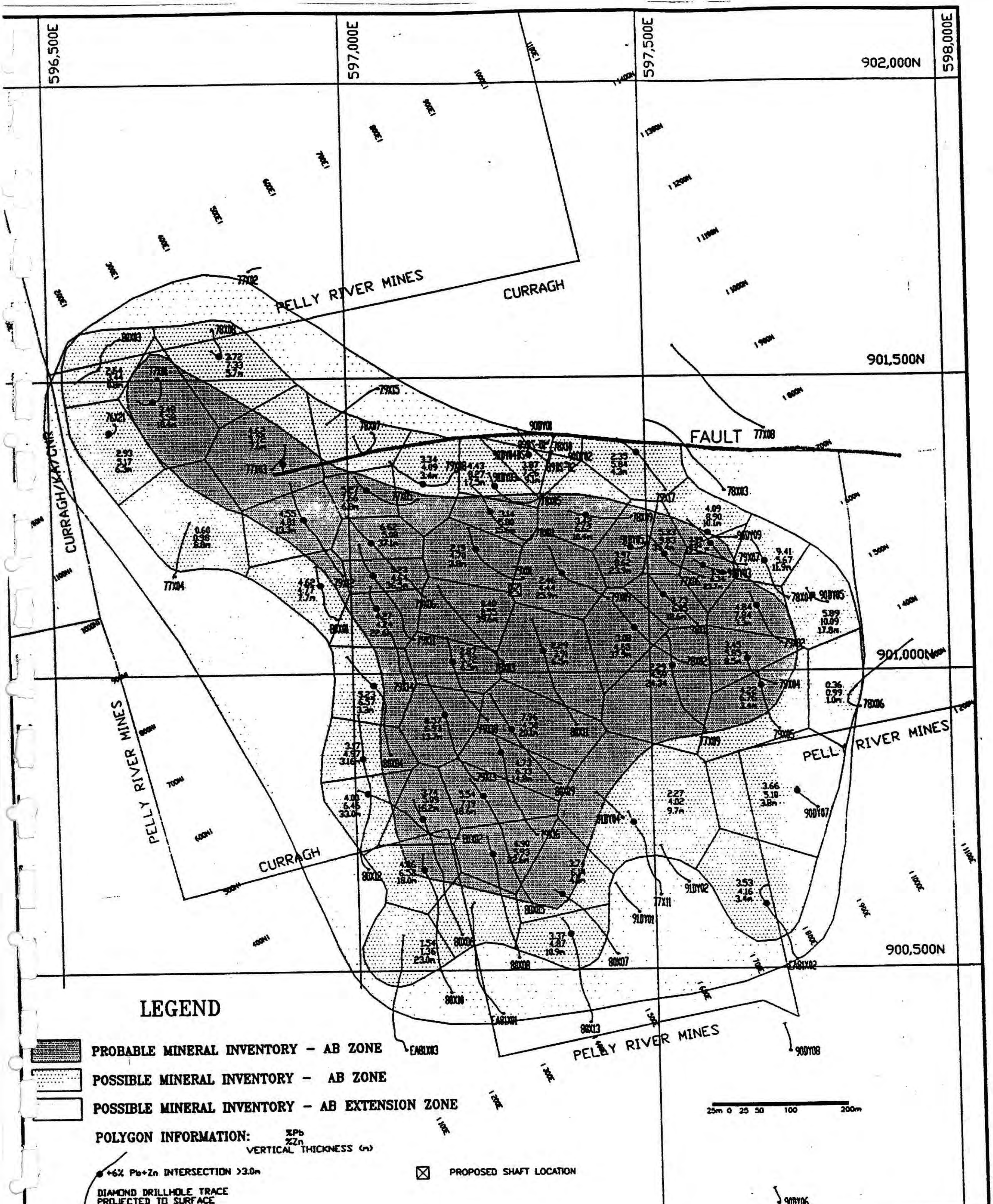
● PROPOSED SHAFT LOCATION

one composite allowed per drillhole. A number of drillholes had more than one qualifying intersection within the AB Zone; these were summed for the drillhole to make up one composite. The composite location plotted on Figures 6, 7 and 8 is the vertical projection of the centre of the total interval. This location will be slightly different for a given drillhole depending on what cutoff grade is used. The location of drillholes which did not intersect the mineralized zone were also plotted on each plan.

In plan, the area of influence of a drillhole intersection is considered to be halfway (maximum of 150 metres) to the adjoining drillhole where the deposit is reasonably defined by drilling. At the edges of the ore zone the area of influence was arbitrarily defined as 60m beyond the most outboard drillhole. Extensions of the AB Zone beyond the arbitrary 60m limit are likely, especially along the west and south edges of the drilling area, and an effort to quantify this material is made below. The outline of the AB Zone thus interpreted is shown on Figures 6, 7 and 8.

Polygon limits (Figures 6, 7 and 8) are defined by the perpendicular bisectors of lines drawn to nearby drillholes. Polygon limits were clipped against the interpreted outline of the AB Zone as defined above. Polygon areas are calculated in the horizontal plane. Polygon limits and areas were calculated and plotted using Gemcom's GEOMODEL software. Since composite locations vary slightly for each drillhole it will be noted that polygon areas also are slightly different for each drillhole depending on the cutoff grade.

Polygon volumes were calculated by multiplying the vertical thickness of the composites by the polygon area. The vertical thickness for each composite is derived (by GEOMODEL) by attempting to correct for the deviation of each drillhole from vertical at the location of each composite centre; essentially GEOMODEL subtracts the elevation of the lower end of the composite from the elevation of the higher end. Vertical thickness is thus generally less than the composite length. In gently dipping orebodies, like Dy, the vertical thickness is generally slightly greater than true thickness. The exaggerated thickness is compensated for by measuring areas in the horizontal plane where they are slightly smaller by the same factor (cosine of dip) than they would be if measured in the plane of the orebody. This results in the volume calculated being a close approximation of the volume of the dipping ore layer. In the case of the Dy deposit, the software's misleading use of "vertical thickness" leads to a conservative volume since most drillholes, due to their great length, tend to deviate until they are close to perpendicular to S_2 and consequently also the ore layers; the composite lengths are thus relatively close to true



Curragh Inc. FIGURE 6	DY DEPOSIT - MINERAL INVENTORY POLYGONAL ESTIMATE 6% LEAD + ZINC CUTOFF

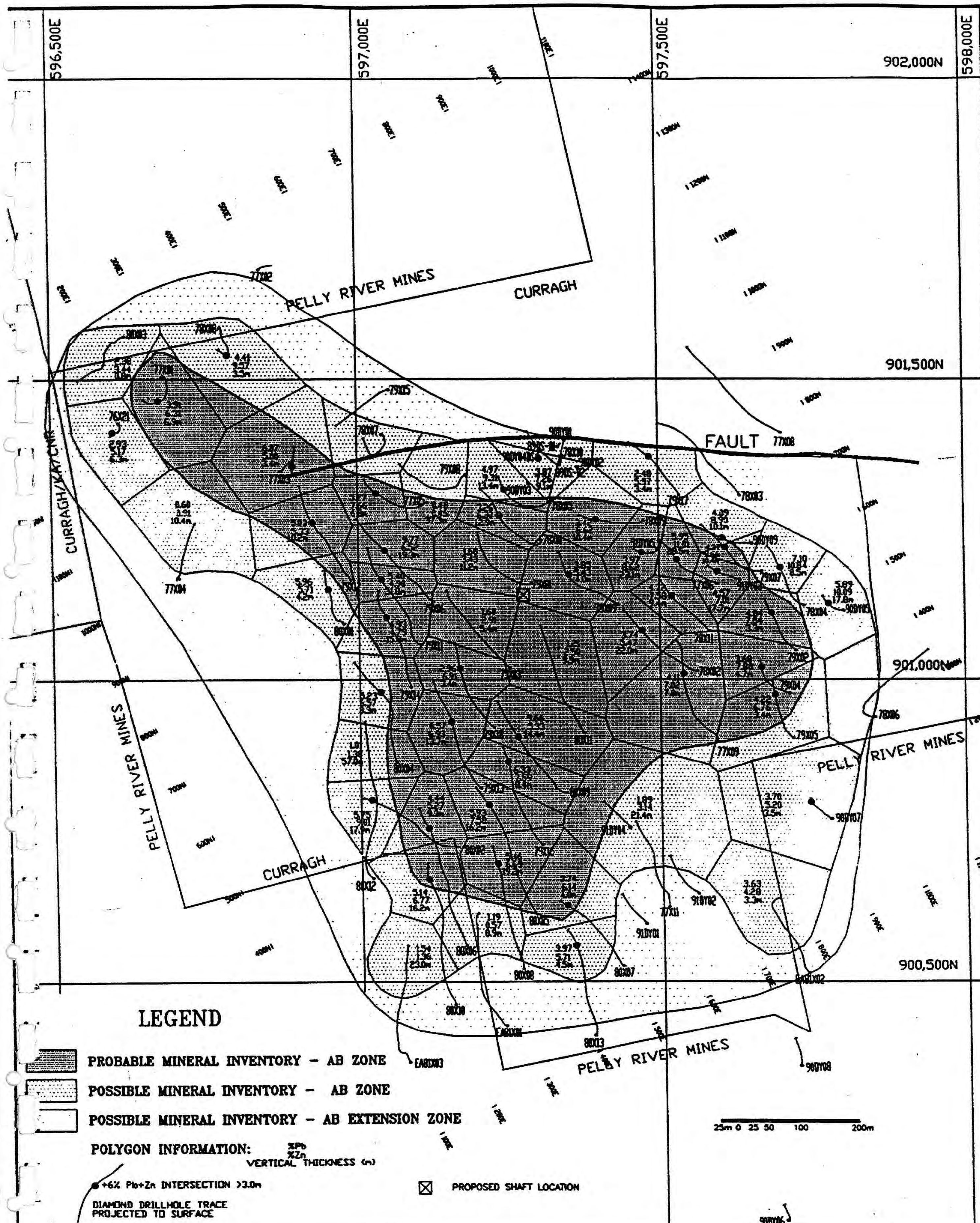


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FIGURE 7

DY DEPOSIT - MINERAL INVENTORY
POLYGONAL ESTIMATE
8% LEAD + ZINC CUTOFF

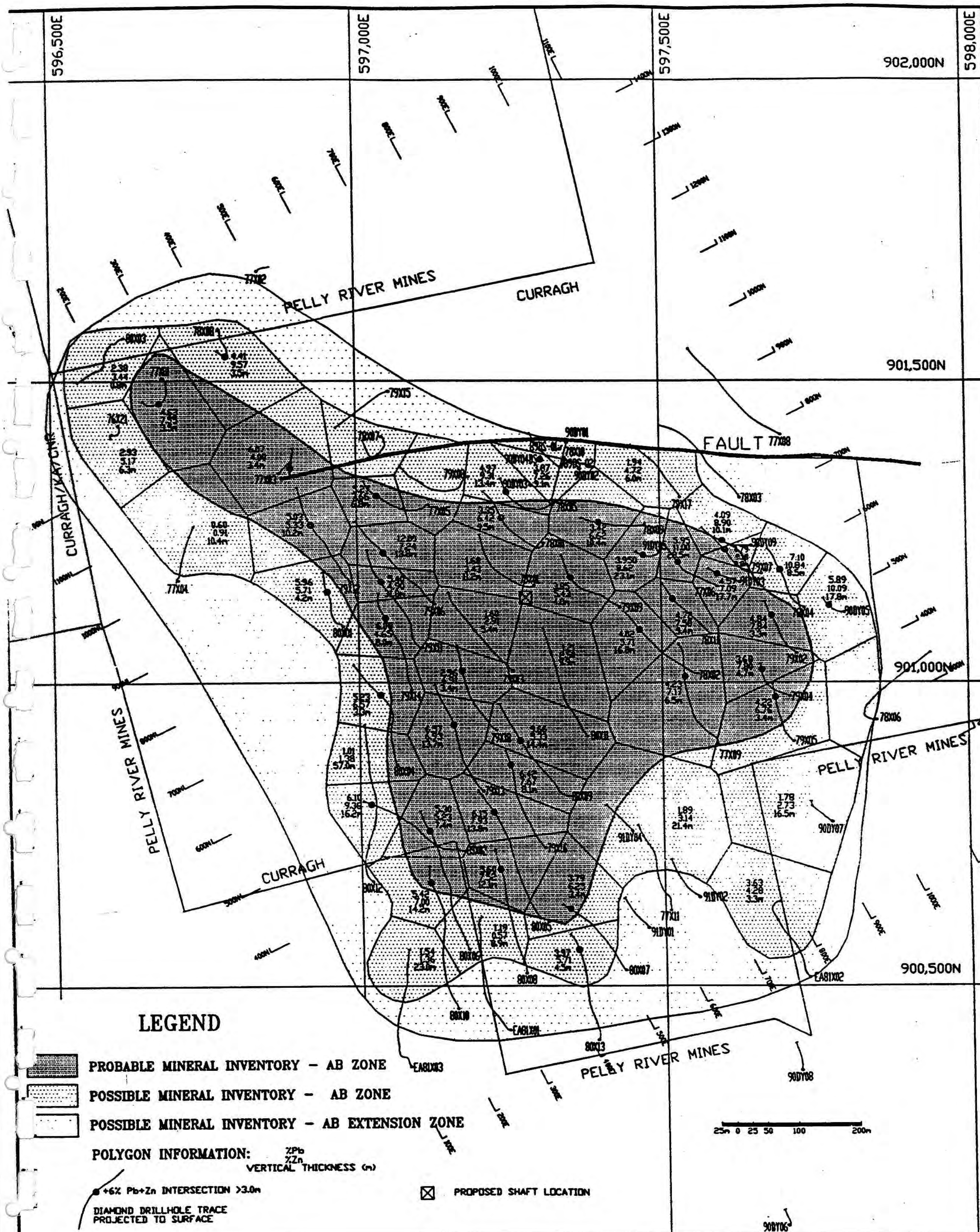

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FIGURE 8

DY DEPOSIT – MINERAL INVENTORY
POLYGONAL ESTIMATE
9% LEAD + ZINC CUTOFF

thicknesses and the conversion to "vertical thickness" by the GEOMODEL software reduces them rather than enlarging them.

Polygon volumes are converted to tonnage using a density of 3.92 tonnes/cubic metre for all ore types. This density is arrived at through examination of averages of pulp specific gravity data for composites at a 6, 8 and 9% Pb+Zn cutoff. To account for porosity of the in-situ rock the pulp specific gravity average was reduced by 2%. This reduction is based on experience at Faro with similar ores and is an empirical factor which gives a good fit between calculated ore tonnage and pit production tonnage as delineated by blastholes.

The AB Extension Zone

The outline of the AB Zone as defined above by the arbitrary 60m limit has an embayed appearance in the west and south due to a few more outboard holes. The vertical projection of deposit limits in the better known deposits of the district, such as Faro, tends to be smoother. An additional, more generous, deposit outline which extends projection of peripheral holes about 30% and smooths out irregularities in the AB Zone outline has been added to Figures 6, 7 and 8 where it is indicated to be the limit of the AB Extension Zone. This additional, possible, peripheral mineralization is highly subjective, it has been made somewhat more generous where bounding holes are too short or contain some sulphides and indications of fault complications (as in the north) or less generous in areas thought to have difficulties with dykes interfering with the ore zones (as in the east). The estimate of tonnage and grade for this area is made following the above procedures and using the composites described above but enlarging the area of influence for each drillhole to a radius large enough to reach the limit of the extension zone. The estimate is thus provided by the grade and thickness of the peripheral holes that have composites above cutoff grade. Since the zone is expected to thin gradually in this area the tonnage has been reduced by half.

Above and Below the AB Zone

There is additional mineralization above and below that assigned to the AB Zone. Some of this mineralization exceeds the grade/cutoff criteria however continuity of horizons could not be established as the intersections are singular or widely spaced. To reflect the potential that these intersections represent a calculation was made on the basis of a radius of influence of 50m and a thickness equal to the composite length. Composites were calculated as described previously. A specific gravity of 3.92 was used for all cutoffs.

Classification of Mineral Inventory

There are currently no underground openings at Dy; furthermore, drillhole spacing is not adequate to consider any portion of the deposit to be proven. The inventory was categorized as probable or possible based on the following criteria which are influenced greatly by experience with other Anvil District mineral deposits. In these deposits the mineralization in the interiors of the deposit is reasonably continuous on a broad scale although highly unpredictable in detail. Extrapolations of general thickness and tenor along the deposit grain of 60 to 150m are not unreasonable. Extrapolations across the grain, however, show more limited success and approximately 30m is reasonable.

Extensive experience in the Faro and Vangorda deposits shows that, in an open pit context, an average drillhole spacing of at least 30.5m (ideally on a 15 to 23m by 30 to 43m grid basis) is required to confidently and accurately define deposit structure, tonnage and grade.

Experience with Faro, Vangorda and Grum shows that drillhole spacings as broad as 60m by 120m are adequate to broadly outline global deposit tonnage and grade although much refinement is required for local confidence. The edges of deposits are more difficult to estimate. Based on these observations the following definition of probable and possible was developed.

Probable mineralization is that in a sulphide horizon which can be correlated with reasonable confidence and is delineated both up and down dip and along strike by diamond drilling or is limited by a well known structural or topographic discontinuity. The range of extrapolation within the zone can be justified by comparison to other deposits of similar nature in the same region.

In plan view this criteria results in restricting the probable material to that within a limit inside of the last peripheral hole in the drill array. That limit is shown on Figures 6, 7 and 8.

Possible mineralization is the result of a quantitative estimate based on widely spaced drillholes and largely on broad knowledge of the geological character of the deposit and similar nearby deposits. The continuity of mineralization is not necessarily confirmed up or down dip or along strike by drillholes or other sample points.

As applied to the Dy deposit this criteria results in all mineralization beyond the probable inventory limit described in the previous section and all mineralization above or below the AB Zone being classified as possible.

Results

The results of the mineral inventory estimation for the individual zones of the Dy deposit and the total deposit are provided in Tables 2 through 6. Results in all cases are presented for 6, 8 and 9% Pb+Zn sample cutoffs. In all cases the figures quoted are for undiluted, in-situ material. Since the material cannot necessarily be extracted as delineated it is not considered a reserve.

AB Zone

Table 2 gives the result of the estimate for probable and possible material within the limit of the AB Zone defined by the arbitrary 60m extrapolation limit on Figures 6, 7 and 8 (i.e. within the inner two more densely stippled areas on those figures).

AB Extension Zone

Table 3 provides the result for the quantification of peripheral mineralization around the AB Zone. Specifically this includes the material in the outermost, sparsely stippled area, on Figures 6, 7 and 8. All this mineralization is considered possible and is considered less firmly defined than the possible mineralization

Above and Below the AB Zone

Quantities of mineralization above and below the AB Zone are tabulated on Table 4. The Above AB Zone mineralization is based on scattered intersections that tend to occur just east or northeast of the A Zone. The Below AB Zone mineralization occurs in scattered intersections just southwest of the B Zone. All this mineralization is classified as possible and is comparable in certainty to the AB Extension Zone. Further underground drilling may elevate this material to proven and probable ore locally.

Total Deposit Summary

A summary of all zones for the total deposit is provided in Table 5.

TABLE 2. Mineral Inventory for AB Zone, Dy Deposit In-situ - Undiluted

	Tonnes	%Pb+Zn	%Pb	%Zn	Ag (g/mt)	Au (g/mt)
6% Pb+Zn Cutoff						
Probable	24,949,000	9.70	4.21	5.49	63.0	0.67
Possible	10,348,000	10.43	4.01	6.42	61.3	0.62
Total	35,297,000	9.91	4.15	5.76	62.5	0.60
8% Pb+Zn Cutoff						
Probable	14,895,000	12.06	5.43	6.63	80.0	0.87
Possible	6,720,000	12.59	4.84	7.75	73.4	0.80
Total	21,705,000	12.23	5.25	6.98	78.0	0.84
9% Pb+Zn Cutoff						
Probable	13,133,000	12.58	5.71	6.87	83.1	0.86
Possible	5,389,000	13.62	5.26	8.36	78.2	0.85
Total	18,522,000	12.88	5.58	7.30	81.7	0.85

TABLE 3. Mineral Inventory for the AB Extension Zone, Dy Deposit In-situ, Undiluted, Possible Mineralization

	Tonnes	%Pb+Zn	%Pb	%Zn	Ag (g/mt)	Au (g/mt)
6% Pb+Zn Cutoff	3,746,000	10.30	4.07	6.23	61.9	0.63
8% Pb+Zn Cutoff	2,094,000	13.39	5.28	8.11	79.4	0.94
9% Pb+Zn Cutoff	1,904,000	13.92	5.52	8.40	82.0	0.98

TABLE 4. Mineral Inventory Above and Below the AB Zone, Dy Deposit In-situ, Undiluted Possible Mineralization

6% Cutoff	Tonnes	%Pb+Zn	%Pb	%Zn	Ag (g/mt)	Au (g/mt)
Above	1,828,800	7.77	3.81	3.96	50.6	0.36
Below	683,500	9.05	3.60	5.45	58.9	0.56
Total	2,512,300	8.12	3.75	4.37	52.9	0.41
8% Cutoff						
Above	606,500	10.27	4.96	5.31	65.2	0.74
Below	541,900	9.78	3.91	5.88	62.9	0.63
Total	1,148,400	10.04	4.46	5.58	64.1	0.69
9% Cutoff						
Above	606,500	10.27	4.96	5.31	65.2	0.74
Below	323,300	10.78	4.50	6.29	74.9	0.89
Total	929,800	10.45	4.80	5.65	68.6	0.79

TABLE 5. Dy Deposit Summary of Mineral Inventory for Entire Deposit In-situu, Undiluted

	Category	Tonnes	%Pb+Zn	%Pb	%Zn	Ag (g/mt)	Au (g/mt)
6% Cutoff							
AB Zone	Probable	24,949,000	9.70	4.21	5.49	63.0	0.67
AB Zone	Possible	10,348,000	10.43	4.01	6.42	61.3	0.62
AB Extension	Possible	3,746,000	10.30	4.07	6.23	61.9	0.63
Above & below AB	Possible	2,512,000	8.12	3.75	4.37	52.9	0.41
Subtotal	Probable	24,949,000	9.70	4.21	5.49	63.0	0.67
Subtotal	Possible	16,606,000	10.05	3.98	6.07	60.2	0.59
Grand Total	Probable+Possible	41,555,000	9.84	4.12	5.72	61.9	0.65
8% Cutoff							
AB Zone	Probable	14,985,000	12.06	5.43	6.63	80.0	0.87
AB Zone	Possible	6,720,000	12.59	4.84	7.75	73.4	0.80
AB Extension	Possible	2,094,000	13.39	5.28	8.11	79.4	0.94
Above & below AB	Possible	1,148,000	10.04	4.46	5.58	64.1	0.69
Subtotal	Probable	14,985,000	12.06	5.43	6.63	80.0	0.87
Subtotal	Possible	9,962,000	12.47	4.89	7.58	73.6	0.82
Grand Total	Probable+Possible	24,947,000	12.22	5.21	7.01	77.4	0.85
9% Cutoff							
AB Zone	Probable	13,133,000	12.58	5.71	6.87	83.1	0.86
AB Zone	Possible	5,389,000	13.62	5.26	8.36	78.2	0.85
AB Extension	Possible	1,904,000	13.92	5.52	8.40	82.0	0.98
Above & below AB	Possible	929,800	10.45	4.80	5.65	68.6	0.79
Subtotal	Probable	13,133,000	12.58	5.71	6.87	83.1	0.86
Subtotal	Possible	8,223,000	13.33	5.27	8.06	78.0	0.87
Grand Total	Probable+Possible	21,356,000	12.87	5.54	7.33	81.1	0.87

Discussion of Inventory

Polygonal calculations are widely recognized to have significant short comings in estimating tonnage and grade of sparsely drilled deposits. This is due to what is essentially a force fitting of the grade distribution for large ore blocks so that it is the same as that of the assay composites. The assay composite population will contain more extreme values than the ore block population which will tend to result in a higher average value above a given cutoff grade from the polygonal calculation than will actually occur in nature. The degree of overestimation has been variously estimated at approximately 10% but, of course, depends on actual grade distributions. The phenomena is known to occur in Anvil District deposits but has not been quantified.

Intuitively it can be appreciated that the polygonal calculation cannot be realistic since it is inherent in the assumptions that the grade of an assay composite can be extrapolated over great distances. A proper range of influence for a drillhole assay composite in any of the Anvil District deposits has not yet been satisfactorily worked out. Preliminary information, however, suggests it may be 30m or less across the deposit grain and twice that along the grain. Experimental semi-variograms at Faro and Vangorda suggest the range may be even smaller than this for drillcore assays. This raises the essential question: "given that it is reasonable to extrapolate ore zones from hole to hole over distances as great as 200m, is it then logical to attempt to weight the grade of ore 100m from a drillhole with the value of assays from that hole if everything indicates the likely range of that hole is well under 30m?". The answer to this question would seem to be no.

An alternative is to assume that the drillholes all have equal weight and to arrive at an average grade for the deposit as the arithmetic average of the grades of all the drillholes in the deposit. This has been done for the AB Zone at an 8% Pb+Zn cutoff giving an even higher grade than the polygonal method. It would appear that the method of global averaging does nothing to refute the applicability of the polygonal calculation; other more complicated calculations have not been attempted. There, of course, remains a possibility that the polygonal calculation will overestimate the inventory, however this may be balanced by the conservatism inherent in the treatment of vertical thickness.

Table 6 compares the results of the 1991 Mineral Inventory to previous work. This table shows that the comparison for tonnage and grade is reasonably good at the 9% cutoff grade. Because of differences in interpreted outline, the AB Zone limit is larger than the limits used previously, but extra tonnage here is compensated by less generous treatment of the Above and Below AB intersections in the 1991 work.

The sensitivity of tonnage to cutoff grade is quite notable. This is due to incorporating more disseminated rock types into the reserve at the lower cutoffs. This sensitivity shows that lower mining costs and, in particular, lower transportation costs to the Faro mill, will pay off not only in greater profits but significant reserve increases.

TABLE 6. Comparison of 1991 Mineral Inventory to Previous Work

Calculation	Cutoff (%Pb+Zn)	Tonnes	% Pb	% Zn	Pb+Zn	Ag (g/mt)	Au (g/m
CRI 91	6	41,555,000	4.12	5.72	9.84	61.9	0.58
CRI 91	8	24,947,000	5.21	7.01	12.22	77.4	0.85
CRI 91	9	21,356,000	5.54	7.33	12.87	81.1	0.87
Cyprus Anvil Hall 81	9	21,334,127	5.68	6.95	12.63	81.6	n/a
" " Rollings 82	9	21,059,980	5.54	6.74	12.28	83.77	0.95
Kilborn 89	9	20,114,825	5.47	6.77	12.44	84.5	0.91

EXPLORATION POTENTIAL

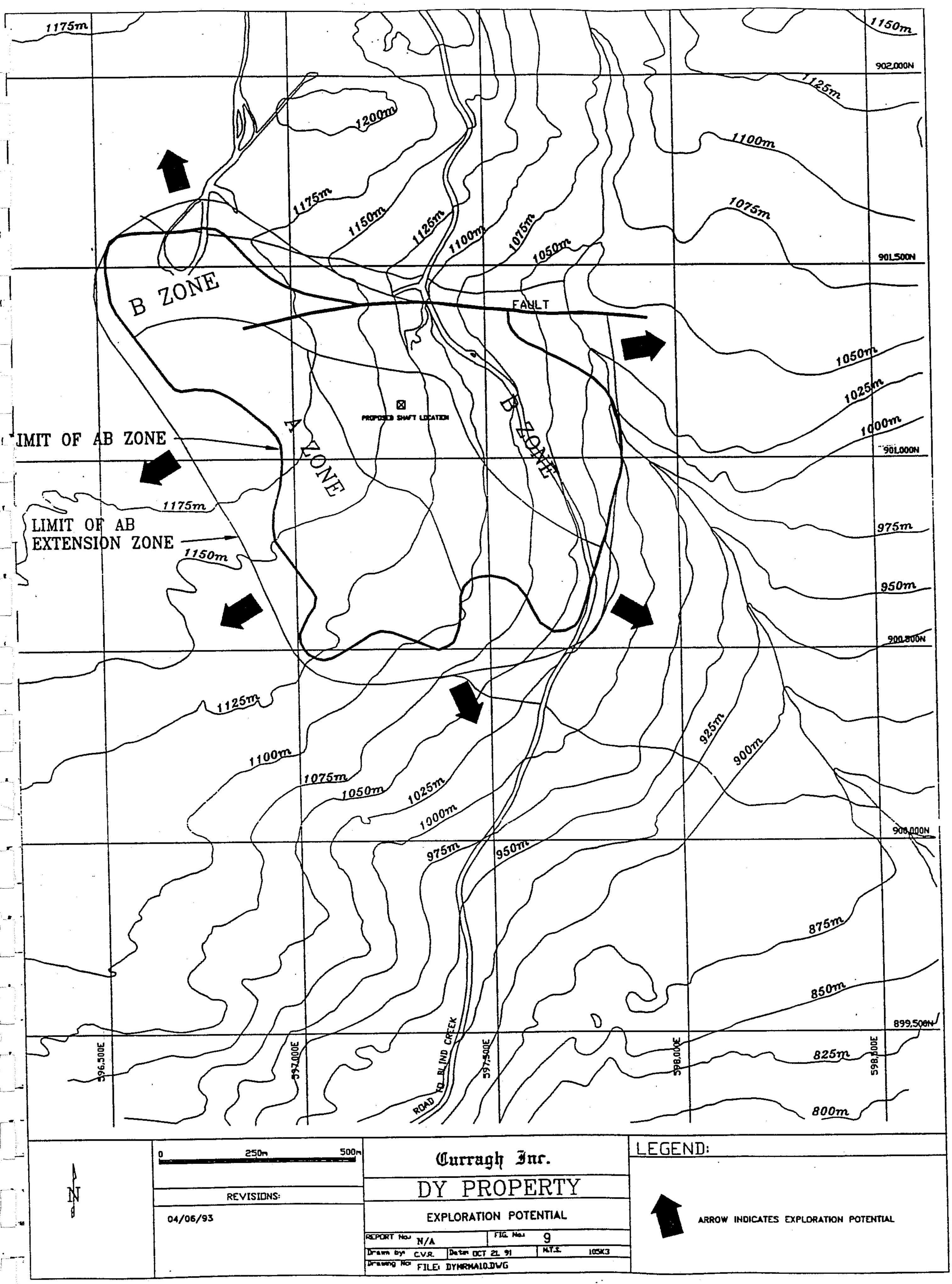
There is considerable potential to extend the deposit by additional stepout drilling. The deposit is closed off by drilling only locally (Figure 9) and it is likely that additional mineralization will be found.

Exploration should proceed to the east of the deposit at least as far as the fault paralleling Blind Creek (not indicated on figures) since some of the peripheral intersections in that area are quite good and drilling in that direction was terminated by Cyprus Anvil because of concern over shallowly dipping dykes or sills interfering with the ore horizons.

Additional stepout drilling will be required to the west and south of the deposit. These areas are deep and larger drills than used to date will be required. This area is of lower priority as this part of the deposit could only be brought into production late in the likely mine life.

In order to quantify the potential additional mineralization that might be added to the Dy deposit, it has been hypothesized that along each of the south and east margins, a 200m further extension is possible along a 600m length of the deposit. No allowance has been made beyond the AB Extension Zone on the north or west because structural complications related to faults are expected in those directions. If average thickness is 5 to 6 metres, this would represent approximately an additional 5 million tonnes of potential mineralization. Approximately 12 holes 600 to 1100m deep (a total of 10,500m or approximately $1.6 million) would be needed to test this area.

There has been considerable speculation over a second deposit approximately one or two km to the north of the Dy beneath the thick sequence of mafic igneous rich Vangorda formation preserved there. These concepts should be evaluated by deep drilling in that area.



B ZONE
FAULT
A ZONE
B ZONE
PROPOSED SHAFT LOCATION
LIMIT OF AB ZONE
LIMIT OF AB EXTENSION ZONE
ROAD TO BLIND CREEK
1175m
1200m
1150m
1125m
1100m
1075m
1050m
1025m
1000m
975m
950m
925m
900m
875m
850m
825m
800m
902,000N
901,500N
901,000N
900,500N
900,000N
899,500N
596,500E
597,000E
597,500E
598,000E
598,500E
0
250m
500m
N
REVISIONS:
04/06/93
Curragh Inc.
DY PROPERTY
EXPLORATION POTENTIAL
REPORT No: N/A
FIG. No: 9
Drawn by: C.V.R.
Date: OCT 21, 91
N.T.S. 105K3
Drawing No: FILE: DYMRMA10.DWG
LEGEND:
ARROW INDICATES EXPLORATION POTENTIAL

Within the area already drilled much additional drilling will be needed to upgrade confidence in the mineral inventory and further define structure.

Consideration should be given to use of deep penetrating Telluric EM to evaluate its possible use as a guide to locating stepout holes in the future. Borehole geophysics may be worth trying. In light of the highly graphitic environment to the south and southwest of the deposit it may not be helpful there.

MINABLE RESERVE

There have been several preliminary studies of mining of the Dy deposit, including a study by Wright Engineers in 1978 for Cyprus Anvil (very early in the understanding of the deposit), Canadian Mine Development in 1988, and Fox Geological in 1992, both for Curragh. These studies tended to focus on variants of room and pillar mining, and predicted mining recoveries of 75% without backfill, or even higher if backfill were used.

Experience at Faro underground suggests that recovery of 65% of the preliminary mineral inventory is reasonable and justified. Kilborn calculated in-situ mineral inventory of 2.61 million tonnes for the Faro underground, and 1.78 million tonnes was actually recovered by the room and pillar mining method. The Faro underground is similar in style to the second phase fold limbs at Dy. Higher recovery and lower dilution would probably be possible in fold nose areas.

The above noted studies projected dilution in the range of 5% to 10% for the room and pillar method. Dilution of 10% has been selected, which is on the high range of the various studies. Since many highgrade intersections at Dy are bordered by mineralization near cutoff grade, it is assumed that the dilutant will have a combined lead-zinc content of 7.5% (distributed as 4.8% Zn 2.7% Pb), 42 g/t Ag, and 0.35 g/t Au).

A mine quantity has been developed for use for long range planning, using the probable as well as possible inventory for the AB Zone. For the purpose of the calculation the deposit has been divided into three phases, Phases I and II above the elevation of the shaft intersection with the AB Zone, and Phase III below it. Phases I and II represent most of the east and west ends of the zinc-rich B-Zone noted previously. Phase III represents the lead-rich A-Zone and the central barren zone, and a small downdip part of the B-Zone. Table 7 details the diluted and in-situ material in each of these phases. It is emphasized that since possible mineralization is included, this should not be considered a reserve.

Using the above assumptions and the probable portion of the mineral inventory given in Table 2, the minable reserve is 9,390,095 tonnes, averaging 6.62% Zn, 5.50% Pb, 80.3 g/t Ag, and 0.82 g/t Au.

FURTHER WORK

Considerable additional fill-in drilling is required from planned underground openings to more reliably define the deposit. A detailed plan for this underground drilling should be laid out as soon as possible.

Preliminary hydrogeological work indicates that most of the rock mass will not make water. Some fault zones, however, can be expected to be significant aquifers which could discharge large quantities of water into the underground workings until pressure is relieved. Scheduling for underground advance should take cognizance of the likely wet conditions and appropriate dewatering facilities should be in place.

Hydrogeological information has proved to be of great use for environmental purposes and further data should be collected on groundwater flows as the deposit is developed.

The role of S_2 in geotechnical stability should be carefully considered. In light of the weakness of S_2 large horizontal spans may be undesirable.

Potentially acid generating rocks will be encountered at approximately 450m depth in the shaft. At that point particularly, close liaison will be needed between the underground contractor and the environmental staff for solid waste management on surface.

This study benefited greatly from a brief re-interpretation of the geology of the Dy deposit. Further work is warranted particularly to address questions of dyke, fold and fault orientation. Once a better structural interpretation is available then a more sophisticated calculation may be in order.

Study of drill logs makes it clear that Dy lags behind other deposit in quality and consistency of logging. The available drill core should be relogged and brought to a common standard compared to the other deposits. Ideally this should precede any major structural reinterpretation.

Nothing discovered in the course of this study suggests that plans for immediate underground access should be changed. It is still evident that early exploration should concentrate on the B Zone as it is slightly higher grade and more zinc rich than the A Zone.

Once underground exploration commences, geological data collection should begin immediately. The workings should be mapped in detail to help clarify fault patterns.

Holes should be drilled to test rock conditions at proposed shaft site. Five holes including both vertical holes near the shaft location, and angle holes drilled across it, are proposed.

Table 7
Estimate of Diluted Recoverable Mineralization
For the Dy Mineplan

Phase	Tonnes	Pb+Zn (%)	Zinc (%)	Lead (%)	Silver (g/t)	Gold (g/t)
1	4,830,584	13.11	8.43	4.68	73.75	0.61
2	1,449,697	12.94	6.97	5.97	88.93	0.87
3	8,298,161	12.20	6.64	5.56	78.84	0.91
Total	14,578,442	12.57	7.26	5.31	78.15	0.81

LISTS of FIGURES

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INTRODUCTION

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

HISTORY

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

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

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

Kerr Addison sold the deposit, along with Vangorda and Swim, to Cyprus Anvil Mining Corporation in 1979. From 1980 to 1982 Cyprus Anvil drilled additional holes in and around the deposit, and relogged, resampled, and reassayed most Kerr Addison drillholes.

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

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

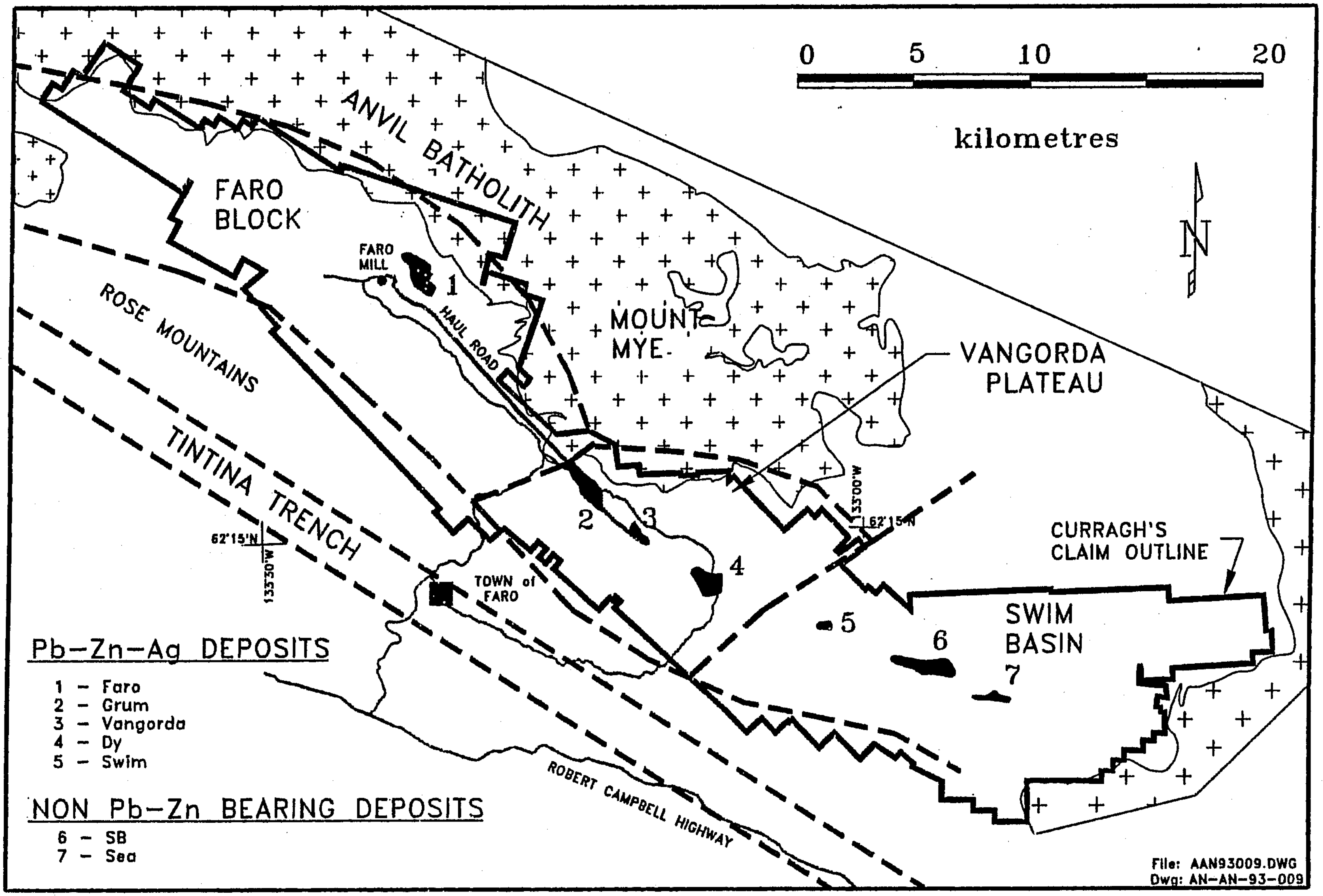


Figure 1. Location of the Grum Deposit Within the Anvil District

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

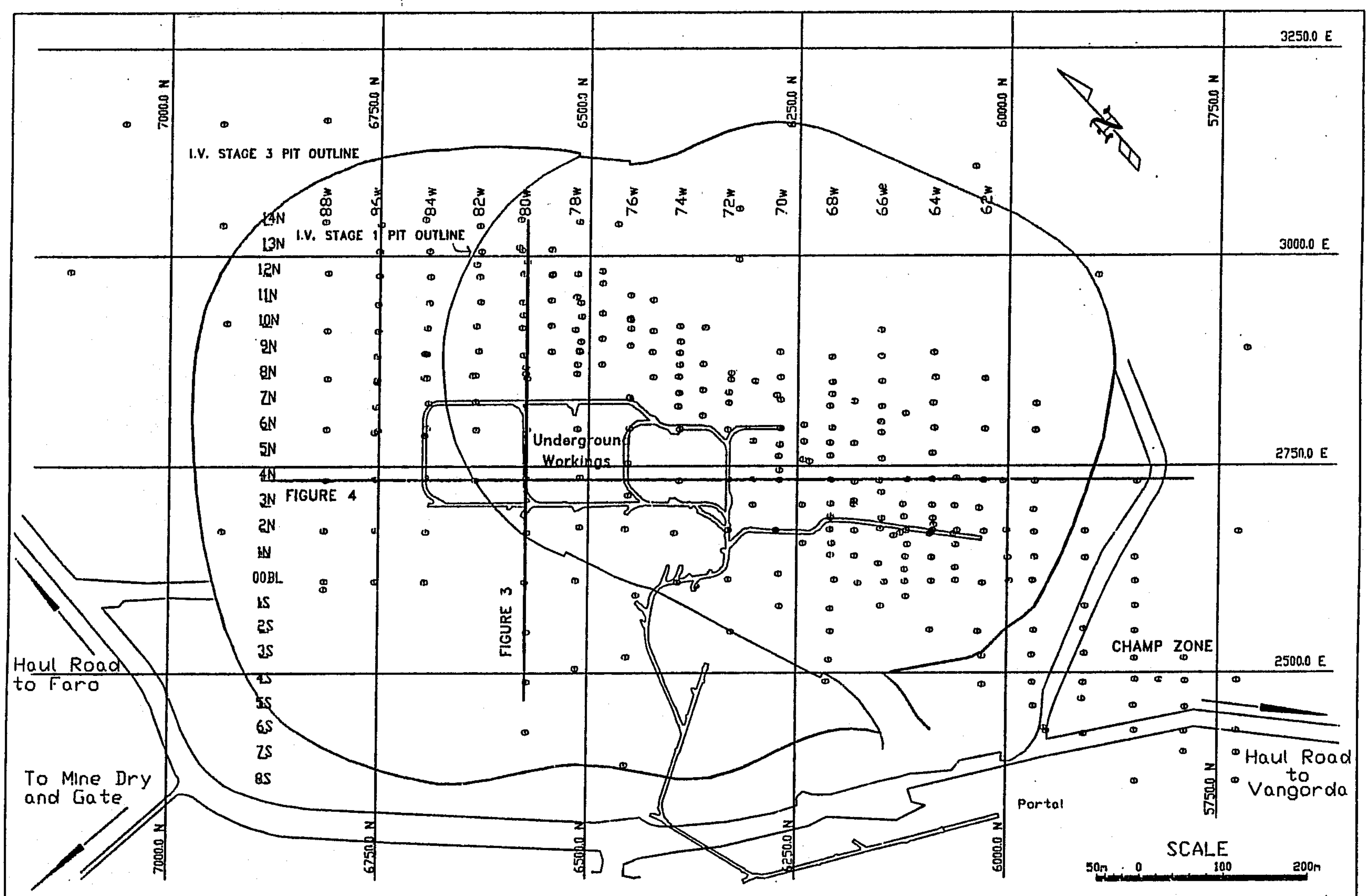
INFORMATION BASE

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

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

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

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



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

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

- The Champ Zone (51W to 60W) in the southeast part of the deposit;
- The central portion 61W to 87W, and
- the Northwest Extension, which is northwest of Section 87W.

Table 2. Summary of Drilling History of the Grum Deposit

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

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

GENERAL GEOLOGY

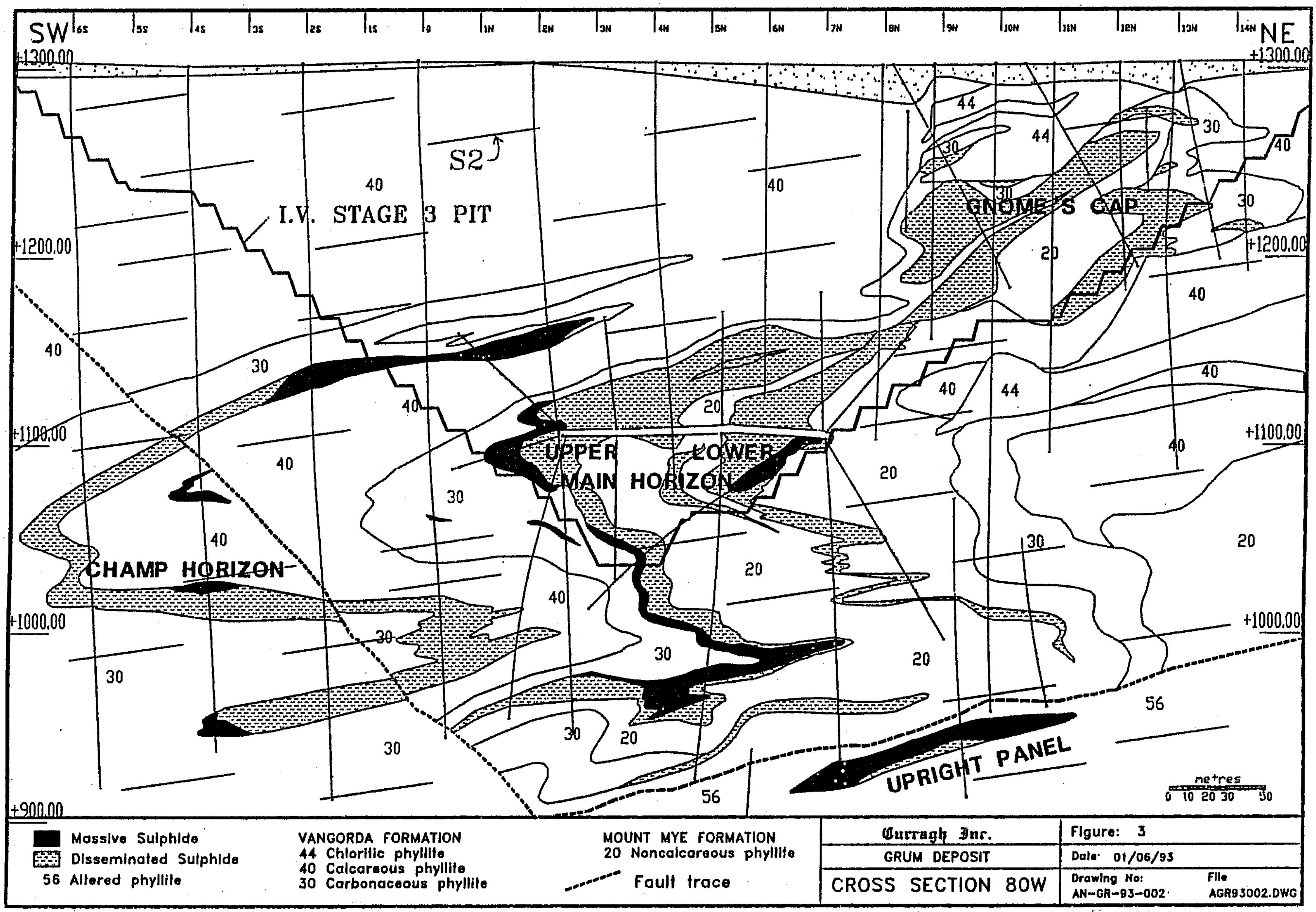
Stratigraphy and Lithology of the Grum Area

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

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

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

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


SW
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+1000.00
+900.00
S2
I.V. STAGE 3 PIT
GNOME'S GAP
UPPER
LOWER
MAIN HORIZON
CHAMP HORIZON
UPRIGHT PANEL
metres
0 10 20 30 50
Massive Sulphide
Disseminated Sulphide
56 Altered phyllite
VANGORDA FORMATION
44 Chloritic phyllite
40 Calcareous phyllite
30 Carbonaceous phyllite
MOUNT MYE FORMATION
20 Noncalcareous phyllite
Fault trace
Curragh Inc.
GRUM DEPOSIT
CROSS SECTION 80W
Figure: 3
Date: 01/06/93
Drawing No: AN-GR-93-002
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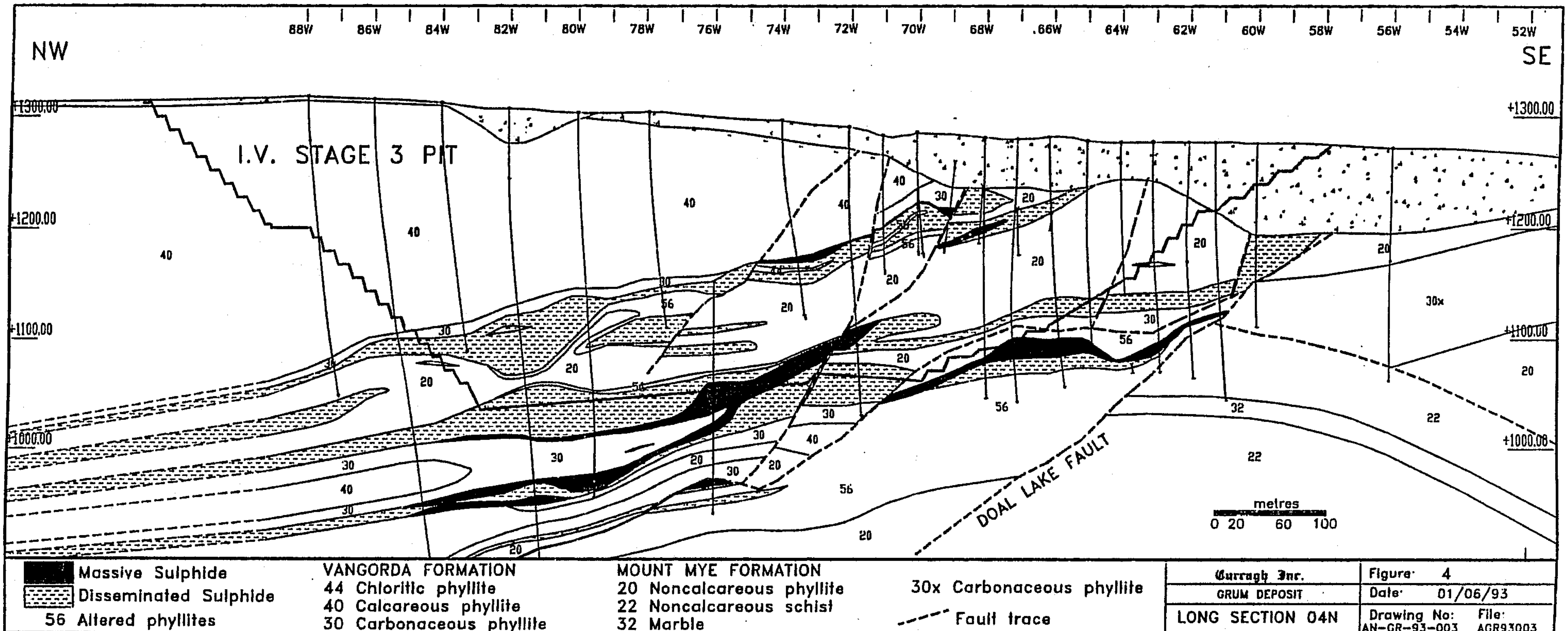
NW
SE
88W 86W 84W 82W 80W 78W 76W 74W 72W 70W 68W 66W 64W 62W 60W 58W 56W 54W 52W
+1300.00
+1200.00
+1100.00
+1000.00
I.V. STAGE 3 PIT
DOAL LAKE FAULT
metres
0 20 60 100
Massive Sulphide
Disseminated Sulphide
56 Altered phyllites
VANGORDA FORMATION
44 Chloritic phyllite
40 Calcareous phyllite
30 Carbonaceous phyllite
MOUNT MYE FORMATION
20 Noncalcareous phyllite
22 Noncalcareous schist
32 Marble
30x Carbonaceous phyllite
Fault trace
Curragh Inc.
GRUM DEPOSIT
LONG SECTION 04N
Figure 4
Date 01/06/93
Drawing No: AN-GR-93-003
File: AGR93003

pyritic/barytic sulphides at the top of the horizon. The upright panel is distinctly different from the other sulphide horizons of Grum in that it has extensive, well developed alteration, (a "bleached" muscovite-chlorite phyllite) below the semi-massive base. The intensity of alteration is similar to the alteration developed beneath the major ore horizon at Vangorda, and is unusually intense for Grum. The upright panel horizon is assumed to correlate structurally and stratigraphically with the Vangorda deposit. A sulphide horizon at the base of the basal carbonaceous member of the Vangorda formation, like the main horizon at Grum, appears to be present at Vangorda but is of insignificant grade and thickness.

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

There are no significant post metamorphic dykes at Grum. The Cretaceous Anvil Batholith crops out 1.5 km northeast of the deposit, but is separated from it by major faults. The Batholith is unrelated to the deposit and does not appear to have significantly affected it, except for the regional scale metamorphism, recrystallization and deformation that accompanied batholith emplacement. Conversely, rocks of the Anvil Batholith do not appear to have been affected by ore formation, as expected, since they are significantly younger than the ores.

Structure

The ore layers at Grum are contorted into a complex, shallowly northwest plunging, polyphase fold structure. The prominent 'S' shaped folds (looking northwest - Figure 3), are second phase structures. They are superimposed on a large 'Z' shaped first phase anticline-syncline pair (Figure 3 -the area labelled "Gnome's Cap" is the crest of the first phase anticline). The dominant plane of fissility (S_2) in the phyllites at Grum is axial planar to the second phase folds and dips shallowly (l0° to 30°) generally to the west or southwest (Figure 3). This fissility is a major factor in assessing slope stability for a Grum pit. The overall deposit elongation parallels the axial direction of the second phase folds (Figure 4 is a section parallel to the 315° trend/11° plunge).

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

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

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

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

Surficial Geology of the Grum Area

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

DEPOSIT GEOLOGY AND ORE TYPES

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

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

Ore Types

Disseminated Carbonaceous Pyritic Quartzite (Unit 2)

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

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

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

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

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

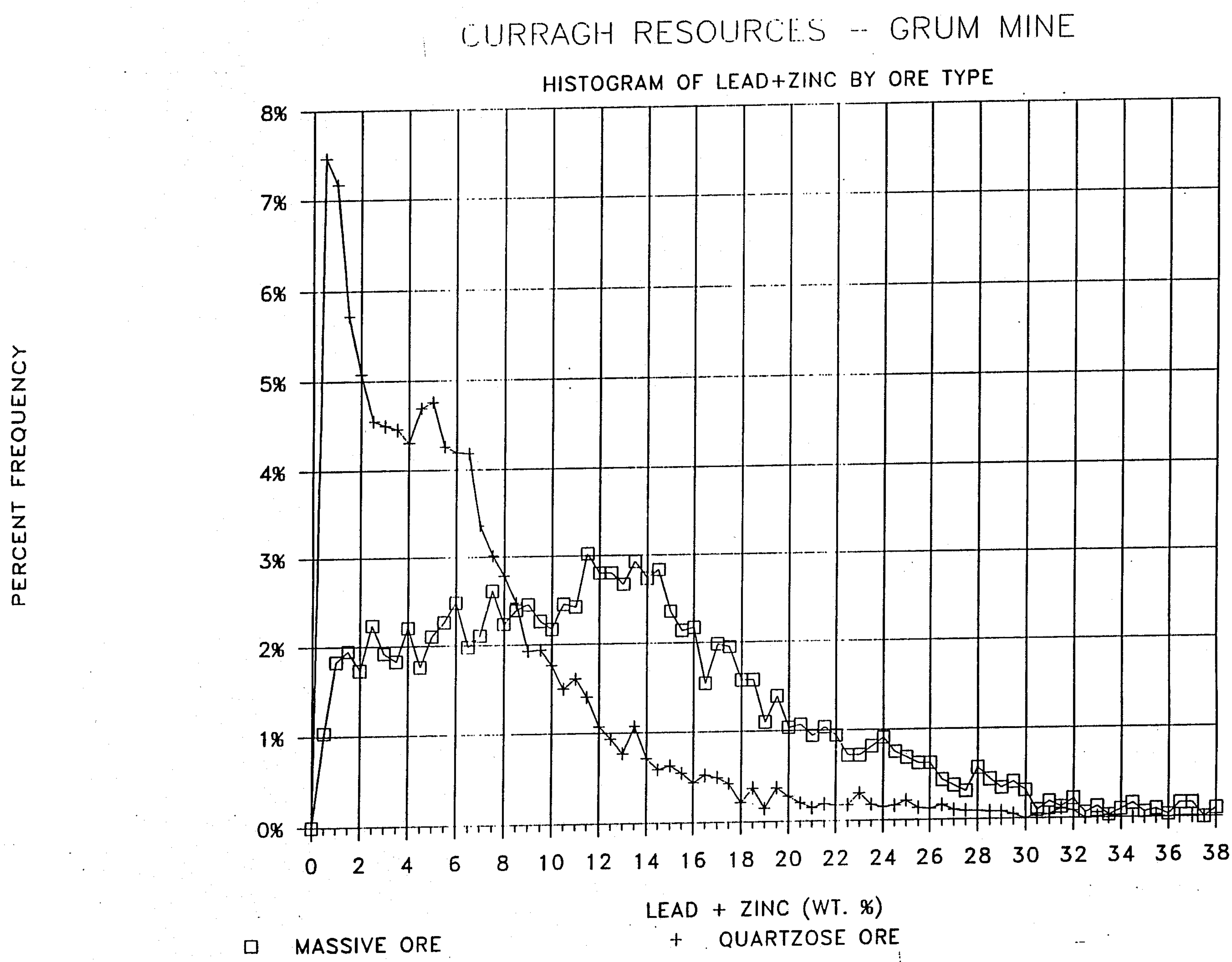

Figure 5 Histogram of Lead and Zinc Grades for Massive Sulphides and Disseminated Sulphide in Quartzite at Grum, based on Raw Core Assays

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

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

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

Massive Sulphide (Unit 5 and 7)

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

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

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

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

Comparison of Grum to Faro and Vangorda

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

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

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

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

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

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

MINERAL INVENTORY

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

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

Co-ordinate System

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

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

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

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

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

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

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

Table 3. Co-ordinates for Survey Station 1404 (VG4)

UTM Grid Co-ordinates	Northing	6,904,623.172
	Easting	593,847.979
	elevation	1,300.062
Grum Local and Co-ordinates	Northing	5,000.000
	Easting	3,500.000
	elevation	1,300.062

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

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

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

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

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

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

Grum Model Property Definition

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

Table 4. Grum G9110 Model Corner Co-ordinates

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

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

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

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

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

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

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

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

Surface Grids

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

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

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

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

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

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

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

Rock Type Model

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

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

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

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

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

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

Table 7. Sulphide Horizon Codes

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

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

Assay Composites

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

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

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

Table 8. 95th Percentile Cutoff Values

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

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

Bench Composites

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

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

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

Geological Composites

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

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

Grade Interpolation

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

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

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

Table 9. Interpolation parameters for G9110 Model

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

Grade interpolation for the bench composite model.

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

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

Table 10. Average SG of Interpolated Blocks

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

Grade Interpolation for the Geological Composite Model

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

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

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

Mineral Inventory Results

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

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

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

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

Table 12

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

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

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

Table 11
GRUM DEPOSIT – IN-SITU MINERAL INVENTORY
in-place mineralization / no dilution / no mining recovery consideration

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

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

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

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

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

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

Classification of Mineral Inventory

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

Table 13
HISTORICAL SUMMARY OF GRUM MINERAL INVENTORY ABOVE A 4% Pb+Zn CUTOFF GRADE
Total resource for the 61W to 87W portion of the deposit – no adjustments for dilution or mining recovery

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

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

Table 14

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

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

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

NOTES:

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

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

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

MINABLE RESERVE

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

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

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

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

GRADE MODEL FOR LABEL 3 [%Zn]

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

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

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

	55	56	57	58	59	60	61	62	63	64	65	66	67	68	69	70	71	72	73	74	75	76	77	78	79
55	.00	.00	.00	3.76	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00
56	.00	.00	.00	3.75	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	3.51	3.35	3.34	2.81	2.97	3.03	.00
57	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	2.86	2.86	2.92	2.88	3.05	2.62	2.77	2.67
58	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	1.81	2.21	2.18	2.54	2.58	1.94	1.23	1.22	3.54
59	.00	.00	4.38	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	2.49	2.19	1.99	1.63	1.20	1.20	1.20	1.20	1.20	1.20	.00
60	.00	.00	3.69	3.45	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	2.51	2.32	2.05	1.72	1.15	1.13	1.18	1.23	1.15	1.17	.00
61	.00	3.75	3.59	3.48	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	2.34	2.06	1.88	1.59	1.20	.98	1.09	1.18	1.24	1.27	.00
62	3.84	3.75	3.67	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	2.30	2.12	1.97	1.81	1.47	1.19	1.13	1.20	1.25	1.28	.00
63	3.76	3.79	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	3.47	2.97	2.75	2.54	2.08	1.87	1.59	1.57	1.62	1.49	1.28	.00	.00
64	4.30	.00	.00	.00	.00	.00	.00	.00	.00	.00	3.98	3.75	3.57	3.27	2.75	2.56	2.28	2.16	1.68	1.65	1.70	.00	.00	4.20	4.40
65	.00	.00	.00	.00	.00	.00	.00	.00	.00	4.17	4.10	3.67	3.45	3.23	2.74	2.38	2.31	2.11	1.84	.00	.00	3.66	3.80	3.89	3.89
66	.00	.00	.00	.00	.00	.00	.00	4.98	4.82	4.47	4.06	3.74	3.30	3.20	2.87	2.43	2.38	.00	.00	.00	3.45	3.36	3.28	3.20	3.39
67	.00	.00	.00	.00	.00	.00	.00	5.09	5.03	4.66	4.05	3.64	3.33	3.09	2.79	2.36	.00	.00	.00	.00	2.98	3.19	2.90	2.30	.00
68	.00	.00	.00	.00	.00	.00	.00	4.98	4.73	4.37	3.95	3.35	2.67	2.71	.00	2.18	.00	.00	.00	3.04	3.00	2.86	2.68	2.56	.00
69	.00	.00	.00	.00	.00	.00	.00	8.94	4.78	4.31	3.85	3.28	2.75	.00	.00	2.53	2.50	.00	.00	2.60	2.82	2.74	2.74	2.71	.00
70	.00	.00	.00	.00	.00	.00	.00	9.00	9.23	4.31	3.84	3.28	.00	.00	.00	2.44	2.41	.00	.00	2.62	2.70	2.75	2.77	2.74	.00
71	.00	.00	.00	.00	.00	.00	.00	.00	9.63	4.70	.00	.00	.00	.00	.00	.00	2.49	.00	.00	2.66	2.65	2.69	2.67	2.73	.00
72	.00	.00	.00	.00	.00	.00	.00	9.55	9.81	5.46	.00	.00	.00	.00	.00	2.46	2.69	.00	.00	2.77	2.76	2.75	2.80	2.80	.00
73	.00	.00	.00	.00	.00	9.52	9.76	9.89	6.92	.00	.00	.00	.00	.00	2.46	2.54	.00	.00	.00	2.69	2.70	2.77	2.79	2.91	.00
74	.00	.00	.00	.00	10.33	10.24	6.93	6.80	6.70	.00	.00	.00	.00	2.29	2.04	.00	.00	.00	.00	2.72	2.80	2.85	2.87	2.91	.00
75	.00	.00	.00	.00	10.34	7.34	7.69	7.15	6.57	.00	.00	.00	.00	2.27	.00	.00	.00	.00	.00	.00	2.88	2.96	3.02	3.26	.00
76	.00	.00	.00	.00	11.05	7.00	7.12	6.72	6.42	.00	.00	.00	.00	2.25	.00	.00	.00	.00	.00	.00	3.05	3.18	3.23	.00	.00
77	.00	.00	.00	.00	.00	11.82	6.99	6.59	6.15	.00	.00	.00	.00	1.90	.00	.00	.00	.00	.00	3.17	3.30	3.35	3.40	.00	.00
78	.00	.00	12.94	12.24	6.49	.00	6.82	6.48	6.09	.00	.00	.00	.00	1.96	.00	.00	.00	.00	.00	3.07	3.44	3.46	3.56	.00	.00
79	.00	.00	12.71	12.27	12.60	5.32	5.54	5.39	5.07	.00	.00	.00	2.05	2.00	.00	.00	.00	.00	.00	3.07	3.35	3.49	3.73	.00	.00
80	.00	.00	.00	11.28	11.48	4.78	4.93	.00	4.16	.00	.00	.00	2.18	.00	.00	.00	.00	.00	2.83	2.91	3.24	3.70	.00	.00	.00
81	.00	.00	.00	10.72	11.23	11.25	5.41	.00	.00	.00	.00	2.27	2.26	.00	.00	.00	.00	.00	3.10	3.33	3.71	.00	.00	.00	.00
82	.00	.00	.00	.00	11.10	10.96	6.57	.00	.00	.00	.00	2.64	.00	.00	.00	.00	.00	3.32	3.22	3.57	3.86	.00	.00	5.39	.00
83	.00	.00	.00	.00	9.52	10.51	5.18	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	3.24	3.46	3.89	4.38	.00	5.93	5.73	.00
84	3.29	.00	.00	.00	9.15	9.37	4.68	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	3.51	3.63	4.09	4.46	.00	6.19	.00	.00
85	.00	.00	.00	.00	8.57	4.40	4.29	4.15	.00	.00	.00	.00	.00	.00	.00	.00	.00	3.75	3.84	4.30	4.55	.00	5.96	.00	.00
86	.00	.00	.00	8.18	8.28	4.00	3.75	3.54	.00	.00	.00	.00	.00	.00	.00	4.30	.00	4.00	4.28	4.70	4.77	.00	5.82	.00	.00
87	.00	.00	.00	8.22	8.45	3.13	2.89	2.72	.00	.00	.00	.00	.00	.00	.00	4.77	.00	4.87	4.89	5.16	4.94	.00	5.99	.00	.00
88	.00	.00	.00	8.30	2.13	1.92	1.98	2.03	.00	.00	.00	.00	.00	.00	.00	.00	.00	5.44	5.36	5.47	5.45	.00	.00	.00	.00
89	.00	.00	.00	8.20	1.80	1.80	1.80	1.80	.00	.00	.00	.00	.00	.00	.00	.00	4.88	4.91	5.15	5.60	5.65	5.64	.00	.00	.00
90	.00	.00	6.26	1.64	1.60	1.58	1.56	1.50	.00	.00	.00	.00	.00	.00	.00	3.86	4.26	4.42	4.98	5.30	5.63	.00	.00	.00	.00
91	.00	.00	.00	1.57	1.56	1.51	1.50	1.53	.00	.00	.00	.00	.00	.00	.00	3.37	3.39	3.20	4.27	5.15	.00	.00	.00	.00	.00
92	.00	.00	.00	5.09	1.47	1.52	1.50	1.52	.00	.00	.00	.00	.00	.00	2.84	3.06	3.52	3.71	4.08	4.35	.00	.00	.00	.00	.00
93	.00	.00	.00	5.33	1.56	1.53	1.55	1.55	.00	.00	.00	.00	.00	2.51	2.81	3.21	3.69	4.24	5.40	.00	.00	.00	.00	.00	.00
94	.00	.00	.00	6.91	1.83	1.79	1.74	1.76	.00	.00	.00	.00	.00	2.51	3.18	.00	4.89	5.69	5.99	6.21	.00	.00	.00	.00	.00
95	.00	.00	8.45	2.02	1.85	1.70	1.68	1.75	.00	.00	.00	.00	2.69	2.93	.00	4.55	5.33	6.18	6.50	6.83	.00	6.88	.00	.00	.00
96	.00	.00	10.09	2.05	2.03	1.89	1.74	.00	.00	.00	.00	.00	.00	.00	3.86	4.48	.00	6.13	6.32	.00	6.89	.00	.00	.00	.00
97	.00	.00	10.67	1.99	2.09	2.07	1.94	.00	.00	.00	.00	.00	.00	.00	4.21	.00	5.57	6.11	6.44	6.27	6.31	.00	.00	.00	.00

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

RADE MODEL FOR LABEL 3 [%Zn dl]

ESCRIPTION :

CREATED ON : 20/ 4/1992

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

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

	55	56	57	58	59	60	61	62	63	64	65	66	67	68	69	70	71	72	73	74	75	76	77	78	79
55	.00	.00	.00	3.01	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00
56	.00	.00	.00	3.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	2.81	2.68	2.67	2.25	2.38	2.42	.00
57	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	2.29	2.86	2.92	2.88	3.05	2.62	2.77	2.14
58	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	1.45	2.21	2.18	2.54	2.58	1.94	1.23	1.22	2.83
59	.00	.00	3.51	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	1.99	1.75	1.99	1.63	1.20	1.20	1.20	1.20	1.20	.96	.00
60	.00	.00	2.95	2.76	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	2.01	2.32	2.05	1.72	1.15	1.13	1.18	1.23	1.15	.93	.00
61	.00	3.00	2.87	2.79	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	1.87	2.06	1.88	1.59	1.20	.98	1.09	1.18	1.24	1.02	.00
62	3.07	3.00	2.94	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	1.84	2.12	1.97	1.81	1.47	1.19	1.13	1.20	1.25	1.02	.00
63	3.01	3.03	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	2.78	2.38	2.75	2.54	2.08	1.87	1.59	1.57	1.62	1.19	1.03	.00	.00
64	3.44	.00	.00	.00	.00	.00	.00	.00	.00	.00	3.18	3.00	3.57	3.27	2.75	2.56	2.28	2.16	1.68	1.32	1.36	.00	.00	3.36	3.52
65	.00	.00	.00	.00	.00	.00	.00	.00	.00	3.34	4.10	3.67	3.65	3.23	2.74	2.38	2.31	1.69	1.47	.00	.00	2.93	3.04	3.89	3.12
66	.00	.00	.00	.00	.00	.00	.00	3.98	3.85	4.47	4.06	3.74	3.30	3.20	2.87	1.95	1.90	.00	.00	.00	2.76	3.34	3.28	3.20	2.71
67	.00	.00	.00	.00	.00	.00	.00	4.07	5.03	4.66	4.05	3.64	3.33	3.09	2.79	1.89	.00	.00	.00	.00	2.38	3.19	2.90	2.00	.00
68	.00	.00	.00	.00	.00	.00	.00	3.98	4.73	4.37	3.95	3.35	2.67	2.17	.00	1.75	.00	.00	.00	2.43	3.00	2.86	2.68	2.05	.00
69	.00	.00	.00	.00	.00	.00	.00	7.15	4.78	4.31	3.85	3.28	2.20	.00	.00	2.02	2.00	.00	.00	2.08	2.82	2.74	2.74	2.16	.00
70	.00	.00	.00	.00	.00	.00	.00	7.20	9.23	4.31	3.07	2.62	.00	.00	.00	1.96	1.93	.00	.00	2.10	2.70	2.75	2.77	2.19	.00
71	.00	.00	.00	.00	.00	.00	.00	.00	7.70	3.76	.00	.00	.00	.00	.00	.00	1.99	.00	.00	2.13	2.65	2.69	2.67	2.18	.00
72	.00	.00	.00	.00	.00	.00	.00	7.64	9.81	4.37	.00	.00	.00	.00	.00	1.97	2.15	.00	.00	2.21	2.76	2.75	2.80	2.24	.00
73	.00	.00	.00	.00	.00	7.62	7.81	9.89	5.53	.00	.00	.00	.00	.00	1.96	2.03	.00	.00	.00	2.15	2.70	2.77	2.79	2.33	.00
74	.00	.00	.00	.00	8.26	10.24	6.93	6.80	5.36	.00	.00	.00	.00	1.83	1.63	.00	.00	.00	.00	2.18	2.80	2.85	2.87	2.33	.00
75	.00	.00	.00	.00	8.27	7.34	7.69	7.15	5.26	.00	.00	.00	.00	1.82	.00	.00	.00	.00	.00	.00	2.30	2.96	3.02	2.61	.00
76	.00	.00	.00	.00	8.84	7.00	7.12	6.72	5.13	.00	.00	.00	.00	1.80	.00	.00	.00	.00	.00	.00	2.44	3.18	2.59	.00	.00
77	.00	.00	.00	.00	.00	9.46	6.99	6.59	4.92	.00	.00	.00	.00	1.52	.00	.00	.00	.00	.00	2.53	3.30	3.35	2.72	.00	.00
78	.00	.00	10.36	9.79	5.19	.00	6.82	6.48	4.87	.00	.00	.00	.00	1.57	.00	.00	.00	.00	.00	2.45	3.44	3.46	2.85	.00	.00
79	.00	.00	10.17	12.27	12.60	5.32	5.54	4.31	4.05	.00	.00	.00	1.64	1.60	.00	.00	.00	.00	.00	2.46	3.35	3.49	2.99	.00	.00
80	.00	.00	.00	9.02	11.48	4.78	3.94	.00	3.31	.00	.00	.00	1.74	.00	.00	.00	.00	.00	2.27	2.91	3.24	2.96	.00	.00	.00
81	.00	.00	.00	8.58	11.23	11.25	4.33	.00	.00	.00	.00	1.81	1.80	.00	.00	.00	.00	.00	2.48	3.33	2.97	.00	.00	.00	.00
82	.00	.00	.00	.00	8.88	10.96	5.25	.00	.00	.00	.00	2.11	.00	.00	.00	.00	.00	2.65	3.22	3.57	3.09	.00	.00	4.32	.00
83	.00	.00	.00	.00	7.61	10.51	4.14	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	2.60	3.46	3.89	3.51	.00	4.74	4.58	.00
84	2.63	.00	.00	.00	7.32	9.37	3.74	.00	.00	.00	.00	.00	.00	.00	.00	.00	.00	2.81	3.63	4.09	3.57	.00	4.95	.00	.00
85	.00	.00	.00	.00	6.86	4.40	4.29	3.32	.00	.00	.00	.00	.00	.00	.00	.00	.00	3.00	3.84	4.30	3.64	.00	4.77	.00	.00
86	.00	.00	.00	6.54	8.28	4.00	3.75	2.83	.00	.00	.00	.00	.00	.00	.00	3.44	.00	3.20	4.28	4.70	3.82	.00	4.66	.00	.00
87	.00	.00	.00	6.57	8.45	3.13	2.89	2.17	.00	.00	.00	.00	.00	.00	.00	3.81	.00	3.90	4.89	5.16	3.95	.00	4.79	.00	.00
88	.00	.00	.00	6.64	2.13	1.92	1.98	1.63	.00	.00	.00	.00	.00	.00	.00	.00	.00	4.36	5.36	5.47	4.36	.00	.00	.00	.00
89	.00	.00	.00	6.56	1.80	1.80	1.80	1.44	.00	.00	.00	.00	.00	.00	.00	.00	3.90	4.91	5.15	5.60	4.52	4.51	.00	.00	.00
90	.00	.00	5.00	1.31	1.60	1.58	1.54	1.20	.00	.00	.00	.00	.00	.00	.00	3.08	4.26	4.42	4.98	5.30	4.50	.00	.00	.00	.00
91	.00	.00	.00	1.25	1.36	1.51	1.50	1.22	.00	.00	.00	.00	.00	.00	.00	2.70	3.39	3.20	4.27	4.12	.00	.00	.00	.00	.00
92	.00	.00	.00	4.07	1.47	1.52	1.50	1.22	.00	.00	.00	.00	.00	.00	2.27	3.06	3.52	3.71	4.08	3.64	.00	.00	.00	.00	.00
93	.00	.00	.00	4.27	1.56	1.55	1.55	1.24	.00	.00	.00	.00	.00	2.01	2.25	2.57	3.69	4.24	4.32	.00	.00	.00	.00	.00	.00
94	.00	.00	.00	5.53	1.83	1.79	1.74	1.41	.00	.00	.00	.00	.00	2.01	2.54	.00	3.91	5.69	5.99	4.97	.00	.00	.00	.00	.00
95	.00	.00	6.76	2.02	1.85	1.70	1.68	1.40	.00	.00	.00	.00	2.16	2.35	.00	3.64	4.26	6.18	6.50	3.47	.00	5.51	.00	.00	.00
96	.00	.00	8.07	2.05	2.03	1.89	1.39	.00	.00	.00	.00	.00	.00	.00	3.09	3.58	.00	4.91	6.52	.00	5.31	.00	.00	.00	.00
97	.00	.00	8.54	1.99	2.09	2.07	1.55	.00	.00	.00	.00	.00	.00	.00	3.37	.00	4.46	6.11	6.44	6.27	5.05	.00	.00	.00	.00

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

Table 15

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

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

	MINERAL INVENTORY									CONTAINED METAL			
	Cutoff (wt%) Pb+Zn	Volume (cu. m.) *1000	Density (tn/m3)	Tonnage (tonnes) *1000	Pb+Zn (wt %)	Pb (wt %)	Zn (wt %)	Ag (g/t)	Au (g/t)	lead tonnes *1000	zinc tonnes *1000	silver kg	gold kg
GEOLOGICAL COMPOSITES	8	2,791.59	3.799	10,604.09	11.77	4.49	7.28	74.2	1.06	475.593	771.978	786,930	11,187
G9110 MODEL–GEOL (2m)	6	4,652.67	3.541	16,476.35	10.04	3.80	6.24	63.1	0.94	625.772	1,028.783	1,039,822	15,537
Total Model (52W–88W)	5	5,947.05	3.430	20,401.23	9.17	3.45	5.72	57.6	0.88	703.842	1,166.134	1,175,927	17,973
all horizons	4	6,939.07	3.376	23,423.28	8.57	3.22	5.35	54.0	0.85	754.698	1,252.209	1,265,560	19,840
	3	7,595.00	3.356	25,488.40	8.16	3.07	5.09	51.7	0.83	783.513	1,296.595	1,317,241	21,206
	0	8,494.97	3.338	28,358.74	7.52	2.83	4.69	48.0	0.81	802.552	1,330.592	1,362,354	22,999

Table 16
Minable Reserve Grum Deposit I.V. Pit
From diluted G9110 geological composite model

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

Table 16
Minable Reserve Grum Deposit I.V. Pit
As of the Start of 1992
From diluted G9110 geological composite model

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

See Table 1 for quantities of waste mined in 1992 and earlier, no mining in 1993 at Grum.

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

ADDITIONAL POTENTIAL

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

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

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

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

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

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APPENDIX A

G9110 MODEL

PROPERTY DEFINITION,

ROCK CODE LISTING

AND

SURFACE GRID LISTING

Curragh Resources Inc. : Whitehorse Office : Property Definition Listing:
93/05/31 : 10:58:54 Page
GEMCOM Services MOD11: DB=E:\PMDBG4 :
: = 1

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

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

Current Units are :

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

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

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

Curragh Resources Inc. : Whitehorse Office : Rock Type Data Listing: 93/05/31
GEMCOM Services MOD14: DB=Rock Type Data Listing :

Listing of Rock Type Information for Selected Rock Types

Summary Listing

Record	Record Description	Code	Density	Pen	Status	Material
1	NONCALCAREOUS PHYLLITE	20	2.7000	6	Live	Waste
2	NONCALCAREOUS BIOTITE-MUSCOVITE-ANDALUSITE SCHIST	22	2.7000	6	Live	Waste
3	CARBONACEOUS PHYLLITE	30	2.7000	0	Live	Waste
4	MARBLE	32	2.7000	9	Live	Waste
5	CALCAREOUS PHYLLITE	40	2.7000	11	Live	Waste
6	GREENSTONE	48	2.7000	2	Live	Waste
7	ALTERED PHYLLITE	56	2.7000	3	Live	Waste
8	WASTE - UNDIFFERENTIATED PHYLLITE	77	2.7000	8	Live	Waste
9	OVERBURDEN - UNDIFFERENTIATED	82	2.2000	10	Live	Waste
10	AIR	99	0.0000	9	Live	Air
11	CARBONACEOUS QUARTZITE	2	2.7000	7	Live	Ore
12	NONCARBONACEOUS QUARTZITE	3	2.7000	14	Live	Ore
13	PYRITIC MASSIVE SULPHIDES	5	2.7000	12	Live	Ore
14	BARITIC MASSIVE SULPHIDES	7	2.7000	13	Live	Ore
15	QUARTZITES - DISSEMINATED SULPHIDES	12	2.7000	4	Live	Ore
16	MIXED QUARTZITES AND MASSIVE SULPHIDES	15	2.7000	4	Live	Ore
17	MASSIVE PYRITIC AND BARITIC SULPHIDES	18	2.7000	12	Live	Ore
67	SEMI-MASSIVE SILICEOUS PYRITIC SULPLHIDES	4	2.7000	12	Live	Ore
68	PYRRHOTITIC SULPHIDE	8	2.7000	12	Live	Ore

SURFACE ELEVATION GRID MODEL

SUMMARY PRINTOUT

FROM RECORD [1] TO RECORD [18]

RECORD	STATUS	TYPE	DESCRIPTION	DATE
1	1	USER	1 TOPOGRAPHY-1979 CAMC ORTHOPHOTO (Geomodel-R&C)	29/10/ 91
2	1	USER	2 TOPOGRAPHY-1990 CRI ORTHOPHOTO (Geomodel-RAC)	29/10/ 91
3	1	USER	3 TOPOGRAPHY-edited 1979 topo-PREFERRED STARTING SURFACE	29/10/ 91
4	1	USER	4 OVERBURDEN/BEDROCK SURFACE-includes 1991 drilling results	29/10/ 91
5	1	USER	5 ION VINTILA STAGE 1 PIT-background=2000	29/10/ 91
6	1	USER	6 ION VINTILA STAGE 2 PIT-background=2000	29/10/ 91
7	1	USER	7 ION VINTILA STAGE 3 PIT-background=2000	29/10/ 91
8	1	USER	8 Mine Survey Pit Status October 1991-bkgnd=2000	29/10/ 91
9	1	USER	9 I VINTILA STAGE 1 PIT (grd 3 merge grd 5)	29/10/ 91
10	1	USER	10 I VINTILA STAGE 2 PIT (Grd 3 merge Grd 6)	29/10/ 91
11	1	USER	11 I VINTILA STAGE 3 PIT (Grd 3 merge Grd 7)	29/10/ 91
12	1	USER	12 CUT SURFACE-OCT 1990 (Grd 2 merge Grd 3)	29/10/ 91
13	1	USER	13 CUT SURFACE - OCT 1991 (Grd 3 merge Grd 8)	29/10/ 91
14	1	USER	14 QUICK SINK PIT 1-(bkgnd=2000m)	5/11/ 91
15	1	USER	15 QUICK SINK PIT 1 (Grd 3 merge Grd 14)	91/11/ 1
16	1	USER	16 QUICK SINK PIT 2 (bkgnd=2000m)	1/11/1991
17	1	USER	17 QUICK SINK PIT 2 (Grd 3 merge Grd 16)	91/11/ 1
18	1		Overburden surface lowered by 6 metres (weathered surface)	5/11/1991

APPENDIX B

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

CURRAGH RESOURCES INC. – NUMERIC ANVIL LITHOSTRATIGRAPHIC CODE

ROCK CODES (OLD CODES INCLUDED FOR COMPARISON)

DISSEMINATED QUARTZITES		
2	4A	Ribbon banded carbonaceous quartzite
3	4C/4D	Pyritic quartzite (<30% pyrite)
SEMI MASSIVE SULPHIDE (Generally low grade)		
4	4EC/4E1/4C3	Siliceous pyritic sulphides (30–60% pyrite, generally <4% Pb+Zn)
MASSIVE SULPHIDES		
5	4E/4F	Massive pyritic sulphides (60–100% pyrite)
6	4K	Massive pyritic sulphide with clasts of dolomite/ankerite
7	4G	Baritic pyrite sulphides (>10% barite)
8	4H	Pyrrhotitic sulphides
9	4J	Nonpyritic sulphides & oxides – pyrite poor
METASEDIMENTS		
20	3G	Noncalcareous, muscovite–chlorite, medium grey phyllite
22	1C/1CD/1D	Noncalcareous, bio–musc–qtz staurolite+andalusite+garnet+fibrolite schist
30	5A/5G/3E/1E	Carbonaceous phyllite/schist
32	5E/3F/1G	Marble + calc–silicate bands
33	1B	Skarn and "silicated" marble
36	3D	Calc–silicate
40	5B	Calcareous, silvery grey, muscovite chlorite phyllite
METAIGNEOUS		
44	5C/3C/1F	Metabasite, poorly foliated greenstone (relict igneous texture)
45	5C/3C/1F	Pyroxenite – commonly serpentinized (relict bastites)
46	5C/3C/1F	Amphibolite – blue–green hornblende + plagioclase + quartz
47	5D/3B/1H	Chloritic phyllite/schist – pale green, homogenous
ALTERED ROCKS		
52	4L0	Muscovite>chlorite quartz phyllite/schist – light cream to white
54	4L6	Chlorite>muscovite quartz phyllite/schist – pale green
CRETACEOUS INTRUSIVES		
60	10Q	Quartz vein – white bull quartz
61	10AB	Anvil Batholith – Mt Mye phase of Anvil plutonic suite. Musc–bio granite
65	10C	Pegmatite
66	–	Aplite
68	10E	Hornblende–biotite quartz diorite – massive and unfoliated
69	10F	Smokey quartz–feldspar porphyry – massive and unfoliated
FAULT ROCK (use only if parent not recognized)		
72		Gouge
74		Tectonic breccia
76		Mylonite
OVERBURDEN		
82		Unclassified – general
84		Triconed – no recovery
86		Till – silt – sand
88		Ferricrete
99		Air
OTHER		
0		No Recovery
&		+/–

MINERAL IDENTIFIERS

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

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

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

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

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

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

Other
- g carbon
- x noncalcareous

ROCK TEXTURES

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

USAGE: ROCK CODE MINERAL IDENTIFIERS TEXTURE GRADE

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

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

GRADE MODIFIERS

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

3

GEOLOGY, MINERAL INVENTORY

and RESERVES of the

GRUM DEPOSIT - Yukon

CURRAGH INC.
Whitehorse, Yukon

Report WH9305

May 1993

EXECUTIVE SUMMARY

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

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

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

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

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

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

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

4

SÄ DENA HES

MINERAL INVENTORY

AND

MINABLE RESERVES

01 JANUARY 1993

Curragh Inc.
Sä Dena Hes Joint Venture
Geological and Engineering Departments

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EXECUTIVE SUMMARY

A revised geological interpretation and mineral inventory (= undiluted in-situ mineralization) dated January 1, 1993, has been completed for the Jewelbox Hill, Burnick and Gribbler mineral deposits on the Sä Dena Hes property in the Yukon (Table 1).

The results of this inventory in Jewelbox are:

	Tonnes	Pb(%)	Zn(%)	g/t Ag(%)
Proven and Probable	652,666	7.2	11.8	60
Possible	269,433	7.3	11.0	61
Total	**922,099**	**7.3**	**11.6**	**60**

After deducting the mined tonnage (487,292 tonnes) at an estimated dilution of 18% (412,959 tonnes) from the total inventory of January 1, 1992 (1,322,468 tonnes) there is a net gain in both categories of 12,590 tonnes. This is due to discovery of new zones,as a result of more extensive drill testing of the periphery of the known zones. The new discoveries have more than compensated for tonnage losses in previously known zones.

The results of this inventory in Burnick are:

	Tonnes	Pb(%)	Zn(%)	g/t Ag(%)
Proven and Probable	1,010,964	1.1	13.3	45
Possible	594,115	0.5	12.8	49
Total	**1,603,079**	**0.9**	**13.1**	**47**

After deducting the mined tonnage (11,068 tonnes) at an estimated dilution of 18% (9,380 tonnes) from the total reserves of Roscoe Postle Associates (2,494,000 tonnes), there is a loss in both categories of 881,541 tonnes. This is due to a revised geologic interpretation caused by fill-in drilling carried out in 1992.

The results of the inventory on Gribbler are:

	Tonnes	Pb(%)	Zn(%)	g/t Ag(%)
Proven and Probable	106,318	8.1	17.5	106
Possible	165,795	8.5	15.4	101
Total	**272,113**	**8.4**	**16.2**	**103**

Comparison with the Roscoe Postle Associates tonnages (347,00 tonnes) in both categories shows a loss of 74,887 tonnes. This loss is due to the G1 (upper) zone having proved to be oxidized. The inventory of G2 and G3 actually increased, both in tonnage and in confidence. More importantly, the average thickness of this high grade ore increased.

There was no change in the Attila mineral inventory as no new data was available and the geological interpretation is reasonably consistent with the revised Burnick interpretation. The Roscoe-Postle inventory was 405,700 tonnes averaging 3.4% Pb, 12.4% Zn and 62 g/t Ag all of which was classified as probable.

A total of 2,393,900 tonnes at a diluted grade of 3.5% Pb, 12.0% Zn and 52 g/t Ag has been delineated for mining in the four ore zones (Table 2). Only the proven and probable (type A) component of this tonnage qualifies as a reserve according to the corporate reserve policy. The start of 1993 proven plus probable minable reserve is thus 1,756,000 tonnes averaging 3.4% Pb, 12.1% Pb and 52 g/t Ag over the four deposits. After allowing for mining in 1992 this represents a decrease of nearly one million tonnes of minable reserve due mainly to the decreased inventory at Burnick, partially compensated by the introduction of the Gribbler Ridge zone into the property reserve base. Grade has increased due to the addition of Gribbler and due to the fact that the revised Burnick reserve is slightly higher grade than the previous reserve.

The following two tables summarize the results of this study by zone and by confidence category.

It is essential that underground drilling continue at Jewelbox and Burnick and that the exploration drift be driven south from Burnick to Atilla in order to drill test the gap between those zones. Given this further exploration work there is good potential to regain some of the lost tonnage as has been the case at Jewelbox.

TABLE 1

SA DENA HES JOINT VENTURE

JANUARY 1, 1993 MINERAL INVENTORY

ZONE NAME	CATETORY A (Proven and Probable)							CATEGORY B (Possible)							CATEGORY A & B (Proven + Probable + Possible)						
	ORE	ORE GRADE			METAL			ORE	ORE GRADE			METAL			ORE	ORE GRADE			METAL		
	TONNAGE (tonnes)	LEAD (%)	ZINC (%)	SILVER (g/t)	LEAD (tonnes)	ZINC (tonnes)	SILVER (kg)	TONNAGE (tonnes)	LEAD (%)	ZINC (%)	SILVER (g/t)	LEAD (tonnes)	ZINC (tonnes)	SILVER (kg)	TONNAGE (tonnes)	LEAD (%)	ZINC (%)	SILVER (g/t)	LEAD (tonnes)	ZINC (tonnes)	SILVER (kg)
MAIN O/P	5,120	17.1	22.7	82	876	1,162	419								5,120	17.1	22.7	82	876	1,162	419
MAIN U/G	8,707	9.9	13.6	56	862	1,184	484								8,707	9.9	13.6	56	862	1,184	484
JB1 – POD	70,430	9.5	13.8	52	6,691	9,719	3,683								70,430	9.5	13.8	52	6,691	9,719	3,683
JB1	135,143	6.8	11.0	37	9,190	14,866	4,933	22,433	8.5	11.8	43	1,907	2,647	974	157,576	7.0	11.1	37	11,097	17,513	5,906
JB1 – LOWER	27,997	9.1	14.3	43	2,548	4,004	1,190	7,765	10.7	14.9	55	831	1,157	426	35,762	9.4	14.4	45	3,379	5,161	1,615
JB – M2								18,343	8.8	13.8	44	1,614	2,531	805	18,343	8.8	13.8	44	1,614	2,531	805
JB2 – WEST	74,059	7.4	13.9	57	5,480	10,294	4,229	24,394	6.5	10.7	53	1,586	2,610	1,298	98,453	7.2	13.1	56	7,066	12,904	5,527
JB2 – EAST	115,330	6.8	9.4	86	7,842	10,841	9,941	12,421	9.7	8.6	149	1,205	1,068	1,820	127,751	7.1	9.3	92	9,047	11,909	11,761
CHIMNEY	19,773	8.6	15.6	52	1,700	3,085	1,022								19,773	8.6	15.6	52	1,700	3,085	1,022
J1	13,400	6.2	9.2	31	831	1,233	410	4,593	6.4	10.0	28	294	459	127	17,993	6.3	9.4	30	1,125	1,692	537
JB – FW	112,099	5.7	10.7	59	6,390	11,995	6,580	22,651	5.3	9.4	47	1,201	2,129	1,065	134,750	5.6	10.5	57	7,590	14,124	7,645
JB – HFW	24,021	4.8	10.4	42	1,153	2,498	1,011	29,544	9.7	14.4	50	2,866	4,254	1,471	53,565	7.5	12.6	48	4,019	6,753	2,483
UFW								12,320	7.5	13.9	43	924	1,712	525	12,320	7.5	13.9	43	924	1,712	525
ASHBIN	38,908	9.5	15.6	119	3,696	6,070	4,634	12,957	10.3	16.1	77	1,335	2,086	999	51,865	9.7	15.7	109	5,031	8,156	5,633
LOWER MAIN	7,679	7.2	9.9	85	553	760	650	57,806	6.5	9.1	70	3,757	5,280	4,023	65,485	6.6	9.2	71	4,310	6,021	4,674
UNDEFINED								44,206	5.1	8.4	67	2,255	3,713	2,949	44,206	5.1	8.4	67	2,255	3,713	2,949
OPEN PIT	5,120	17.1	22.7	82	876	1,162	419								5,120	17.1	22.7	82	876	1,162	419
UNDERGROUND	647,546	7.2	11.8	60	46,936	76,548	38,769	269,433	7.3	11.0	61	19,773	29,629	16,481	916,979	7.3	11.6	60	66,709	106,177	55,249
JEWELBOX – TTL	652,666	7.3	12.0	60	47,812	77,710	39,188	269,433	7.3	11.0	61	19,773	29,629	16,481	922,099	7.3	11.6	60	67,585	107,339	55,668
GRIBBLER	106,318	8.1	17.5	106	8,633	18,616	11,291	165,795	8.5	15.4	101	14,126	25,516	16,696	272,113	8.4	16.2	103	22,759	44,132	27,987
ATTILA	406,000	3.4	12.4	62	13,804	50,344	25,172								406,000	3.4	12.4	62	13,804	50,344	25,172
BURNICK	1,010,964	1.1	13.3	45	11,121	134,458	45,898	595,115	0.5	122.8	49	2,961	75,791	28,718	1,603,079	0.9	13.1	47	14,081	210,249	74,615
MINE TOTAL	2,175,948	3.7	12.9	56	81,369	281,129	121,549	1,027,343	3.6	12.7	60	36,859	130,935	61,894	3,203,291	3.7	12.9	57	118,229	412,064	183,442

TABLE 2

SA DENA HES JOINT VENTURE

JANUARY 1, 1993 MINABLE RESERVES

ZONE NAME	CATETORY A (Proven and Probable)								CATEGORY B (Possible)								CATEGORY A & B (Proven + Probable + Possible)						
	ORE	ORE GRADE			METAL			REC/DIL	ORE	ORE GRADE			METAL			REC/DIL	ORE	ORE GRADE			METAL		
	TONNAGE (tonnes)	LEAD (%)	ZINC (%)	SILVER (g/t)	LEAD (tonnes)	ZINC (tonnes)	SILVER (kg.)	(vol %)	TONNAGE (tonnes)	LEAD (%)	ZINC (%)	SILVER (g/t)	LEAD (tonnes)	ZINC (tonnes)	SILVER (kg)	(vol %)	TONNAGE (tonnes)	LEAD (%)	ZINC (%)	SILVER (g/t)	LEAD (tonnes)	ZINC (tonnes)	SILVER (kg)
MAIN O/P																							
MAIN U/G	7,880	8.8	12.0	49	693	948	387	0/0									7,880	8.8	12.0	49	693	948	387
JB1 – POD	37,303	7.7	11.1	43	2,872	4,141	1,611	97/14									37,303	7.7	11.1	43	2,872	4,141	1,611
JB1	109,737	6.1	10.1	33	6,694	11,083	3,621	79/19	20,055	6.6	9.1	35	1,324	1,825	692	70/31	129,792	6.2	9.9	33	8,018	12,908	4,313
JB1 – LOWER	25,081	8.2	12.8	38	2,057	3,210	958	81/17	6,617	9.1	12.5	46	602	827	306	72/20	31,698	8.4	12.7	40	2,659	4,037	1,264
JB – M2									13,297	7.5	11.9	37	997	1,582	489	81/22	13,297	7.5	11.9	37	997	1,582	489
JB2 – WEST	60,473	6.3	11.7	50	3,810	7,075	2,993	75/18	23,476	5.0	8.3	41	1,174	1,949	967	80/31	83,949	5.9	11.7	47	4,984	9,024	3,961
JB2 – EAST	67,155	5.5	7.6	71	3,694	5,104	4,755	66/22	9,203	6.8	5.8	88	626	534	809	84/46	76,358	5.7	7.4	73	4,319	5,638	5,564
CHIMNEY	15,717	7.4	12.9	43	1,163	2,027	674	66/22									15,717	7.4	12.9	43	1,163	2,027	674
J1	7,566	6.2	8.4	31	469	836	244	70/15									7,566	6.4	32.0	32	469	836	244
JB – FW	103,630	5.1	9.5	50	5,285	9,845	5,182	93/15	16,550	4.7	8.3	39	778	1,374	639	84/19	120,180	5.0	9.3	48	6,063	11,219	5,820
JB – HFW	23,506	3.9	8.3	35	917	2,021	811	83/26	22,663	7.8	11.4	40	1,769	2,586	896	61/30	46,168	5.8	10.0	37	2,686	4,607	1,707
UFW									11,581	6.4	11.8	36	741	1,367	419	80/20	11,581	6.4	11.8	36	741	1,367	419
ASHBIN	31,732	7.6	12.6	94	2,412	3,998	2,978	72/23	9,430	8.8	13.2	46	830	1,245	429	69/36	41,162	7.9	12.7	83	3,241	5,243	3,406
LOWER MAIN	7,956	3.5	8.5	73	493	676	582	89/19	27,472	5.8	7.6	55	1,593	2,088	1,522	43/30	35,428	5.9	7.8	59	2,087	2,764	2,104
UNDEFINED									20,519	3.4	6.9	36	698	1,416	720	83/31	20,519	3.4	6.9	35	698	1,416	720
OPEN PIT																							
UNDERGROUND	497,735	6.1	10.2	50	30,559	50,763	24,796	79/18	180,883	6.2	9.3	44	11,132	16,791	7,888	68/29	678,618	6.1	10.0	48	41,690	67,554	32,684
JEWELBOX – TTL	497,735	6.1	10.2	50	30,559	50,763	24,796	79/18	180,883	6.2	9.3	44	11,132	16,791	7,888	68/29	678,618	6.1	10.0	48	41,690	67,554	32,684
GRIBBLER	93,560	7.4	15.9	97	6,906	14,893	9,033	80/12	145,900	7.7	14.0	92	11,301	20,413	13,357	80/12	239,459	7.6	14.7	94	18,207	35,306	22,390
ATTILA	386,300	3.1	11.5	60	11,975	44,425	23,178	84/15									386,300	3.1	11.5	60	11,975	44,425	23,178
BURNICK	778,969	1.0	12.4	43	7,790	96,592	33,729	83/12	310,556	0.4	11.7	37	1,242	36,336	11,366	79/14	1,089,524	0.8	12.2	41	9,032	132,927	45,096
MINE TOTAL	1,756,564	3.4	12.1	52	57,230	206,673	90,736		637,338	3.7	11.5	51	23,675	73,539	32,611		2,393,901	3.5	12.0	52	80,904	280,212	123,347

INTRODUCTION

Starting in late December, 1992, a study was undertaken to update the mineral inventory of the Sä Dena Hes mine. Its purpose was to produce a new mineral inventory and from this a mineable ore reserve effective January 1, 1993.

LOCATION

The Sä Dena Hes mine is located in the southeastern Yukon, 50 km north of the Town of Watson Lake (Figure 1). Access is by an all-weather road to the minesite, from the Robert Campbell Highway. The Jewelbox Hill ore deposit is the first of four ore occurrences developed on the property. It is located approximately 1 km southwest of the mill site (Figure 2). Additional ore zones, include the Burnick zone (presently being developed), and the Attila Zone, on North Hill, 4 km to the north of Jewelbox Hill, and on Gribbler Ridge 1 km to the northwest of Jewelbox Hill.

HISTORY

Lead/zinc mineralization was discovered and the property first staked in 1962. The area was intermittently explored in the early sixties. After over a decade the property was reactivated again in 1979 by Cima Resources. From 1979 to 1981 the Main Zone on Jewelbox Hill was delineated by surface drilling, and several other zones were tested. A proven reserve of 263,000 tonnes was established, and a feasibility study was carried out. The property was purchased in 1984 by Canamax Resources which carried out aggressive exploration from 1985 to 1988, drilling nearly 20,000 metres in 112 holes. The Canamax work was important in that it indicated much more extensive mineralization on Jewelbox Hill, on Gribbler Ridge, and on North Hill in the Burnick and Attila zones. The property was purchased by the Mt. Hundere Joint Venture in 1989. Extensive surface diamond drilling was carried out in 1989 (140 holes), and the property was committed to production the following year. Geological reserves were calculated by Roscoe-Postle Associates based on this surface work. Using the same information base underground minable reserves were determined by Canadian Mine Development and open pit reserves by H.S. Clarke. These reserve estimates were presented in the Kilborn March 1990 Project Development Plan and were the basis for the feasibility study. Early in the construction phase, underground exploration and diamond drilling of the Chimney, JB1-Pod and part of the JB1 zone was carried out, as well as further surface drill definition of the Main zone and condemnation drilling at the mill site (142 holes - 4,460m). The start of mining reserve was recalculated in mid 1991 by the mine-site geological and engineering staff using the results of this initial phase of underground work. The first concentrate was shipped on August 1, 1991. The mine was officially opened on September 24, 1991. Total capital cost for the project, including acquisition and exploration, was $95 million.

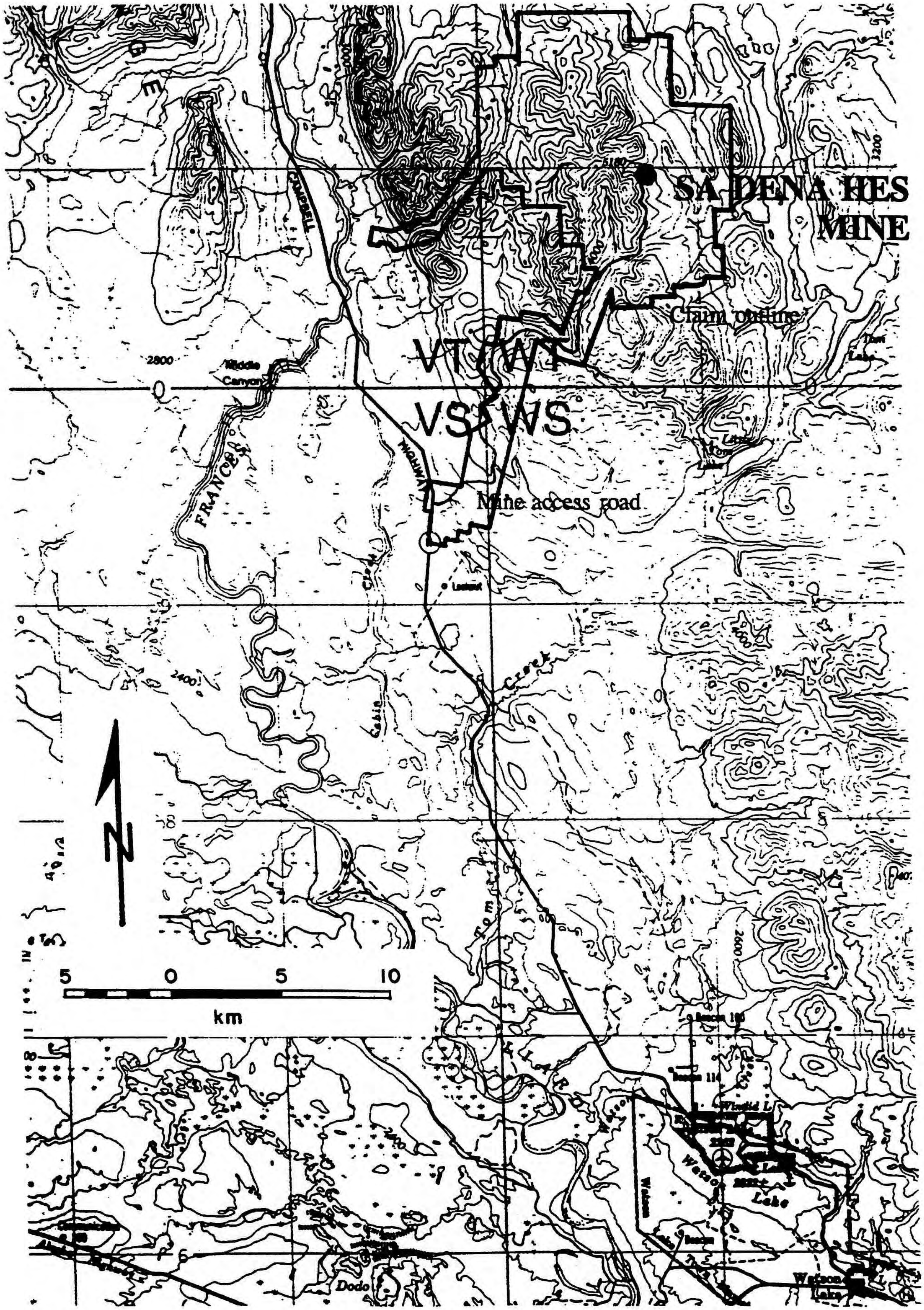

Figure 1. Location of the Sä Dena Hes Mine

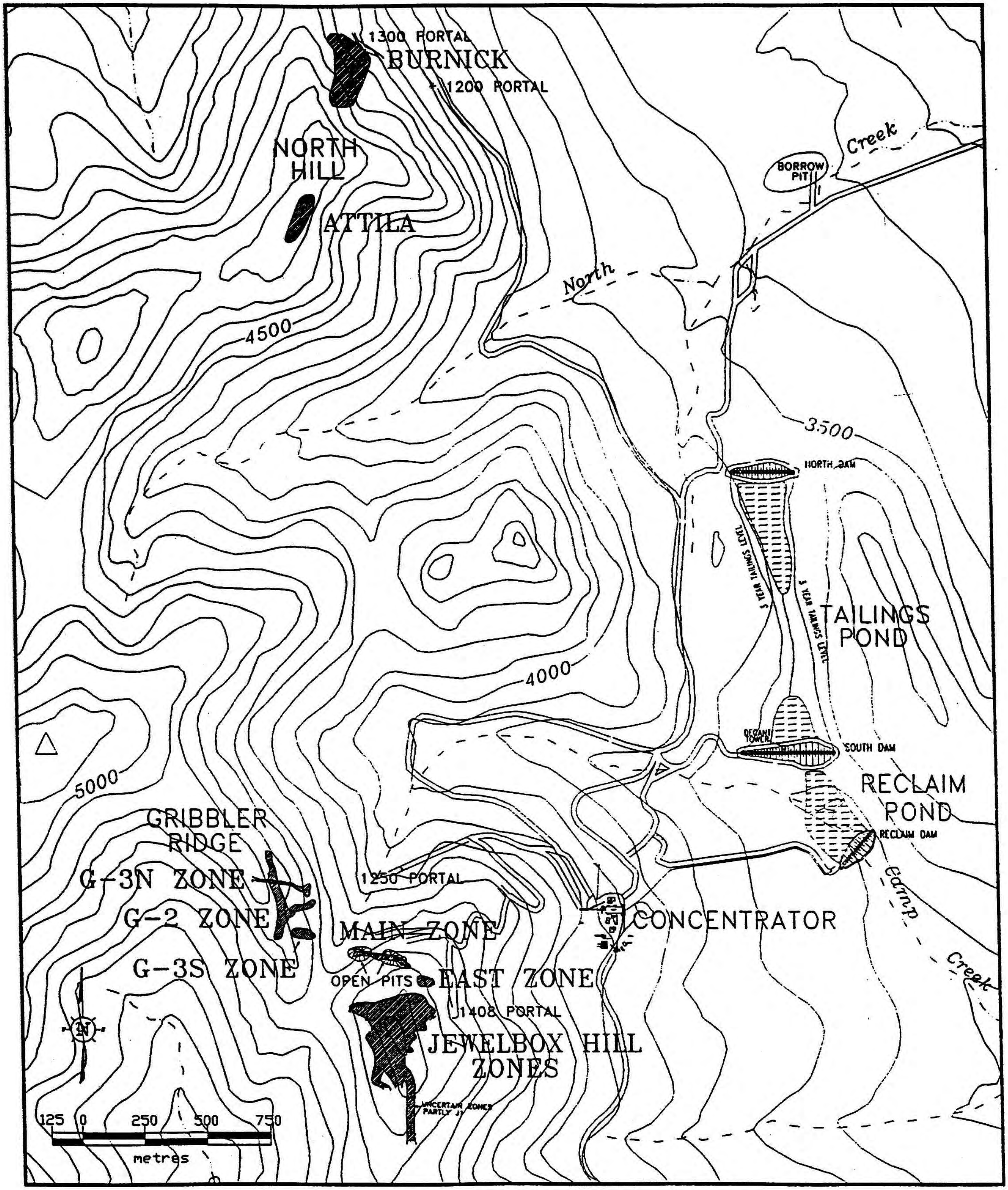

Fig. 2: Sa' Dena Hes mine site, showing location of ore zones relative to the Jewelbox Hill deposit.

To December 31, 1992, a total of 726,758 tonnes, averaging 7.2% Pb and 11.7% Zn, have been mined by underground and open pit methods. All but 11,000 tonnes of this is from Jewelbox Hill, the remainder is from Burnick. Extensive definition diamond drilling has been done underground in Jewelbox (approximately 970 holes - 42,300m) and a limited amount of definition drilling (56 underground holes - 4,093 m., and 15 surface holes - 791 m.) on Burnick.

Additional ore reserve and/or mineral inventory studies that have been completed include the start of 1992 inventory and reserve for the entire property and an interim mineral inventory for parts of Jewelbox Hill as of August 1, 1992.

PROPERTY GEOLOGY

The Sä Dena Hes property is underlain by lower Paleozoic metasedimentary phyllites. Approximately 5% of the section consists of lower Cambrian archaeocyathid bearing limestone which forms discontinuous bodies a few m. to 100 m. thick.

The area is structurally complex, having been regionally deformed and metamorphosed to greenschist facies during the Mesozoic in a east directed polyphase fold/thrust regime. Late normal faults in several directions are important on the property. Post deformation intrusive rocks exposed on the property are confined to volumetrically minor mafic and felsic dykes of Cretaceous or Tertiary age. The property is located on a fault bounded uplift exposing the lower Paleozoic strata; this uplift has been hypothesized to be underlain by a Cretaceous granitic pluton.

The Sä Dena Hes ores are skarn lead-zinc-silver replacements developed along the margins of the lower Cambrian limestones. Mineralization is hosted by actinolite-hedenbergite-diopside-epidote-garnet (grossularite-andradite)-chlorite-calcite-quartz skarns. The best grades of lead-zinc appear to be related to retrograde actinolite $\pm$chlorite dominant skarns, rather than to prograde garnet-pyroxene dominant skarns. Most important skarns are formed from a limestone protolith; however, there are good examples of skarn developed from phyllite and locally from intrusive rocks. Generally these skarns are low grade, although locally they do make ore. The skarns bear little relation to intrusive contacts. Indeed, many of the best ore zones appear totally lacking in any intrusive association.

Sulphide mineralization consists mainly of medium to coarse-grained sphalerite and galena heavily disseminated in skarn layers. There is little or no iron sulphide present. Toward the periphery of areas of lead-zinc bearing skarn, magnetite skarns are developed locally. In places these peripheral skarns also contain pyrrhotite and pyrite and, more locally, chalcopyrite.

Within 130m of surface the ores are commonly heavily oxidized to soft incompetent rusty masses of clay, quartz, smithsonite, anglesite and cerussite. Commonly some relict sulphides, especially galena, are retained in the oxidized skarn. Although oxidation is generally a near-surface phenomena, some oxides have been encountered at depths of 300m or more; conversely some outcropping ore is only weakly oxidized. Locally, smithsonite has been mobilized from the oxidized skarns and deposited in nearby open fractures. Smithsonite cemented overburden has been noted in places.

Thermal alteration of the phyllites to harder, blocky hornfels is widespread on the property and is spatially related to recrystallization of the limestone to marble, skarn formation and ore emplacement.

The following features are thought to be significant in localization of ore on the property. The list is preliminary and will evolve as further knowledge is gained of the ore zones and their geologic framework. Not all these features are necessary for the formation of ore; however, the first two are nearly universal, whereas the following are less so.

1. presence of hornfels alteration zone;
2. presence of limestone protolith within alteration zone, particularly the equivalents of the Main Limestone;
3. proximity to limestone/marble-phyllite contacts;
4. presence of white coarse-grained marble, as opposed to grey limestone;
5. proximity to steeply dipping fractures and faults trending 090° to 110° and/or 000° to 020°;
6. proximity to the intersections of the above fractures sets;
7. proximity to axial regions of folds of any generation;
8. proximity to changes in thickness of carbonate units or the attitude of phyllite contacts, regardless of the cause of the changes.

JEWELBOX HILL MINERAL INVENTORY

Geology

The Jewelbox Hill deposit consists of a number of zones of skarn lead/zinc/silver mineralization, each of which has its own structure and grade distribution characteristics (Figure 3 - Page 18).

The zones are developed in several massive carbonate bodies near their contacts with phyllite. Near most of the ore zones, and through much of Jewelbox Hill, the carbonate is a coarse grained, white, calcite marble but toward the edges of the system and at depth fine grained, grey limestone is found. The carbonates are underlain and overlain by various phyllites. Intrusive dykes of felsic and mafic composition occur on Jewelbox Hill but are of minor importance.

The structure of Jewelbox Hill is dominated by a flat lying carbonate unit, the "Main Limestone", approximately 100m thick (Figure 5). This carbonate unit pinches out toward the east in the subsurface for reasons that are not fully understood. The pinch-out

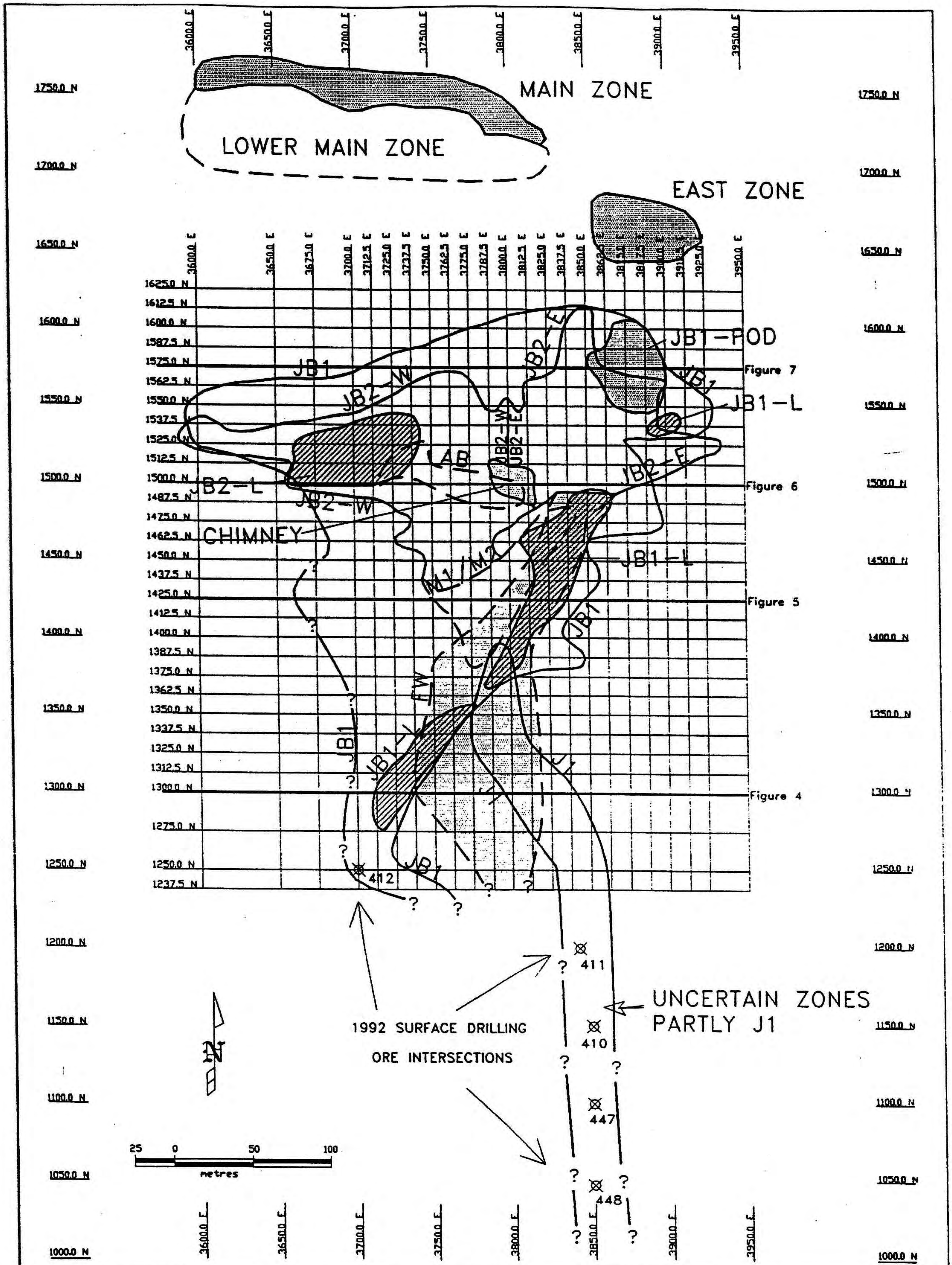

Figure 3: Vertical projection of generalized outlines of the various ore zones on Jewelbox, as shown on the cross sections. See Figures 4 to 7 for generalized sections.

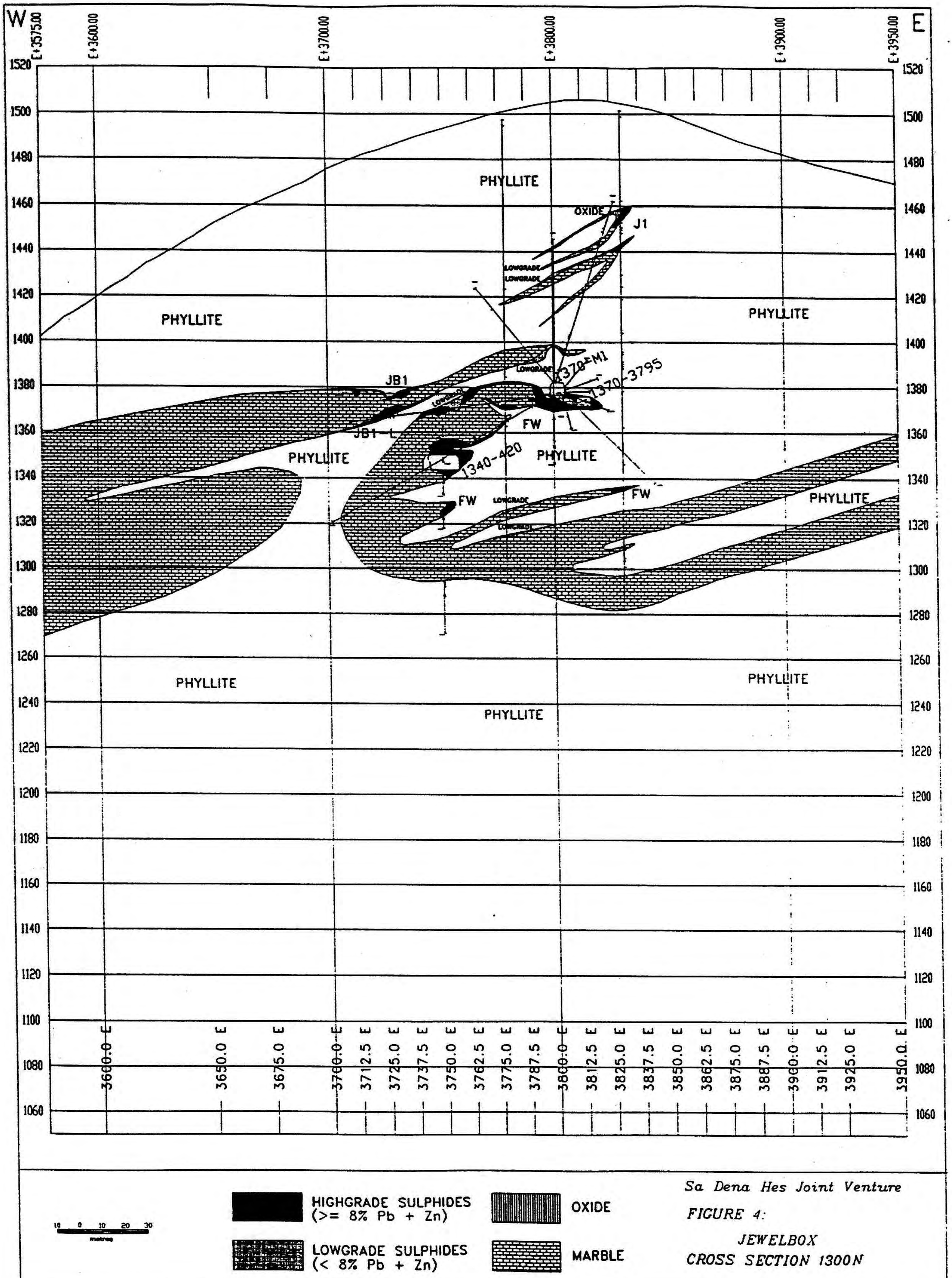

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FIGURE 4:

JEWELBOX CROSS SECTION 1300N

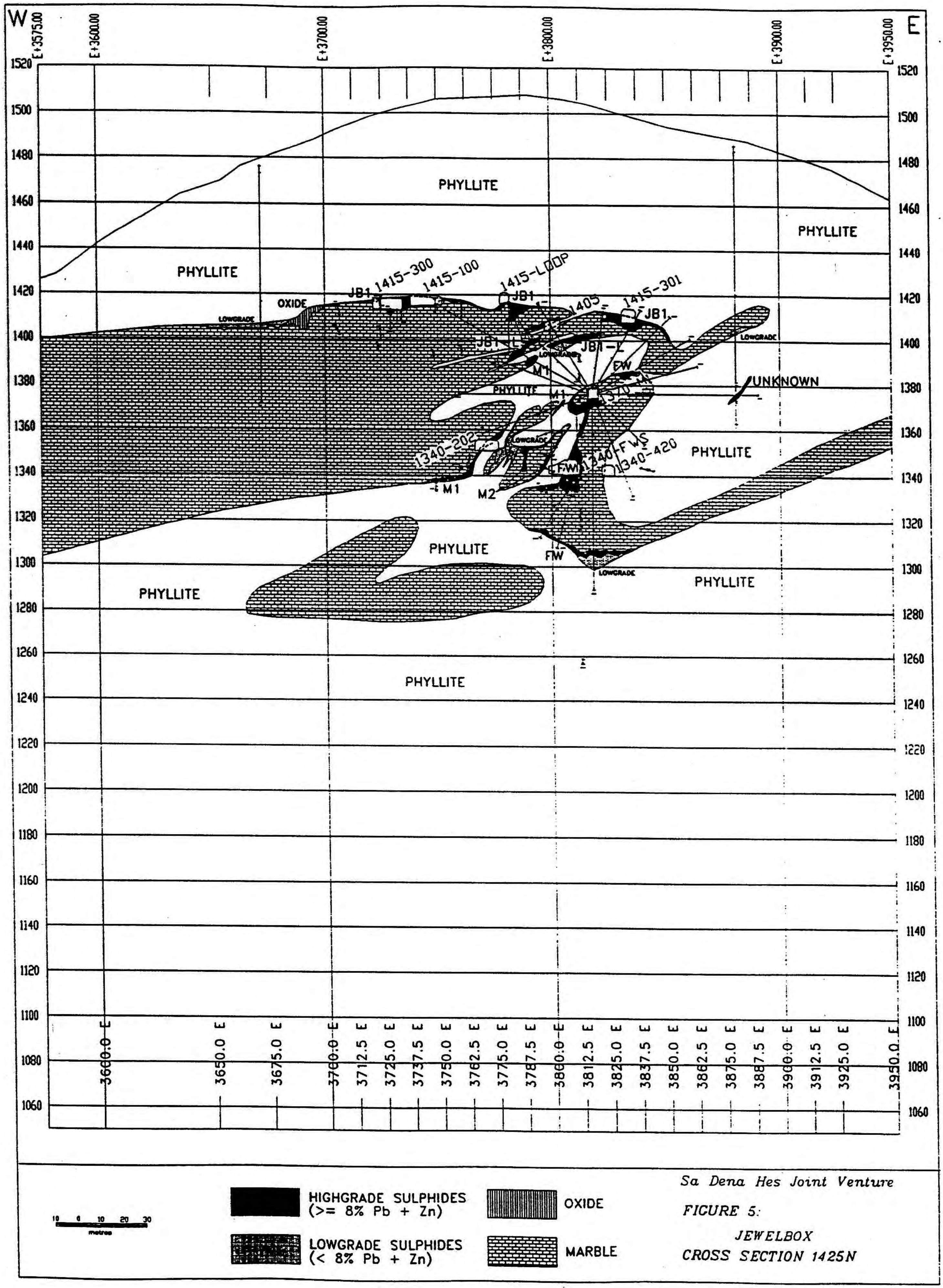

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FIGURE 5:

JEWELBOX
CROSS SECTION 1425N

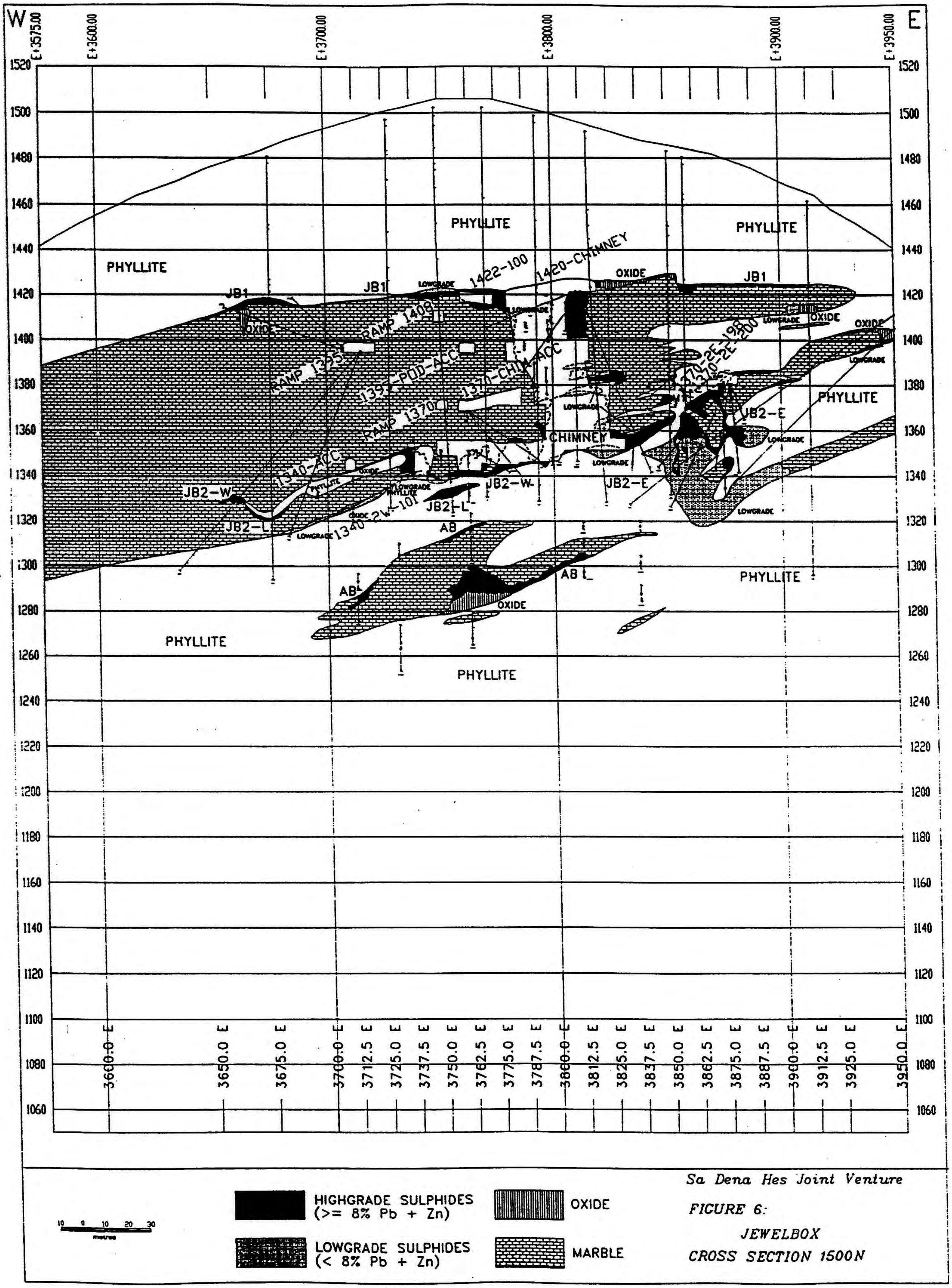

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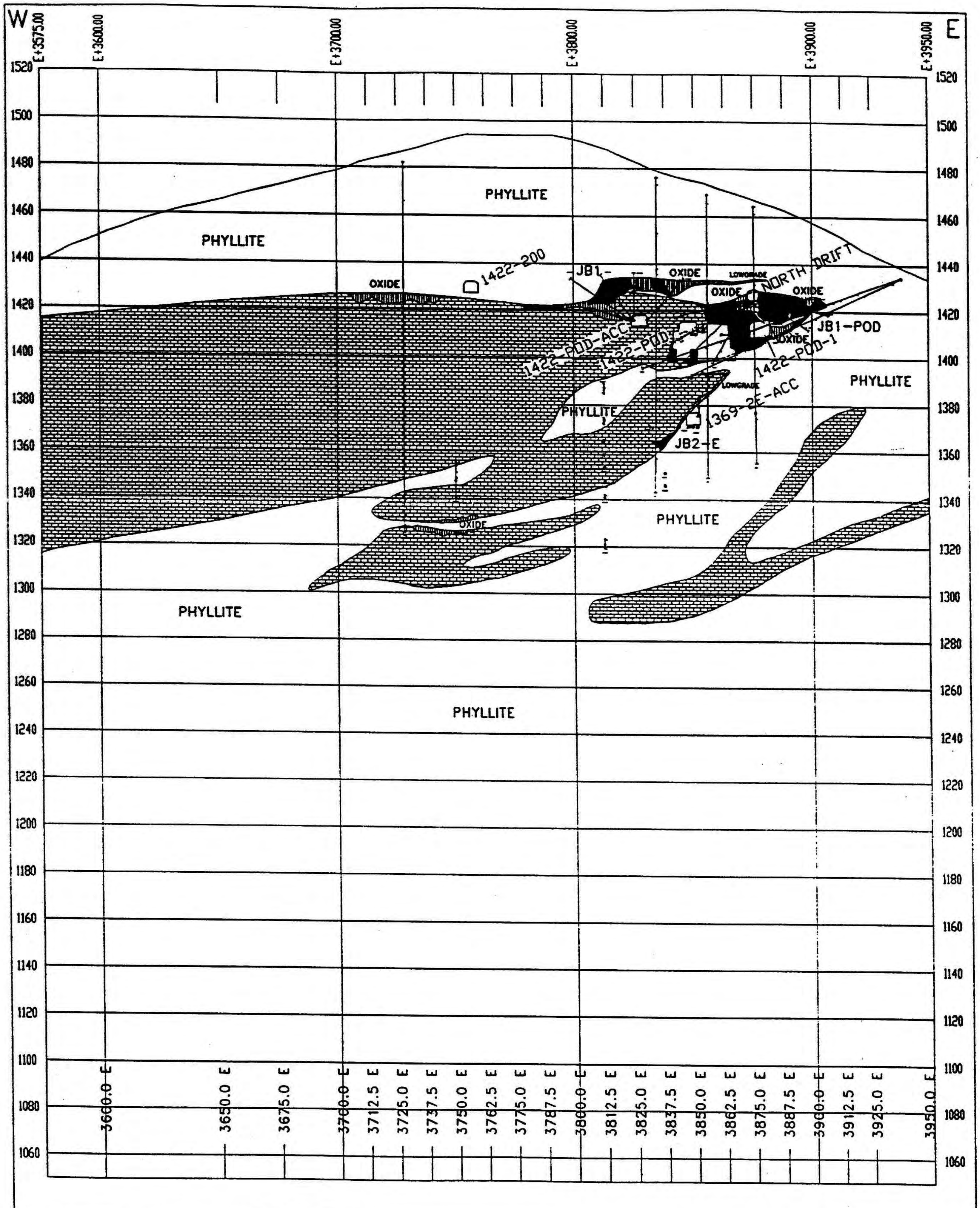
FIGURE 6:

JEWELBOX CROSS SECTION 1500N



HIGHGRADE SULPHIDES (>= 8% Pb + Zn)

LOWGRADE SULPHIDES (< 8% Pb + Zn)

OXIDE

MARBLE

Sa Dena Hes Joint Venture

FIGURE 7:

JEWELBOX

CROSS SECTION 1575N

follows a N35°E trend, swinging to a north-west trend in the north part of the Hill. The Main Limestone interfingers complexly with phyllite in the pinch-out area (Figures 6 and 7). A second carbonate of unknown stratigraphic correlation, the "FW Limestone", forms a steeply dipping layer east of the Main Limestone pinch-out (Figure 5). This is interpreted to be the steep limb of a second phase fold with its axial plane dipping approximately 45° west, and with axial trend 035°. The FW limestone in the steep limb of this fold pinches out upwards, possibly due to an earlier phase fold. At least one other limestone, possibly equivalent to the FW Limestone, occurs below the Main Limestone.

Steeply dipping faults and fracture zones, trending 090° to 110°, are important at Jewelbox Hill and appear to control many of the zones. There is also a north-south bias to mineralization that cuts across the N35°E trend of several of the limestones. Important faults on either side of Jewelbox Hill also trend northerly.

Jewelbox Hill skarns are dominantly coarse grained, dark green, actinolite rich assemblages also containing diopside, garnet, calcite, quartz and locally significant chlorite. A thin rind of pale green garnet succeeded inward by very coarse acicular sprays of actinolite in turn succeeded by high grade ore is a commonly observed sequence of skarn assemblages at the edges of the Jewelbox ore zones. The ore consists of the above assemblages with heavily disseminated, coarse grained, brown sphalerite and lesser galena. Some high grade zones are nearly massive coarse grained sphalerite and galena. Iron sulphide minerals are very minor or absent. The Jewelbox Hill ores are the most lead rich on the property with about 40% of the contained metal as lead compared to only 5% at Burnick.

There are several ore zones delineated at Jewelbox Hill. The zones are outlined in plan on Figure 3, and identified on several generalized cross sections (Figures 4 - 7). Briefly, the zones are:

J1 — Skarns associated with a complexly folded limestone and graphitic phyllite above the Main Limestone. Skarns of this zone are thin, discontinuous and may dip moderately west overall.

JB1 — Skarns occurring along the upper contact of the Main limestone. These skarns are <1m to over 5m thick and pinch and swell rapidly. Oxidation is widespread, affecting approximately half of the zone.

JB1-L — Below the JB1, generally on the lower contact of the uppermost of the easterly protruding noses of limestone.

JB2E and JB2W — Along the lower contact and easterly pinch-out of the Main limestone. The distinction between E and W is basically east and west of the Chimney (see below) respectively. These zones are relatively thick and continuous and are two of the more important zones on Jewelbox Hill. The ore is generally silver-rich and contains elevated bismuth content. The JB2W is relatively thick along a N-S trend at 3762.5E and also follows a 110° trend to the west.

JB2L Skarn below a phyllite intertongue into the JB2W area, this has been included with the JB2W for mineral inventory calculations.

M Zones (M1 and M2) Thin skarn zones of complex and uncertain structure related to intertonguing of phyllite and limestone, and possibly to rootless fold hinges of carbonate between the Main and FW limestones (M2 in particular - Figure 5).

FW Skarns developed on either side of the FW limestone and locally in its interior. These zones locally appear to be related to a pinch out of the FW limestone upward, perhaps due to a fold nose. This zone is also silver-rich. The FW is structurally equivalent to the easterly JB2E north of section 1487.5N (compare Figures 5 and 6 - Pages 8 and 9) The FW has been divided into 3 sub-zones, the Main FW, HFW and UFW. The UFW is on the upper contact of the FW limestone, above the Main FW, south of Section 1325N. The HFW is in the fold nose area (or pinch out) east of the Main FW. The remainder of the mineralization in the FW limestone is considered the Main FW.

JB1 Pod A thick sub-horizontal pod of high grade skarn between sections 1550 and 1600N, which appears to be related to the intersection of the Main Limestone pinch-out, and an 090° to 110° fault (along section 1562). The ore is moderately oxidized due to proximity to surface.

Chimney A vertical pipe of high grade skarn which penetrates the Main Limestone, and connects the JB1 and JB2E/W. The Chimney is approximately 15m in diameter and locally is elongate in a southeast-northwest direction.

Ash Bin A local, approximately 110° trending zone in a limestone below the Main Limestone. Developed in the vicinity of and below the base of the Chimney in basically the same location, and following the same trend, as the westerly extension of the JB2W.

Main Zone The Main Zone is the most northerly zone yet discovered on Jewelbox Hill. It is an approximately east-west trending tube of mineralization following a limestone phyllite contact for 175m. The tube plunges gently west and is roughly 10m to 15m in diameter. Much of the Main Zone has already been mined in two small open pits, but some material remains in the pit floor and in an underground area between the two pits.

Lower Main The Lower Main zone is a recent discovery on a deep marble phyllite contact below the Main zone. Its geometry is uncertain. Scattered drill intersections suggest it could have considerable tonnage potential.

The JB1 and JB2 zones are essentially mirror images of one another, although the JB2 is thicker and more continuous, as well as less oxidized. Both occur along the Main Limestone-phyllite contacts. They tend to have a relatively smooth, but locally distorted or folded, contact against phyllite, but a highly irregular, lobate, ragged contact against marble. The nature of the marble/skarn contact is suggestive of a "corrosional" boundary due to replacement of the carbonate by skarn. The contact against phyllite is, by its very nature, conformable. In contrast, the contact against marble is strongly discordant in detail. On a larger scale the skarn-carbonate contact tends to follow lithologic trends, locally exploiting particular layers in the carbonate. The asymmetry and geometry of the skarn contacts, described above, is typical of most skarn zones on phyllite-limestone contacts, regardless of orientation. The ragged irregular contacts lead to high waste dilution. The continuity of the skarn zones is limited and they pinch and swell over short distances. Due to the ragged contacts the thickness inferred from drilling can be highly variable over short distances, adding to difficulties of inventory estimation. There is a danger of over correlation of skarns in poorly drilled areas, and of assigning too large an area of influence to drill intersections. The strong EW control to some skarn zones introduces difficulties in representing the zones on EW cross-sections, and longitudinal sections must also be consulted. The approach used in this inventory effectively deals with these issues by representing the zones on levels derived from both the cross and longitudinal sections. Underground mapping has also been used to help delineate the zones in plan and define zone trends.

Preparation of Mineralized Blocks

Geological inventory blocks were constructed from cross and long sections which were, in turn, developed from the database of 985 underground drill holes, 398 surface drill holes, and extensive underground geological mapping. The mineral inventory area involved extends from section 1175N to section 1775N (Figure 3). Most sections are spaced 12.5 m. apart.

Ore grades were generated using diamond drill hole intersections, assays were weighted by recovered core length. Where 5 meter cuts were used to define an ore zone, grades were calculated by averaging all intersections within that elevation in that zone.

Tonnes were calculated using a specific gravity of 3.2 for ore and 2.8 for waste.

The inventory blocks were laid out using a minimum true thickness of 2.2m and a cutoff grade of 8% Pb and Zn. Lead and zinc were combined arithmetically.

Completely oxidized zones have been separately delineated and are not included in the mineral inventory. The mineral inventory is based on total lead-zinc grade. Some minor oxide lead-zinc may be included but if there is significant oxidation or partial oxidation, the zone has been so classified and excluded. To account for weakly oxidized blocks, an allowance for potentially unrecoverable minerals is made in defining mill recovery.

Classification of Mineralization

Inventory blocks were classified as either Type "A" or Type "B" to categorize the certainty of the estimate of the blocks tonnage and grade of the block.

Type "A" mineral inventory is that mineralization considered as proven or probable as defined in the corporate policy on reserves and mineral inventory.

(1) Proven (measured) Reserves: Reserves for which (a) quantity is computed from dimensions revealed in outcrops, trenches, workings, or drill holes; grade and/or quality are computed from the results of detailed sampling, and (b) the sites for inspection, sampling and measurement are spaced so closely and the geologic character is so well defined that size, shape, depth and mineral content of reserves are well-established.

As applied at Jewelbox, this category includes mineralization defined by closely spaced drill holes approximately 12.5m or less apart, and/or the presence of workings showing demonstrable good geological continuity.

(2) Probable (indicated) Reserves: Reserves for which quantity and grade/or quality are computed from information similar to that used for proven (measured) reserves, but the sites for inspection, sampling, and measurement are farther apart or are otherwise less adequately spaced. The degree of assurance, although lower than that for proven (measured) reserves, is high enough to assume continuity between points of observation.

This category includes mineralization defined by moderately spaced drill holes, approximately 12.5m - 25m apart, and good geological continuity confirmed by excavation nearly in the zone.

Proven and Probable mineralization have in the past been separated, but here are collectively reported as proven + probable in Type "A". The reason for this was to simplify the calculations and to deal more realistically with the nature of proven mineralization at Sä Dena Hes.

The mineralization that might be considered proven is a relatively small proportion of the overall inventory (approximately 20%). This material consists essentially of two components, that tied up in pillars and an additional component. The amount of this additional component tends to remain fairly constant as mining and definition drilling proceeds, but moves about in the mine depending on the rate of drilling and mining. Since material is mined almost immediately after it is proven, separate quantification does not seem warranted from an operational perspective.

Type "B" mineralization is that considered as "possible" and is based upon widely spaced drill holes and/or assumed geological continuity. This category includes mineralization defined by drill holes more than 25m apart and with geological continuity not well defined.

Methods Used for Each Zone

Each zone was individually assessed for the most appropriate calculation methodology, based on its geometry and drill data. The methods used are as follows:

Main Zone:	Open Pit - unchanged from January 1992 calculations
	Underground - based upon January, 1991, calculation, less tonnage mined in 1992.
J1 Zone:	Sectional polygons based on surface and underground diamond drilling on sections ranging from 12.5m to 50m apart. This zone appears to be very irregular and discontinuous.
JB1 Zone:	Polygons constructed on the JB1 horizon using surface and underground diamond drilling results and geological mapping in underground workings.
JB1-L Zone:	Polygons constructed on the JB1-L horizon using surface and underground diamond drilling results and geological mapping in underground workings.
JB1-Pod Zone:	Polygons on 5m horizontal cuts developed from cross and longitudinal sections which were developed from surface and underground diamond drilling and underground geological mapping.
Chimney Zone:	Polygons on 5m horizontal cuts developed from cross and longitudinal sections which were developed from surface and underground diamond drilling and underground geological mapping.

M Zones:	Polygons on 5m horizontal cuts developed from cross sections based upon surface and underground diamond drilling.
JB2-West Zone:	Polygons on 5m horizontal cuts generated from cross and longitudinal sections, underground geological mapping and surface and underground diamond drilling.
JB2-East Zone:	Polygons on 5m horizontal cuts generated from cross and longitudinal sections, underground geological mapping and surface and underground diamond drilling.
FW Zone:	Has been broken down into three smaller zones, HFW, UFW and Main FW, based upon geometry of the zone. <u>HFW Zone</u>: Polygons constructed along the fold nose area east of the Main FW zone. Polygons based upon cross sections developed from underground diamond drilling and geological mapping of the workings. <u>UFW Zone</u>: Located above the Main FW zone, south of section 1325N. Polygons on section based on limited underground diamond drilling. <u>Main FW Zone</u>: The core of the FW defined by two steeply dipping marble phyllite contacts that strike approximately northeast, southwest and dip to the west. Polygons on 5m horizontal cuts constructed from cross sections based upon surface and underground diamond drilling and geological mapping.
Ash Bin Zone:	Polygons on longitudinal sections developed from underground diamond drilling below the JB2W Zone.
Lower Main Zone:	Polygons on a marble phyllite contact situated below the Main Zone. Defined by a few widely spaced surface and underground drill holes.
Undefined Zones:	Zones based on one or a few widely spaced holes where continuity is not obvious due to a very limited drill data.

Results

A total inventory of 922,099 tonnes at an average undiluted in-situ grade of 7.3% Pb, 11.6% Zn and 60 g/t Ag was delineated. Approximately 60% of this material is considered proven plus probable (type A). After adjusting for mining during 1992 this represents a slight increase in the mineral inventory over the start of 1992 inventory.

The results of this study compare relatively closely to the August 1, 1992 mineral inventory when compared on the basis of the pre-mining in-situ inventory for only those zones examined in the August 1, 1992 study. On that basis this work estimates 1.182 million tonnes (excluding the Main, lower Main and Chimney Zones) compared to 1.213 million tonnes for the August study, a difference of 2.5%. The grades are also very close if it is assumed that what remains is close to the starting grade (lead is identical, zinc lower by 4 relative percent and silver by 8%).

The results of the mineral inventory are summarized by zone and category (A, B, and A + B) on Table 3. Appendix I provides a list of the tonnage and grade of each mineralized block used in the inventory, and a set of diagrams that identify the location of each block. Appendix I also includes an additional set of tables listing the assay data used to calculate grade for each block or horizontal cut.

Comments on Changes in Mineral Inventory

In the Main Zone there was no new data collected and no changes in mineral inventory, other than those brought about by limited mining in 1992.

In the J1 Zone a considerable number of new holes were drilled from underground, trying to establish ore on this horizon; on only two sections, 1375N and 1362.5N, was this successful. This resulted in a loss of about 71,000 tonnes in "Type B" mineralization.

In the JB1 Zone considerable drilling and underground development took place to the North and South in 1992. After mining there was a moderate increase in "Type A" mineralization of about 37,000 tonnes; some of this came from "Type B" mineralization which had a loss of about 67,000 tonnes, and a new ore extension found to the north.

The JB1-L Zone underwent a major loss in "Type A" mineralization of about 121,000 tonnes. This is due largely to two reasons: better definition of the zone by mining that indicates a smaller extent than expected, and a reinterpretation of the geology which has moved a large portion of the zone, with its tonnage, into the new, better defined FW Zone.

TABLE 3

SA DENA HES JOINT VENTURE
JEWELBOX HILL MINE
JANUARY 1, 1993 MINERAL INVENTORY

NAME	CATEGORY A (proven+probable) ORE TONNAGE (tonnes)	CATEGORY A ORE GRADE LEAD (%)	CATEGORY A ORE GRADE ZINC (%)	CATEGORY A ORE GRADE SILVER (g/t)	CATEGORY B (possible) ORE TONNAGE (tonnes)	CATEGORY B ORE GRADE LEAD (%)	CATEGORY B ORE GRADE ZINC (%)	CATEGORY B ORE GRADE SILVER (g/t)	CATEGORY A&B (proven + probable + possible) ORE TONNAGE (tonnes)	CATEGORY A&B ORE GRADE LEAD (%)	CATEGORY A&B ORE GRADE ZINC (%)	CATEGORY A&B ORE GRADE SILVER (g/t)
MAIN O/P	5,120	17.1	22.7	82					5,120	17.1	22.7	82
MAIN U/G	8,707	9.9	13.6	58					8,707	9.9	13.6	56
JB1 – POD	70,430	9.5	13.8	52					70,430	9.5	13.8	52
JB1	136,143	6.6	11.0	37	22,433	8.5	11.8	43	157,676	7.0	11.1	37
JB1 – LOWER	27,997	9.1	14.3	43	7,765	10.7	14.9	55	35,762	9.4	14.4	45
JB – M2					18,343	8.8	13.8	44	18,343	8.8	13.8	44
JB2 – WEST	74,059	7.4	13.9	57	24,394	6.5	10.7	53	96,453	7.2	13.1	56
JB2 – EAST	115,330	6.8	9.4	86	12,421	9.7	8.6	149	127,751	7.1	9.3	92
CHIMNEY	19,773	8.6	15.6	52					19,773	8.6	15.6	52
J1	13,400	6.2	9.2	31	4,593	6.4	10.0	28	17,993	6.3	9.4	30
JB – FW	112,099	5.7	10.7	59	22,651	5.3	9.4	47	134,750	5.6	10.5	57
JB – HFW	24,021	4.8	10.4	42	29,544	9.7	14.4	50	53,585	7.5	12.6	48
UFW					12,320	7.5	13.9	43	12,320	7.5	13.9	43
ASHBIN	38,908	9.5	15.6	119	12,957	10.3	16.1	77	51,865	9.7	15.7	109
LOWER MAIN	7,679	7.2	9.9	85	57,806	6.5	9.1	70	65,485	6.6	9.2	71
UNDEFINED					44,206	5.1	8.4	67	44,206	5.1	8.4	67
OPEN PIT	5,120	17.1	22.7	82					5,120	17.1	22.7	82
UNDERGROUND	647,546	7.2	11.8	60	269,433	7.3	11.0	61	916,979	7.3	11.6	60
JEWELBOX – TOTAL	652,666	7.3	12.0	60	269,433	7.3	11.0	61	922,099	7.3	11.6	60

In the Pod, with mining and limited underground drilling, there has been limited change which results in an increase of about 7,000 tonnes of "Type A" mineralization.

In the Chimney, mining with limited diamond drilling indicated a modest increase of about 10,000 tonnes in "Type A" mineralization.

In JB2W there was a small increase in "Type A" mineralization after mining of about 28,000 tonnes. This was the result of mining and underground diamond drilling extending the zone to the west. This also resulted in an increase of "Type B" mineralization of about 23,000 tonnes.

In JB2E there was an increase of "Type A" mineralization of approximately 47,000 tonnes, the result of underground diamond drilling, underground mapping, and geological reinterpretation of some areas. There was a small loss of 4,000 tonnes in "Type B" mineralization.

The FW Zone was broken down into three sub-zones on the basis of local geology, the "FW" proper, the "HFW" and "UFW". The three zones had a substantial increase in "Type A" mineralization of approximately 118,000 tonnes. This is due to better geological data available from numerous new drill holes, underground mapping and the gaining of mineralization from the JB1-L Zone and M zones. There was also a considerable increase in "Type B" mineralization of about 46,000 tonnes.

The M zones contained no "Type A" mineralization on January 1, 1992, but yielded 7,694 tonnes of ore in 1992. There was, however, a substantial loss in "Type B" mineralization of about 94,000 tonnes. Some of this is due to movement of material to the FW Zone, but the majority is due to the reinterpretation of the zone which underground drilling has indicated is quite discontinuous along strike.

The Ash Bin Zone is a new zone found about 40 meters below the JB2-W Zone. It is estimated to contain about 39,000 tonnes of "Type A" ore and about 13,000 tonnes of "Type B" mineralization. It is still open to the west.

The Lower Main Zone is a new zone found late in 1992. It is situated about 70m below the Main Zone open pits. "Type A" mineralization is estimated at about 8,000 tonnes, but "Type B" mineralization is estimated at about 58,000 tonnes. The zone itself is open to the north and west and has significant potential for increased reserves.

The undefined zones are based upon miscellaneous drill hole intersections, primarily below and to the east of the FW Zone which are not correlatable over a large area. They are estimated to contain about 44,000 tonnes of "Type B" mineralization.

BURNICK ZONE MINERAL INVENTORY

Geology

The geology of the Burnick Zone has been considerably revised in light of recent fill-in drilling mand underground exploration (Figure 8 - Page 21). The previous synclinal interpretation of the upper part of the zone has been abandoned since fill-in drilling did not confirm continuity of ore zones.

The zone is now interpreted to be a series of steeply west dipping lenses of skarn ± limestone from 5m to 30m thick (Figures 9 and 10 - Pages 22 and 23). Local thicker elongate pods are interpreted in a few areas which could be due to medium scale parasitic folds. The lenses seem to parallel metamorphic foliation in the phyllites.

Limestone bodies in the Burnick zone are generally thinner than at Jewelbox Hill and are typically grey fine grained limestone or dark grey, argillaceous limestone rather than marble. Not all skarns on North Hill are derived from limestone; some appear to be derived from phyllite. This has been observed at Jewelbox but seems to be more important at Burnick.

Other differences between Jewelbox and Burnick include the skarn mineral assemblage which at Burnick is more garnet-pyroxene rich and less actinolite-chlorite rich. This may impact operations variably in that the ore is likely to be harder but less prone to handling problems. Burnick ores are, of course, generally poorer in lead compared to zinc, than at Jewelbox. The silver content is comparable and since silver reports to the lead concentrate, a very silver (and bismuth) rich lead concentrate is produced.

The Burnick ores are visually finer grained than Jewelbox ores.

Two important north/south trending, near vertical, faults are interpreted east and west of the mineralized zones and a third fault dipping 50° east and striking 020°, has been interpreted to pass through the mineralized zones, although most zones are above this fault. Several other smaller faults have been interpreted. The structural interpretation is preliminary and will be refined as further work is done; however, it seems unlikely that there will be much variance in volume from the steep west dipping layer interpretation in the area drilled off underground.

Sä Dena Hes Joint Venture
Burnick Infrastructure Plan

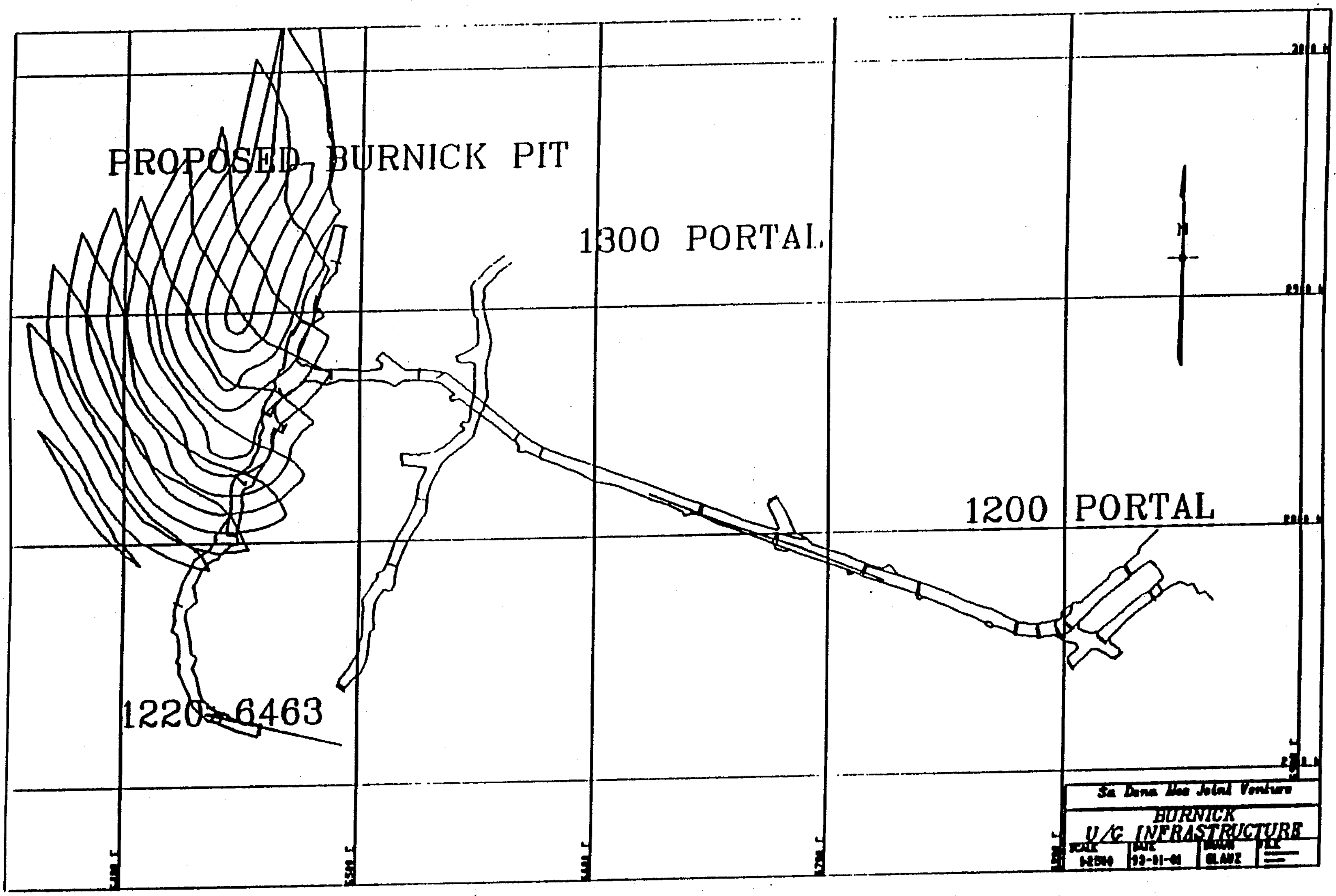


Figure 9

Sä Dena Hes Joint Venture
Burnick Section 2850N

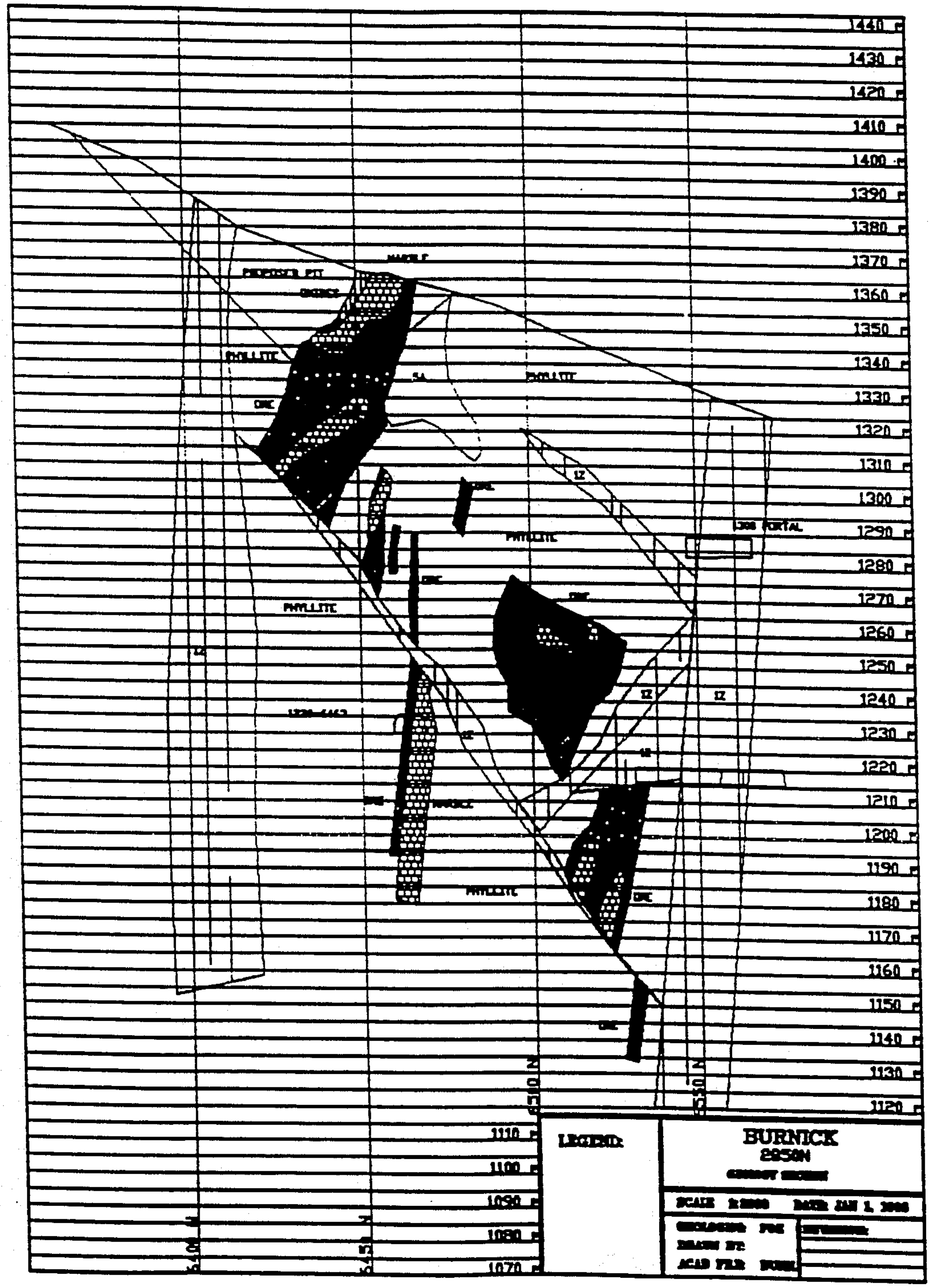
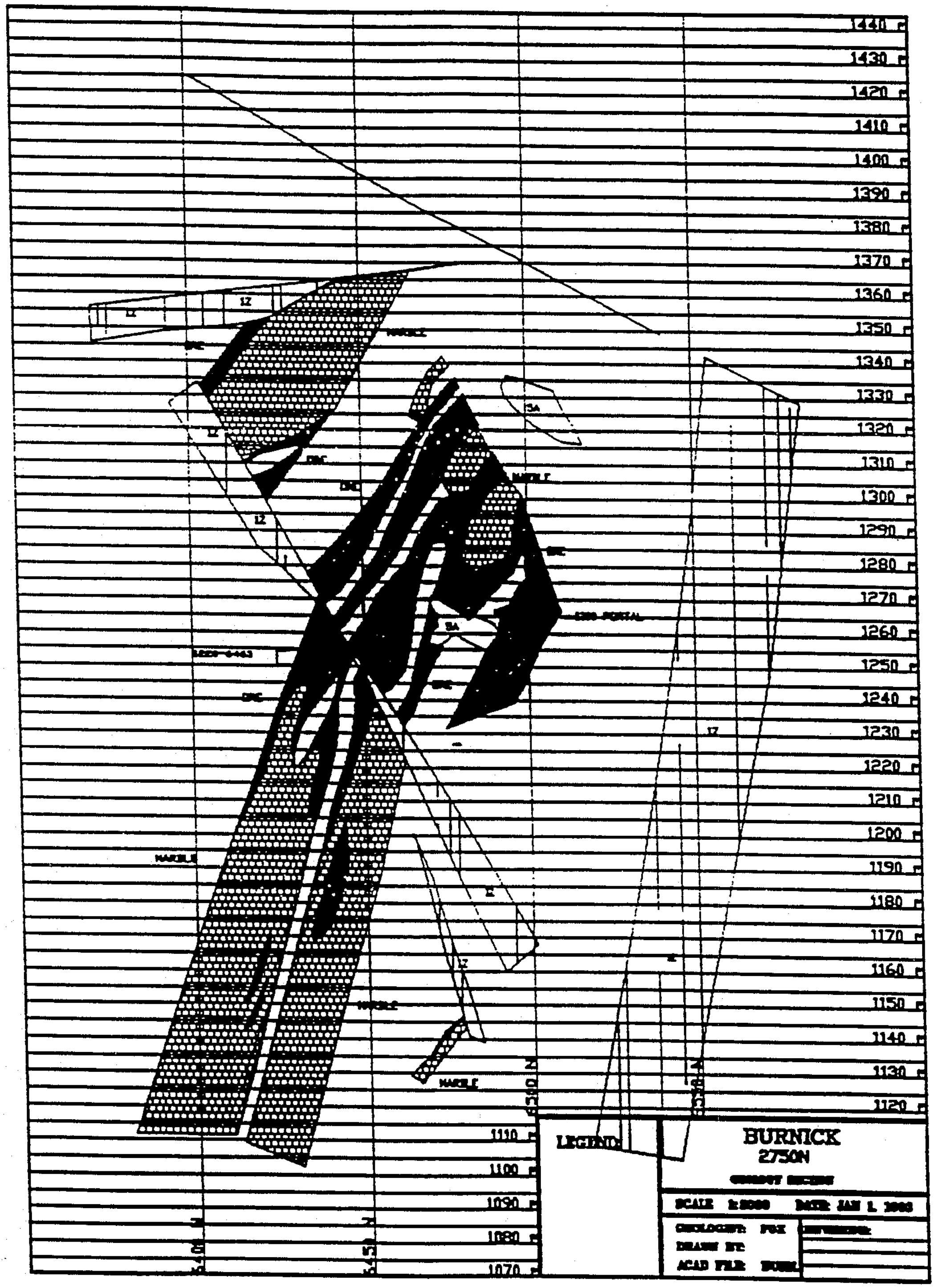


Figure 10

Sä Dena Hes Joint Venture
Burnick Section 2750N



Drilling at Burnick is limited compared to Jewelbox. Drilling is on thirteen 25m spaced sections from 2650N in the south, to 2950N in the north, a distance of 300m. Underground drilling was not done on all sections.

	Holes	Metres
Pre-1992 Surface Drilling	84	15,914
1992 Surface Drilling	15	792
1992 Underground Drilling	56	4,093
Total	**155**	**20,798**

Jewelbox, in comparison, has 1,223 holes totalling 67,105m.

Preparation of Mineralized Blocks

Mineralized inventory blocks were constructed on cross sections at 25m intervals from 2650N to 2975N, using data from surface and underground drill holes and geological mapping in the development drifts. Longitudinal sections were not used in the interpretation. The cross sections were checked for consistency by preparing selected level plan interpretations. Because of time considerations the geological interpretation and preliminary block layout was completed by Fox Geological and later revised and completed by the mine staff. The geological interpretation varies in several ways from the mine staff "in-house" interpretation but it is similar in style and the differences are not material.

Ore grades were generated using diamond drill assays, except in a few locations in drifts where visual estimates, based on underground muck samples, were used.

Tonnes were calculated using a specific gravity of 3.1 for ore and 2.8 for waste.

The inventory blocks were laid out using a minimum true thickness of 2.2m and a cutoff grade of 8% Pb and Zn. Lead and zinc were combined arithmetically.

Blocks were subdivided into 5 m. levels, assuming constant grade through the entire block, in order to report the reserve by lift.

Completely oxidized zones have been separately delineated and are not included in the mineral inventory. The mineral inventory is based on total lead-zinc grade. Some oxide lead-zinc may be included, particularly in the open pit area, but if there was significant oxidation or partial oxidation, the zone was so classified and excluded. For weakly oxidized blocks, allowance for potentially unrecoverable minerals is made in defining mill recovery. In general oxidation of the underground ore at Burnick is minor and is less than at Jewelbox Hill.

Classification of Mineralization

Type "A" mineral inventory is a combination of proven and probable as defined in the corporate policy on reserves and mineral inventory as discussed previously.

The Type "A" category includes material with a drill hole spacing of 25m or less, and/or 25m from an excavation. Mineralized blocks are projected a maximum of half the distance between the diamond drill holes. Where trends, history or geological information demonstrates otherwise, the distance may be reduced.

As with Jewelbox, there has been no separate accounting of Proven and Probable mineralization for simplicity. At Burnick, Proven mineralization would amount to a small proportion of the material near the limited underground openings to date.

Type "B" mineralization is all material for which there is sufficient information to attempt a correlation, but drill control is less dense than for Type "A". Type "B" is considered synonymous with "possible" mineralization.

Results

The results of this inventory in Burnick are:

	Tonnes	Pb(%)	Zn(%)	g/t
Proven and Probable	1,010,964	1.1	13.3	45
Possible	594,115	0.5	12.8	49
Total	**1,603,079**	**0.9**	**13.1**	**47**

This compares to total geological reserves of Roscoe Postle Associates of 2,494,000 tonnes and indicates a loss of nearly 900,000 tonnes. This is due to a revised geologic interpretation caused by fill-in drilling carried out in 1992.

The results of the mineral inventory are summarized by section and category (A, B, and A + B) on Table 4. Appendix II provides a list of the tonnage and grade of each mineralized block used in the inventory, and a set of small scale cross-sections that identify the location of each block. Appendix II also includes an additional set of tables listing the assay data used to calculate grade for each block.

Table 4
Sa Dena Hes Joint Venture
Burnick Zone, North Hill
Summary of the In-situ Mineral Inventory
or ("Geological Reserve") by Section

Category A mineralization (i.e. proven+probable)

SECTION	TONNAGE (tonnes)	Pb (%)	Zn (%)	Ag (g/t)
2650	0			
2675	34,736	0.5	14.7	33.4
2700	0			
2725	59,435	2.2	10.5	39.2
2750	114,406	0.4	13.4	49.3
2775	103,548	3.5	12.3	34.7
2800	67,720	0.4	16.0	37.8
2825	127,418	0.8	13.0	40.6
2850	191,371	0.4	13.8	56.1
2875	130,154	0.3	12.7	44.7
2900	97,991	2.7	13.2	42.7
2925	50,174	0.4	13.4	68.3
2950	34,015	0.2	15.6	37.1
Total	1,010,964	1.1	13.3	45.4

Category B mineralization (i.e. possible)

SECTION	TONNAGE (tonnes)	Pb (%)	Zn (%)	Ag (g/t)
2650	14,578	0.4	10.8	58.3
2675	47,748	1.0	15.0	45.3
2700	123,233	0.9	11.2	34.4
2725	61,930	0.5	10.7	36.4
2750	100,277	0.4	13.5	55.8
2775	23,165	0.2	11.5	31.1
2800	24,645	0.3	15.4	56.8
2825	38,642	0.2	14.2	47.0
2850	12,850	0.4	11.8	36.6
2875	87,730	0.2	14.0	52.1
2900	13,298	0.1	13.3	53.4
2925	26,389	0.1	14.0	30.9
2950	17,631	1.3	10.8	178.3
Total	592,115	0.5	12.8	48.6

Category A and B mineralization combined

SECTION	TONNAGE (tonnes)	Pb (%)	Zn (%)	Ag (g/t)
2650	14,578	0.4	10.8	58.3
2675	82,484	0.8	14.9	40.3
2700	123,233	0.9	11.2	34.4
2725	121,365	1.3	10.6	37.8
2750	214,682	0.4	13.4	52.3
2775	126,712	2.9	12.2	34.0
2800	92,364	0.4	15.8	42.9
2825	166,060	0.7	13.3	42.1
2850	204,221	0.4	13.7	54.9
2875	217,884	0.3	13.2	47.7
2900	111,289	2.4	13.2	44.0
2925	76,563	0.3	13.6	55.4
2950	51,646	0.6	14.0	85.3
Total	1,603,079	0.9	13.1	46.6

Comments on Changes in Mineral Inventory

The Burnick Zone shows a substantial reduction in both Type "A" and "B" mineralization. This is attributable to a total change in interpretation of the geology of the deposit, brought about by underground development and diamond drilling.

The original interpretation assumed a synclinal shape with a thick "keel" in the hinge of the fold. Detailed underground mapping and diamond drilling have indicated that this is incorrect and the mineralization consists of a series of steeply dipping lenses with waste between them. Although the new interpretation caused a significant drop in tonnage, the overall drop in the tonnage in the upper part of the deposit (where the syncline was interpreted) was limited compared to the Roscoe-Postle calculation (decrease of 14% for Type "A" and "B"combined), the bulk of the tonnage loss at Burnick (75%) was at depth, below the syncline, where there was actually no new drilling done. This was due to extrapolating the new interpretation downwards. It is conceivable that further work may change the structural interpretation at depth.

The large drop in tonnage at Burnick may be partly compensated by future discoveries to the west and at depth. Of particular importance is the area south of Burnick towards Attila where there are currently only 2 holes. The steep lens interpretation opens considerable potential there. It is important that future underground drilling at North Hill test these areas in order to realize this potential. In the case of exploring the Burnick-Attila gap, it is essential that a heading be driven south from Burnick towards Attila as Burnick is being developed.

GRIBBLER RIDGE MINERAL INVENTORY

Geology

Considerable drilling was done in Gribbler Ridge in 1992 (48 holes - 10,969m). This has resulted in significant changes to the geologic interpretation and to the nature and correlation of mineralized zones.

Gribbler Ridge mineralization includes the G1, G2 East and West, G3 South and North, and the G4 zones. The G1 is hosted within a limestone of variable thickness that rarely exceeds 50m. This limestone, the Upper Limestone, is the structurally highest of all other identified zones at Gribbler Ridge. The G2 occurs at the upper contact of a thick limestone package, the Middle Limestone. This unit is typically white to light grey and altered to marble. The lower contact of this limestone hosts the G3 zone. The G4 zone occurs below the G3 and is hosted near the upper contact of the dark grey argillaceous Lower Limestone. All units generally dip gently the west or southwest. The G1 is extensively oxidized. The G2, G3 and G4 mineralization is dominantly sulphide with variable oxidation where these zones are near surface. This mineral inventory includes the G2 and G3 zones only.

The controlling structures that host the thickest mineralized zones at Gribbler Ridge are believed to be, in most cases, the intersection of faults and limestone contacts. The faults are considered conduits for mineralizing fluids that used the carbonate contacts as nucleation sites for mineralization. The mineralized zones are interpreted to be elongate tube-shaped bodies, similar to the Main Zone at Jewelbox Hill. The thickest portion of these bodies are in close proximity to the controlling structures. G3 South and G2 East are examples of these thick tube-shaped bodies proximal to the controlling structure. Laterally away from the structure the mineralization thins out, but, in some cases, forms sheet-like replacements along lithologic contacts. These sheet-like lenses could maintain a minable thickness over a significant distance. There has not been a mineralization-controlling structure identified for the G2 West zone. The G2 West is the best example of sheet-like mineralization that traces a lithologic contact.

Methods

Assay composites for Gribbler Ridge were selected with a cut-off grade of 8% combined Pb+Zn, and a 3.0m minimum width. Assays are composited by length weighting individual assays. Cored intervals not meeting minimum width were still used in the inventory if their 3-meter diluted equivalent met the grade requirement. Internal waste was averaged into the composite unless the waste interval was over 3.0m thick. Internal waste not represented by analytical data was assigned a zero grade. A limited data set of oxide lead and zinc analyses exists for Gribbler Ridge. Oxidation has generally been found to be minimal. The inventory is based on total metal content. Strongly oxidized intersections have been quantified in the calculation but have been treated independently of sulphides, and are not reported with the sulphide mineral inventory.

Polygons representing mineralized bodies were determined in conjunction with a preliminary interpretation of both cross and longitudinal sections. The polygon limits are defined by the 3.0m isopach. Polygon volumes have been determined by construction of isopachs of the vertical thickness of mineralization which is, in most cases, the same as the cored intervals. Inclined holes have been corrected for vertical thickness. Isopachs for the three, five, ten, twenty and thirty meter thickness, have been constructed. These polygons were assigned an average thickness midway between the upper and lower contours. The isopachs were then digitized and the areas reported in Autocad.

Density values used for the tonnage calculation are 3.4 tonnes/cu. m. for sulphide, 2.5 for oxide zones and 3.0 tonnes/cu. m. for G2 West sulphides. The 3.4 tonnes/cu. m. sulphide density is the same as that used for the Jewelbox Hill Main Zone. Main Zone and Gribbler Ridge mineralization are of similar tenor. The sulphide density is also in reasonable accord with production experience. The 3.0 tonnes/cu. m.density value for G2 West has been used because of the thin speculative nature of the zone and because most intersections in this zone had to be diluted by waste to reach the minimum width.

Classification

The mineral inventory for Gribbler Ridge includes both probable and possible categories. Probable mineralization is that for which drill hole spacing allows reasonably inferred continuity, based on comparison to better known nearby zones of similar nature. In this case limited evidence of continuity also exists in G3 South where electrical continuity has been established via a *mise-la-masse* survey conducted from hole to hole. The probable mineralization is on the weak side of the classification however the mineralization has been intersected by so many holes it is no longer appropriate to consider it as possible. Possible mineralization includes all other intersections correlated.

Results

The Gribbler Ridge mineral inventory completed at the end of the 1992 exploration season reports an undiluted, in place, probable sulphide tonnage of 106,300 tonnes of 8.1% Pb, 17.5% Zn and 106 g/t Ag in the G2 and G3 zones. The possible category contains 165,800 tonnes of 8.5% Pb, 15.4% Zn and 101 g/t Ag of sulphide mineralization. Total inventory of probable and possible is 272,100 tonnes of 8.4% Pb, 16.2% Zn and 103 g/t Ag. Results are tabulated by zone and by classification on Table 5). Further detail on the Gribbler Ridge inventory (including drawings of zone limits and isopachs) can be found in Appendix II. A separate Exploration Department memo dated Dec. 1, 1992 provides complete documentation of the mineral inventory.

Discussion

The inventory quoted herein includes only the G2 and G3 zones. Previously the "geological reserve" calculated by Roscoe-Postle was 347,600 tonnes, distributed as shown below:

	Tonnes	Pb(%)	Zn(%)	g/t Ag(%)
G1	128,900	14.6	14.2	79
G2	84,900	10.0	15.1	106
G3	133,800	11.1	16.1	106
Total	**347,600**	**12.1**	**15.2**	**96**

No 1993 G1 zone inventory was calculated since 1992 drilling showed it to be extensively oxidized, as had been expected. The write-off of the G1 zone reduced the overall inventory by 128,900 tonnes.

Structural and stratigraphic interpretation of Gribbler Ridge, in light of the 1992 drilling, suggests that hole 233, correlated by Roscoe Postle with the G3 zone, is actually a deeper horizon, the G4, which has been intersected at scattered localities through

Table 5

Sä Dena Hes Joint Venture
Gribbler Inventory Summary by Zone

GRIBBLER RIDGE MINERAL INVENTORY SUMMARY				23-Nov-92
	TONNES	Pb %	Zn %	Ag (g/t)
PROBABLE				
G-3 SOUTH	58,100	7.0	20.3	128
G-3 NORTH	21,800	12.4	17.5	114
G-2 EAST	26,400	7.0	11.4	52
TOTAL	106,300	8.1	17.5	106
POSSIBLE				
G-3 SOUTH	37,000	7.0	20.3	128
G-3 NORTH	39,100	11.4	17.6	123
G-2 EAST	21,500	7.0	11.4	52
G-2 WEST	68,200	8.2	12.7	88
TOTAL	165,800	8.5	15.4	101
PROBABLE + POSSIBLE				
TOTAL	272,100	8.4	16.2	103

Results present in-place sulphide mineralization with no dilution.

Gribbler Ridge. The G4 horizon has not been quantified at this time since there is still considerable uncertainty over its continuity. The G4 zone represents future tonnage potential which may develop into a minable ore zone once underground drilling begins. The G3 tonnage calculated by Roscoe Postle, based on the intersection in hole 233 was 17,200 tonnes.

Given the above, the G2 and G3 zone geological reserve calculated by RPA, expressed in terms comparable to the current inventory would be:

	Tonnes	Pb(%)	Zn(%)	g/t Ag(%)
G2	84,900	10.0	15.1	106
Revised G3	116,600	12.6	15.4	94
Total G2 and G3	**201,500**	**11.5**	**15.3**	**98.9**

This compares to 272,100 tonnes calculated by the current inventory. The increase in both zones is due to having located thicker east/west trending shoots within the horizon and tracing them with the 1992 drilling. Although the lead grade has dropped slightly, the very high grade native of the Gribbler zones has been maintained. The potential for further shoots to be discovered at the G2 and G3 level is excellent, thus it may be possible to compensate for the tonnage lost in the G1 zone.

MINING RESERVES

Methods

The mining reserves were done assuming similar mining methods as are presently used on the property. The mining reserve was calculated using the same inventory blocks established for the mineral inventory and assessing each block for its minability, recovery and dilution. The classification of the minable reserves is the same as that determined by the geologists for the corresponding inventory blocks.

Some of the geological inventory was excluded from the mining reserve calculation as it was either, 1) physically inaccessible, 2) of uneconomic grade after dilution, 3) within a permanent pillar, 4) under fill, 5) in the back of an open stope, or 6) uneconomic based on the distance from present or future workings.

The "geologic tonnes used" is that portion of the total geologic tonnage (= mineral inventory) that is minable. The recovery factor is applied to this tonnage.

The recovery is defined as the percentage by volume of the "geologic tonnes used" that is minable. This portion is termed the "Actual Minable Geologic Tonnes" (AMGT).

The dilution is the percentage <u>by volume</u> of waste that must be mined with the "recoverable" geological reserves. As the density of the ore and waste are not identical, the diluted minable tonnes are prorated, based on the specific gravities. The specific gravities used are:

	Ore	Waste
Jewelbox	3.2	2.8
Burnick	3.1	2.8

The dilution is calculated with zero grade. To arrive at the diluted tonnes the following formulae apply:

ACTUAL MINABLE GEOLOGIC TONNES = (GEOLOGIC TONNES USED * RECOVERY)

$$\text{DILUTED TONNES} = \text{AMGT} * \left(1 + \left(\frac{\text{S.G. WASTE}}{\text{S.G. ORE}} * \frac{\text{\% DILUTION}}{100} \right) \right)$$

The correction by the ratio of waste density to ore density is necessary to avoid over calculating the minable tonnes.

Some of the minable blocks included in the minable reserves are not economic on a "stand alone" basis but are dependant on blocks that have enough reserves to warrant development. These blocks are incremental ore.

Jewelbox

The mining methods used in Jewelbox are open mechanized stoping, cut and fill, and a minor amount of longhole around the chimney. The recovery and dilution were done in a volumetric fashion, mainly based on a scaled or visual evaluation of the recovery or dilution.

Based on past experience and the complex nature of the orebody, the maximum recovery for the blocks that are bound by waste is 95%. A reduced recovery is determined based on the additional complexity. The dilution is based on the additional nature of waste that must be added to bring the excavation to the minimum operating width and height parameters.

In a stoping environment the minimum mining width is 3.5m wide by 3.5m high for low profile, and 4.0m by 4.0m in larger excavations where trucks have to enter. If the mining block is to be used as an access the dimensions are 4.25m by 4.25m.

Each block is assessed and the values of the recovery and dilution are assigned to it.

Burnick

The mining methods proposed in Burnick are 15m high open stopes with intermediate sill pillars should the vein be more than 30m in vertical height., The recovery is never assumed to be more than 95% where the block is in contact with waste.

All intermediate sill pillars and crown pillars for the pit are assumed 50% recoverable as they would be blasted longhole.

The dilution is based on the complexity of the ore and the necessary waste that must be mined to attain a mining width of 4.0m.

The open pit recovery assumed that selective blasting would be used to remove the ore first and then the waste. The dilution was influenced by the dip of the vein and the complexity of the ore. The minimum mining width was assumed to be 4.0m.

All mining blocks were assessed individually and given a recovery and dilution.

Gribbler

The minable reserve is derived using recovery and dilution factors determined by C.M.D. Overall recovery is 80% and dilution 10% (by mass, or 12% by volume) by waste of zero grade.

Results

A total of 2.4 million tonnes of diluted minable material averaging 3.5% Pb, 12.0% Zn and 53 g/t Ag has been delineated in the four zones. In order to use the term reserve only type A material may be quoted. The total proven plus probable (type A) minable reserve for the start of 1993 is thus 1.76 million tonnes averaging 3.4% Pb, 12.1% Zn and 52 g/t Ag in the four zones. The start of 1992 minable reserve was 3.317 million tonnes averaging 3.1 % Pb, 11.3 % Zn and 48 g/t Ag. There is thus a decrease of nearly one million tonnes after accounting for 498,360 tonnes mined during 1992. This decrease is largely due to the change in the Burnick inventory (minable decreased by 1.06 million tonnes) with a much smaller decrease in Jewelbox (30,00 tonnes) and a gain of 93,500 tonnes in Gribbler. The increase in grade is due to a slight grade increase at Burnick and to adding the high grade Gribbler ore to the reserve base.

The minable reserve for Type "A" and "B" mineralization for the various Jewelbox Hill ore zones and for Burnick and Gribbler is presented in Table 2 (Page iv). Table 2 also provides a minable reserve for Attila, quoted from the *1990 Kilborn Project Definition Study*. Tables 6 and 7 provide details of the minable reserve derivation for Jewelbox and Burnick by zone and classification.

As with the mineral inventory, it is likely that some of the tonnage lost this year will be regained by further underground drilling.

TABLE 6A
SA DENA HES JOINT VENTURE
JAN. 1, 1993 MINERAL INVENTORY/ORE RESERVES
JEWELBOX HILL – SUMMARY BY ZONE – CATEGORY A – (PROVEN + PROBABLE)

ZONE NAME	Mineral Inventory (Geological tonnes)	Ore Grade			Geologic Tonnes Used	Ore Grade			Mining Recovery (%)	Mining Dilution (vol %)	Diluted Tonnes	Diluted Grade		
		Pb (%)	Zn (%)	Ag (g/t)		Pb (%)	Zn (%)	Ag (g/t)				Pb (%)	Zn (%)	Ag (g/t)
ASHBIN	38,908	9.5	15.6	119.1	36,828	9.4	15.5	109.5	72	23	31,732	7.6	12.6	93.8
CHIMNEY	19,773	8.6	15.6	51.7	19,773	8.6	15.6	51.7	67	22	15,717	7.4	12.9	42.9
FW	112,099	5.7	10.7	58.7	98,174	5.7	10.7	56.5	93	15	103,630	5.1	9.5	50.0
HFW	24,021	4.7	10.4	42.1	23,048	4.8	10.6	43.1	83	26	23,505	3.9	8.6	34.5
J1	13,400	6.2	9.2	30.6	9,520	6.9	9.4	35.5	70	15	7,566	6.2	8.4	32.3
JB1	135,143	6.8	11.0	36.5	119,201	7.2	11.8	38.5	79	19	109,737	6.1	10.1	33.0
JB1–L	27,997	9.1	14.3	42.5	27,052	9.3	14.6	43.3	81	17	25,081	8.2	12.8	38.2
JB1–POD	70,430	9.5	13.8	52.3	34,440	8.6	12.5	48.4	97	14	37,303	7.7	11.1	43.2
JB2–E	115,330	6.8	9.4	86.2	84,880	6.4	9.2	80.6	66	22	67,155	5.5	7.6	70.8
JB2–W	74,059	7.4	13.9	57.1	69,944	7.4	13.8	56.9	75	18	60,473	6.3	11.7	49.5
LOWER MAIN	7,679	7.2	10.0	84.7	7,679	7.2	10.0	84.7	89	19	7,956	6.2	8.5	73.2
MAIN ZONE OP	5,120	17.1	22.7	81.9	0	0.0	0.0	0.0	0	0	0	0.0	0.0	0.0
MAIN ZONE UG	8,707	9.9	13.6	55.6	8,707	9.9	13.6	55.6	80	15	7,880	8.8	12.0	49.1
TOTAL	652,666	7.3	11.9	60.0	539,246	7.2	11.9	58.1	79	18	497,735	6.1	10.2	49.8

TABLE 6B
SA DENA HES JOINT VENTURE
JAN. 1, 1993 MINERAL INVENTORY/ORE RESERVES
JEWELBOX HILL – SUMMARY BY ZONE – CATEGORY B – (POSSIBLE)

ZONE NAME	Mineral Inventory (Geological tonnes)	Ore Grade Pb (%)	Ore Grade Zn (%)	Ore Grade Ag (g/t)	Geologic Tonnes Used	Ore Grade Pb (%)	Ore Grade Zn (%)	Ore Grade Ag (g/t)	Mining Recovery (%)	Mining Dilution (vol %)	Diluted Tonnes	Diluted Grade Pb (%)	Diluted Grade Zn (%)	Diluted Grade Ag (g/t)
ASHBIN	12957	10.3	16.2	77.1	7997	11.6	17.5	60.1	89	38	9,430	8.8	13.2	45.5
FW	22651	5.4	9.4	47	16923	5.4	9.6	45.1	84	19	16,550	4.7	8.3	38.6
HFW	29544	9.8	14.4	49.8	29544	9.8	14.4	49.8	61	30	22,683	7.8	11.4	39.5
J1	4,593	6.4	10.0	27.7	0	0.0	0.0	0.0	0	0	0	0.0	0.0	0.0
JB1	22,433	8.5	11.9	43.4	22,433	8.5	11.9	43.4	70	31	20,055	6.6	9.1	34.5
JB1-L	7,765	10.7	14.9	54.8	7,765	10.7	14.9	54.8	72	20	6,617	9.1	12.5	46.2
JB2-E	12,421	9.7	8.6	146.5	7,829	9.3	8.3	120.8	84	46	9,203	6.8	5.8	87.9
JB2-W	24,394	6.5	10.7	53.2	23,229	6.6	10.8	54.0	80	31	23,476	5.0	8.3	41.2
LOWER MAIN	57,806	6.5	9.1	69.6	51,034	7.0	9.4	70.6	43	30	27,472	5.8	7.6	55.4
M	18,343	8.8	13.8	43.9	13,820	9.0	14.3	44.3	81	22	13,297	7.5	11.9	36.8
UFW	12,320	7.5	13.9	42.6	12,320	7.5	13.9	42.6	80	20	11,581	6.4	11.8	36.2
UNDEFINED	44,206	5.1	8.4	66.7	19,488	4.4	8.7	44.5	83	31	20,519	3.4	6.9	35.1
TOTAL	269,433	7.4	11.0	61.2	212,382	7.7	11.5	56.1	68	29	180,883	6.2	9.3	43.6

TABLE 6C
SA DENA HES JOINT VENTURE
JAN. 1, 1993 MINERAL INVENTORY/ORE RESERVES
JEWELBOX HILL – SUMMARY BY ZONE – CATEGORY A +B – (PROVEN + PROBABLE + POSSIBLE)

ZONE NAME	Mineral Inventory (Geological tonnes)	Ore Grade Pb (%)	Ore Grade Zn (%)	Ore Grade Ag (g/t)	Geologic Tonnes Used	Ore Grade Pb (%)	Ore Grade Zn (%)	Ore Grade Ag (g/t)	Mining Recovery (%)	Mining Dilution (vol %)	Diluted Tonnes	Diluted Grade Pb (%)	Diluted Grade Zn (%)	Diluted Grade Ag (g/t)
ASHBIN	51,865	9.7	15.7	108.6	44,825	9.8	15.9	100.7	75	26	41,162	7.9	12.7	82.7
CHIMNEY	19,773	8.6	15.6	51.7	19,773	8.6	15.6	51.7	67	22	15,717	7.4	12.9	42.9
FW	134,750	5.6	10.5	56.7	115,097	5.7	10.5	54.8	92	16	120,180	5.0	9.3	48.4
HFW	53,565	7.5	12.6	46.3	52,592	7.6	12.7	46.9	71	28	46,188	5.8	10.0	37.0
J1	17,993	6.3	9.4	29.9	9,520	6.9	9.4	35.5	70	15	7,566	6.2	8.4	32.3
JB1	157,576	7.0	11.1	37.5	141,634	7.4	11.8	39.3	78	21	129,792	6.2	9.9	33.2
JB1–L	35,762	9.4	14.4	45.2	34,817	9.6	14.7	45.9	79	18	31,698	8.4	12.7	39.9
JB1–POD	70,430	9.5	13.8	52.3	34,440	8.6	12.5	48.4	97	14	37,303	7.7	11.1	43.2
JB2–E	127,751	7.1	9.3	92.1	92,709	6.6	9.1	84.0	68	25	76,358	5.7	7.4	72.9
JB2–W	98,453	7.2	13.1	56.1	93,173	7.2	13.1	56.2	76	21	83,949	5.9	10.7	47.2
LOWER MAIN	65,485	6.6	9.2	71.4	58,713	7.0	9.5	72.4	49	27	35,428	5.9	7.8	59.4
M	18,343	8.8	13.8	43.9	13,820	9.0	14.3	44.3	81	22	13,297	7.5	11.9	36.8
UFW	12,320	7.5	13.9	42.6	12,320	7.5	13.9	42.6	80	20	11,581	6.4	11.8	36.2
UNDEFINED	44,206	5.1	8.4	66.7	19,488	4.4	8.7	44.5	83	31	20,519	3.4	6.9	35.1
MAIN ZONE OP	5,120	17.1	22.7	81.9	0	0.0	0.0	0.0	0	0	0	0.0	0.0	0.0
MAIN ZONE UG	8,707	9.9	13.6	55.6	8,707	9.9	13.6	55.6	80	15	7,880	8.8	12.0	49.1
TOTAL	922,099	7.3	11.6	60.4	751,628	7.3	11.8	57.5	76	21	678,618	6.1	10.0	48.2

TABLE 7
SA DENA HES JOINT VENTURE
JAN. 1, 1993 MINERAL INVENTORY/ORE RESERVES
BURNICK ZONE – SUMMARY BY ZONE – CATEGORY A – (PROVEN + PROBABLE)

ZONE NAME	Mineral Inventory (Geological tonnes)	Ore Grade Pb (%)	Ore Grade Zn (%)	Ore Grade Ag (g/t)	Geologic Tonnes Used	Ore Grade Pb (%)	Ore Grade Zn (%)	Ore Grade Ag (g/t)	Mining Recovery (%)	Mining Dilution (vol %)	Diluted Tonnes	Diluted Grade Pb (%)	Diluted Grade Zn (%)	Diluted Grade Ag (g/t)
CATEGORY A														
OPEN PIT	165,106	0.4	12.6	45.8	139,802	0.4	13.2	50.3	92	9	138,442	0.4	12.3	47.3
UNDERGROUND	845,858	1.2	13.5	45.3	713,690	1.3	13.8	46.3	81	12	640,527	1.1	12.4	42.4
TOTAL	1,010,964	1.1	13.4	45.4	853,492	1.2	13.7	47.0	83	11	778,969	1.0	12.4	43.3
CATEGORY B														
OPEN PIT	9,432	3.1	8.1	69.7	9,432	3.1	8.1	69.7	72	15	7,662	2.7	7.2	61.6
	8,594	0.1	13.7	22.3	7,835	0.1	14.1	22.5	88	6	7,287	0.1	13.4	21.3
UNDERGROUND	574,089	0.5	12.9	48.6	330,724	0.4	13.3	40.3	79	14	295,607	0.3	11.8	36.4
TOTAL	592,115	0.5	12.8	48.6	347,991	0.5	13.2	40.7	79	14	310,556	0.4	11.7	36.7
CATEGORY A+B														
OPEN PIT	183,132	0.5	12.4	45.9	157,069	0.5	12.9	50.1	91	9	153,391	0.5	12.1	46.8
UNDERGROUND	1,419,947	0.9	13.3	46.6	1,044,414	1.0	13.6	44.4	80	13	936,134	0.8	12.2	40.5
TOTAL	1,603,079	0.9	13.2	46.6	1,201,483	1.0	13.5	45.1	82	12	1,089,525	0.8	12.2	41.4

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Curragh Inc.

SÄ DENA HES JOINT VENTURE

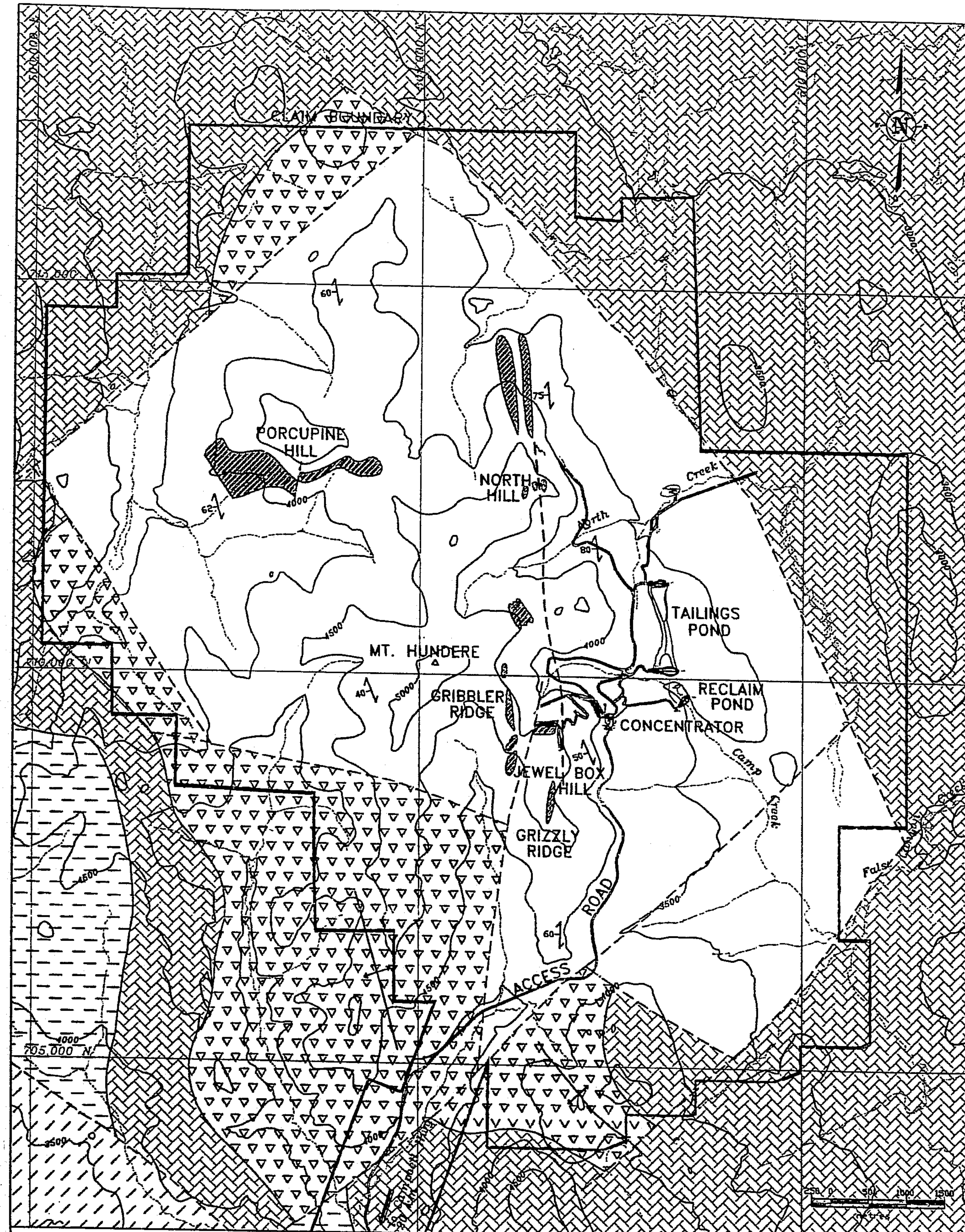
EXPLORATION POTENTIAL

The Sä Dena Hes property covers 14,300 hectares, only a small part of which has been well explored. The skarn ores at Sä Dena Hes are hosted by a stratigraphic package very similar in age and style to that of the Anvil District. As in the Anvil District, deformation is complex and late-stage post metamorphic faulting was significant. Ore occurs as pods of skarn containing coarsely crystalline galena and sphalerite along the margins of the lower Cambrian limestone. Skarn formation is particularly well developed near north-south or east-west trending steeply dipping faults, and especially near their intersections. Intrusive rocks are very limited on the property although they are thought to be present at depth based on geophysics or regional geologic grounds. There is, thus, little association between skarn formation and intrusive contacts. The hot ore forming fluids hornfelsed the meta-sedimentary phyllites leaving a clear indication of the past working of the ore-forming system. Ore formation, as opposed to skarn formation, appears to be late in the paragenetic sequence and related to the waning retrograde stages of the hydrothermal system.

Exploration in this district is oriented toward areas of strong alteration where limestone may occur in the sub-surface. There are no diagnostic geophysical methods applicable to these skarns, thus discovery depends on geochemistry for near-surface mineralization, or, more reliably, on drilling. Only drilling is reliable for deeper ore. Currently there are no known mineralogical or geochemical guides to ore, however, further research may all such exploration tools to be developed.

Surficial exploration on the property is incomplete but geologic and soil geochemical surveys have outlined a number of interesting alteration zones and geochemical anomalies in addition to those associated with the known ore deposits. Currently diamond drilling has been primarily in the immediate vicinity of the four known ore zones. There is relatively little drilling between them or on the several additional alteration zones on the claims. There are now a large number of targets to be drill tested, and the potential for discovery of new ore zones is excellent.

Persistent exploration drilling around the periphery of Jewelbox Hill has already resulted in the discovery of additional ore and it is likely that more will be found there. Further drilling around the other zones will likely have similar success. There are particularly attractive opportunities between, and to the west of, Burnick and Attila on North Hill which will be tested by an extension of the newly developed Burnick underground workings. Gribbler Ridge also offers good potential for discovery of new ore shoots by continuing the work done there in 1992.



Fault

Layering

Lower Cambrian: limestone

Cambrian-Ordovician: phyllite

Silurian: siltstone, dolomite, quartzite

Devonian-Mississippian: shale, chert, conglomerate

Permian-Pennsylvanian: chert, argillite

Triassic: shale

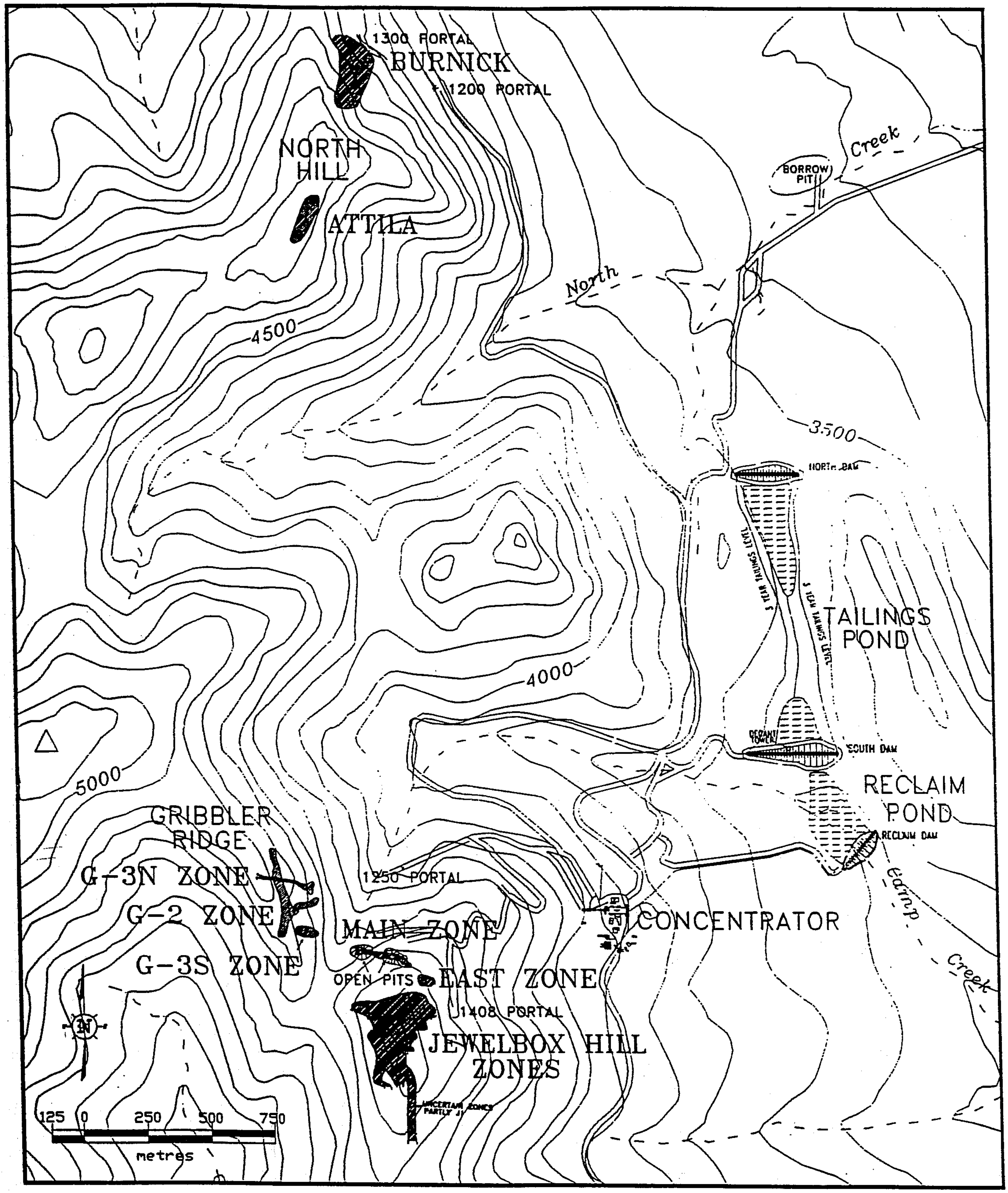


Fig. 2: Sa' Dena Hes mine site, showing location of ore zones relative to the Jewelbox Hill deposit.

Table 1

Sa Dena Hes Geological Inventory as of start of Second Quarter 1993

Total in-situ mineralization, no adjustments for mining recovery or dilution

Deposit Name	Cutoff % Pb+Zn	Classification	Tonnage (tonnes)			Zn (%)	Pb (%)	Ag (g/t)
Jewelbox Hill	8	Prov,Prob.+Poss	922,099			11.6	7.3	60.0
North Hill, Burnick Zone	8	Prob. + Poss.	1,603,079			13.1	0.9	47.0
North Hill, Attila Zone	8	Probable	406,000			12.4	3.4	62.0
Gribbler Ridge	8	Prob. + Poss.	273,113			16.2	8.4	103.0
Stockpiles	8	Proven	26,186			7.5	2.7	42.5
Total Defined Inventory			**3,230,477**			**12.8**	**3.7**	**57.3**
Additional Potential in areas of known deposits and ii areas of known anomalies			Low End Potential (tonnes)		High End Potential (tonnes)	Zn (%)	Pb (%)	Ag (g/t)
Jewelbox Hill	NA	Potential	50,000	to	200,000	10.0	6.0	50.0
North Hill, Burnick Zone	NA	Potential	25,000	to	250,000	12.0	1.0	40.0
North Hill, Burnick – Attila Gap	NA	Potential	50,000	to	300,000	10.0	5.0	50.0
North Hill, Attila Zone	NA	Potential	0	to	200,000	11.0	3.0	60.0
Gribbler Ridge	NA	Potential	25,000	to	150,000	15.0	7.0	90.0
Grizzly Ridge	NA	Potential	0	to	250,000	10.0	5.0	50.0
Porcupine Hill	NA	Potential	0	to	250,000	10.0	5.0	50.0
Total Potential Mineralization in known anomalous areas			**150,000**	to	**1,600,000**			
Grand Total Inventory and Potential					3,380,477	12.7	3.7	57.2
					to			
					4,830,477	12.2	3.9	56.0

Table 2
Start of 2nd Quarter Reserves and Minable Inventory
SA DENA HES DIVISION

Start of 1993 and Start of 2nd Quarter, all at a 8% Pb+Zn cutoff over minimum 2.2 m.

	Classification	(metric tonnes)	zinc (%)	lead (%)	silver* (g/t)	gold (g/t)	total zinc (million lbs.)	total lead (million lbs.)	total silver (thousand oz.)	total gold (thousand oz.)
Jewel Box Hill	Prov. + Prob.	497,735	10.20	6.10	50.0		111.926	66.936	800.084	
Burnick	Prov. + Prob.	778,969	12.40	1.00	43.0		212.949	17.173	1,076.852	
Gribbler	Probable	93,560	15.90	7.40	97.0		32.796	15.264	291.762	
Attila	Probable	386,300	11.50	3.10	60.0		97.939	26.401	745.150	
Stockpile Total	Proven	26,186	7.50	2.68	42.5		4.332	1.549	35.772	
Jewel Box Hill	Possible	180,883	9.3	6.2	44		37.086	24.724	255.869	
Burnick	Possible	310,556	11.7	0.4	37		80.105	2.739	369.410	
Gribbler	Possible	145,900	14	7.7	92		45.032	24.767	431.530	
Attila	Possible	0	0	0	0		0.000	0.000	0.000	
Total Minable Reserve	Prov. + Prob.	**1,782,750**	**11.70**	**3.24**	**51.5**		459.942	127.323	2,949.620	
Total Defined Minable Mineralization	Prov.,Prob.&Poss	2,420,089	11.66	3.37	51.5		622.165	179.553	4,006.429	

* s/p Ag is est.

Stockpiles:

	Classification	(metric tonnes)	zinc (%)	lead (%)	silver (g/t)	gold (g/t)	total zinc (million lbs.)	total lead (million lbs.)	total silver (thousand oz.)	total gold (thousand oz.)
UG broken ore		6,549	8.2	4.3	45		1.184	0.621	9.474	
Portal s/p		10,407	4.5	2.3	40		1.032	0.528	13.383	
Crusher s/p		3,000	10.8	4.7	50		0.714	0.311	4.822	
Fine ore		500	6.8	5.8	45		0.075	0.064	0.723	
Jewel Box S/P Total		20,456	6.7	3.4	43.2		3.006	1.523	28.403	
Burnick		5,730	10.5	0.2	40		1.326	0.025	7.369	
Stockpile Total		26,186	7.5	2.7	42.5		4.332	1.549	35.772	

6

CIRQUE CLAIMS POTENTIAL

Discovery of the North and South Cirque deposits supports the geologic modelling of several depositional sub-basins on the Cirque property within which barite-sulphides were deposited. Exploration to date has not fully tested mineralization potential of these sub-basins along the North Cirque-South Cirque trend within favourable Devonian-Mississippian strata.

The area immediately northwest and southeast of the North Cirque deposit has not been drill tested. In the R-Creek area, northwest of North Cirque, only the down-dip potential of the favourable strata has been explored; the area toward the north in Cirque Valley which is more directly along trend of the North Cirque deposit and in the same thrust panel still contains exploration potential.

Potential for additional mineralization at the South Cirque deposit lies updip to the northeast as supported by lead-zinc ratios and trends of the ore-host rock facies. Narrow, very high grade sulphide mineralization associated with siltstone breccias have been intersected in drillholes southwest of the South Cirque deposit. The significance of the intersections has not been worked out nor has the area further to the southwest been drill tested. Further drilling in this area is needed.

In the Paul River valley, the siliceous Gunsteel formation, host to the North and South Cirque deposits, has been mapped into areas of increasing overburden. Similar to the area on the north end of the Cirque claims, this favourable geology has not been fully tested for stratiform mineralization.

FLUKE CLAIMS POTENTIAL

The 1980 drilling program failed to located any significant mineralization down-dip of the discovery showing on the Fluke claims. A continuing program of detailed mapping in 1981 located a barite-sulphide horizon within a structural panel of host rock associated with semi-continuous lead-barium soil anomalies. Soil geochemical anomalies extend for at least 1 kilometre beyond the area which as been partially drill-tested.

During the 1982 field season, one drill hole was completed on the Fluke property prior to mobilization of the drill to the South Cirque area. The drill hole intersected sulphide mineralization with sporadic lead-zinc values roughly 200 metres down-dip from the surface baritic mineralization. This mineralization, consisting of narrow laminated pyrite horizons with minor laminae of sphalerite and galena occurs throughout a 20-metre section of siltstone breccia.

Although the mineralization is low grade, the style of mineralization is identical to the lithological character and geological setting of the mineralization in the South Cirque area. Indication of sub-basin development accompanied by significant mineralization of the Fluke supports existence of another potential deposit in the district.

ELF CLAIMS POTENTIAL

Narrow, high grade barite-galena-sphalerite mineralization occurs on a steeply dipping overturned fold limb on the Elf claims. Mineralization has been traced by deep drilling for 800 metres along strike and 600 metres down-dip. Tonnage potential in the showing area occurs within a sub-horizontal, south raking target zone located between surface and a depth of 300 metres. The best intersection to date, occurring 300 metres down-dip from the showing, contains 13.8% combined lead-zinc with 27 grams per tonnes silver over 11 meters. Continued drilling of short sub-horizontal holes is required to test the tonnage potential of this attractive mineralization.

Potential for additional deposits is excellent, as only a portion of an eight kilometre horizon of favourable host rock, with associated lead-zinc anomalies, has been tested. Although there is very limited outcrop, surface exposures are identical to the host units at Cirque that subcrop updip from the barite-sulphide deposits and mineralization intersected on the property is similar to intersections at the south end of the North Cirque-South Cirque trend.

FLUKE CLAIMS

The Fluke claims are located 16 km. southeast of the Cirque deposit. The property is 4,000 hectares in area.

A small number of drill holes on Fluke were completed from 1980-1982. These holes have partially tested two barite-sulphides showing on the property. The following table summarizes drilling to date.

PROPERTY	NO. OF DRILL HOLES	METRES
Fluke	7	3295.4

The most recent drill hole intersected sulphide mineralization with sporadic lead-zinc values roughly 200 m down-dip from a surface baritic showing. This mineralization, consisting of narrow laminated pyrite horizons with minor laminae of sphalerite and galena occurs throughout a 20 m section of siltsone breccia. Although the mineralization is low grade, the style of mineralization is identical to the lithological character and geological setting of the mineralization in the South Cirque area.

All drillhole collars were visually located on detailed orthophoto topographic maps. Downhole deviations have been measured at regualr intervals down the hole using a Sperry-Sun single shot camera instrument. Core for mineralized intersections is stored in Fort St John. Unmineralized core is stored in racks on the property. Surplus assay sample material is stored at KRAL in Kamloops, British Columbia.

In addition to drilling, gridded soil geochemical sampling and detailed geology mapping at a scale of 1:2000 have been completed on the property. The mapping and sampling programs have outlined an extensive zone of favourable stratigraphy with associated geochemical anomalies. This zone has only been partially tested in one location by drilling. Significant exploration potential remains for the property. Expenditures to date are $690,000.

- ELF CLAIMS

The Elf claims cover 3,200 hectares, 35 km southeast of Cirque. Mineralization on the Elf claims is stratiform and, like Cirque, is hosted by Devonian sedimentary rocks.

Limited drilling from 1979-1981 was largely directed towards testing the downdip and along strike extensions of a thin, high-grade sulphides-barite surface showing.

The drilling has traced the mineralization for 800 m along strike and 600 m downdip. The best intersection to date, occurring 300 m downdip from the surface showing, contains 13.8% (Pb+Zn) with 27 g/tonne Ag over 11 m. The mineralization remains untested from the surface showing to a depth of 100 m. The following table summarizes drilling to date.

PROPERTY	NO. OF DRILL HOLES	METRES
Elf	26	10445.9

All drill hole collars were carefully visually located on detailed orthophoto topographic maps. Downhole deviations were measured at regular intervals down the drillhole using a Sperry-Sun single shot camera instrument. Core for mineralized intersections is stored in Fort St John. Unmineralized core is stored in racks on the property. Surplus assay sample material is stored at KRAL in Kamloops, British Columbia.

Gridded soil geochemical sampling and geologic mapping at a scale of 1:5000 have outlined an 8 kilometre long horizon of favourable host rock with associated lead-zinc geochemical soil anomalies. This strike length represents a significant exploration target which has not been tested on the Elf claims. Total expenditures to date for exploration and drill are $2,200,000.

7

From:

A report summarizing the 1989 - 1991 Advanced Exploration Program and the Geological Reserve Calculation for the North Cirque Deposit, British Columbia

STRONSAY CORPORATION

May 1991

Report #WH9102

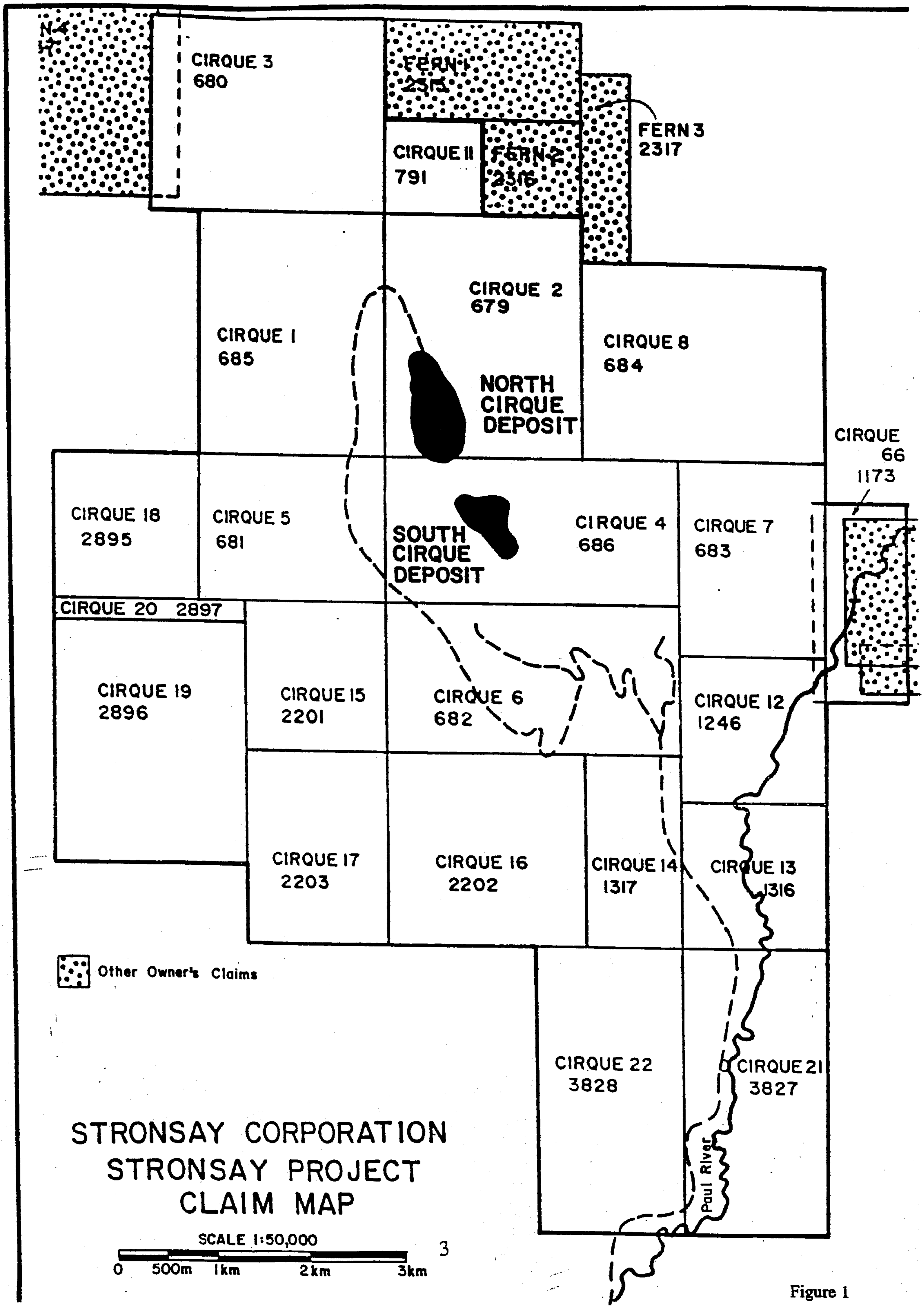
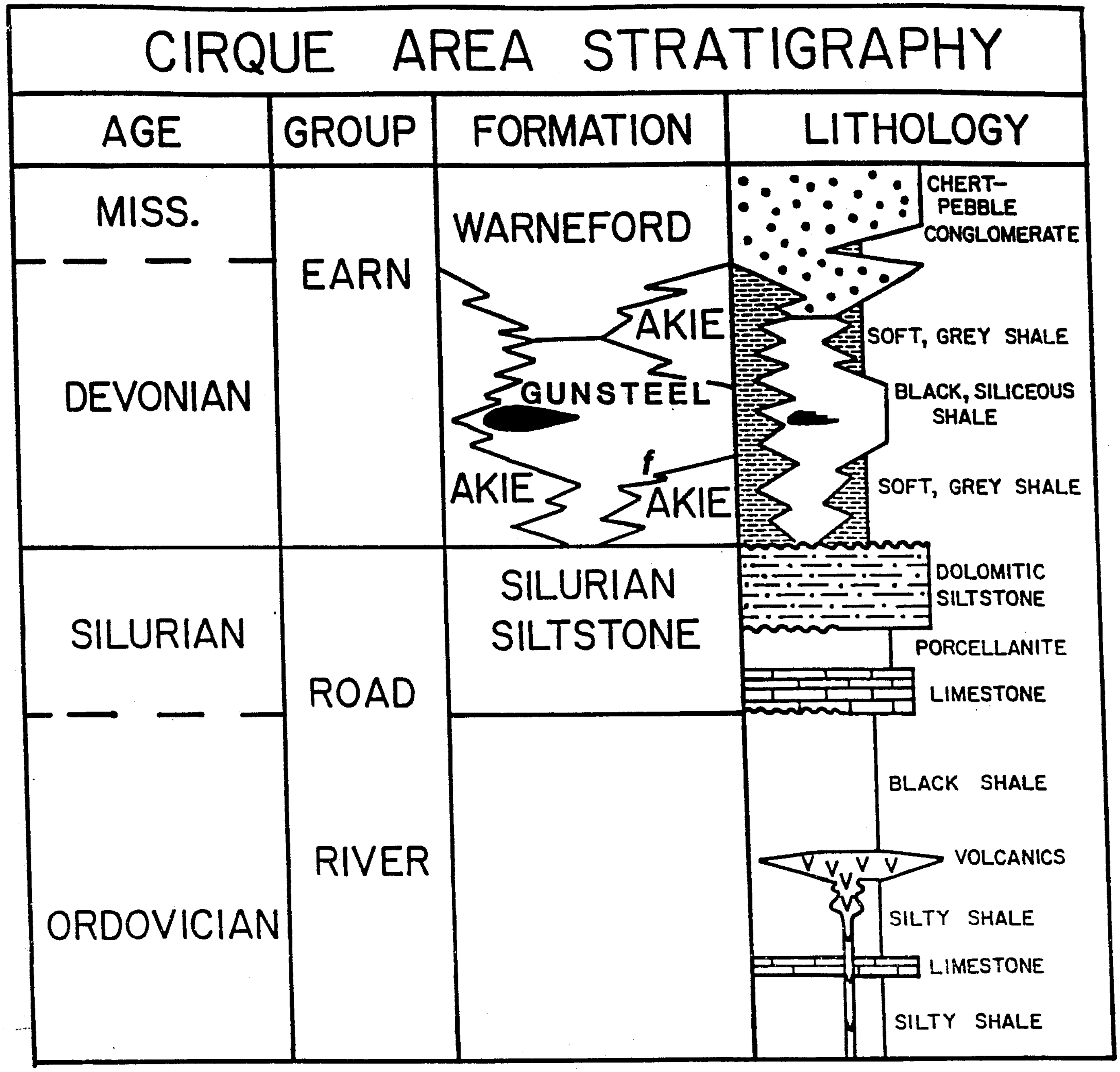
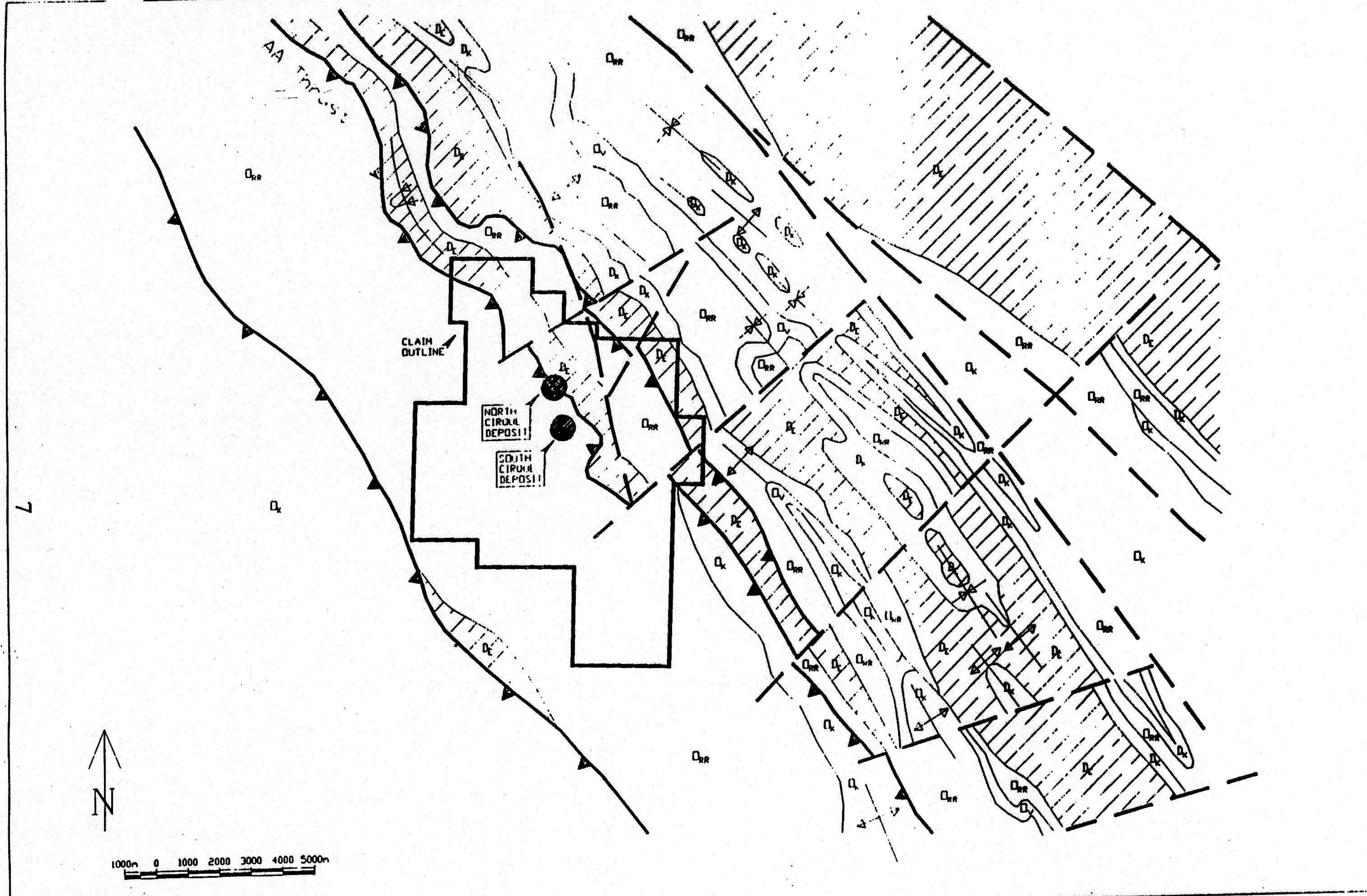

CIRQUE 3
680
FERN 1
2315
FERN 3
2317
CIRQUE 11
791
FERN 2
2316
CIRQUE 2
679
CIRQUE 1
685
CIRQUE 8
684
NORTH CIRQUE DEPOSIT
CIRQUE 66
1173
CIRQUE 18
2895
CIRQUE 5
681
SOUTH CIRQUE DEPOSIT
CIRQUE 4
686
CIRQUE 7
683
CIRQUE 20 2897
CIRQUE 19
2896
CIRQUE 15
2201
CIRQUE 6
682
CIRQUE 12
1246
CIRQUE 17
2203
CIRQUE 16
2202
CIRQUE 14
1317
CIRQUE 13
1316
Other Owner's Claims
CIRQUE 22
3828
CIRQUE 21
3827
Paul River
STRONSAY CORPORATION
STRONSAY PROJECT
CLAIM MAP
SCALE 1:50,000
0 500m 1km 2km 3km

Figure 1



Stratigraphic section of the Cirque Property vicinity after Jefferson et. al. (1983)

Figure 3



ROCK TYPES

- D_E Earn Group — black shale, chert, chert congl
- D_K Kwadacha reef — limestone
- O_{RR} Road River Group — calc black shale, limestone, chert, siltstone
- O_V Ospika volcanics — basalt
- O_K Kechika Group — calc shale

REVISIONS

Curragh Resources Inc.

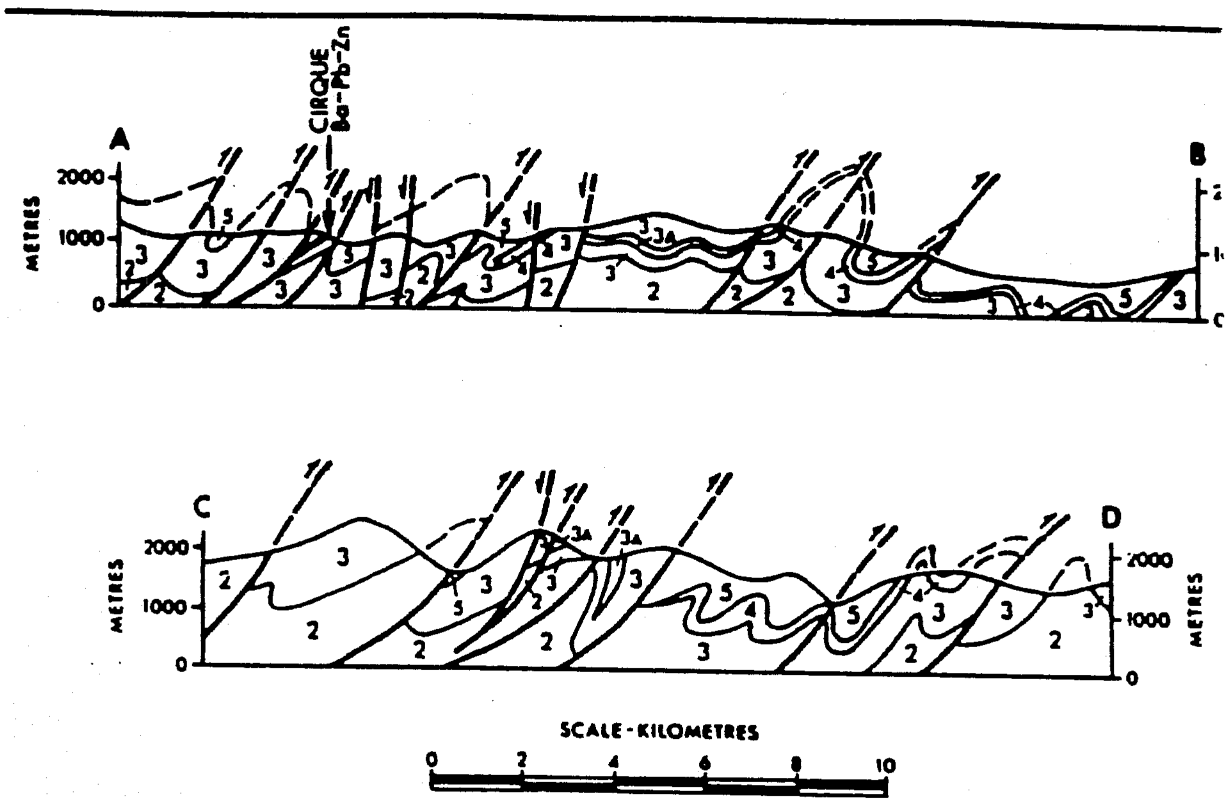
CIRQUE PROJECT

Geology of Cirque Claims

Figure: 4

Date 31/12/1990 Geologist L Pigage

Drawing No:



LEGEND

TRIASSIC

| 6 | DOLOMITIC SILTSTONE, LIMESTONE |

DEVONIAN

| 5 | BLACK SHALE, SILTY SHALE, SILTSTONE, MINOR SANDSTONE, CONGLOMERATE |

| 4 | LIMESTONE, LIMESTONE TURBIDITES, QUARTZ SANDSTONE |

MIDDLE ORDOVICIAN – SILURIAN

| 3 | DOLOMITIC SILTSTONE, BLACK GRAPTOLITIC SHALE, CALCAREOUS SILTY SHALE, LIMESTONE TURBIDITES; 3A VOLCANIC ROCKS |

CAMBRIAN – LOWER ORDOVICIAN

| 2 | NODULAR PHYLLITIC MUDSTONE, LIMESTONE QUARTZITE |

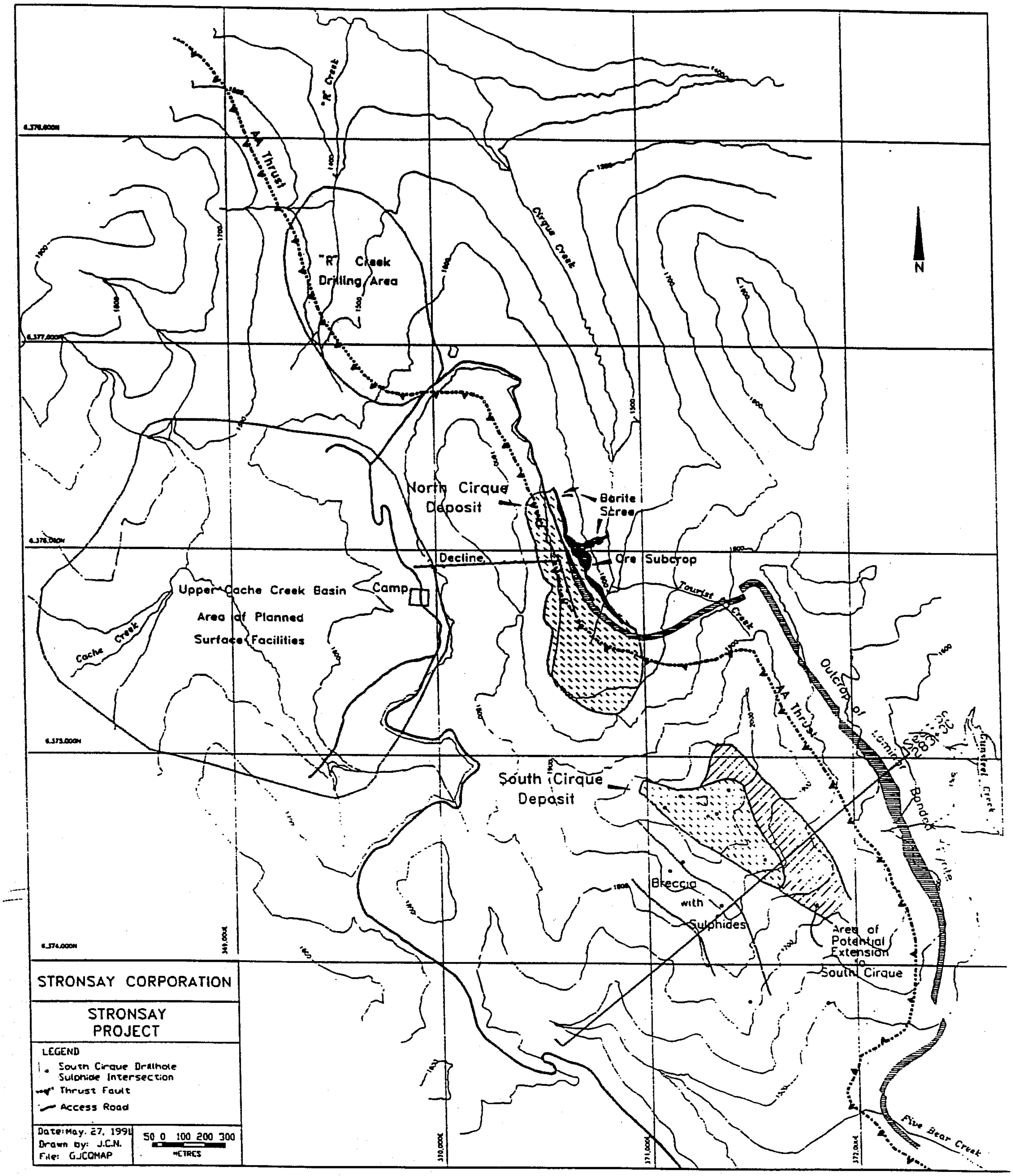
PROTEROZOIC

| 1 | PHYLLITE, SCHIST |

SYMBOLS

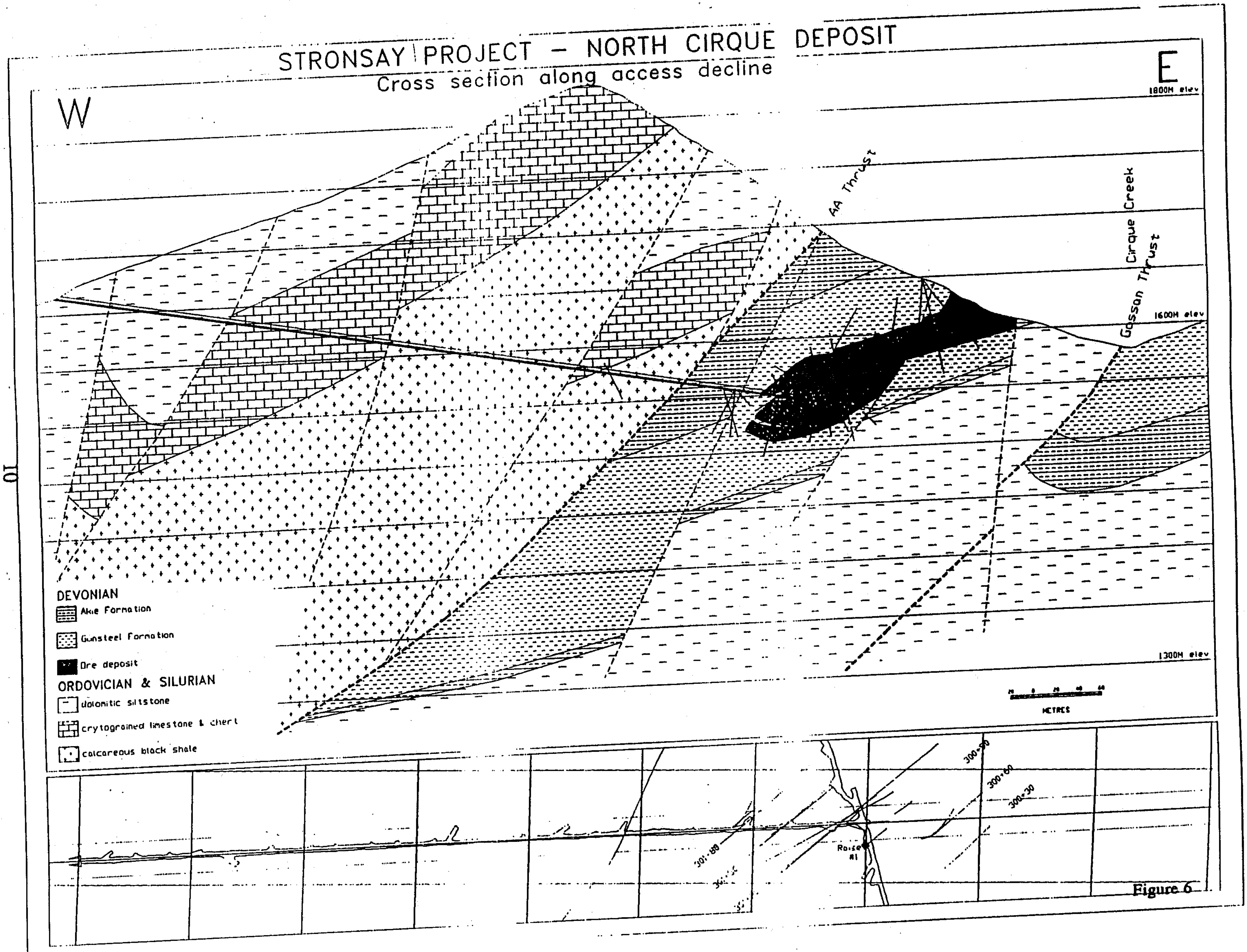
THRUST FAULT . . . .

SHALE-HOSTED LEAD-ZINC . . . .



Location of the North Cirque and South Cirque deposits in relation to the area of minesite surface development. The deposit outline shown is the vertical project of the approximate 2m isopach. See Figure 6 for a section along the decline and Figure 52 for section 283+00 through South Cirque.

Figure 5



STRONSAY PROJECT – NORTH CIRQUE DEPOSIT
Cross section along access decline
W
E
1800M elev
1600M elev
1300M elev
AA Thrust
Cirque Creek
Gossan Thrust
DEVONIAN
Akie Formation
Gunsteel Formation
Ore deposit
ORDOVICIAN & SILURIAN
dolomitic siltstone
crytograined limestone & chert
calcareous black shale
METRES
300+90
300+60
300+30
Raise #1

Figure 6

STRONSAY PROJECT
SUMMARY FACTS SHEET

GEOLOGY

North Cirque	Continuous wedge shaped, bedded barite sulphide lens approximately concordant with Devonian siliceous black shale host sediments; 1000 m n-s, 400 m e-w, 2-70 m thick; dips 30° to 55° southwest; contains all presently known mineable reserves.
South Cirque	500 m south of North Cirque, extent has not been fully defined. Not included in mineable reserves.
Mineralization	Sphalerite, galena, pyrite and barite; banded, varying from thin laminations to broad layering.
Ore Types (3)	Based on % barite: Massive sulphide (barite <20%); sulphide with barite (20%<barite<60%); barite with sulphide (barite >60%).
Specific Gravity	Based on pulps with allowance for porosity: Massive sulphide 4.41 Sulphide with barite 4.35 Barite with sulphide 4.20

Pb+Zn% Comparisons for Lithology 1, 4 & 5

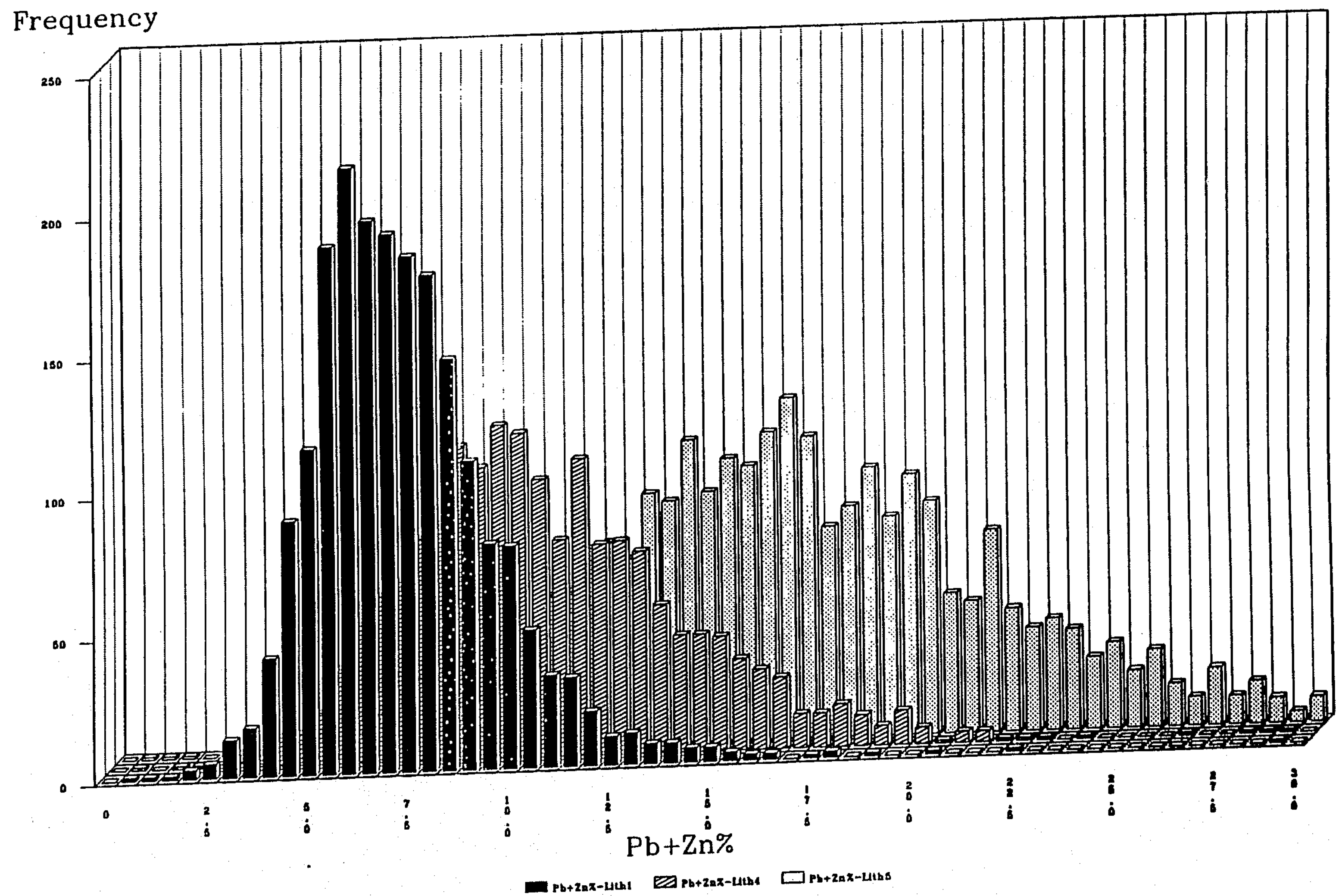
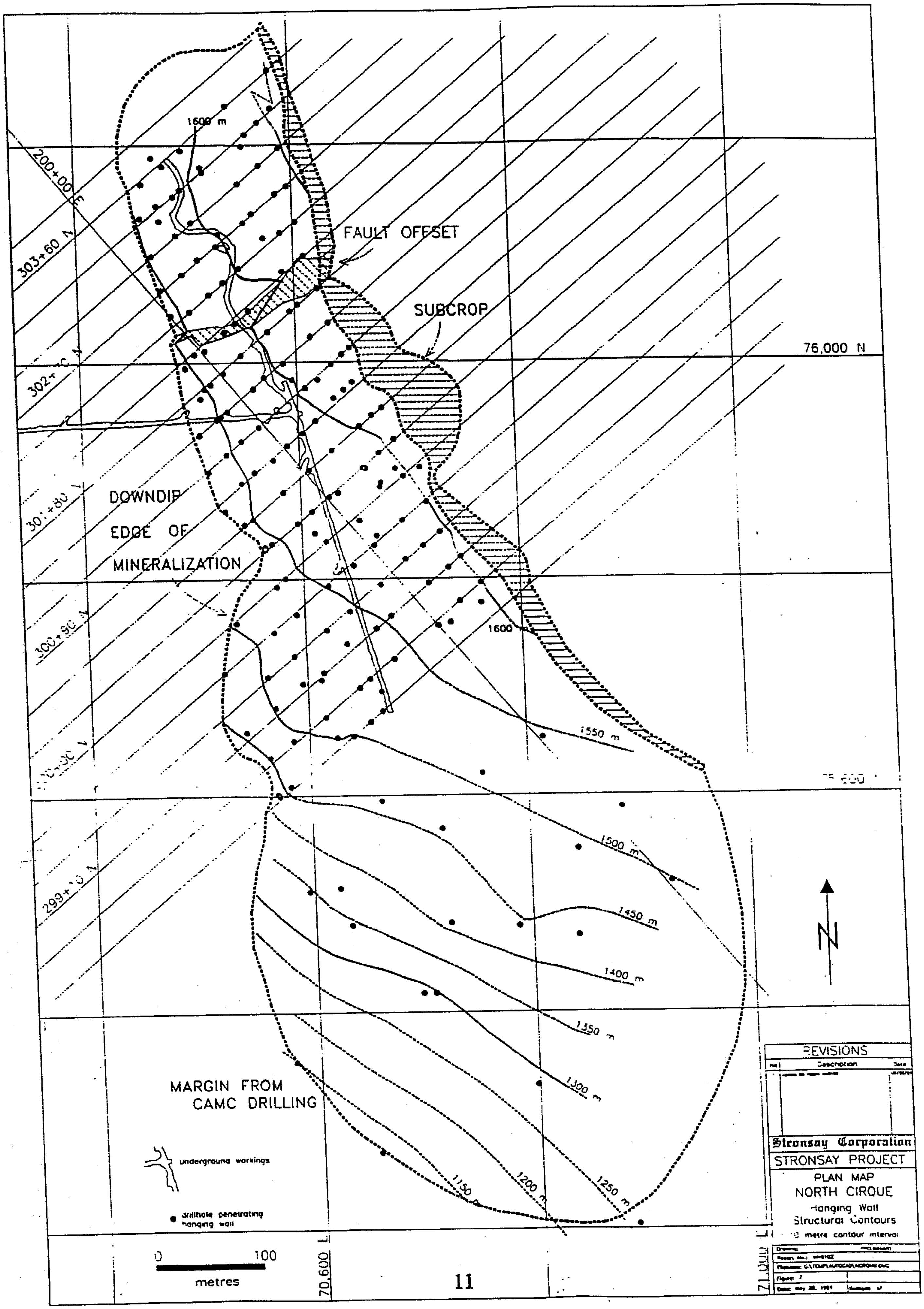


Figure 8


1600 m
200+00 E
303+60 N
302+
FAULT OFFSET
SUBCROP
76,000 N
DOWNDIP
EDGE OF
MINERALIZATION
300+90
1600
1550 m
1500 m
1450 m
1400 m
1350 m
1300 m
1250 m
1200 m
1150
MARGIN FROM
CAMC DRILLING
underground workings
drillhole penetrating
hanging wall
0
100
metres
70,600
REVISIONS
Description
Date
Stronsay Corporation
STRONSAY PROJECT
PLAN MAP
NORTH CIRQUE
Hanging Wall
Structural Contours
metre contour interval
Drawing:
Figure:
Date:

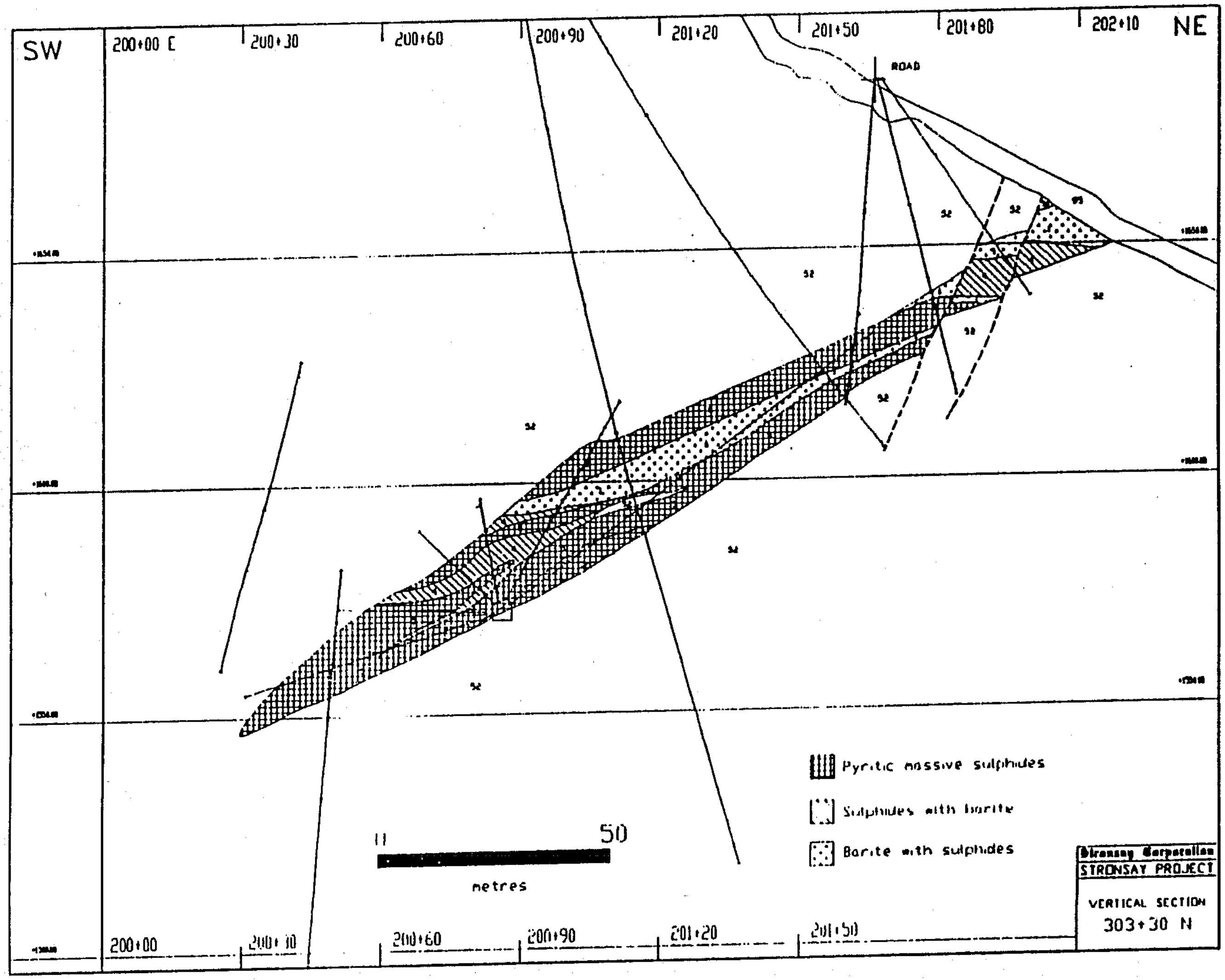
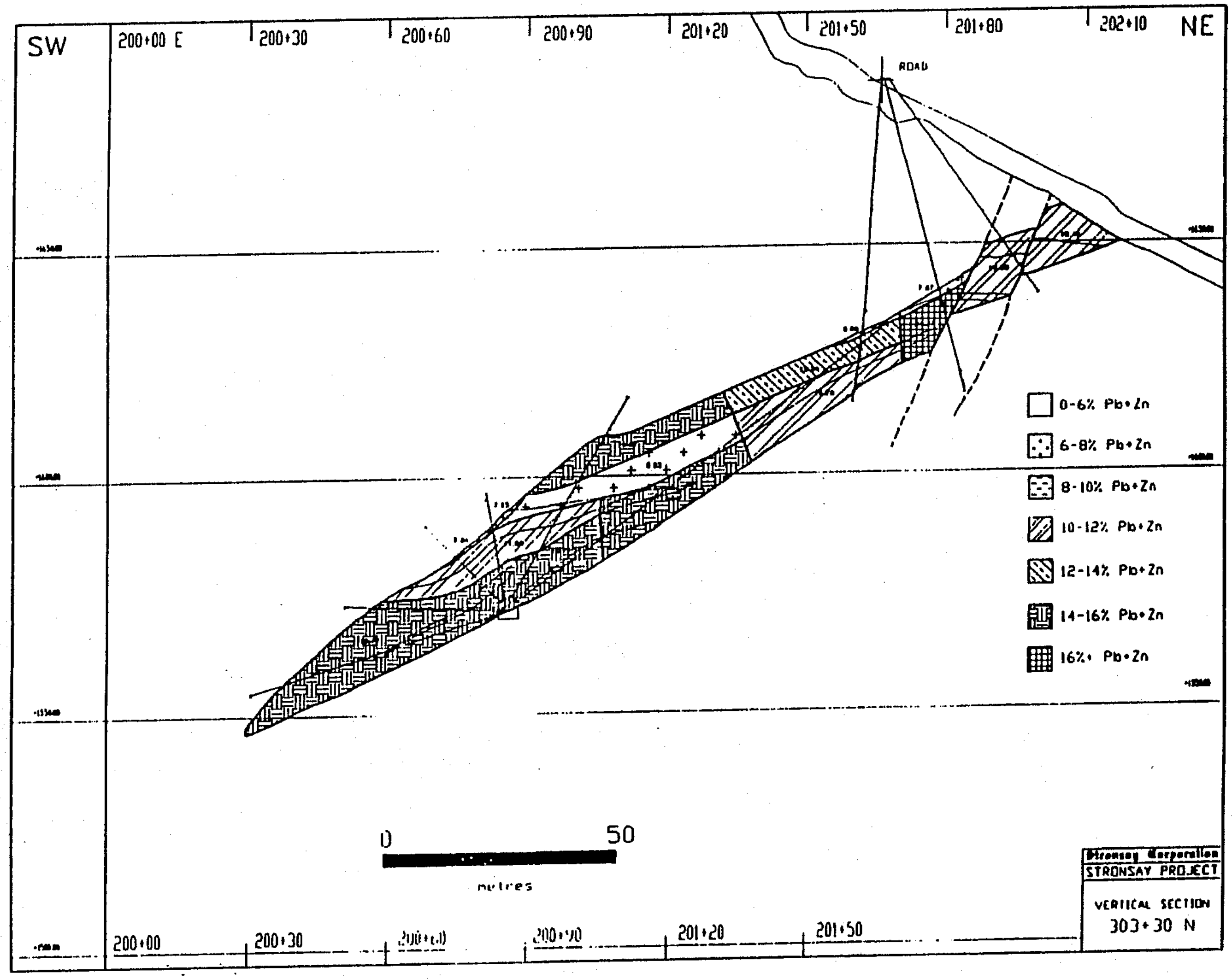

SW
NE
200+00 E
200+30
200+60
200+90
201+20
201+50
201+80
202+10
ROAD
Pyritic massive sulphides
Sulphides with barite
Barite with sulphides
50
metres
STRONSAY PROJECT
VERTICAL SECTION
303+30 N
200+00
200+30
200+60
200+90
201+20
201+50

Figure 27



SW
NE
200+00 E
200+30
200+60
200+90
201+20
201+50
201+80
202+10
ROAD
0-6% Pb+Zn
6-8% Pb+Zn
8-10% Pb+Zn
10-12% Pb+Zn
12-14% Pb+Zn
14-16% Pb+Zn
16%+ Pb+Zn
0
50
metres
Stronsay Corporation
STRONSAY PROJECT
VERTICAL SECTION
303+30 N
200+00
200+30
201+20
201+50

Figure 44

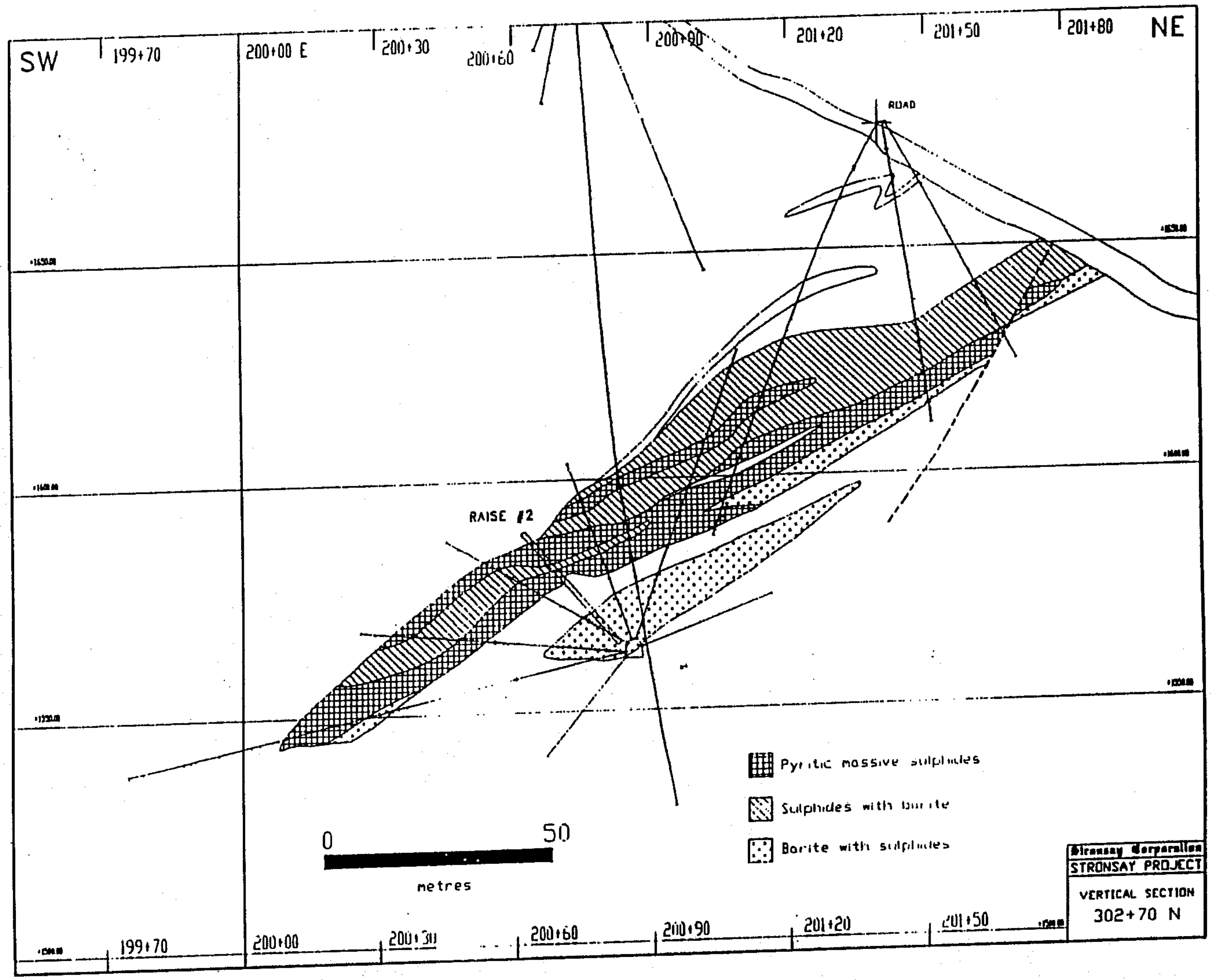
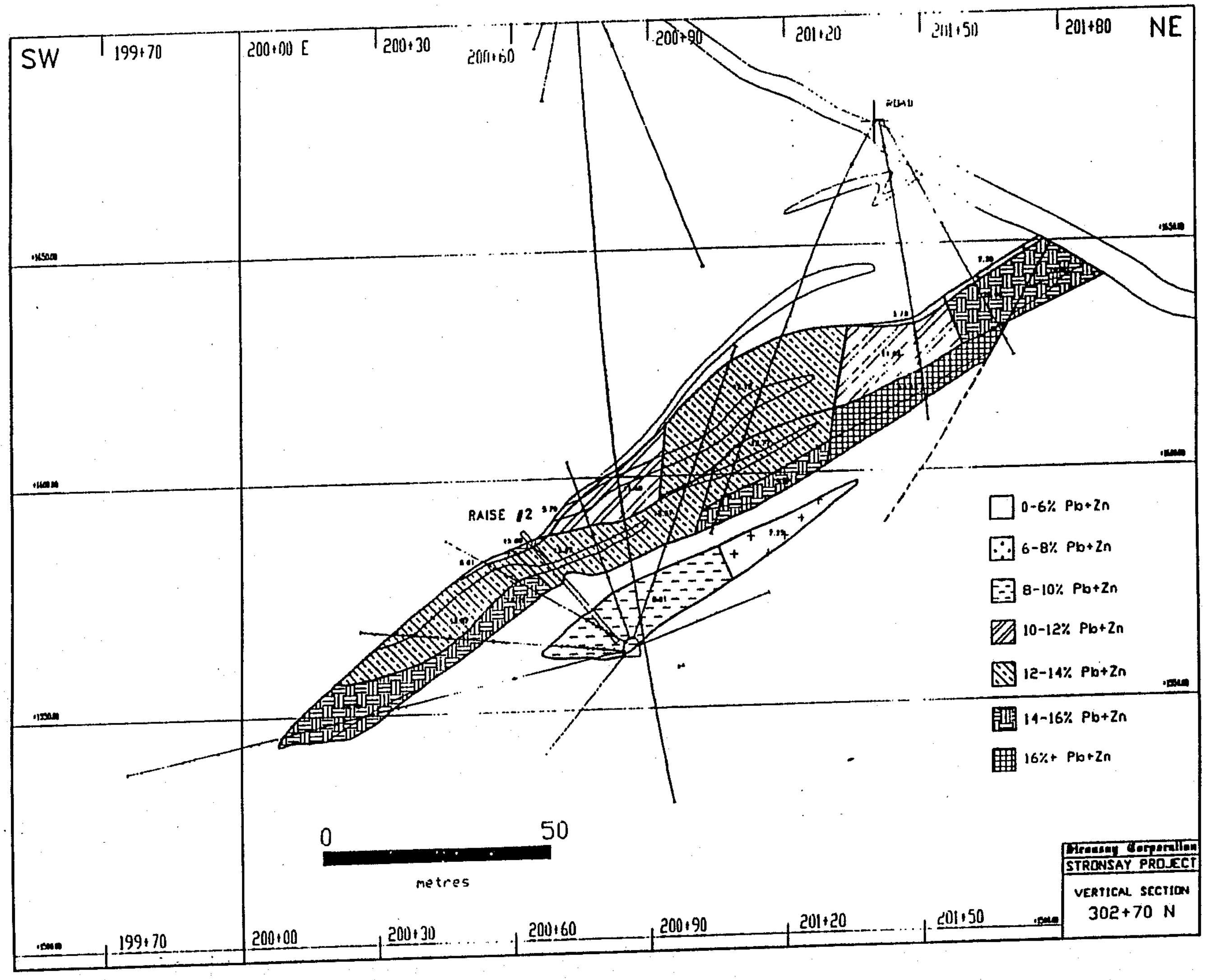

SW
NE
199+70
200+00 E
200+30
200+60
200+90
201+20
201+50
201+80
ROAD
RAISE #2
Pyritic massive sulphides
Sulphides with barite
Barite with sulphides
0
50
metres
STRONSAY PROJECT
VERTICAL SECTION
302+70 N
199+70
200+00
200+30
200+60
200+90
201+20
201+50

Figure 28



SW
NE
199+70
200+00 E
200+30
200+60
200+90
201+20
201+50
201+80
ROAD
RAISE #2
0-6% Pb+Zn
6-8% Pb+Zn
8-10% Pb+Zn
10-12% Pb+Zn
12-14% Pb+Zn
14-16% Pb+Zn
16%+ Pb+Zn
0
50
metres
Stronsay Corporation
STRONSAY PROJECT
VERTICAL SECTION
302+70 N
199+70
200+00
200+30
200+60
200+90
201+20
201+50

Figure 45

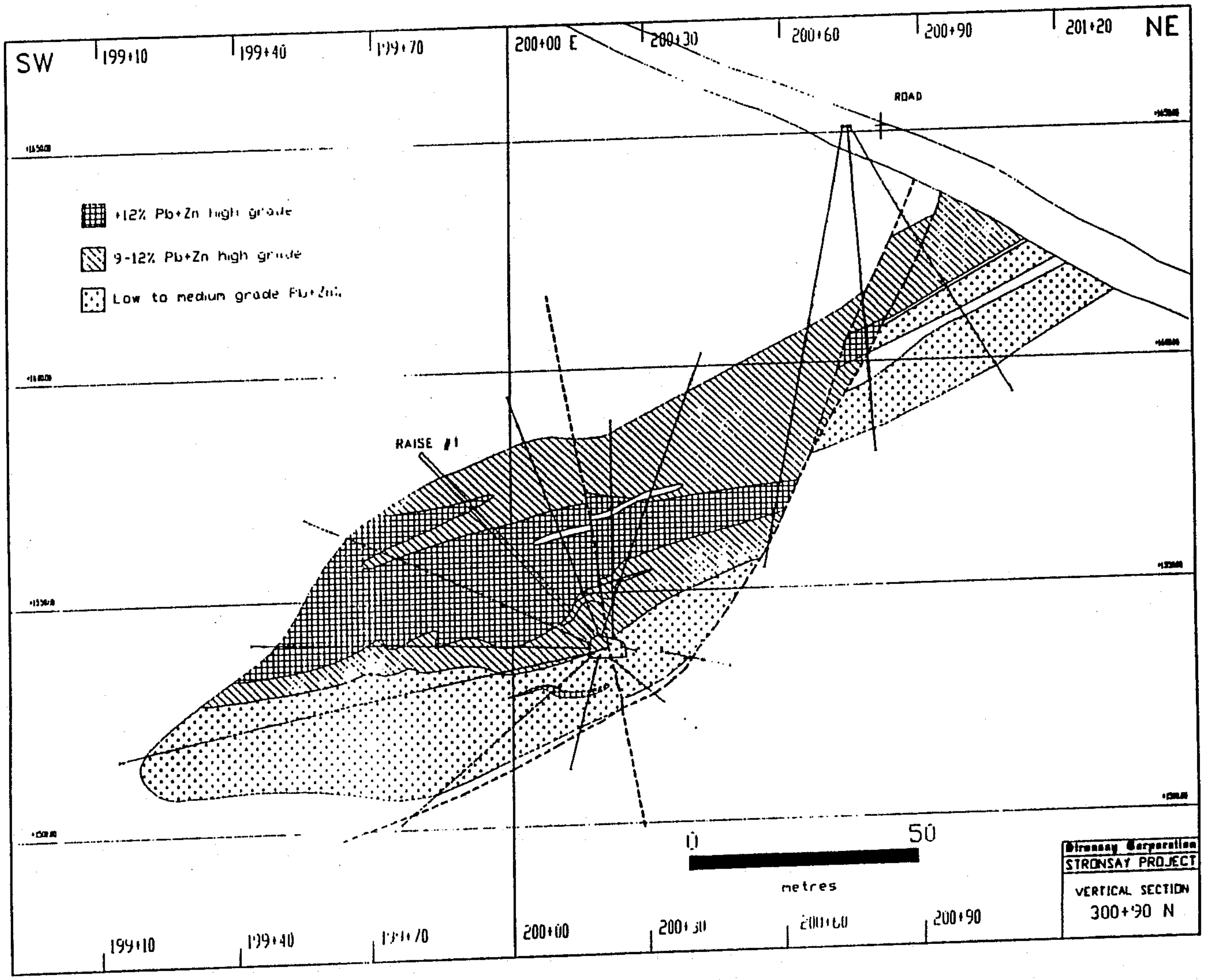
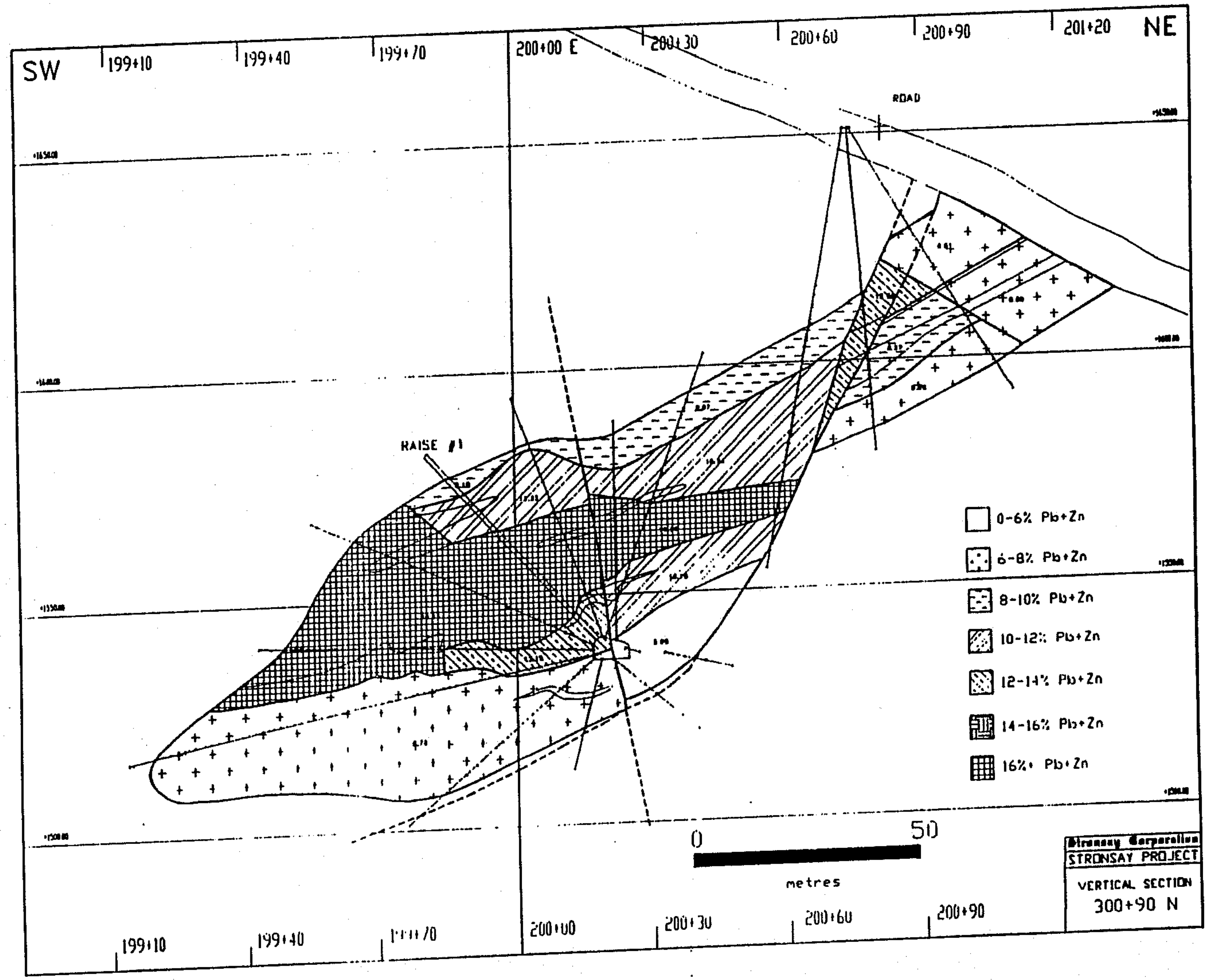
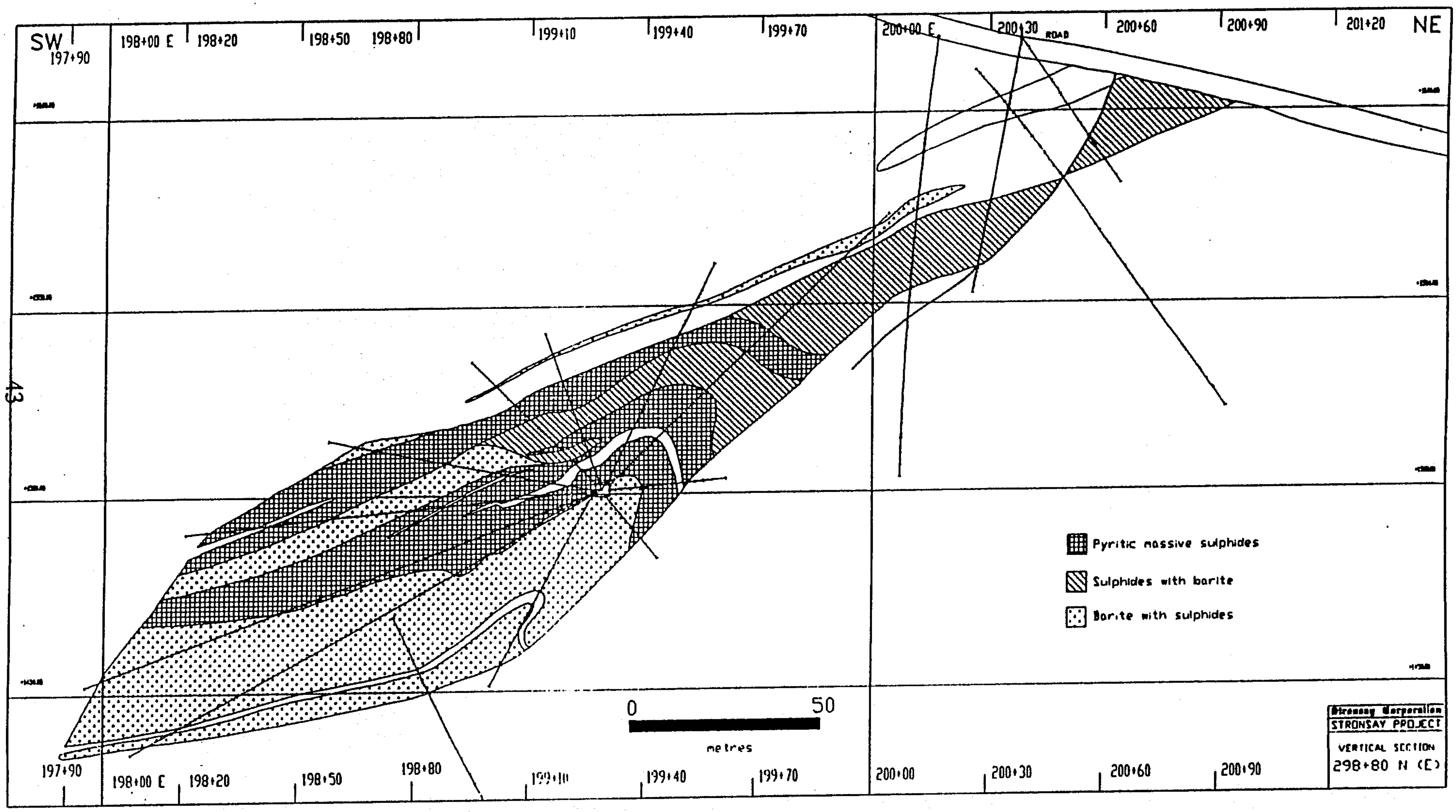

SW
NE
199+10
199+40
199+70
200+00 E
200+30
200+60
200+90
201+20
ROAD
+12% Pb+Zn high grade
9-12% Pb+Zn high grade
Low to medium grade Pb+Zn
RAISE #1
0
50
metres
Stronsay Corporation
STRONSAY PROJECT
VERTICAL SECTION
300+90 N
200+00
200+90

Figure 30



SW
NE
199+10
199+40
199+70
200+00 E
200+30
200+60
200+90
201+20
ROAD
RAISE #1
0-6% Pb+Zn
6-8% Pb+Zn
8-10% Pb+Zn
10-12% Pb+Zn
12-14% Pb+Zn
14-16% Pb+Zn
16%+ Pb+Zn
0
50
metres
200+00
STRONSAY PROJECT
VERTICAL SECTION
300+90 N

Figure 47



SW
NE
197+90
198+00 E
198+20
198+50
198+80
199+10
199+40
199+70
200+00 E
200+30
ROAD
200+60
200+90
201+20
Pyritic massive sulphides
Sulphides with barite
Barite with sulphides
0
50
metres
Stronsay Corporation
STRONSAY PROJECT
VERTICAL SECTION
298+80 N (E)

Figure 32

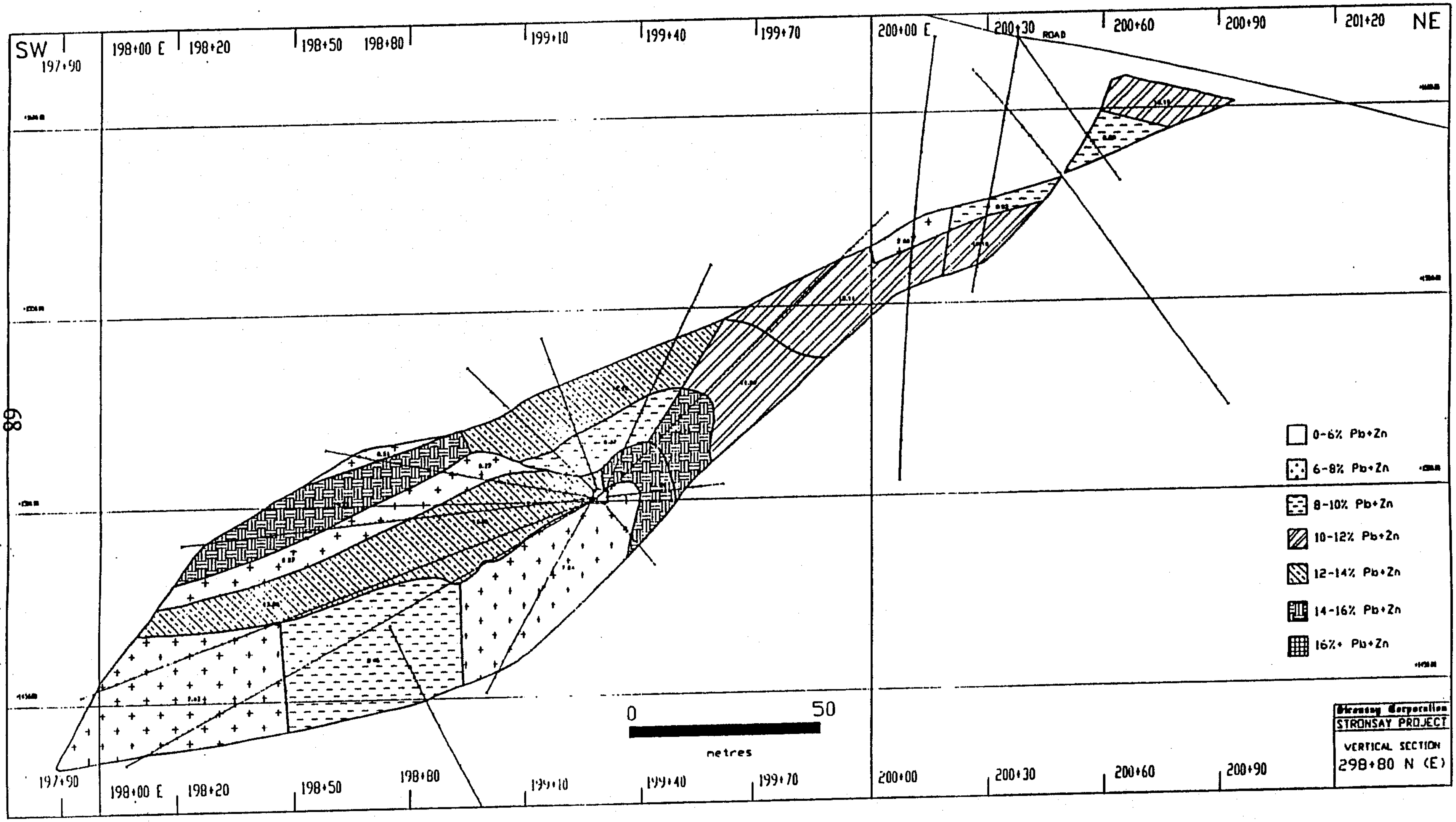
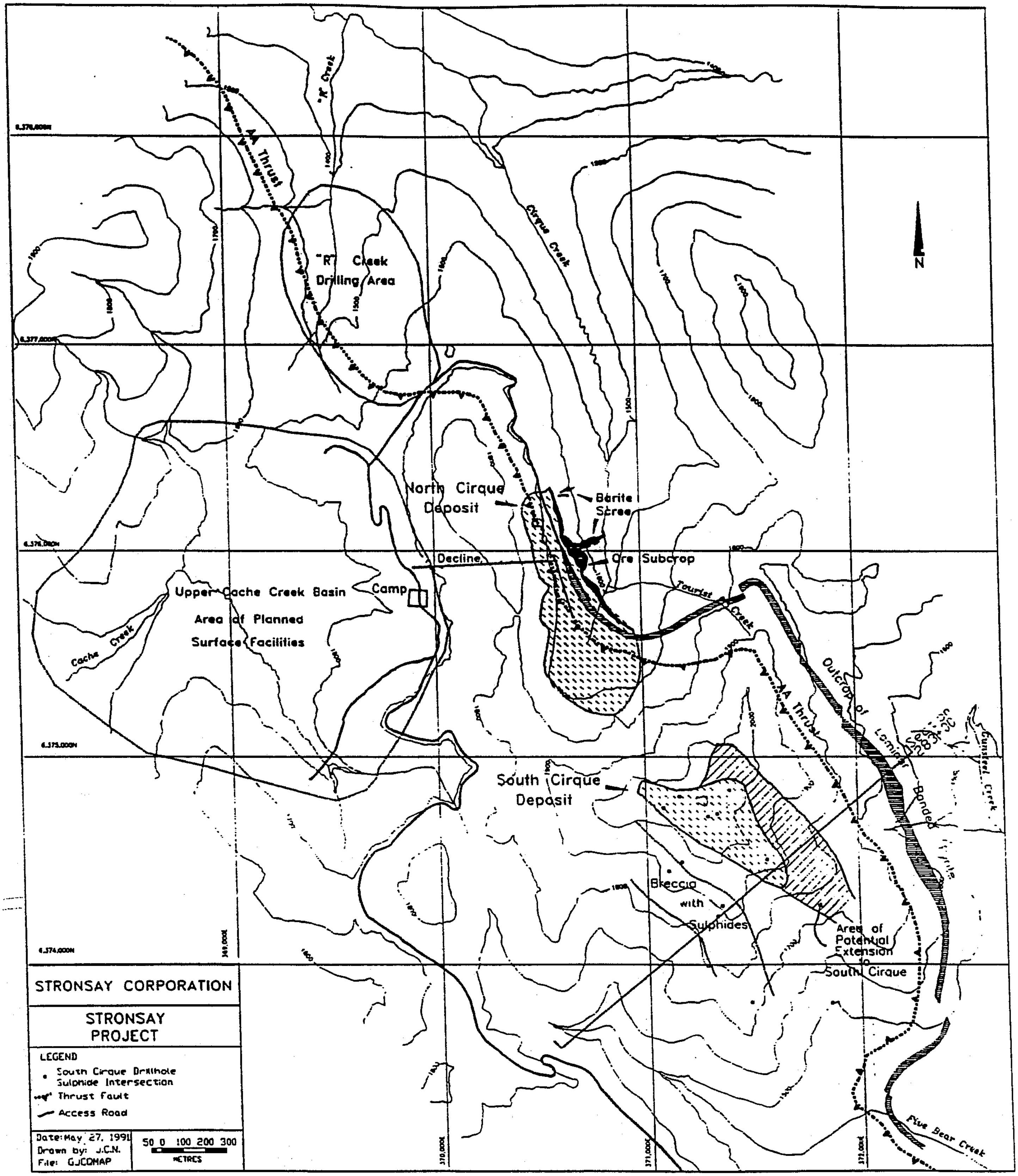
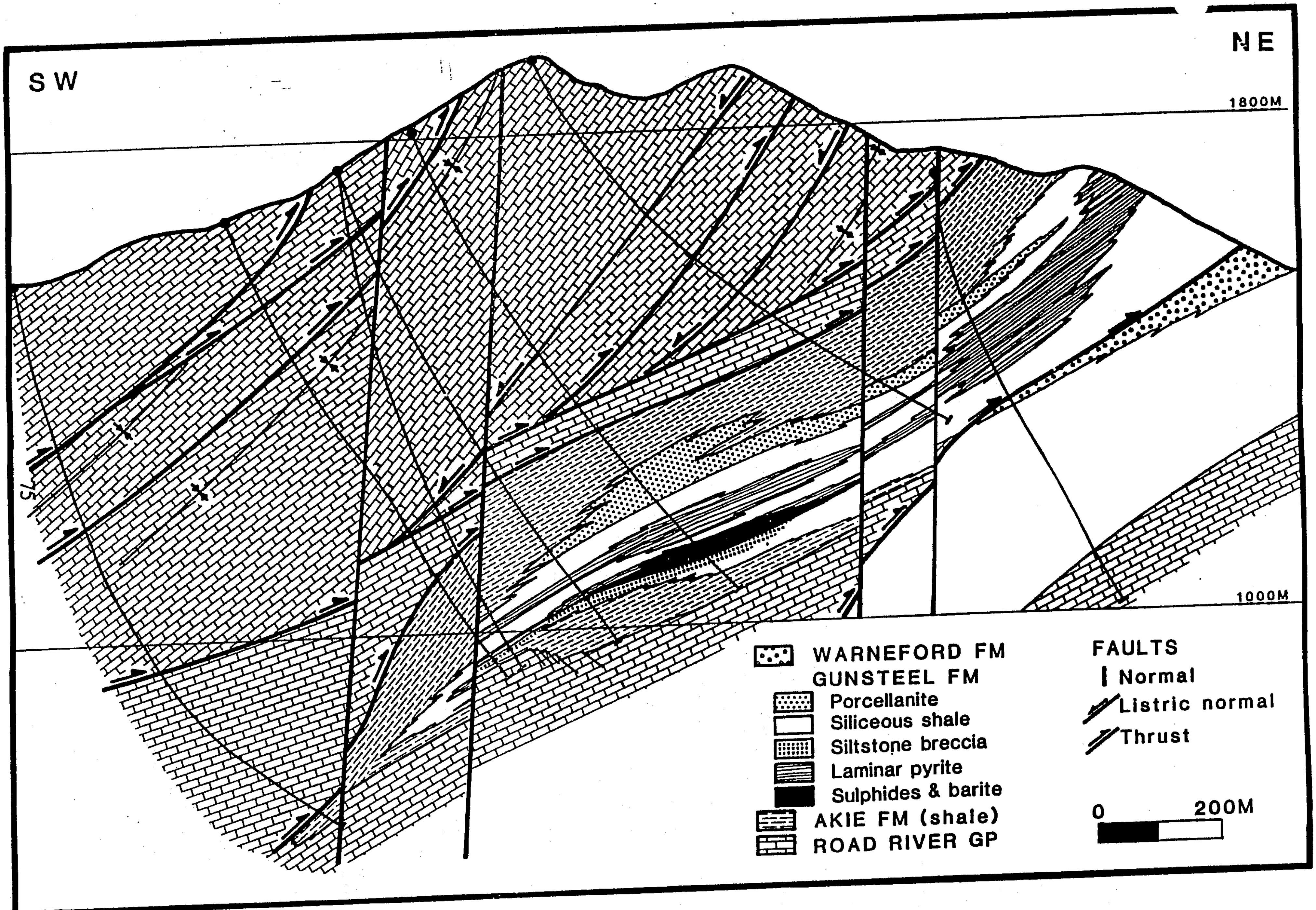

SW
NE
197+90
198+00 E
198+20
198+50
198+80
199+10
199+40
199+70
200+00 E
200+30
ROAD
200+60
200+90
201+20
0-6% Pb+Zn
6-8% Pb+Zn
8-10% Pb+Zn
10-12% Pb+Zn
12-14% Pb+Zn
14-16% Pb+Zn
16%+ Pb+Zn
0
50
metres
STRONSAY PROJECT
VERTICAL SECTION
298+80 N (E)

Figure 49



Location of the North Cirque and South Cirque deposits in relation to the area of minesite surface development. The deposit outline shown is the vertical project of the approximate 2m isopach. See Figure 6 for a section along the decline and Figure 52 for section 283+00 through South Cirque.

Figure 5



Vertical cross section 283+00 of the South Cirque Deposit

Figure 52

STRONSAY PROJECT – RESERVES AND INVENTORY

MINERAL INVENTORY – 0% Cutoff		Proven + Probable				Possible				Total			
Source	Area	Tonnes	Pb%	Zn%	Ag g/t	Tonnes	Pb%	Zn%	Ag g/t	Tonnes	Pb%	Zn%	Ag g/t
CMD	North Cirque 298+35N to 303+75N	17 263 730	2.37	8.81	51.7					17 263 730	2.37	8.81	51.7
CMD	North Cirque 296+25N to 298+35N	11 848 483	2.07	7.84	47.3					11 848 483	2.07	7.84	47.3
										0	0.00	0.00	0.0
	Total North Cirque	29 112 213	2.25	8.41	49.9	0	0	0	0	29 112 213	2.25	8.41	49.9
CRI	Total South Cirque					20 013 083	1.94	6.41	29.6	20 013 083	1.94	6.41	29.6
	Total	29 112 213	2.25	8.41	49.9	20 013 083	1.94	6.41	29.6	49 125 296	2.12	7.60	41.6

MINEABLE RESERVES – 6% Cutoff		Proven + Probable				Possible				Total			
Source	Area	Tonnes	Pb%	Zn%	Ag g/t	Tonnes	Pb%	Zn%	Ag g/t	Tonnes	Pb%	Zn%	Ag g/t
CMD	North Cirque 298+35N to 303+75N	14 859 672	2.42	8.84	51.9					14 859 672	2.42	8.84	51.9
CMD	North Cirque 296+25N to 298+35N	9 818 344	2.20	8.05	49.1					9 818 344	2.20	8.05	49.1
										0	0.00	0.00	0.0
	Total North Cirque	24 678 016	2.33	8.53	50.8	0	0	0	0	24 678 016	2.33	8.53	50.8
	Total South Cirque									0	0.00	0.00	0.0
	Total	24 678 016	2.33	8.53	50.8	0	0	0	0	24 678 016	2.33	8.53	50.8

8

STRONSAY PROJECT

PROJECT EXPENDITURE SUMMARY

The work, and costs thereof, conducted to date on the Stronsay Property are summarized annually as follows:

1977	Staked Cirque claims, mapping and geochemistry.	
	Exploration Expenditures (1977$)	$ 84,000
1978	Staked Elf and Fluke claims, mapping, geochemistry, geophysics.	
	Diamond drilling 6 holes/882 m.	
	Exploration Expenditures (1978$)	$ 333,000
1979	Mapping, geology, geochemistry.	
	Diamond drilling 29 holes/9,018 m.	
	Exploration expenditures (1979$)	$1,566,000
1980	Mapping, geology, geochemistry, geophysics.	
	Diamond drilling 43 holes/16,408 m.	
	Exploration expenditures (1980$)	$4,370,000
	Finbow airstrip construction (1980$)	$ 190,000
1981	Geology.	
	Diamond drilling 53 holes/21,816 m	
	Exploration expenditures (1981$)	$5,235,000
	Finbow airstrip (1981$)	$ 116,000
	Akie/Paul Valley Road (1981$)	$4,990,000
	Capital Equipment (1981$)	$1,600,000
1982	Geology.	
	Diamond drilling 14 holes/13,088 m.	
	Exploration Expenditures (1982$)	$2,726,000
	Road Construction (1982$)	$ 93,000
1983 - 1988	No activity	
1989	Camp and equipment mobilization, access decline, mapping, environmental studies.	
	Diamond drilling 4 holes/99 m.	
	Exploration Expenditures (1989$)	$8,045,000
1990	Underground exploration development, surface and underground diamond drilling, environmental studies, engineering studies.	
	Diamond drilling 125 holes/7,978 m.	
	Exploration Expenditures (1990$)	$8,868,000
1991	Underground exploration, mapping, bulk sample, environmental studies, engineering studies.	
	Diamond drilling 86 holes/4,973 m	
	Exploration Expenditures (1991$)	$3,826,000
1992	Permitting, environmental studies.	
	Exploration Expenditures (1992$)	$ 940,000

In dollars of the day, the total expenditures on the Stronsay Property are:

1977 - 1982	$21,303,000
1989 - 1992	$21,679,000
TOTAL	$42,982,000

In 1992 dollars, the total spent from 1977 through 1982 is $37,457,000, and from 1989 through 1992 is $24,104,000 for total expenditures of $61,561,000 (1992$).

Environmental

1

FARO DECOMMISSIONING

OVERVIEW OF THE ENVIRONMENTAL PLANS

REPORT #WH9108

CURRAGH RESOURCES INC.
Whitehorse, Yukon
December 1991

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1 EXECUTIVE SUMMARY

Since 1986, Curragh Resources Inc. has operated one of the world's largest open-pit lead-zinc mines at Faro, in Yukon. Now (December 1991) mining in the original Faro Pit is close to ending. An interrelated plan involving closure of the Faro Pit, conversion of the pit to a tailings pond, utilization of the Faro mill and tailing facilities for new mines at Grum and Vangorda, recycling of tailings, and the final decommissioning of the tailings facilities, has been devised. Regulatory approval has already been received for the operational aspects of this plan: the new mines, continued use of the Faro Mill and use of the Faro Pit for tailings storage. Now under consideration are the closure aspects of the Faro Site and, in particular, the Rose Creek Valley tailings facility and the Faro Pit Tailings facility.

In April 1991 Curragh Resources Inc. submitted to the Yukon Territory Water Board a plan for decommissioning of the Rose Creek Tailings facility (also referred to as the Down Valley Tailings Impoundment). The report was the result of several years of research and analytical effort, with assistance from experienced engineers and scientists, and in consultation with government officials, regulators, and the public.

In July 1991 an additional report was submitted to the Yukon Territory Water Board. This report (Kilborn 1991) presented Curragh's plans for the conversion of the Faro Pit to a tailings facility and the plans for water recycle related to that conversion. The July 1991 report dealt with some aspects of decommissioning. In particular, plans for closure of the Faro Pit tailings facility, demonstration that the Faro Pit has sufficient volume to complete the reprocessing/recycling operation outlined in the April 1991 report, and examination of some aspects of tailings management related to the reprocessing/recycling operation.

A decommissioning plan must serve multiple objectives. The Faro plan has been developed to:

- provide for environmental protection both in the immediate future and longer term;
- comply with all Government regulations;
- employ state-of-the-art technology;
- ensure the full utilization of a valuable resource. The plan must exploit every opportunity to recycle materials so that useful minerals are not wasted;
- be consistent with the available financial resources.

Curragh's decommissioning plan for Rose Creek Valley Tailings area is designed to ensure a final storage place for the tailings in a low-risk environmental location: submerged under a permanent water cover so that oxidation of the sulphides in the tailings is severely impeded by lack of available oxygen. The plan also creates direct employment for 90 persons for six to eight years after mining operations cease, excluding any multiplier effect.

Curragh's report to the Yukon Territory Water Board examined several alternatives and it was concluded that the most environmentally secure--and most financially viable--alternative involved recycling most of the tailings at Faro for additional extraction of metal, with the transfer of residual tailings into the mined-out pit which would be flooded. Costs would be recovered by revenues generated from the sale of the mineral concentrates derived from the recycling/reprocessing operations.

At the conclusion of reclamation, tailings still remaining in Rose Creek Valley would be isolated and protected with a water cover, and to the extent practicable, the entire site would be restored to nature.

Plans and supporting technical data were filed with the Yukon Territory Water Board in April and July, 1991. Several areas requiring additional work were identified and commitments were made to gather further information with definitive studies proposed. Some of the additional test work has already been completed while other studies are on-going. This report is a supplement to the April 1991 and July 1991 Yukon Territory Water Board filings. It summarizes technical aspects of the overall closure plan, describes the results of additional test work, and provides supporting detail.

Included in this report is consideration of:

- Plans for <u>Reclamation of Pits, Ponds, Dams and Waste Dumps.</u>

- <u>Tailings Transfer</u>, which involves the recovery and transfer of tailings back to the mill for reprocessing, using techniques of "hydromonitoring" and slurry pumping. Significant quantities of tailings will be pumped from the current tailings ponds back to the mill for reprocessing in the mill, following which the tailings will be placed in the Faro Pit. In support of this plan we provide working examples, a schedule for startup and shutdown, and identification of the required capital equipment and operating costs.

- <u>Metallurgical Reprocessing,</u> which involves the re-treatment of tailings in the Faro concentrator to produce marketable bulk concentrate. This process will require modifications to the existing Faro mill (or "concentrator"). Sampling and test work in support of the plan are described. The required process technology is defined in general terms.

- <u>Marketing Bulk Concentrate</u> The decommissioning plan is funded from metals markets on a pay-as-you-go basis. Accordingly, the state of metal markets will govern the magnitude of cash flows available to pay for the plan. Included in the report are assumptions derived from reviews of the lead and zinc metal markets, historic prices and expected future prices. Projected revenues from bulk concentrate sales are compared to projected costs derived from detailed calculation of project operating, transportation, and smelter treatment charges. This comparison indicates that at reasonable future prices the project will break even.

- Environmental Management. It is planned to maintain the environmental integrity of the Faro site by controlling effluent from the site during tailings reprocessing, in periods between seasonal reprocessing, and throughout the inevitable metal market cycles which may force suspension of operations from time to time. As well it is planned to collect and treat water, provide surveillance and maintenance of the site and its facilities, and monitor water quality on an on-going basis.

- Biological Treatment of Waste Dump Drainage This section deals with Curragh's planned biological treatment of waste dump drainage, employing a new and exciting technology with great potential applicability. Biological technology is altering industry thinking with respect to the troublesome problem of mine drainage.

- Water Quality expected of the water management incorporated within Curragh's plan has been studied with encouraging results. Those findings are summarized in this report.

- Timing and Sequencing Curragh's decommissioning plan incorporates a carefully sequenced series of projects, until final decommissioning is achieved. A complicating factor is the need to inter-weave the Faro decommissioning plan with the ongoing mining at the adjacent Grum and Vangorda Pits, since common facilities are involved. The resulting inter-connected schedule is presented in this report. It is important to note that "final" decommissioning will not occur until well past the end of this decade.

The essence of the proposed plan will be to:

- ☐ remove all the tailings above the 1044.3m (3426 ft.)* elevation in the Rose Creek Valley;
- ☐ recover the base and precious metals in a marketable bulk concentrate;
- ☐ condition secondary tailings arising from reprocessing in the mill with lime (to precipitate any water soluble metals);
- ☐ store the secondary tailings in the mined-out Faro Pit where they will be submerged beneath a water cover;
- ☐ maintain a water cover over tailings below elevation 1044.3m (3426 ft.) in the Rose Creek Valley so they will remain non-reactive;
- ☐ treat dump seepage biologically.

* see addendum on elevations - Appendix F

So that the reader may obtain an overview of the plan for decommissioning the Faro site two artist's sketches are shown herein, one showing the site facilities before decommissioning (Plate 1) and the other after decommissioning (Plate 2).

In addition, artist's sketches of the Rose Creek Valley tailings area have been prepared. They depict the current impoundment facilities (Plate 3), the same area during tailings reclamation (Plate 4) and how it will look after decommissioning (Plate 5).

2 THE REGULATORY SETTING

The Yukon Territory Water Board Public Hearing

The Yukon Territory Water Board has called for a hearing on January 15, 1992, for public discussion of the implications of various technical studies and engineering reports submitted by Curragh Resources Inc. in connection with it's plan for Rose Creek Tailings Area decommissioning and the closure aspects of its plan for water recycle and tailings deposition. More particularly, the hearing will consider the implications of two reports submitted by Curragh in April 1991 and June 1991, respectively, as a condition of Curragh's water license. These studies focus on environmental management of tailings ponds and waste dumps, and other water quality issues.

The Purpose of This Report

The decommissioning of a large facility, such as exists at Faro, in today's regulatory and technical milieu, is no small undertaking. In preparing for the January 1992 hearing, Curragh officials began to appreciate that the accumulating mass of technical and scientific studies could be overwhelming and possibly hinder the proper assessment of the decommissioning plans.

Curragh officials concluded that it might be helpful to produce an all-encompassing "umbrella" summary report in non-technical language, so that various stakeholders could more readily grasp the broad outlines of the plan. The goal of this report is to provide this overview of the Faro environmental plan in all of its various interrelated aspects.

This report also presents new information and technical data which could improve the environmental plan. As well, this report further attempts to clarify any inconsistencies between the various other reports that have been produced on decommissioning as those plans have evolved during Curragh's operation of the Faro Mine.

Status of Faro Water License

The current Rose Creek water license (No. IN89-001) applicable to the Faro Mine and Mill was granted to Curragh Resources by the Yukon Territory Water Board on January 23, 1990 and expires on January 30, 1997. This was a renewal of the water license originally granted to Cyprus Anvil Mining Corporation seventeen years ago, and assigned by the Yukon Territory Water Board to Curragh Resources Inc. in 1985.

Scope of the Water License

The existing water license, and this report, pertain to all of Curragh's operations affecting water resources in the Rose Creek Valley, including the Faro Pit, Mill and the tailings facilities which will serve the Grum and Vangorda Mines and possibly later the Dy Mine. The Faro water license is not otherwise concerned with the Grum and Vangorda pits and dumps, nor with the expected future Dy Mine, all of which are located in other nearby watersheds.

3 PITS AND PONDS, DAMS AND DUMPS

The Rose Creek Valley watershed around Faro encompasses several medium-to-small sized streams, and an intricate complex of "pits and ponds, dams and dumps". Over the years, a carefully balanced system of dams and diversion ditches, canals, settling basins, tailings impoundments, special drainage systems, and pumping arrangements, have evolved at Faro. These have performed several functions: providing water for the mill, keeping the mining area dry, and protecting the environment.

The dominant water course of the area is Rose Creek, which is more than a "creek" as it is the source of drinking and process water for the Faro operations. The broad (1.5 km wide) Rose Creek Valley is also the home of Faro's tailings facilities. Upstream from the Faro mine site, Rose Creek splits into a "North Fork" and a "South Fork". Faro Creek is a small creek which, before mining commenced, flowed directly over the Faro ore body, running down the hill into Rose Creek. In the early stages of mine development Faro Creek was diverted around the Faro Pit and into North Fork to create dry ground for mining.

The various structures which have been developed at the Faro site over the years can be broadly categorized into the following functional areas:

a) Pits and Underground Workings: from which the ore is removed;
b) Rock Dumps: where rock removed from the mines to get at the ore has been placed;
c) Roads: on which ore is carried to the mill and waste rock is carried to the dumps;
d) Mill: in which the ore is ground to a powder and most (about 80%) of the valuable minerals are separated to produce a mineral concentrate containing either lead, zinc or both together (bulk concentrate);
e) Water Supply: which provides a source of water, year round, to the mill for use in the mineral separation process. The water is used to make a slurry of the pulverized ore and is the medium in which the mineral separation occurs. The water is also used for drinking, washing and fire protection;
f) Rose Creek Tailings Area: where the leftover minerals from the milling process (along with 20% of the lead and zinc in the ore) are deposited and the leftover water separated;
g) Faro Pit Tailings Area: when the Rose Creek Tailings Area is full and the Faro Pit is mined out the pit will be converted to a tailings impoundment for the newer mines.
h) Other Facilities: which include the shops, office complex, guard house, warehouse, explosives magazines, lube shack, petroleum storage and all the other structures needed by a mine and the people who work there.

The sections which follow elaborate on the various components of the Faro site in the context of the above functional categories. We describe first what the structure is, or what it does, then how it will be dealt with at final closure and decommissioning of the complex. The location of all of these structures can be found on Figure 1. Many of the structures are also identified on Plates 1 through 5.

A. PITS AND UNDERGROUND WORKINGS

The Faro Pit. The Faro Pit is approximately three-quarters of a kilometre wide, one and one-quarter kilometres long, and about one-third of a kilometre deep. For approximately 23 years, Cyprus Anvil Mining Corporation and later Curragh Resources Inc. excavated material from the pit. Currently there is approximately six (6) months of ore supply remaining and the pit will soon be freed for other uses. The obvious option was to utilize the pit as a place for permanent storage of tailings. This change was approved by the Yukon Territory Water Board and is now integral to the Faro environmental plan. The pit as a tailings facility is discussed in more detail later. After final decommissioning, the pit will remain flooded and a lake 4.25m deep will cover the tailings. We will refer to this lake as "**Faro Lake**".

The Zone II Pit. Zone II was a separate, smaller pit mined by Cyprus Anvil between 1978 and 1981. It lies to the southeast of the Faro Pit. Once depleted, it was filled with waste rock from the Faro Pit, first by Cyprus Anvil and then by Curragh. In Curragh's environmental plan the Zone II Pit plays a key role in the collection of seepage waters from various sources, particularly the Waste Dumps, so that such water can be treated before release to the environment.

Faro Underground Mine. Since early 1990 an underground mine has been operated through a opening in the southwest wall near the bottom of the Faro Pit. The mine openings follow the ore deposit to the southwest and exploit areas where the deposit was too thin to be mined by open pit methods. The fate of the underground workings is discussed later where they are shown to be an environmental solution rather than a problem.

Metal-Rich Water from the Pits. Naturally occurring sulphide rock remains in both the Faro Pit walls and the Zone II Pit walls. With the passage of time, and when exposed to the combination of air, water and bacteria, rocks containing sulphides will oxidize, forming soluble sulphates and turning adjacent water acidic. The acidic water dissolves various metals from the host rocks. If not controlled, these metals will eventually enter the environment. This undesirable chain of events must be prevented--or at least inhibited--from occurring.

In the Faro decommissioning plan, most exposed sulphides in the pit walls are inundated by tailings, or, by a cover of water so they cannot readily oxidize. Nevertheless, small quantities of sulphide material at the northwest end and along the northeast wall of the Faro pit will be exposed. Due to the insignificant area exposed, the amount of metal picked up by surface run-off water will be minimal.

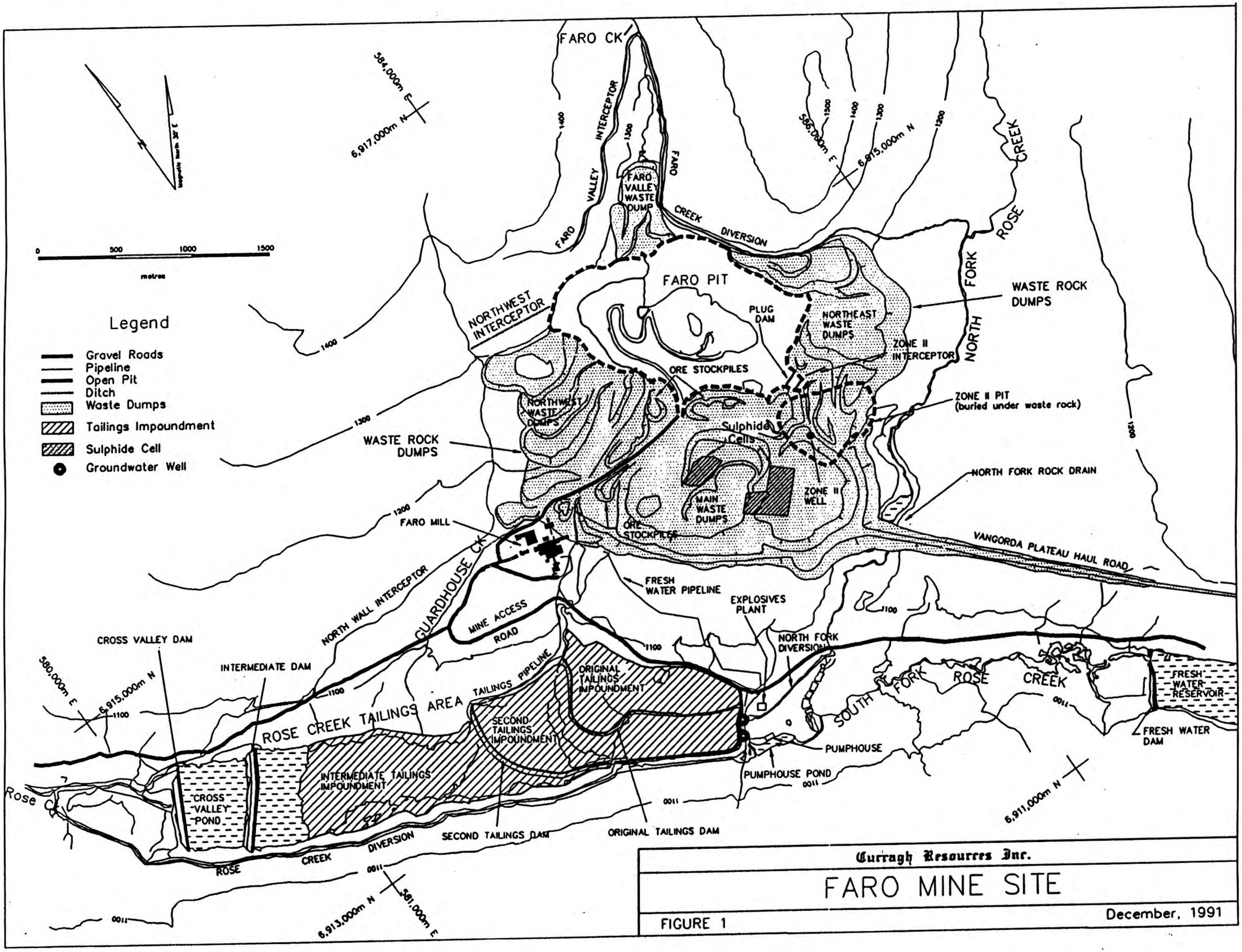
FARO CK
FARO VALLEY INTERCEPTOR
FARO VALLEY WASTE DUMP
FARO CREEK DIVERSION
NORTH FORK ROSE CREEK
FARO PIT
NORTHWEST INTERCEPTOR
PLUG DAM
NORTHEAST WASTE DUMPS
WASTE ROCK DUMPS
ZONE II INTERCEPTOR
ORE STOCKPILES
NORTHWEST WASTE DUMPS
ZONE II PIT (buried under waste rock)
Sulphide Cells
WASTE ROCK DUMPS
ZONE II WELL
NORTH FORK ROCK DRAIN
MAIN WASTE DUMPS
FARO MILL
ORE STOCKPILES
VANGORDA PLATEAU HAUL ROAD
GUARDHOUSE CK
NORTH WALL INTERCEPTOR
FRESH WATER PIPELINE
EXPLOSIVES PLANT
MINE ACCESS ROAD
CROSS VALLEY DAM
INTERMEDIATE DAM
NORTH FORK DIVERSION
ORIGINAL TAILINGS IMPOUNDMENT
ROSE CREEK TAILINGS AREA
TAILINGS PIPELINE
SOUTH FORK ROSE CREEK
FRESH WATER RESERVOIR
SECOND TAILINGS IMPOUNDMENT
PUMPHOUSE
FRESH WATER DAM
PUMPHOUSE POND
INTERMEDIATE TAILINGS IMPOUNDMENT
CROSS VALLEY POND
Rose Ck
SECOND TAILINGS DAM
ORIGINAL TAILINGS DAM
ROSE CREEK DIVERSION
584,000m E
6,917,000m N
586,000m E
6,915,000m N
580,000m E
6,915,000m N
6,913,000m N
581,000m E
6,911,000m N
0 500 1000 1500
metres
Legend
Gravel Roads
Pipeline
Open Pit
Ditch
Waste Dumps
Tailings Impoundment
Sulphide Cell
Groundwater Well
Curragh Resources Inc.
FARO MINE SITE
FIGURE 1
December, 1991

PLATE 1 - FARO MINE SITE - EXISTING - The Faro Pit and Waste Rock Dumps when mining is completed and before tailings deposition in the Pit begins. The Rose Creek Tailings Area is very near completion. Faro Creek is diverted around the pit into the North Fork of Rose Creek which flows through a rock drain under the Vangorda Plateau Haul Road. The Rose Creek Diversion carries the flow above and around the Rose Creek Tailings Area.

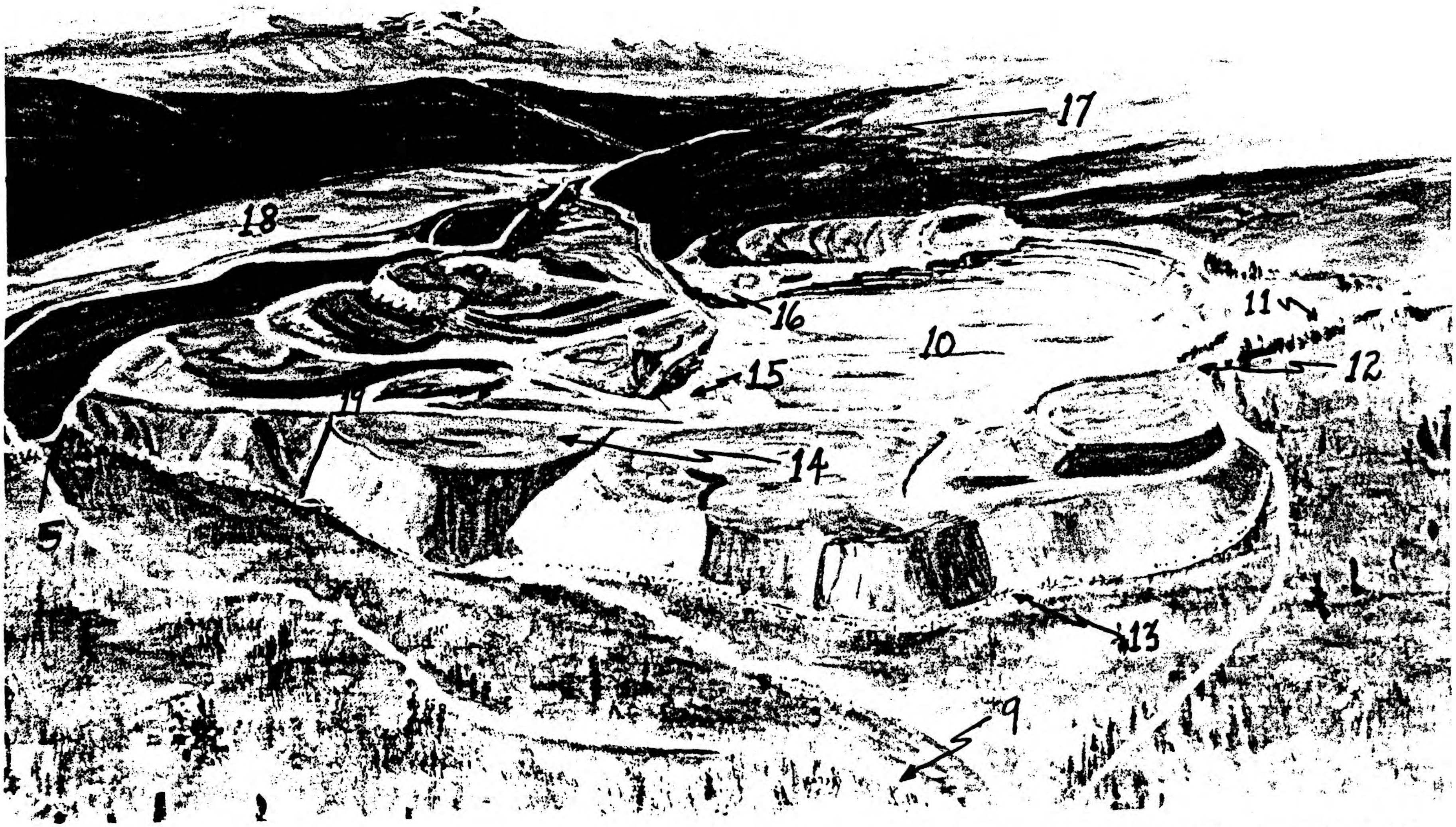

PLATE 2 - FARO MINE SITE - POST DECOMMISSIONING - Faro Lake is shown at its final water elevation with tailings deposited below water surface (year 2026). Faro Creek is permanently diverted into the lake over the pit wall and its outflow is directed to Rose Creek. The Plug Dam prohibits surface flow and restricts seepage to the Zone II Pit. The North Fork of Rose Creek flows through a partial breach in the Vangorda Plateau Haul Road, joins the South Fork and enters the Valley Lake. The Rose Creek Diversion is removed. The Mill, other buildings and facilities are dismantled and the area revegetated with grass/legume cover crop. The Waste Rock Dumps and area around the Rose Creek Tailings Area are also revegetated.

Seepage from Faro Pit into the Zone II Pit. A fault zone exists at the southeast limit of the Faro Pit which could provide a natural pathway for water to seep from Faro Lake into Zone II where it not only would become contaminated but also would add to the total volume of water needing treatment. In order to limit this seepage, a "Plug Dam" (see below) has been designed which will cover the fault zone with highly impermeable fine tailings and effectively seal the pit wall.

Use of Zone II Pit as Sump. Paradoxically, while considerable money and effort will be expended to minimize seepage out of Faro Lake into Zone II Pit, the Faro environmental plan also uses Zone II Pit as a sump to collect seepage from various waste dumps, prior to being treated.

Faro Pit Wall Stability. The steep, high-standing (over 300 metres) northeast wall of the Faro Pit has shown signs of instability when wet. This rock face was stabilized during mining operations through surface water control and internal dewatering (via drill holes). Under the decommissioning plan, the Faro Pit will be flooded with water. Initially water will cover the toe of the pit wall. Three years later the wall will be inundated to its maximum level but only 60m of tailings will be deposited in the pit. Eventually the pit will be nearly full of tailings with only 4.25m of water covering them.

The stability of the pit wall, when saturated with water initially, prior to the pit being full of tailings, could be in question. Current geotechnical advice suggests that if the pit wall collapses during the early backfilling stages, the consequences are not serious since a wave could not go over the top of the pit walls. Once the pit wall is substantially flooded with water its stability will actually increase because of the combined result of two opposite effects a) water in the wall will reduce the rocks resistance to movement, however this will be offet by b) the buoyancy of the rock in the water which will reduce the weight of the wall and reduce its tendency to move downward. After the pit is filled with tailings, stability will be further increased and the impact of any pit wall subsidence would be negligible due to the minor volumes which could collapse unhindered. Nevertheless, monitoring of the pit wall will be continued.

The complete geotechnical opinion, as provided by Mr. A. Stewart of Piteau & Associates Ltd, is included in Appendix D.

B. DUMPS

Waste Rock Dumps. The existing Faro Mine has generated approximately 100 million cubic metres of waste rock. Today, all of this material resides in "waste dumps" around the perimeter of the Faro Pit and within the former Zone II Pit.

Fortunately for the environment, the vast majority of this material is benign; that is, it presents no hazard to the environment. However, mixed within the dumps, more or less at random, are low-grade sulphide rocks which (like the sulphide rocks left behind in the pit walls) will oxidize. When in contact with rainwater and snow melt percolating through the dumps, the oxidized rock will generate metal-rich seepage which will emerge from the toe of the dump.

The oxidation process is proceeding slowly at present and the technical estimate is that, due to the inherent nature of the rock and low temperature conditions which prevail in Yukon, oxidation will continue to be slow. Nevertheless, waste rock seepage, if unattended and unmanaged, could become deleterious to the environment. Thus, an over-riding environmental management issue is to ensure collection and appropriate treatment of this drainage.

Dump tops and slopes will be revegetated with a grass/legume cover crop to accelerate their natural reversion to wild terrain.

Sulphide Cells. During Curragh's tenure at the Faro Site, sulphide rocks have been concentrated into specific areas of the dumps. There are two such areas identified on Figure 1. Both the sulphide cells will be covered by a layer of compacted phyllite as described in the 1988 Closure Plan. Some sulphide dumps have been established within the Faro Pit. These will be inundated by tailings and water at final closure.

Faro Valley Interceptor. This is a minor tributary diversion ditch located northwest of Faro Creek. It prevents surface water from running through the Waste Dump areas during mining operations. During decommissioning, this Interceptor will be upgraded and maintained, so as to cut off water before it runs into the waste dumps where it could become contaminated.

Northwest Interceptor. This is another ditch to be constructed above the Northwest dumps to intercept surface runoff before it contacts the dumps. It will be built as part of the post closure water management plan.

Ore Stockpiles. In Curragh's plan, the various ore stockpiles, including low-grade ores, oxidized ores, and Skagway gravels returned to the mine for reprocessing, are to be fed into the mill during the later stages of regular operations. They will be fully consumed.

C. ROADS

Vangorda Plateau Haul Road. This 14-kilometre causeway was constructed from non-acid-generating materials extracted from the Faro Pit. At closure, the roadway will be breached to accommodate the North Fork of Rose Creek. Several culverts on the haul road will be breached if it becomes evident that future blockage of the culverts is a clear threat. Elsewhere the haul road will be scarified and seeded.

North Fork Rock Drain. This rock drain permits water in the North Fork to flow under the haul road and causeway running from Vangorda Plateau to the Faro Mill. During decommissioning, the haul road embankment will be breached where it crosses North Fork and a spillway will be established to accommodate freshet water flows in North Fork. This spillway will be close to the original stream elevation and will allow fish passage to resume.

D. MILL

The mill is one of the major assets of the Faro Site. It produces mineral concentrates from ores from the mines. The concentrates have more unit value than the ore and can be shipped to smelters around the world to produce refined metal. The Faro Mill is one of the largest lead-zinc concentrators in the world. It was originally built in the late sixties but has been expanded several times, most recently in 1981. Since Curragh re-opened the mine in 1986 the mill has been further modernized and improved through installation of new equipment and repair of old.

The Faro Mill will be refitted to reprocess tailings once all ores are consumed as described in this report. When all tailings have been reprocessed the Mill will be disassembled and sold for salvage. The above ground concrete structures will be demolished and/or buried and concrete debris as well as scrap metal will be buried. Below ground level excavations will be filled in. Areas with significant contamination will be covered with soil and the area revegetated.

E. WATER SUPPLY

Fresh Water Dam. During operations, fresh water for the mill, particularly needed during arid winter months, has been stored in a reservoir behind the Fresh Water Dam on the South Fork of Rose Creek. During decommissioning the overflow ditch bottom will be lowered slightly, so as to preserve valuable fish habitat in the reservoir while allowing fish passage to the lower South Fork. The dam will be breached sufficiently, however, to provide a very high factor of dam safety and stability.

Freshwater Reservoir. The reservoir will be partly retained since it has created valuable over-wintering habitat for Arctic grayling fish. The South Fork Reservoir, reduced in size, will continue as permanent fish habitat. The reservoir water level will be 4m lower than it is today, however its area will be little changed. The reservoir will have substantial area of water depth greater than 8m.

Pumphouse Pond. This pond, about three kilometres downstream from the Fresh Water Reservoir, feeds water into the process water supply system for the mill. In decommissioning, the pump house will be removed, the impoundment structure will be breached, and Rose Creek will be returned to its natural elevation.

North Fork Diversion. This is a short diversion channel which carries the water flow of the North Fork of Rose Creek around the Pumphouse Pond during periods of spring freshet. The purpose is to reduce suspended sediment load in the process water supply for the concentrator. The diversion channel is not used in winter when water flows are low and the North Fork is directed into Pumphouse Pond to augment the water supply. During decommissioning, the diversion channel will be upgraded to accommodate flood events while the original course will be maintained in the flood plain as an emergency overflow.

Groundwater Wells. Several wells were developed in the area of the junction of the north and south forks of Rose Creek. These wells augment surface water supply in winter. They are connected by pipelines to the Pumphouse Pond. All surface facilities connected with the wells will be removed at final decommissioning.

Fresh Water Pipeline. This pipeline conveys fresh water from the pumphouse pond to the mill and will be dismantled at the conclusion of operations.

F. ROSE CREEK TAILINGS AREA

The Rose Creek tailings area is shown in Plate 3 in essentially its current configuration.

Tailings Impoundments and Dams. Tailings produced by mill operations at Faro have been stored behind three retaining dams stretching across Rose Creek Valley, as follows:

- The **Original Tailings Impoundment** behind the **Original Tailings Dam** was used for tailings between 1969 and 1975. It is located just north of the junction of the North and South Forks of Rose Creek. The low retaining dam was built during Cyprus Anvil's early years of operation and raised several times.

- The **Second Tailings Impoundment** behind the **Second Tailings Dam** was used for tailings between 1975 and 1981 and occasionally since Curragh reopened the mine. It was built and used by Cyprus Anvil and is located below the original impoundment dam.

- The **Intermediate Tailings Impoundment** behind the **Intermediate Dam** was constructed by Cyprus Anvil (1981) and used for tailings storage by Curragh Resources (1986-present). It is the newest and most sophisticated of the tailings facilities and has been raised on two occasions since 1986.

Cross Valley Dam. Downstream in the Rose Creek Valley, below the three tailings dams, there is a final dam: the Cross Valley Dam. This dam was built to create a final "polishing pond" which provides the settling for suspended sediments which might escape the tailings ponds. It thereby helped to ensure suitable water quality before final release to the environment. The Cross Valley Pond will be retained

PLATE 3 - ROSE CREEK TAILINGS AREA - EXISTING - The Original and Second Tailings Impoundments were almost entirely filled by Cyprus Anvil Mining Company and are no longer in use. The Intermediate Tailings Impoundment is the site of current tailings deposition and contain exclusively Curragh tailings. The Cross Valley Pond is for water clarification only and contains no significant tailings. The Rose Creek Diversion carries the flows of Rose Creek around the tailings area. The Pumphouse Pond is the source of water for mill operations and is fed by the South Fork of Rose Creek whose flow is regulated by the upstream Freshwater Reservoir.

throughout the tailings reprocessing operation to assist in maintaining water quality. During decommissioning, the Cross Valley Dam will be breached as it is no longer required. Accumulated fines behind the dam will be removed and placed behind the Intermediate Dam under permanent water cover.

Rose Creek Diversion. In order to develop Cyprus Anvil's second tailings impoundment area and the currently used Intermediate Tailings Impoundment area in the Rose Creek Valley, it was necessary to divert Rose Creek from its natural course by constructing a canal running along the south wall of the Valley. This canal will be retained during reprocessing; however, after decommissioning Rose Creek will be re-routed into a permanent pond behind the Intermediate Dam. Rose Creek Diversion will then be abandoned.

North Wall Interceptor. The North Wall Water Interceptor is a ditch established to re-direct stream flow and surface water run-off which might have otherwise entered the Intermediate Dam tailings impoundment. With decommissioning this ditch will no longer be required. Some flows will be returned to their natural courses and portions of the ditch will be abandoned. Other parts will be upgraded and will be used for Faro Lake discharge.

Tailings Pipelines. The pipelines which conveyed tailings to the tailings ponds will be retained during the reprocessing operation as an emergency discharge line and to deliver lime slurry as needed to the Intermediate Tailings Impoundment. At final closure the pipelines will be dismantled.

The Valley Lake. During reprocessing, most of the tailings in the original and second impoundment and a small amount of the tailings behind the Intermediate Dam, will be removed. This will result in all tailings in the Rose Creek Valley above the elevation of 1044.3m (3426 ft.) being reclaimed. Flooding over the residual tailings will therefore be feasible to an elevation of 1046.3m (3433 ft.). This water cap will permanently isolate the tailings from contact with oxygen in the atmosphere. A lake, **Valley Lake**, will be created in Rose Creek Valley which will be a minimum of two metres deep.

This plan avoids the extensive capital expenditure which would be required to flood all of the Rose Creek Valley Tailings area to the current elevation. This was a financially unrealistic option which was rejected in the past.

Spillway on Intermediate Dam. A side channel spillway will be constructed at the northwest edge of the Valley Lake at the north abutment of the Intermediate Dam. The spillway is sized to safely pass the magnitude of flood that would be expected to occur once every 500 years in Rose Creek.

Rose Creek Inlet. During decommissioning Rose Creek will be returned to its original course, flowing through the (to be breached) Pumphouse Pond and entering Valley Lake. An energy dispersing rip-rap structure will be provided to prevent any disturbance of the submerged tailings as Rose Creek enters the Valley Lake.

Clean Up of Tailing Pond Shorelines. Underlying the tailings ponds of Rose Creek Valley are the natural valley bottoms. The bottoms behind the original and secondary dams were overlaid with tailings by Cyprus Anvil . Since this environmental plan specifies that these tailings are in large measure to be removed, the exposed tailing pond shoreline will be cleaned during decommissioning and returned to nature. The cleaning technique to be employed will involve hydromonitoring with mechanical excavation in selected areas. The area will then be re-vegetated.

G. FARO PIT TAILINGS AREA

Faro Creek and Faro Creek Diversion. Upon completion of mining in the Faro Pit, the pit will be flooded and backfilled with tailings. Commencing in 1992 and for approximately three years thereafter, Faro Creek will be diverted back to its natural course, flowing into the Faro Pit to assist in its flooding. Once the pit is flooded, and while the pit is still being used for active tailings deposition, the Creek will be temporarily re-routed back to its diversion channel. Later, when the pit has been filled with tailings, Faro Creek will be returned to its natural course and run into Faro Lake (see below) formed by the flooding of the pit, and exit over a constructed spillway to rejoin Rose Creek in the Valley.

Creation of Faro Lake. The minimum elevation of the bottom of the Faro Pit is approximately 3200 feet (976m) above sea level. Tailings are to be protected with a permanent water cover to prevent oxidation. Flooding will create a lake three-quarters of a kilometre across by one and one-quarter kilometre wide; which we will refer to as "Faro Lake".

At the conclusion of Faro Pit backfilling, the tailings surface will be approximately 3836 feet (1169m) above sea level. The tailings deposited under the Lake will be approximately 636 feet (196m) thick, or about the height of one of the taller office towers in Toronto. The surface of Faro Lake will be 3850 feet (1173m) above sea level. Water depth in the lake will be 4.25m (14 ft.).

The surface water in Faro Lake will be used as a mill water supply source once the Pit is flooded to capacity. Water level in Faro Lake will be controlled by a siphon structure until a permanent spillway is constructed upon decommissioning.

Water quality in Faro Lake will be excellent during the filling stage due to the high alkalinity of the mill tailings discharged into it, thereby inhibiting any uptake of metals during pond initiation. Once the lake is filled, water quality will remain high due to the limited sources of contamination which could enter the lake. Discharge from the lake will meet the relevant water quality standards.

The Plug Dam. In order to eliminate water seepage from the flooded Faro Pit into the Zone II Pit, a Plug Dam must be installed between them. This Plug Dam will consist of two dams plus a water barrier: a downstream impermeable dam, an upstream rock-fill dam, and fine materials (cycloned out of the tailings) placed in-between to ensure an adequate water barrier thereby preventing leakage. The development of the Plug Dam is illustrated in Figure 2.

Faro Lake Spillway. The spillway from Faro Lake will be constructed to the southwest of the Faro Pit. It will lead into a channel about 300m long excavated through fill material, followed by an additional 1 km of channel at the current ground level, all lined with impermeable clay. The remaining 2.9 km of channel will be an open ditch in natural ground. The spillway structure itself will be rip-rap armoured.

The south side of the flooded Faro Pit and the above two water management structures are illustrated in Figure 3.

Faro Creek Cascade. Following decommissioning, Faro Creek will be allowed to cascade over the pit wall into Faro Lake below. An energy-dispersing structure will be placed at the bottom of the cascade to avoid agitation of the tailings at the Lake bottom.

H. OTHER FACILITIES

Buildings and Structures. Several buildings around the minesite must eventually be decommissioned and removed. All buildings not needed for an ongoing environmental maintenance and monitoring, will be dismantled and removed. Cash raised through the dismantling and sale of equipment will be available for environmental funding. The area will be made safe for human and wildlife passage by backfilling foundations and demolishing and/or burying concrete structures. Certain areas will be scarified and covered with soil, including those, such as parking lots, which might contain superficial hydrocarbon or metal contamination.

COMPARISON WITH THE 1988 CLOSURE PLAN

Certain aspects of the 1988 Closure Plan filed with the Yukon Territory Water Board differ from the plans submitted more recently. The more current plans replace those provisions of the 1988 plan. The most significant changes are:

- The Faro Pit will be flooded with tailings and water, rather than simply water as in the 1988 Plan. The Pit will be filled to a higher elevation: 1173m (3850 ft.) ASL versus 1161.5m (3811 ft.) ASL previously, thus water level will be 12m (39 ft.) higher. All but the top 4.25m (14 ft.) of the filled pit will contain tailings which will seal the pit walls, reduce the effect of the higher water level and reduce seepage, improving groundwater control.

- Discharge from the Faro Pit will be to the southwest rather than the southeast as in the 1988 plan avoiding the possibility of leakage from the discharge spillway into Zone II.

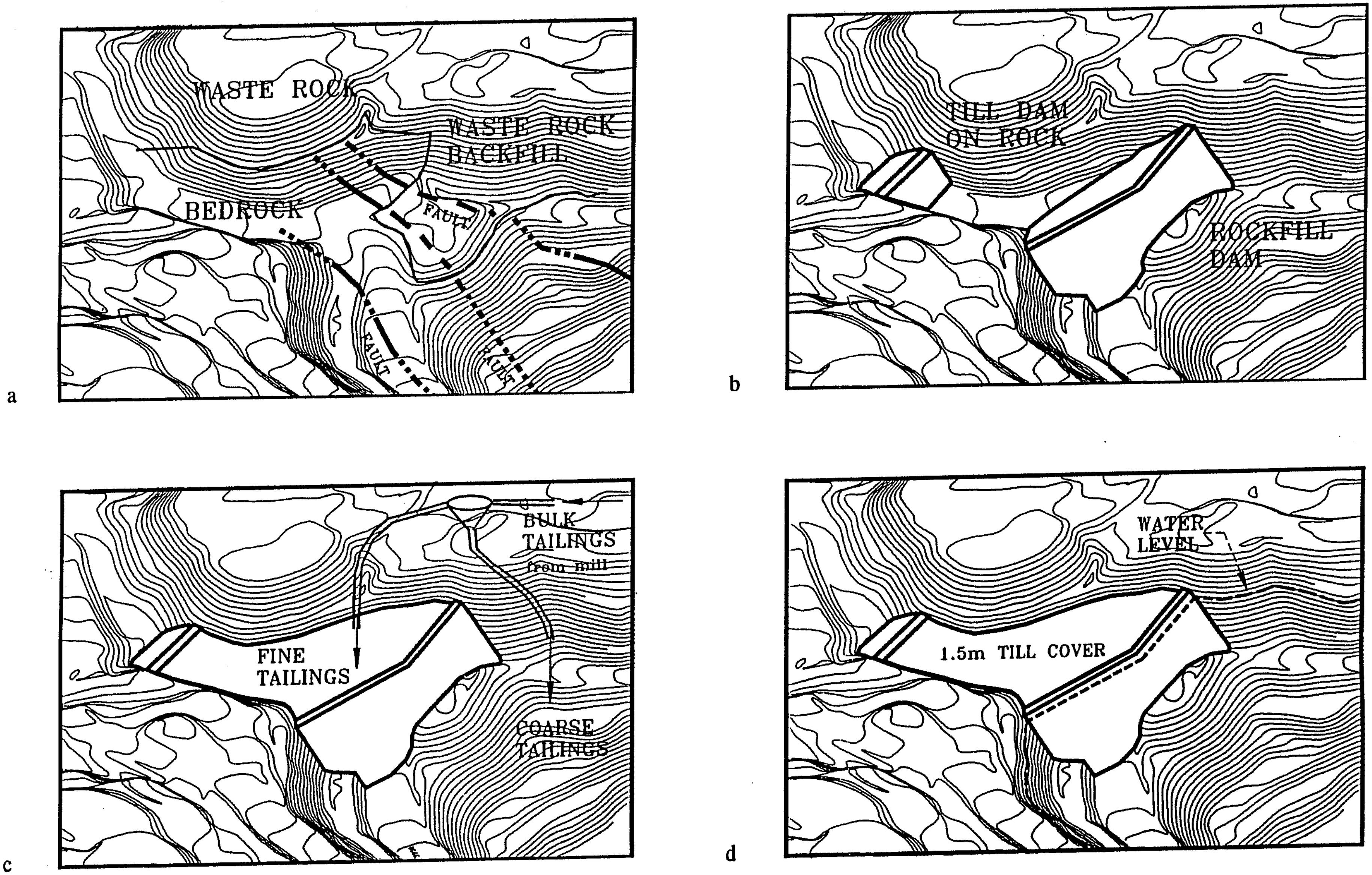

FIGURE 2 - Sequence showing the development of the Plug Dam at the southeast end of the Faro pit **a)** area as it is today showing area already backfilled and the two major faults **b)** construction of the two dams **c)** placement of cycloned fine tailings between the two dams **d)** placement of a 1.5m thick till cover over the consolidated fine tailings, water level in the flooded pit is indicated.

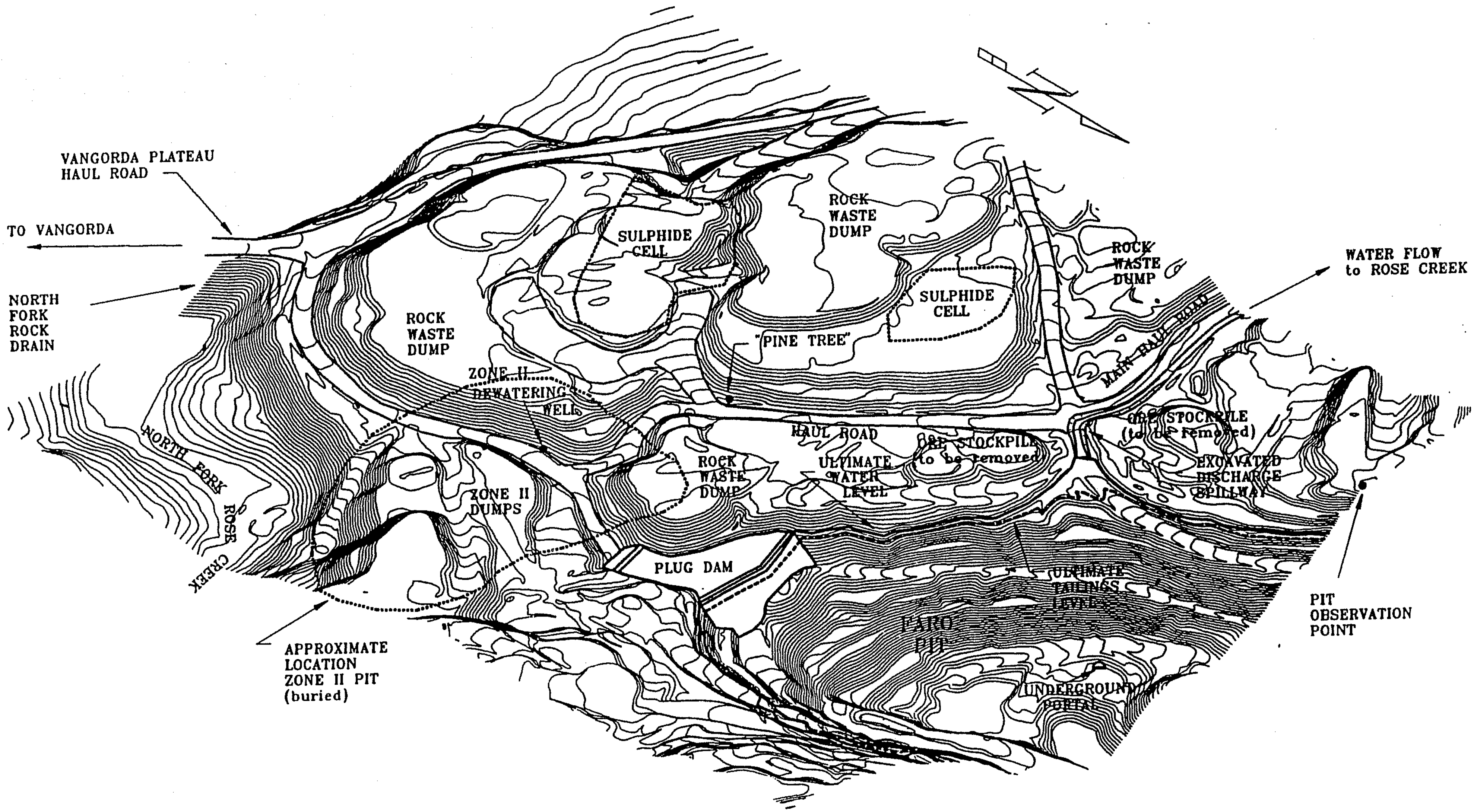

FIGURE 3 - Oblique view (looking southwest) of the south side of the Faro Pit and a portion of the Waste Rock Dumps to the southwest showing the water control system that will be in place when the pit is flooded. The level of pit flooding will be raised by the Plug Dam so that water will flow out of the pit to the west, rather than the southeast, to the North Fork of Rose Creek as proposed in the 1988 plan. Discharge from the Lake in the pit will be through an excavated spillway on the south pit wall and then via a channel following the current main haul road and portions of the North Wall Interceptor to Rose Creek (now shown).

- The current plan incorporates a Plug Dam to the southeast of the Faro Pit, in order to prevent water seepage into Zone II. This represents a significant further improvement in groundwater control and a structure much improved over the dam in the 1988 Plan.

- Water quality concerns, attendant to the filling of Faro Pit, are reduced in the current plan due to the alkalinity introduced by process water flowing into the Pit with the tailings. In the 1988 plan, the pit would have been flooded with only natural stream run-off. This change alleviates one of the major uncertainties of the 1988 plan.

- North Fork Rock drain will be breached to a substantially lower elevation than previously, so that impounded water cannot back up into the Zone II area and interfere with dump seepage collection ditches. The bulk of the rock drain will be removed and the channel of the North Fork will be suitable for fish passage.

- Faro Creek diversion will be managed as described in this report, with resultant changes from the 1988 report.

- Collection and treatment of dump seepage was proposed as a contingency measure in the 1988 Report. Seepage collection and treatment has been greatly elaborated in the current plan (through specification of ditches, sumps, pumps, and biological sulphate reduction).

- Zone II interceptor ditches are changed, as outlined in this report, in order to take advantage of a dewatering well which has been installed in Zone II since the 1988 report was written, and to enhance water control.

- Water level in Zone II is now controlled by a dewatering well rather than a drain providing improved control over the water level in the pit.

- Fresh water reservoir elevation at closure will be 2m lower in the current plan than in the 1988 plan. This will provide increased dam stability, preserve the arctic grayling overwintering habitat within the reservoir and allow fish passage between the reservoir and the lower South Fork.

4 TAILINGS RECYCLING/REPROCESSING: PHYSICAL

Between 1969 and 1991, a large volume of tailings was produced as a byproduct of the milling operations at Faro. In total, approximately 56 million tonnes of tailings were deposited behind the three tailings dams in Rose Valley. Of this total, approximately 32 million tonnes were produced by Cyprus Anvil and approximately 24 million by Curragh Resources Inc.

Tailings in the Rose Creek Valley have economic value and studies conducted by Curragh Resources, its various technical consultants and advisors have demonstrated that prospects for the economic recovery of zinc and lead sulphides from the tailings are good. The minerals to be recovered were rejected during the original milling process at Faro, for a variety of technical reasons:

- Earlier milling technology under Cyprus Anvil employed a coarser grind which failed to successfully liberate as much metal from the ore as is currently achieved;
- Cyprus Anvil operations gave highest priority to high-grade ores with correspondingly richer tailings;
- Initial operations were subject to frequent mill upsets resulting in wastage of mineral product into tailings;
- Curragh has "forced" mill throughput, maximizing production tonnages at some sacrifice of minerals to the tailings pond.

The lower the efficiency of past processing, the greater the opportunity for future reprocessing. It is estimated that approximately 800,000 tonnes of metal is contained in the Faro tailings. A portion of this metal can be reclaimed and marketed.

Curragh's environmental plan involves reclaiming approximately two-thirds of the accumulated Rose Creek Valley tailings for reprocessing in the Faro Mill and sending the residual tailings into the Pit. The primary environmental goal is to remove sufficient tailings (37.7 million tonnes) to bring the elevation of the final tailings surface (behind all three Rose Creek Valley tailings dams) down to at least 1044.3m (3426 ft.). Therefore not all of the tailings are to be reprocessed. Skimming off the highest portion of the tailings will permit the remaining Intermediate Dam Impoundment tailings to achieve the presently accepted optimum form of abandonment: permanent under-water storage. The permanent water cover will be established behind the existing Intermediate Dam. It should be noted that the 1044.3m (3425 ft.) cutoff elevation was determined to be within the storage capacity volume of Faro Pit, and consistent with the tailings facility needs of the Grum, Vangorda and Dy mines.

The motivation of the tailing reprocessing plan is environmental; however, it is also financial for it must be affordable. Previous closure plans for Faro promised flooding behind a new Cross Valley Dam of enormous scale, to be built after the mine had closed. The cost was impossibly high. Using the existing Intermediate Dam, as opposed to some new or greatly enlarged and costly structure, is an extremely important factor in the plan economics.

At decommissioning, virtually all of Cyprus Anvil tailings and some of the Curragh tailings from Faro orebodies will have been reprocessed and permanently protected under about 4.25m (14 ft.) of water in Faro Lake; while most of Curragh's tailings from the Faro orebodies will have been permanently stored under a minimum 2m of water behind the Intermediate Dam. Bulk concentrates produced in the mill will have paid for virtually all of the operating costs involved.

In order to recycle the tailings, it is first necessary to recover them physically for transfer back to the mill. Kilborn Engineering Inc. was retained by Curragh Resources Inc. to evaluate various physical transfer options, including:

- Dredging the tailings.
- Truck and shovel excavation.
- Hydromonitoring.

The most cost effective option was hydromonitoring. This involves the use of high-pressure water guns to liquefy the tailings and transfer them as a slurry to pumping points downstream. The process is not unlike hydraulic placer mining common to Yukon and Alaska, however in this case there will be a closed system with no water escaping without having been treated, if necessary. The slurry is screened to remove oversize, non-tailings material and then pumped to the mill for re-processing.

Kilborn Engineering staff has extensive experience with hydromonitoring through their involvement in the ERG project in Timmins, Ontario and the ERGO project in South Africa. At ERGO large tailings impoundments have been cleaned up by hydromonitoring. The slurried tailings have been pumped as far as 20-30km to a mill for reprocessing. At ERG, the technique was highly successful in moving large quantities of tailings and the work there gave valuable cold weather experience.

The equipment involved is fairly simple: three hydraulic monitors (two for cutting, and one cutting part time and helping to wash tailings down the ditch to the pumping sump) which can be remotely controlled; a booster pump to feed high-pressure water to the monitors; a trench along which the remobilized tailings slurry moves away from the cutting face; a concrete basin (sump) for collecting the slurry; two low lift pumps capable of moving material up to eight inches in diameter; a oversize rejection screen; a battery of high-head booster pumps to transfer the tailings to the mill (lifting it an average of 75 metres), and the transfer pipeline to the mill.

At the conclusion of reprocessing, the final tailings (or secondary tailings) are sent to the Faro Pit for permanent disposal.

This will be an environmentally desirable closed circuit. The source of water for the high-pressure monitors will be Faro Lake, and the recovery operation returns this water, with residual tailings, to the Lake. Water quality will be controlled through the addition of lime in the mill to remove base metals from solution before the water emerges from Faro Lake.

The hydromonitoring operation will employ approximately 25 persons on a 2-shift, 24-hour per day, 7-days per week operation. Some will also be employed in ongoing rehabilitation and grooming of the Down Valley tailings shoreline.

Tailings reprocessing will be a seasonal operation as hydromonitoring is not feasible when temperatures drop below freezing. Thus, both hydromonitoring and milling will be confined to approximately six months per year. This necessitates a spring mobilization and an autumn demobilization of hydromonitoring and mill processing with the inactive months providing an opportunity for equipment maintenance.

Hydromonitoring may also be interrupted by the business cycle. Since reclaiming is being funded through the sale of bulk concentrates, world metal markets may not allow the generation of sufficient cash flows from time to time, causing temporary shutdown. In this regard, the economics of reclamation will resemble the economics of most mines in a competitive, cyclical market. However, unlike most mines, this operation will be designed to shutdown and startup on a regular basis thus will have the option of much more flexible response to business cycles.

The tailings reprocessing operation anticipates the movement of 800,000 tonnes of tailings per month, or 4.8 million tonnes per year. Kilborn Engineering estimates the capital cost of the hydromonitoring equipment at approximately \$6.6 million. The cost of the tailings transfer operation will be 39 cents (Cdn) per tonne of material moved or about \$1.8 million (Cdn) per annum. This figure excludes the expense of metallurgical reprocessing in the mill as well as conditioning of the water with lime.

With slightly under 5.0 million tonnes of tailings being reprocessed each year, the requisite 38 million tonnes of Down Valley tailings can be reprocessed and transferred to permanent storage under Faro Lake in about 8 operating years.

The engineering study by Kilborn is included as Appendix A of this report where more detailed information can be found on hydromonitoring as contemplated for the Faro site. Plate 4 shows an artist's conception of the Rose Creek Valley when hydromonitoring is in progress. Plate 5 shows the Rose Creek Valley when hydromonitoring is finished and the area has been completely reclaimed.

PLATE 4 - ROSE CREEK TAILINGS AREA - HYDROMONITORING IN PROGRESS - The area is shown shortly after the hydromonitoring process has begun (year 2010). The water supply pipeline from the Faro Pit is feeding the three hydromonitors which are advancing a cut face in the tailings from right to left. The slurried tailings wash to a sump, are pumped through a classifying screen and then are pumped to the mill for reprocessing. The tailings line from the Mill to Faro Pit completes the closed system.

PLATE 5 - ROSE CREEK TAILINGS AREA - POST DECOMMISSIONING - An artists' conception of the Valley Lake formed behind the Intermediate Dam (year 2018). The Cross Valley Dam is breached. The Mill and all other buildings and facilities are removed. Disturbed area around Valley Lake is revegetated and wetland vegetation is established along the shoreline.

5 TAILNGS RECYCLING/REPROCESSING: METALLURGICAL

Value of the Rose Creek Valley Tailings

Rose Creek Valley tailings to be reprocessed have the following composition, in comparison with typical Faro ore:

	Tailings	Faro Ore
Lead	0.79%	3.50%
Zinc	1.23%	5.00%
Copper	0.15%	0.15%
Silver	15 g/t	30 g/t
Gold	0.12 g/t	0.12 g/t

Viewing the Rose Creek Valley tailings dump as just another orebody, we see that in comparison with original Faro ore, it has a much-reduced lead and zinc content, a somewhat reduced silver content, but virtually an unchanged gold content. The Rose Creek Valley "low-grade ore" has several advantages: it has already been crushed and ground, no other material needs to be moved to mine it, it is physically accessible, can be cheaply moved, and it can be metallurgically processed in facilities whose capital cost have already been paid back.

At the prices assumed for this study one tonne of tailings contains metal with a gross value of about $28 (US) as shown below:

Lead @ 30¢ US per lb.	= $ 5.22 US
Zinc @ 60¢ US per lb.	= $16.27 US
Copper @ $1.06 US per lb.	= $ 3.51 US
Silver @ $4.50 US per oz.	= $ 2.17 US
Gold @ $365 US per oz.	= $ 1.41 US
Gross Value	= $28.58 US per tonne of tailings

In comparison, at today's prices for lead (24¢ US/lb) and zinc (55¢ US/lb), the gross value of a tonne of tailings is $26.18 (US). In the first quarter of 1989 prices were considerably higher (lead averaged 28.6¢ US/lb and zinc 86.9¢ US/lb) and the gross value was $35.63 US per tonne.

The calculated worth of the metal in all 56 million tonnes of Down Valley tailings at $28.58 US per tonne is $1.5 billion dollars. However, this sum must be balanced against the cost of recovering only a fraction of those metals, shipping that fraction to market, smelting it, as well as paying for environmental shutdown at the end of the day. The question to be answered is what, realistically, can be recovered?

In order to answer that question, Curragh Resources Inc. retained a number of metallurgical consultants and a test laboratory, as well as its own metallurgists and laboratory facilities.

The Metallurgical Test Program

In September 1991, the original and second tailings impoundment areas were sampled by digging 17 holes on a grid plan with a backhoe. Samples were cut, bagged and shipped for laboratory testwork. The initial laboratory program was designed to determine the physical and chemical description of each sample, metal distribution in each screen fraction, pH variation with depth of the hole and analysis of the supernatant water leached from each grid hole composite.

Screen analyses of each sample gave varied results. Some samples indicated mineral enrichment of the finer fractions. However this could not be used to advantage in reprocessing because the indication was not universal. It was concluded that the entire tailings would have to be reprocessed rather than one specific grain size range.

Microscopic examination of the composite sample indicated that some of the zinc sulphide (sphalerite) and lead sulphide (galena) present in very fine tailings could be recovered in a commercial milling process. The balance of mineral values were bound up in grains that are a mixture of pyrite (iron sulphide), sphalerite and galena crystals which would be difficult to recover under present technology.

A composite of samples was prepared at the Faro laboratory and sent to Lakefield Laboratories in Ontario for preliminary flotation testwork. The pH of the composite tailings was 4.0 to 5.2 and indicated the need for lime for neutralization.

The Lakefield program subjected the composite sample to <u>selective</u> flotation procedures (endeavouring to recover a separate zinc concentrate and separate lead concentrate) and <u>bulk</u> flotation procedures (endeavouring to recover a mixed lead-zinc concentrate). Tests were run on the total tailings and on separated sand and fines fractions of the tailings.

Because of unfavourable economics (reagent costs exceeded metal values) the selective flotation approach was abandoned and bulk concentrate production became the focus of testwork.

Batch flotation tests with the sand size fraction and finer fraction of the tailings treated separately produced flotation results similar to the total tailings results. Reagent consumptions were high so the sand-and-fines approach was also abandoned and further work focused on test of the total tailings.

In the next series of flotation tests at Lakefield, various flowsheets and reagent schemes were tested. Total tailings were used in the evaluation of several different collectors (a reagent) to produce a bulk concentrate. The reagents tested are experimental but can be purchased commercially. Results to date are encouraging and testwork is ongoing. This work offers potential to further improve the reprocessing plan however at this time a simpler reagent scheme is proposed.

A second laboratory campaign was designed to produce a bulk concentrate from the total tailings with the simplified reagent scheme. Preliminary flotation test results indicated that it is possible in the laboratory to produce a bulk concentrate assaying 2.2% lead and 12.9% zinc at a calculated recovery of 24.5% and 63% respectively. These are encouraging results which call for further testwork to optimize concentrate grade and metal recovery. This testwork is ongoing. Considering the metallurgical response obtained, Curragh concluded that a marketable bulk concentrate is achievable.

Expected Plant Metallurgical Balance

The expected metallurgical balance was derived from the preliminary test data outlined above, a review of sample mineralogy, knowledge of the Faro plant performance and discussions with laboratory metallurgists at Lakefield and Faro, as well as outside metallurgical consultants.

The conclusion is that a bulk concentrate with the following specifications can be produced at the Faro mill from Rose Creek Valley tailings feedstock:

Lead content	14.2%
Zinc content	37.1%
Silver content	165 grams per tonne
Metal Recovery	25% (combined Pb/Zn)
Weight Recovery	1% of feed weight will be recovered; i.e. it is necessary to process 100 tonnes of tailings to recover one tonne of concentrate

Reprocessing Flows in the Mill

While it appears technically possible to produce a bulk concentrate from Down Valley tailings, it is necessary to accomplish this within the processing capabilities of a modified Faro mill. Metallurgical engineering at Curragh has concluded that a simple, modified circuit can do the job.

Hydromonitored tailings from the Down Valley will be pumped to a slurry surge tank (converted fine ore bin) in the mill. Lime will be added to the slurry, as well as the following reagents: an activator, a collector and a frother. The slurry will be pumped to six banks of flotation cells where preliminary bulk concentrate will be recovered. The preliminary bulk concentrate will be subjected to finer grinding; then it will be treated again with the same suite of reagents; and subjected to three stages of further flotation (cleaning). The final bulk concentrate will be thickened, filtered, and thermally dried so that it is ready for truck and ship transport to market. The residual tailings will be pumped to the Faro Pit.

Mill Operations

In order to assess the economics of this operation, it was assumed that the Faro Mill circuitry would be modified (simplified) to produce bulk concentrate, and that it would process Rose Creek Valley tailings 2-shifts per day, 24 hours per day, 6 months per year. Other key operating assumptions are:

Plant availability:	90%
Tailings processed:	1,234 tonnes (dry) per hour
	4.8 million tonnes (dry) per year
Concentrate output:	48,000 tonnes (dry) per year

It may be noted that, during reprocessing, the 6 month concentrate tonnage output will be slightly under one-tenth the output achieved by the Faro Mill in a typical 12 month year during regular operations.

A crew consisting of operators and contractors will mobilize the equipment each year before the spring start up, and winterize the equipment after operations cease in the fall.

The operating crew will comprise of:

Management/supervision	9
Operators, Tradesmen, Service	20
Total	29

Two employees will be retained on the payroll at the end of each operating season to caretake the minesite (buildings and equipment) and perform any monitoring and other environmental duties necessary. Caretakers will be required to make daily inspections of the minesite.

The electric power contract will have to be renegotiated to reflect seasonal use (summer usage only) which should be advantageous to the utility company.

For environmental monitoring, sampling will be done in the winter by the caretakers and in the summer by a designated staff person. During periods of more intense sampling, assistance will be organized by the Curragh Resources Inc. regional office in Whitehorse. All data will be recorded and interpreted at the Curragh regional office.

Total employment sustained by the tailings recycling/reprocessing operation will be 90 persons, excluding any multiplier impact on the Yukon. This will consist of 25 persons in hydromonitoring, 29 persons in milling and 35 persons in trucking and shipping.

Capital Cost Estimate

Capital costs for Mill modifications are estimated to be $500,000. The existing mill circuit will have to be modified to accommodate the reprocessing of Rose Creek Valley tailings, the production of a bulk concentrate, and the pumping of the tailings to the Faro Pit for final deposition. The capital cost estimate does not include provision for a second tailings pipeline and pumping station nor does it provide for a second reclaim water siphon pipeline from the open pit to the mill since these capital costs have been included in the hydromonitoring capital costs estimated by Kilborn Engineering.

Operating Cost

The operating cost for reprocessing has been based on reagent consumptions derived from the metallurgical tests described above, the staffing requirements outlined above and on extensive operational experience within the mill. The cost projection for hydromonitoring is based on the Kilborn Engineering Inc. estimate described above. Costs are presented in Canadian dollars per metric tonne of tailings reprocessed. The costs tabulated below include all costs required to move the tailings from Rose Creek valley, reprocess them, provide water conditioning to maintain water quality and pump the secondary tailings to the Faro Pit for final disposal.

Cost Centre	Cdn Dollars per Tonne of tailings
Manpower	$0.20
Reagents	$1.28
Supplies	$0.10
Power	$0.31
Subtotal Mill operating cost	$1.89
Hydromonitoring operating cost	$0.39
Total Operating Costs	$2.28

Minesite "Net Back" Value of the Bulk Concentrate

The estimated bulk concentrate dollar value returned to the minesite is based on projected smelter treatment charges and actual historical transportation and marketing costs as incurred by Curragh Resources in Yukon. The metal prices assumed were:

zinc	$0.60 US per pound
lead	$0.30 US per pound
silver	$4.50 US per ounce
Rate of exchange	$1.14 Canadian per US dollar

In valuing the netback at the Faro site for Rose Creek Valley bulk concentrate, we have assumed the standard ISF smelter contract with a zinc payment calculated on the basis of the lesser of 85% of the contained zinc metal or a deduction of 7% from the grade of the concentrate (due to the low grade of the bulk concentrate the 7% grade deduction gives the lower payable metal). Similarly the lead payment is the lesser of 95% of the contained lead or a deduction of 3% off the lead grade (again the grade deduction gives the lowest grade payable lead). Silver payment is the lesser of 95% or a deduction of 75 grams per tonne from the silver grade (the grade deduction again gives the lower payable silver). No payment for contained copper or gold was included.

The net-back value at mine site in Faro, under these assumptions, is $199 (US) per tonne of bulk concentrate shipped. Because 100 tonnes of tailings must be reprocessed to produce one tonne of concentrate this is equivalent to $1.99 (US) OR $2.27 Cdn at the assumed exchange rate, per tonne, of Rose Creek Valley concentrate processed.

The mine site net back represents the gross value of the metal less a) the payable metal deductions; b) the smelter treatment charge; c) marketing costs; d) ocean freight; e) storage and loading of the concentrate at the Skagway port facility; and f) trucking to Skagway. All site costs must be paid for from the net back value. If the site costs equal the net back the project breaks even. As noted above, site operating costs are $2.28 Cdn per tonne of tailings reprocessed. Thus, at the prices assumed the project will breakeven. At higher prices the project will make money and at lower it will lose money. The key to operating this project is to average the prices assumed.

Annual average zinc prices for the last twenty years are tabulated below. The average price for 20 years expressed in 1991 dollars is 60¢ US per pound, thus the assumed prices are reasonable. The average price for lead for the same period is 43¢ US per pound in 1991 dollars. The lead price selected is lower and more representative of the price average for the last 10 years in 1991 dollars.

Lead and zinc markets are expected to remain healthy for the foreseeable future. Zinc will see increased use in galvanizing and its pricing can be expected to follow activity cycles in heavy industry, automobiles and construction. Lead will see increased use as electric cars become more widespread but this demand may decline as viable alternative technologies for storage batteries are developed in the future. Little new supply of lead has been developed thus prices may remain stable or rise.

Technical detail and backup to support and elaborate upon the statements in this section can be found in Appendix C of this report.

	Zinc LME* SHG Cash US cents/lb	Zinc price In 1991 US cents/lb	1990 GDP Deflater
1970	13.9	45.3	37.88
1971	15.5	47.8	40.01
1972	17.7	52.1	41.93
1973	19.7	54.4	44.65
1974	35.3	89.6	48.62
1975	36.9	85.2	53.44
1976	36.1	79.4	56.08
1977	32.4	66.0	60.61
1978	27.5	52.2	65.01
1979	35.8	62.4	70.82
1980	36.1	57.7	77.27
1981	41.3	60.2	84.71
1982	38.0	52.0	90.14
1983	37.8	49.9	93.60
1984	45.4	57.7	97.10
1985	38.3	47.3	100.00
1986	36.3	43.7	102.59
1987	37.5	43.7	105.84
1988	56.6	63.8	109.36
1989	78.0	84.6	113.84
1990	68.9	71.8	118.48
1991**	52.9	52.9	123.45
Average	38.1	60.0	

* London Metal Exchange Super High Grade - European Producer Price used for years between 1970 and 1987

** 1991 Price is the year's average to July 3, 1991

6 ENVIRONMENTAL MANAGEMENT

Curragh Resources has a commitment to environmental management and monitoring throughout the life of mining at Faro and thereafter. Our primary environmental objective is to ensure high water quality.

Environmental Policy

Curragh has adopted the Mining Association of Canada Environmental policy and has committed the Company to this strong statement on environmental protection (see Appendix E).

Management Organization for the Environment

The organizational structure for environmental management of the Faro complex currently encompasses:

- A full-time Manager of Environmental Affairs, based in Whitehorse
- A full-time Environmental Engineer at Faro
- A full-time Environmental Technician at Faro
- Long standing relationships with the following consultants with special technical and environmental expertise:
  - --Steffen, Robertson and Kirsten
  - --Kilborn Inc.
  - --Richard F. Down, P. Eng.
  - --A. S. (Daniel) Webster
  - --Lakefield Research
  - --Rescan Environmental Services
  - --Cominco Engineering Services Ltd.
  - --Golder and Associates
  - --Piteau and Associates Ltd.
- Regulatory input from the Yukon Territory Water Board
- Technical input from the following Federal Government Agencies:
  - --Department of Indian Affairs and Northern Development
  - --Environment Canada, Environmental Protection Service
  - --Department of Fisheries and Oceans
- Technical input from the following Yukon Territory Government Agencies:
  - --Department of Renewable Resources
  - --Department of Economic Development
- A senior management and Board of Directors committed to funding the required environmental activities (in recent years at least 10% of Faro's capital expenditures have been environmental).

This network of private and public expertise assists Curragh in discharging its responsibilities with respect to environmental management issues for the Faro site.

In the context of the decommissioning plan for the Faro site these include the following areas.

Rerouting Tailings Deposition to the Faro Pit from Rose Creek Valley

When tailings are deposited into the Faro Pit instead of Rose Creek Valley, water and alkalinity inputs to the Rose Creek Valley tailings will decrease, however, the tailings pond water quality will be maintained. In the event that soluble metals values increase, Curragh will employ lime as a neutralizing factor and corrective, probably discharged as lime slurry from the mill via the current tailings line. Other methods of lime application as outlined in the April 1991 report are being investigated. Regular monitoring of water quality will allow for the detection of any harmful zinc enrichment and initiate the appropriate remedial action .

Handling Excess Water from the Faro Pit

Inflows to the Faro Pit, once flooded, will be greater than outflows. To prevent overtopping of Faro Lake the water-recycling siphon to be installed has been designed with excess capacity. Excess water will be siphoned off to the mill, and treated if necessary before discharge to the Intermediate Tailings Impoundment and Cross Valley Pond where it will be held before discharge to the environment.

Larger than normal inflows to the pit will be handled by operating the siphon so that the Faro Lake water level is drawn down and excess flow can be stored in the pit until discharge.

Closed Circuit During Reprocessing

During reprocessing of the Rose Creek Valley tailings, a closed circuit will exist among the Faro Pit, Mill and Hydromonitoring. No water will be released into Rose Creek directly unless it meets discharge standards. Lime will be added in the mill to prevent excessive acidity and to ensure that dissolved base metals are precipitated into the pit tailings. Excess flows will be managed as noted above.

Environmental Design Standards After Decommissioning

During the decommissioning at Faro, Curragh will construct, modify and/or eliminate various elements of the complex water-handling system described earlier in this report (Section 3). For example, new dams, new spillways, and new interceptor ditches must be constructed. The design standard will vary, but at a minimum new structures in Rose Creek Valley will withstand a one in 500 year flood. The seismic event (earthquake) standard for the Intermediate Dam in Rose Creek Valley will be one event in 475 years. The intake and outlet for Faro Lake will be designed to accommodate a one in 500 year event.

Environmental Management Organization after Decommissioning

Upon final closure, now estimated to occur in the year 2026, the Trusteed Environmental Fund which has been established for Faro will create sufficient revenues to support an on-going environmental management organization and program. A site supervisor and two full-time technicians will be employed to:

- operate the biological treatment system
- monitor water quality
- implement the downstream aquatic life monitoring program
- maintain site infrastructure (pumps, pipelines and instruments, etc.)
- inspect physical structures for safety and stability
- provide site security
- communicate and report

7 WATER TREATMENT

An important feature of the Faro environmental plan is provision for the treatment of water seeping from the waste rock dumps and from exposed sulphide areas at in the pits. The first line of environmental defence is to minimize water and oxygen contact with sulphide materials. Since much of the Faro waste dumps were constructed before acid mine drainage was realized to be a significant concern, the first defense is not available and we must retreat to a second and more expensive line of defense: to collect such seepage and to render it harmless.

Dump Seepage Collection

Dump seepage must be collected before it can be treated. There will be five collection points:

- a sump on the northwest corner of Faro Pit
- the Zone II Pit
- interceptor ditches at the toe of northeast dumps
- interceptor ditches at the toe of south dumps
- interceptor ditches at the toe of the southeast corner of the dumps immediately below the Vangorda Plateau Haul Road

From these collector points, contaminated water will be pumped to the underground biological treatment system. The system is capable of treating a sizeable volume of water at comparatively high base metal loadings.

Biological Water Treatment

In view of the importance of the water treatment issue, Curragh Resources Inc. requested SRK to evaluate the potential of establishing an in-situ sulphate reduction process as a water treatment system. Curragh's motivation was reports of a similar natural phenomenon occurring at the Løkken Mine in Norway.

The study indicated that, upon closure, the underground mine can be used as a "biological precipitator" in a sulphate-reduction system.

It should be noted that while the Faro Mine has been mainly open-pit, underground mining has also been carried out at Faro since 1990. Underground mining was the most efficient way to extract the lower extremities of ore. Today, a network of tunnels extend from the bottom of the pit. When underground mining terminates at Faro in the spring of 1992, a complex of abandoned tunnels will remain.

In mine water, sulphate originates from the naturally occurring chemical and bacterial oxidation of sulphide minerals exposed by mining to air and water. The scientific community has recognized the potential for application of a naturally occurring biological process to reverse the oxidation of sulphide minerals. Such biological process can remove the contaminants from mine and waste dump water and convert undesirable metal-rich water into environmentally acceptable water. Bacteria therefore return dissolved metals to the minerals from which they started: to zinc sulphide and lead sulphide, only now it is sludge deposited safely on the floor of the underground workings.

As applied to Faro, potentially harmful drainage will be collected as it emerges from the waste dumps, and will be pumped into the underground workings which will become "biological precipitators". The underground mine workings at Faro will be fitted with appropriate drill holes from the surface, pumps and instrumentation to manage water in-flows and out-flows. Multiple inlet and outlet drill holes will make process control more efficient and optimize use of the available volume in the mine workings. A schematic view of the treatment system is provided in Figure 4.

The SRK study discusses the overall biological sulphate reduction process, its success in other locations, and its potential at Faro. Once set into motion, the process is a model of simplicity: untreated, acidic, metal-rich, and potentially hazardous waste dump drainage is pumped in; benign water acceptable for discharge to the environment is pumped out.

A preliminary water management system was developed by SRK primarily for the evaluation of the biological water treatment system, but also to demonstrate the feasibility of overall water handling systems at Faro. The system provides for the collection of contaminated run-off in five retention structures. From these retention sumps, contaminated water will be pumped into the "biological precipitators". Naturally clean run-off will be diverted to prevent contamination.

The calculated rate of sulphate reduction will generate sufficient hydrogen sulphide to remove effectively all of the base metals in solution. Total treatment capacity will be considerably in excess of the expected volume of drainage and metals load.

The bacteria which sustain the reaction must be fed. Evaluation of potential food sources to sustain the sulphate-reducing bacterial culture indicates that feeding with sugar will be most cost-effective. Approximately one truck-load of raw industrial sugar will be pumped down from the surface every 4 to 6 weeks to keep the bacteria on the job.

An analysis of the stability of the existing underground mine shows that the existing workings should provide an adequate network of natural travel paths for efficient operation although local instabilities could alter the flow path over time. Sludge build-up (from precipitation of metal sulphides) will not significantly affect the process for many years, after which redistribution of the sludges by flushing may be required to maintain high reaction rates.

Details of the Biological Treatment system are provide in Appendix B of this Report.

BIOLOGICAL TREATMENT SYSTEM

FARO UNDERGROUND

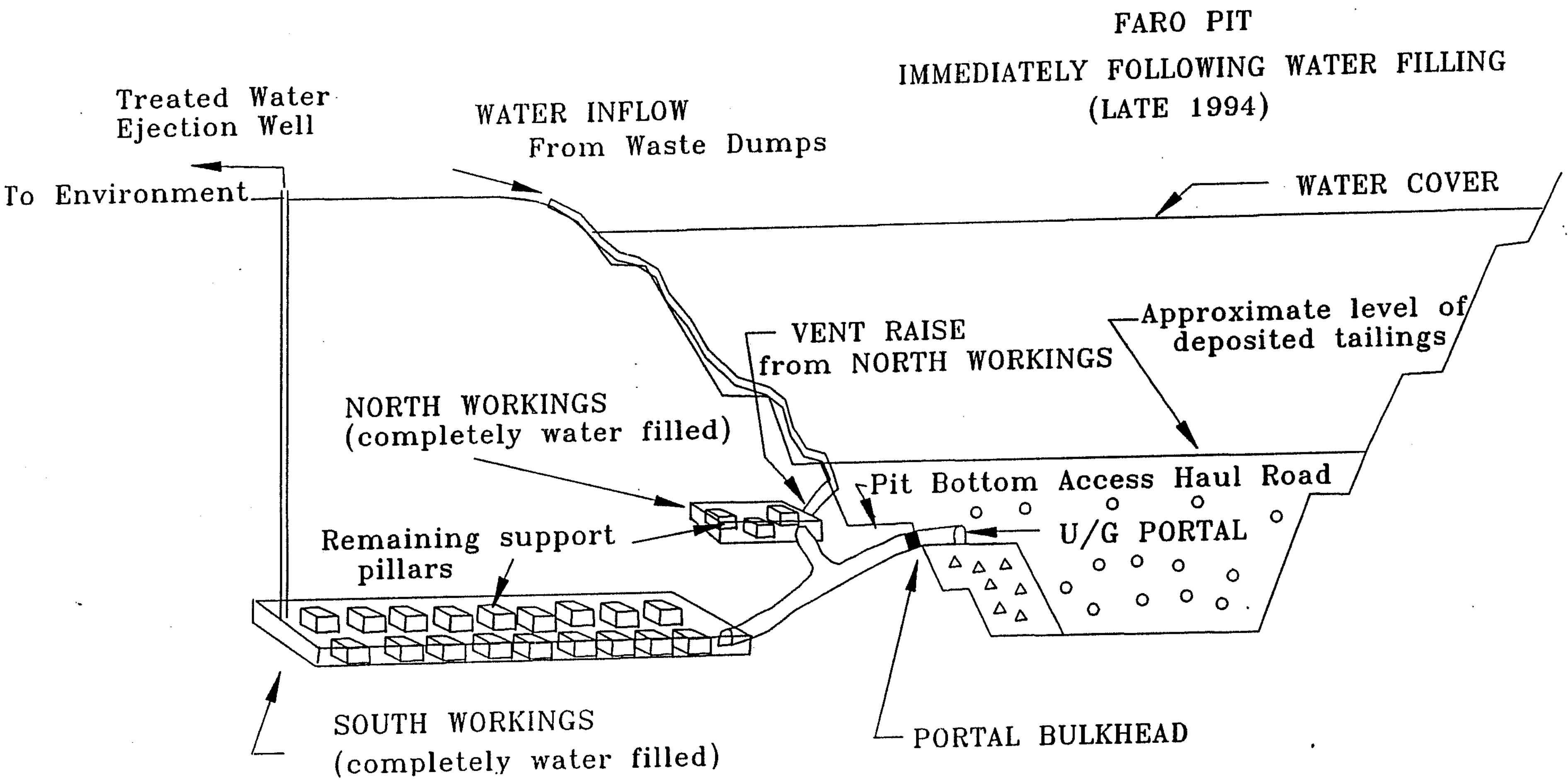

FIGURE 4 - BIOLOGICAL TREATMENT SYSTEM SCHEMATIC - Contaminated, dump seepage is pumped underground through a pipeline connected to the vent raise of the North Workings. This area increases the water temperature such that bacterial sulphate reduction may occur at an accelerated rate within the South Workings. Dissolved metals in the water combine with the free sulphide produced by the bacteria and settle out as a small volume of sludge. Water, suitable for release to the environment, is then pumped from the ejection well at the downstream end of the South Workings.

8 WATER QUALITY

Preservation of water quality is a primary objective in Curragh's environmental plan. Schedule D of license IN89-001 is a series of studies and investigations into various closure issues at the Faro Site. The Schedule D researches culminate in the development of an all inclusive water quality model for the site. This model will be submitted to the Yukon Territory Water Board with the final Schedule D report in March, 1992. We have used a preliminary version of this model to make the following estimates of downstream water quality in Rose Creek.

Final water quality will be a blend of the quality of water discharged in three different areas:

1. Water Quality from the Rose Creek Tailings Area

Five alternatives were examined for the decommissioning of the Rose Creek Tailings Area:

- Tailings exposed to the atmosphere
- Cover the tailings with natural soil material
- Water cover without reprocessing of tailings (the original Cyprus Anvil "large dam" approach)
- Composite soil/fine tailings/coarse rock cover on Original and Second Tailings Impoundments, and water cover over the Intermediate Tailings Impoundment
- Reprocessing of tailings and permanent water cover

Of the five alternatives evaluated, the one selected (re-processing of tailings and permanent water cover) yielded the most favourable final water quality results. This was because the selected alternative:

- minimizes zinc and other base metals concentration in surface waters by placing a water cover over the tailings to prevent further oxidation;
- minimizes the volume of seepage into the groundwater in comparison with other water-cover alternatives; and
- totally removes the oldest and coarsest tailings piled above Rose Creek rather than simply covering them.

Water quality modelling by SRK applied to Rose Creek downstream from the tailings area suggests that peak monthly zinc concentrations will be very low (0.007 mg/l) and flow-weighed average zinc concentrations will be correspondingly low (0.001 mg/l). Drainage from the tailings area at Faro has in the past been essentially neutral (pH about 7.5) and is expected to continue to be so in the future.

2. Water Quality from Faro Pit (Faro Lake)

During the placement of tailings in the Faro Pit, a water cover exists through the diversion of Faro Creek. This water cover will prevent oxidation of the tailings and subsequent metal pickup in the water. The alkalinity of the fresh tailings will suppress any soluble base metals released from the pit walls and floor. This plan is far more effective than the previous plan (1988 Abandonment Plan) which allowed the pit to flood with natural surface water.

The plan incorporates filling (hence sealing) the pit with tailings, and the building of a Plug Dam of low permeability which will restrict the leakage of pit water into Zone II where it would become contaminated.

Water quality modelling indicates that downstream flows out of Faro Lake after decommissioning will be of a high quality.

3. Water Quality from the Waste Dumps

Almost all seepage from the waste dumps will be collected and treated in the biological water treatment system. The previous (1988) plan called for some form of collection and treatment, but details were not provided. The core assumption, at that time, was treatment by adding lime to the effluent. While lime is effective, it is expensive and creates a sludge disposal problem.

The technology underlying Curragh's current (1991) plan--biological treatment--is based on a process demonstrated to occur naturally in the Løkken underground mine in Norway. Conditions at the Faro minesite are conducive to biological treatment since pH of the contaminated water is near-neutral and copper content is very low. The North Workings of the Faro Underground Mine will be used primarily to warm the water to make the biological process most efficient in the larger South Workings.

The input to the biological precipitator will be quite contaminated while the output is anticipated to be benign.

Total Impact on Water Quality

The total impact of the Faro site after implementation of the decommissioning plan on water quality is encouraging. Application of a preliminary water quality model to the entire Faro complex indicates that expected water quality downstream in Rose Creek will be very good.

Water quality will meet the CCREM (Canadian Council of Resource and Environmental Ministers) aquatic environment toxicity guidelines for zinc (maximum zinc 0.030 mg/l) below the tailings in Rose Creek. With a discharge quality from the underground biological precipitator of 0.020 mg/l, <u>peak</u>

monthly zinc values below the tailings area are estimated at 0.029 mg/l, and the flow-weighted average for the year is estimated at 0.022mg/l. As a result, there should be no long term toxic impact on downstream aquatic life.

9 TIMING

The scheduling of the Decommissioning at the Faro site must take several inter-related factors into account:

- the need to find storage for the tailings produced from Grum, Vangorda and the potentially new mine, Dy;
- Curragh's inability to reprocess Faro tailings in the mill while processing other ores;
- the need to secure the environment on an interim basis, until the full decommissioning program has been completed;
- the vagaries of world metal markets, which will shape the cash flows achievable from bulk concentrate sales; and
- new technology, which could dramatically improve the economics of tailings reprocessing and accelerate the schedule.

Time sequence of events are planned as follows:

1992	Cessation of mining the Faro Pit
	Cessation of mining in Faro Underground
	Modify underground facilities for future water treatment
	Accelerate mining at Vangorda Pit
	Stripping program commences at Grum Pit
	Faro Pit prepared for use as tailings facility
	Diversion of Faro Creek into Faro Pit
	Commence deposition of Grum/Vangorda tailings in Faro Pit and halt deposition of tailings in Rose Creek Valley on regular basis
	Continued lime treatment of waste dump seepage
	Commence lime stabilization of Rose Creek Tailings
1993	Commence construction of the Zone II Plug Dam
1994	Faro Lake at final water elevation
	Commence recycling of process water from Faro Lake

From this point in time onward all dates are assumed on the basis of the current mine plan.

1997	Commence development of Dy Mine
1998	Commence placement of Dy tailings under Faro Lake
2008	Completion of mining at Grum, Vangorda and Dy
	Completion of mining in the Anvil District
	Terminate placement of Grum/Vangorda/Dy tailings in Faro Lake
	Commence conversion of Faro Mill to reprocessing circuitry
2009	Commence hydromonitoring of Rose Creek Tailings
	Commence reprocessing of Rose Creek Tailings
	Placement of Rose Creek Tailings into Faro Lake
	Marketing of bulk concentrates
2017	Reprocessing of tailings completed
	Tailings in Faro Lake at final submerged elevation
2018	Commence final decommissioning program on structures
	Build spillways for Faro Lake inlet and outlet
	Build Intermediate Dam spillway
	Breach Cross Valley Dam
	Reduce height of Freshwater Reservoir Dam
	Breach the North Fork Rock Drain
	Construct waste dump interceptor ditches and sumps
	Drill holes into underground precipitator
	Complete facilities for underground water treatment
	Commence biological treatment of seepage from waste dumps
2020	Divert Rose Creek into Intermediate Dam
	Flooding of Rose Creek Tailings
	Re-vegetation of waste dumps and disturbed areas
	Create wetlands environment in Rose Creek Valley
2021	Begin intensive post-closure monitoring
2023	Begin long-term maintenance monitoring program

While the above program is consistent with best available technology and known economics as of 1991, it should be recognized that metallurgical and reclamation technology continues to evolve rapidly, as do world markets. Accordingly, the timing and methodology of closure may change in the future, necessitating modifications to the plan.

APPENDICES

APPENDIX A. *(bound separately)*

Curragh Resources, Faro Mine Tailings Relocation Project, Preliminary Design and Cost Estimates, Report Number 3509-62, Kilborn Inc., Toronto, Ontario, December 1991

APPENDIX B. *(bound separately)*

Sulphate Reduction as a Water Treatment Alternative at the Faro Mine, Report Number 60643, Steffen, Robertson & Kirsten (B.C.) Inc., Vancouver, B.C., December 1991

APPENDIX C. *(bound separately)*

Curragh Resources Inc., Faro Division, Reprocessing Tailings, Report Number WH91-07, R.F. Downs, P. Eng. and G.W. McDonald, Toronto, Ontario, December 1991

APPENDIX D. *(bound separately)*

Letter Report on Geotechnical considerations for Faro Pit Decommissioning, File #85-80413, Alan F. Stewart, P. Eng., Piteau & Associates Engineering Ltd., December 13, 1991

APPENDIX E.

Signed copy of the Mining Association of Canada Environmental Policy adopted by Curragh Resources Inc.

APPENDIX F.

A cautionary note to the reader on Faro Elevations

THE MINING ASSOCIATION OF CANADA ENVIRONMENTAL POLICY

Member companies of The Mining Association of Canada are committed to the concept of sustainable development which requires balancing good stewardship in the protection of human health and the natural environment with the need for economic growth. Diligent application of technically proven and economically feasible environmental protection measures will be exercised throughout exploration, mining, processing and decommissioning activities to meet the requirements of legislation and to ensure the adoption of best management practices. To implement this policy, whether in Canada or abroad, the member companies of The Mining Association of Canada will:

- assess, plan, construct and operate their facilities in compliance with all applicable legislation providing for the protection of the environment, employees and the public;
- in the absence of legislation, apply cost-effective best management practices to advance environmental protection and to minimize environmental risks;
- maintain an active, continuing, self-monitoring program to ensure compliance with government and company requirements;
- foster research directed at expanding scientific knowledge of the impact of industry's activities on the environment, of environment/economy linkages, and of improved treatment technologies;
- work pro-actively with government and the public in the development of equitable, cost-effective and realistic laws for the protection of the environment; and
- enhance communications and understanding with governments, employees and the public.

Curragh Resources Inc.

hereby agrees with The Mining Association of Canada
to adhere to the above policy.

FOR THE COMPANY

Chairman

Vice Chairman

DATE May 29th, 1990

FOR THE MINING ASSOCIATION OF CANADA

CHAIRMAN

PRESIDENT

APPENDIX F

A CAUTIONARY NOTE TO THE READER ON FARO ELEVATIONS

Elevations used in Pit operations at Faro have historically been expressed in feet, and are often but not always given with respect to "Mine Datum". Elevations used in Tailings Pond operations at Faro have historically been expressed in metres, and are often but not always given with respect to "Down Valley Datum". The benchmarks for elevations used in the Pit and the Tailings Area are not the same (the differential being approximately 100 feet). The mean sea level datum is not quite consistent with either of the two benchmarks. These surveying peccadilloes derive from the original wilderness staking of these claims, and they long predate Curragh Resources Inc. involvement at Faro. For clarity in this report, all elevations are expressed in metres and feet, translated to the same common and identical benchmark. That benchmark is the most recent mean sea level datum for the area and is approximately the same as the elevation datum for the most recent Government of Canada 1:50,000 maps for the Region. Readers of reports on the Faro area, including technical reports tabled before the Yukon Territory Water Board by Curragh Resources Inc., its consultants, and its predecessors, are advised to confirm the elevation data used before comparing elevations.

Elevation conversion rules:

To convert from mine datum to mean sea level subtract 33.3m or 109.2 ft.
To convert from mean sea level to mine datum add 33.3m or 109.2 ft.
To convert from Down Valley datum to mean sea level substract 32.3m or 106.0 ft.
To convert from mean sea level to Down Valley datum add 32.3m or 106.0 ft.

REFERENCES

("CRI Abandonment April 1988") Faro Mine Abandonment Plan, Whitehorse, Yukon, April 1988; prepared by R. McLenehan, Environmental Engineer and J. Bower, Mine Engineer.

("CRI Other Facilities 1989) Curragh Resources Inc., E. Soprovich, Environmental Engineer, Other Facilities Abandonment Plan, Whitehorse, Yukon, June 1989

("Kilborn Recycle 1991") Kilborn Inc., Curragh Resources Inc., Faro Mine, Water Recycle and Tailings Deposition Plan, Kilborn Engineering, Report Number 350928, Toronto, Ontario, June 1991. Filed with Yukon Territorial Water Board in July, 1991.

("SRK Down Valley 1991") Steffen Robertson & Kirsten, Down Valley Tailings Impoundment Decommissioning Plan, Faro, Yukon. SRK Report Number 60635, Vancouver, B. C., April 1991.

Volume I: Report
Volume II: Appendices A and B
Volume III: Appendices C to K

Filed with Yukon Territorial Water Board in April 1991.

("SRK Water Quality 1991") Steffen Robertson and Kirsten, Water Quality Assessment, Tailings Disposal in the Faro Pit, SRK Report No 81703, July 1991, included as Appendix A to Kilborn Recycle 1991 report.

2

ENVIRONMENTAL PERMITS/LICENSES/APPROVALS
NEEDED TO MINE IN YUKON

In Yukon, once one has a mineral claim (a quartz claim) granted under the Yukon Quartz Mining Act, one may mine on that claim. There is no additional mining permit or mine development permit needed. (Note - This situation will change within the next 2 years). The mining operation must be conducted in compliance with other relevant Acts, primarily the *Northern Inland Waters Act*. Most of the statutes listed below are Federal, unless otherwise noted.

Water License Under *Northern Inland Waters Act* (soon to be replaced by *Yukon Waters Act*)

A water license is required to:

- to divert, store, or otherwise use water
- to discharge waste into water

A water license is issued by The Yukon Territory Water Board (YTWB) and is not effective until signed by the Minister of the Department of Indian Affairs and Northern Development (DIAND). A water license defines quantity of water that may be used, rate of use, times of use, sources of water, methods of disposal of waste, monitoring programme, compliance levels for various metals and chemicals in discharge, reporting of spills, studies related to abandonment, financial security for closure, and defines abandonment measures. A water license normally requires monthly, annual and special non-periodic reports of water quality, and other matters related to work at the minesite and abandonment of the mine site. DIAND is responsible for enforcing compliance with the license. A water license for a major mine normally requires a public hearing. An amendment to a license to change <u>any</u> term and condition can also require a public hearing.

Decision Report Under *Environmental Assessment and Review Process Guidelines Order (EARP)*

The EARP Guidelines require federal agencies, such as DIAND, to screen the environmental effects of projects before making a decision (such as signing a water license). The screening is to determine if the environmental effects are known, and if known, if significant, and if significant then whether the effects can be mitigated. The Decision Report confirms that the screening is done and identifies what mitigation is required, and what form of commitment the project proponent should show. The EARP Decision Report is not binding. It is advisory in nature but the screening **must be done**. In the next year the EARP Guidelines will be replaced by the screening process defined by the *Canadian Environmental Assessment Act* which is more formalized and provides for public participation.

Two agreements (one for Vangorda/Grum and the other for Sã Dena Hes) between Curragh and the Crown have been executed to secure commitment to reclamation issues, principally financial security, which was recommended by the EARP screening, but were beyond the statutory powers of Government to demand. One of these agreements on the Vangorda site is currently before the Federal Court of Appeals.

Land Use Permit Under *Territorial Lands Act - Land Use Regulations*

Land use permits are **not required** for work done on mineral claims or leases; however, any work done off of the claims or leases would require a permit. In practice all Curragh activity is confined to the mineral property and no land use permits are required. A coal mine or quarry not on quartz claims would require a permit. Carmacks coal is an example. Similarly, harvesting or clearing timber is allowed on quartz claims **without** a permit.

Surface Lease Under *Territorial Lands Act - Lands Regulations*

A surface lease is required to give exclusive right to the surface and to deny trespass, etc. A surface lease is not strictly required to mine in Yukon since a mineral claim carries with it the right to use the surface for mining purposes as well as subsurface rights to minerals, but this right is not exclusive.

All operations must be carried out in compliance with the *Fisheries Act* which states that no discharge of a substance deleterious to fish is allowed into waters frequented by fish, and provides significant penalties for unauthorized damage to fish habitat. The allowable discharge for most metal mines is defined by the *Metal Mine Liquid Effluent Regulations* issued under the Act. No permits are required by the *Fisheries Act*, however, approvals may be required from the **Department of Fisheries and Oceans (DFO)** for stream crossings or other fish habitat disturbance. The *Fisheries Act* is a very powerful piece of legislation and draws the **Department of Environment (DOE)** into the scene as DOE is responsible for assisting DFO in enforcing the Act.

The *Canadian Environmental Protection Act (CEPA)* does not have direct and specific impact in mining, except for regulating specific toxic substances and certain special circumstances such as ocean dumping. The Act could be used to regulate activity on all Federal land but has not been so used.

Closure of a mine must be in compliance with the *Mine Safety Regulations of the Occupational Health and Safety Act* (a Territorial statute).

OTHER PERMITS OR APPROVALS
MAINLY SAFETY AND HEALTH RELATED

All of the following statutes are Territorial unless otherwise listed. Copies of these have not been provided as they are relatively routine matters.

- Explosive Magazine Permit & Yukon Blasters Permit under *Yukon Blasting Ordinance*
- Compliance with Mine Safety Regulations issued under *Occupational Health and Safety Act*
- Permit from **Federal Department of Energy Mines and Resources** to transport over 1000 kg of explosives
- Permit to burn under *Forest Protection Ordinance*
- Compliance with *Public Health Ordinance and Camp Sanitation Regulations*, and *Public Health Act*
- Permit to operate a private sewage disposal system under *Private Sewage Disposal Systems Regulations*
- Compliance with *Rubbish Disposal Regulations*
- Building permit and plumbing permit under *Yukon Building Standards Act* is required to construct site facilities
- Electrical permit under *Yukon Electrical Protection Act*
- Gas permit for propane facilities under *Yukon Boiler and Pressure Vessels Act*
- Compliance with *Yukon Fire Prevention Act*
- Permit required under *Yukon Act, Archaeological Sites Regulations* to investigate an archaeological site

Copies of major environmental licenses have been provided. The following table summarizes the current status of these licenses.

	Water License	EARP Screening	Surface Leases	Annual Fees
Faro	IN89-001 granted January 12, 1990	Decision Report - January, 1990	Mine Site Expires Dec 01, 2018 Fresh Water Resources Expires Apr 01, 2019 No. Fork Rose Creek Rock Dump Expires Jan 01, 2000 Faro Valley Rock Dump Renewal in progress	Approx $350 (approx $5,000 under YWAR)
	Amended Oct. 2, (#1) 1991	Decision Report/Memorandum of Understanding - September 1991		
	Amended Dec. 11 (#2) 1991			
	Amended (#3) but amendment not in effect yet (issued November 20, 1992)	Amendment #3 - Screening is still pending EARP screening and Decision Report		
	Expiry renewal January 30, 1997			
Vangorda	IN89-002 granted October 25, 1990	Decision Report - October 2, 1990	Mine site - Under application Haul Road - Under application Radio Transmitter - Under application	Approx $100 (approx $1,000 under YWAR)
	No amendments	Additional Security Agreement - Oct. 1990		
	Renewal Dec.31, 2003			
Sä Dena Hes	IN90-002 granted January 31, 1991	Decision Report December 31, 1990	Mine site - Under application	Approx $100 (approx $300 under YWAR)
	No amendments	Reclamation and additional Security Agreement - January, 1991		
	Renewal Sept 15, 2000			

3

Introduction

In the last three and one-half years British Columbia has seen the introduction of more legislation which has had a greater impact on the mining industry than any legislation in the last one hundred and thirty odd years when legislation to regulate the mining industry was first introduced in British Columbia.

The first of these Acts affecting the mining industry in British Columbia was the Mineral Tenure Act which came into effect in August of 1988.

The Mineral Tenure Act ("MTA") had the effect of combining the former Mineral Act and Mining (Placer) Act into one statute and as a result significant changes were introduced.

The second of these acts was the Mining Right-of-Way Act ("MRW") which was proclaimed effective on September 29, 1989. The Mining Right-of-Way the rights of a free miner to use public and private lands and created mechanisms whereby land could be acquired for mining purposes.

The third act proclaimed was the Mine Development Assessment Act ("MDAA") on August 30, 1991. The effect of this statute was to create one mechanism whereby mining projects could be dealt with from all aspects including environmental, land and development review processes.

The fourth act being The Mines Act ("MA") came into effect on July 15, 1990 and with it came the health safety and reclamation code for mines in British Columbia.

The changes in the MA were not as far reaching or broad as those occurring as a result of the introduction of the Mineral Tenure Act and the Mine Development Assessment Act.

Mines Act

The MA which was given royal assent on July 17, 1989 and made effective July 15, 1990 pursuant to B.C. Regulation 197/90 has had a significant impact not only on the mine development but also on mineral exploration. The act not only provides regulations for operating mines but also covers all activities which involve mechanical disturbance of the earth. It is difficult to conceive of anything other than perhaps geochemical sampling on a small scale, geophysical exploration or prospecting and sampling which would not be covered by the MA. The MA also established the health safety and reclamation code for the operation of mines in British Columbia.

This code deals with such things as industrial hygiene, emergency preparedness, buildings, machinery and equipment, electrical power systems, mine design and procedures, hoists and shafts, explosives, dams and waste emplacement, reclamation enclosure and exploration. Many people did not consider the MA applicable in carrying out exploration activities because at first glance most of the provisions appear related to the day-to-day operations of active mines.

The definition of "mine" however includes a place where mechanical disturbance of the ground or any excavation is made to explore for or produce coal, mineral bearing substances or placer minerals etc.

The provisions of this act govern all activities including exploratory drilling, excavation, processing, concentrating, waste disposal and site reclamation. Several definitions are important in appreciating the significance of the MA and these include "work place" which includes any place where work is carried out on or about a mine (and one must keep in mind that the definition of mine is not necessarily an active day-to-day operation).

The definition of "owner" includes a holder, proprietor, lessee or occupier of a mine or any part of it but does not include a person who receives a royalty or rent from a mine or is the owner of the surface rights of land under which a mine exists, provided the person does not also own the mineral rights underneath the land.

The Inspectors appointed under the MA have very broad powers and in fact have the powers under the Inquiry Act of British Columbia to take evidence under oath, gather evidence and generally investigate mining matters. These powers however extend only to accidents which cause serious personal injury, loss of life or property or environmental damage.

The Inspectors have the right to compel the attendance of witnesses, to require them to provide evidence under oath and to produce documentation in their possession.

There is no restriction on when an Inspector might carry out an inspection although on a health and safety inspection, the Inspector shall request the manager on the Inspector's arrival at the mine to appoint a representative of the owner to accompany him on the inspection. The Inspector can in fact shut a mine down under the Act. It is the obligation of the owner to provide the Inspector with every facility necessary for the purposes of an investigation or an inspection.

There is a right of appeal from the decision of the Inspector however that right of appeal is to the Chief Inspector and the right of appeal does not carry with it stay of the Inspector's order.

Before commencing any work on a mine, the owner, agent or manager must obtain a permit from the Chief Inspector and file a plan outlining the details of the proposed work. A permit is then issued. A security deposit may be required for reclamation purposes and there may be conditions attached to the approval for such work. The permit may be cancelled or the security deposit used to rectify any deficiencies in any conditions attached to the permit.

It is important to note that under Section 10(8) of the Act no work is to take place without a valid and subsisting permit.

The Lieutenant Governor-in-Council may establish by regulation a mine reclamation fund and such funds must be kept separate from the general mine funds.

Section 14 of the Act provides protection for "whistle blowers" and the Chief Inspector can order a fired employee to be reinstated and that the fired employee be reimbursed for his lost wages and costs.

The Inspector can (under Section 17 of the Act) do work around a closed or abandoned mine which is necessary to avoid danger to persons or property in the area or to abate pollution. If the work carried out is not paid for, a charge may be registered against the mineral title.

It is a requirement that every mine appoint a mine manager whose name should be supplied to the District Inspector and whose qualifications will be established by regulation or the code. Mine plans must be updated every three months and prepared in accordance with good engineering practice and must contain particulars established by the code all of which are required to be kept in the mine office.

The offence provision provides for a fine of not more than $100,000 or imprisonment for more than one year or both for offences under the Act.

Section 11 of the code under the MA deals with exploration. Exploration is defined as the search for coal, minerals, rock etc. by drilling, trenching, excavating, blasting, disturbance of the ground by mechanical means or prescribed geophysical equipment including underground work. "Prescribed geophysical equipment" means exposed electrodes used on IP surveys.

The application in writing for the work permit must be made at least 30 days before the intended commencement of the exploration work. This time frame in fact is being extended because the current practice in British Columbia, in areas where there may be aboriginal land claims, (and most of British Columbia may well be cover by such claims), is such that the native bands in the area are also given notice of the work application permits and their input is also being obtained by the Ministry of Mines before a work permit is issued.

There are also rules dealing with exploration for uranium and thorium (which is band in British Columbia), underground exploration, operation of aircraft, digging of pits and trenches and removal of excavated material.

Mining Right of Way Act

The second piece of legislation I wish to deal with is the Mining Right-of-Way Act. This act came into effect on September 29, 1989 and it provides the right to expropriate and use necessary rights-of-way on private land and on crown land for mining purposes. Power to take or use land under the Act is not exercisable unless the recorded holder (a claim holder or lease holder) first files with the Minister a plan showing the land proposed to be taken or used and obtains the written approval of the plan from the Minister.

There is also a provision regarding the use of access roads owned or built by other parties, for example, forest companies. In that instance the user of the road is responsible for reasonable payment for actual capital costs and maintenance costs.

Where the use of an access road is for mining purposes, that is to say where a lease has been issued or more than 10,000 tonnes of ore will be produced, the owner of the access road may require the miner to make a reasonable payment to reimbursement for a portion of the actual capital costs incurred in constructing the access road.

A miner may use an existing road whether on private land or crown land for exploration, development and operation of the mineral title or removal of minerals or production purposes including transportation of machinery, materials and supplies subject to the above. However, the miner must give written notice to the owner or operator of the road, pay compensation and abide by all lawful conditions respecting the use of the road.

<u>Mine Development Assessment Act</u>

The Mine Development Assessment Act ("MDA") came into effect on August 30, 1991. The effect of this legislation is to eliminate the old Mine Development Review Process ("MDRP"). The Mine Development Review Process was a non-legislative process developed over the years to deal with mining projects within a regulatory framework which had not been established by statute. The effect of the Mine Development Assessment Act was to create a formal procedure whereby all coal and mineral mine development within the provinces is subject to the procedure set down in the legislation. It is important to note that the MDA is intended to dovetail with the federal legislation being Bill C-13 (an act to establish a federal environment assessment process). It must be remembered that the federal legislation will be very far reaching and will allow the federal government or any of its departments to have the Canadian Environmental Assessment Agency (which is to be established by the new federal legislation) to establish review panels for the review of any project which might affect the environment. It must be remembered that environment includes any impact on health or socio-economic conditions. Federal legislation might also be used to deal with comprehensive native land claims where there has been acceptance by the Government of Canada to negotiate a land claim settlement.

If a project within British Columbia has already been through the Mine Development Review Process it would be exempt provided it is substantially complete before the Act came into force or that construction has substantially commenced or the review of the mine development was substantially complete before the Act came into effect.

If any proposed mining development has not yet been commenced it will be deemed to be a reviewable mine development under the new Act.

The Minister may also specify that a proposed mining development that was in the process of being reviewed when the Act comes into force qualifies as a reviewable mine development and that all submissions previously filed can be used as part of the application under the new Act. The Act defines a "reviewable mine development" as being a coal mine or mineral mine that is capable of producing 10,000 tonnes per year and is constructed after the date on which the Act came into force or any other mine that is designated a reviewable mine development by the Chief Inspector. It also includes all of the off-site facilities as may be designated by the Chief Inspector.

The major project review process guidelines remain in effect for projects such as ferro-alloys and primary steel industries, primary aluminum industries, primary smelting of non-ferrous industries and related industrial type products. The Mine Development Assessment Act is not intended to deal with such projects.

Projects which would not ordinarily be reviewable under this legislation include placer mines, sand, gravel and quarry operations.

An application for a mine development certificate is made to the Minister and the application shall contain information, analyses and an environmental protection plan approved by the Minister with the concurrence of the Minister of Environment.

The Minister may provide direction and advice on the information in the application or require the applicant to participate in public consultation or mediation with any person or groups as the Minister directs.

The Minister may with the concurrence of the Minister of Environment:

1. Refer the application to an assessment panel for an enquiry; or

2. Accept the application with terms or conditions; or

3. Return the application with the direction to make modifications an provide additional information; or

4. Reject the application.

The additional terms and conditions imposed under part 2 may include a condition that the applicant include one or more mine operation certificates.

If the matter is referred to an assessment panel the assessment panel may appoint or engage persons who have special technical knowledge to assist the panel. The terms of reference for the enquiry shall be public and they may include a requirement that the assessment panel consider whether or not any approvals etc. under the Waste Management Act, Water Act, Mines Act or Mining Right-of-Way Act should be given or issued under those acts. There are very broad powers given under Section 5 of the Act with respect to the conduct of the enquiry and most evidence is admissible whether or not it would be admissible in a court of law.

The assessment panel may dispose of an application by accepting the application subject to the terms and conditions that the Minister and the Minister of Environment specify, including the need to obtain one or more mine operation certificates or they may reject the application. If the assessment panel recommends that amendments to any mine operation certificate be made the Lieutenant Governor-in-Council may order such amendments. The effect of this is to overrule existing permits even if those permits were obtained in compliance with the applicable legislation. This determination is final and binding and is not subject to any further review.

If a mine operation certificate is required under the mine development certificate, an application is made to the Minister and the Minister who issues a mine operation certificate if the relevant terms and conditions of the mine development certificate have been complied with.

The costs of the assessment panel may be charged to the applicant.

Certificates may be suspended or cancelled where rights are not exercised under the certificate for a period of five years, monies are owing to the Crown, where there has been a failure to comply with conditions or orders or there has been a material misstatement or misrepresentation in the application.

Failure to comply with the provisions of the Act may result in an injunction application by the Minister for an order of the courts restraining a person from carrying on any activity if the Minister has made an order banning such activity.

It is also open to the panel to make any agreements consistent with the Act, with the Government of Canada and with other jurisdictions with respect to the conduct of assessments under the Act.

There are very broad reaching provisions with respect to the regulations to be made under the Act.

The concern with respect to the regulations is that these can be put in force by Order in Council as opposed to the legislature and obviously regulations can be done quickly with Cabinet approval, which should cause some trepidation for the operators of any proposed mining operation.

The Mineral Tenure Act

The Mineral Tenure Act came into effect on the 15th day of August 1988 in British Columbia and it is a combination of the former Mining (Place)r Act and Mineral Act.

Some of the major features of this legislation include the following:

(a) the addition of industrial minerals to the definition of minerals;

(b) partnership free miner certificates are available;

(c) access rights are spelled out in the Act;

(d) surface and subsurface disputes are to be resolved by the Mediation Arbitration Board;

(e) there is no limit to the number of two post claims or placer claims which may be staked.

There are now 20 days to record the claims after completion of staking and the anniversary date of new mineral titles is the completion date of the staking.

Approval under the MA is required when work is to be done which mechanically disturbs the land.

Placer claims were introduced and these are designed to replace old placer leases. Complaints are now allowed with respect to the staking and recording of assessment work on placer claims. It should also be noted that there is no grace period for filing assessment work after the anniversary date which formerly existed.

The MTA adopted the regulations contained under the old Mineral Act and the Placer (Mining) Act and essentially combined them.

There are several provisions of the MTA which have become or in my view will become quite significant in the coming years and these are as follows:

1. Section 16 dealing with the right of entry on private land and compensation therefor.

2. Section 10 dealing with disputes between free miners for example a placer miner and a lode miner.

3. Section 8 dealing with the suspension of a free miners certificates.

4. Section 35 dealing with disputes regarding the location of claims and assessment work carried out on them.

In accordance with Section 8 of the Act the Minister, where he is satisfied that a free miner has contravened the Act or Regulations can give the free miner 30 days notice of a hearing and after the hearing or the expiration of the 30 day period the Minister may suspend or revoke the free miners certificate and his right to apply for another.

Where the certificate has been revoked or suspended the Minister shall serve a notice of the suspension on the person affected and there is a right of appeal to a judge of the Supreme Court within 30 days after service of the notice.

Section 10 of the Act outlines the method whereby placer miners and lode miners can resolve disputes as to the ownership of the mineral (whether it is a placer mineral or mineral substance), or a dispute respecting the exercise of rights conferred under the Mineral Tenure Act or former acts. It must be remembered that a mineral claim and a placer claim or placer lease may exist independently on the same mineral lands.

In the event of a dispute the question is to be decided by the Chief Gold Commissioner on an application to him by a party to the dispute and the Chief Gold Commissioner has all the powers of a commissioner under the Inquiry Act, which allows him to call evidence, require parties to produce documents and require evidence to be given under oath.

If there is insufficient evidence to enable the Chief Gold Commissioner to determine whether a substance is a mineral, a placer mineral or a mineral substance, the Chief Gold Commissioner or the Judge as the case may be is obligated to decide the issue in favour of the person whose mineral title was first located.

A decision of the Chief Gold Commissioner can be appealed within 30 days to a Judge of the Supreme Court.

Section 16 of the Act deals with the right of entry on private land and compensation.

It is unlawful for a person to commence exploration development or production of minerals by mechanical means unless the recorded holder first gives notice to the owner of every surface area on which he intends to work or utilize the right of entry.

The free miner is liable to compensate the owner of the surface for any damage caused by his activities.

The Mediation and Arbitration Board which has been established under the Petroleum and Natural Gas Act has the authority on the application of a free miner, recorded holder, owner or other person having a material interest in the surface to settle matters of dispute between them arising from rights acquired under the Act regarding the right-of-way, use or occupation, security, rent and compensation and for the purposes of the Act provisions of the Petroleum and Natural Gas Act apply.

Before making the application however the Chief Gold Commissioner is to use his best endeavours to settle the issues between the parties. The procedure for a hearing before the Board is that notice is given and all interested parties must be served with the notice and usually the Board will actually attend the site to see for themselves what the situation is.

Witnesses can be called, evidence is given under oath and documents can be produced including expert evidence if that should be necessary.

The Board has the authority to award costs and the Board in reaching its decision must take into account which of the rights was applied for first and, except where an injustice would result, must give the holder of those rights due priority in its consideration of the dispute between the parties.

The next section of the Act which has been getting a great deal of attention lately and mainly because of the "Eskay Creek Virus" is Section 35.

This provision of the Act, which has had many precursers over the years including Section 50 of the former act, deals with disputes between free miners as to the ownership of claims. The disputes can arise in four ways:

1. The claim has been located or recorded contrary to the Act and regulations.

2. A person has made a false statement regarding assessment.

3. A claim has been acquired for purposes other than mining.

4. Production on a claim has exceeded the allowable limits.

It should be noted that a complaint under staking must be made within one year of the date of recording of the claim and those under assessment within one year of the date of the assessment statement having been filed with the Gold Commissioner.

The Ministry itself has authority to file complaints. In order to file a complaint one must set out particulars of the complaint and the interest the party has in making the complaint.

When the Chief Gold Commissioner receives a complaint he is to provide a copy of the complaint to the recorded holder of the claim and he may order an inspection to take place.

If an inspection takes place both parties can be present at the inspection, however the inspection is merely a fact gathering process.

When the report has been made available to both parties they have an opportunity to provide comments and provide additional evidence to the Chief Gold Commissioner and to comment on the other party's submissions with regard to that. There is no formal hearing process.

After the Chief Gold Commissioner has received the report and the submissions of all parties concerned, he makes his decision which is either to uphold the complaint or dismiss it. In certain circumstances the Chief Gold Commissioner may dismiss the complaint but may require the recorded holder of the claim to do remedial work such as abandoning and relocating the claim in a proper manner.

Any party can appeal within 30 days within the date of the Chief Gold Commissioner's order to a Judge of the Supreme Court. The hearing in Supreme Court is not a full hearing, but rather is a hearing on the material before the Chief Gold Commissioner. It is therefore imperative in such hearings, if there is a possibility of an appeal, that all relevant evidence be before the Chief Gold Commissioner prior to his making a decision otherwise the evidence may not be admissible in the court. There is no hearing of witnesses in the court, nor is additional evidence usually allowed to be introduced at the court hearing.

I am hopeful that the information provided will be of benefit to you and it should be noted that this in intended to be an overview of the current legislation in British Columbia and is not intended to deal with specific instances.

Stronsay

All data and drawings in this booklet were presented to KZ/S for review.

Given to KZ Samsung June 13/93

STRONSAY PROJECT

Report List

1. Development Plan, Executive Summary of Technical and Cost Aspects, Kilborn Inc., March 1992. 2 copies

2. Development Plan, Volume I, Technical Report, Kilborn Inc., May 1991. 1 copy

3. Development Plan, Volume II, Capital and Operating Costs, Kilborn Inc., May 1991. 2 copies
Vol 3 1 copy

4. Mineable Reserves, Mine Plan, Capital and Operating Costs, Canadian Mine Development, July 1991.

5. Mine Development Certificate and Reasons for Decision, Province of British Columbia, Ministry of Energy, Mines and Petroleum Resources, December 1992.

6. Waste Management Permit, PE-10940, Province of British Columbia, Ministry of Environment, December 1992.

7. Reclamation Permit, M-193, Province of British Columbia, Ministry of Energy, Mines and Petroleum Resources, December 1992.

8. Cirque Drill Hole Logs

a)	Volume I	90CS01 - 90CS45
b)	Volume II	90CU01 - 90CU60
c)	Volume III	90CU61 - 90CU120
d)	Volume IV	90CU121 - 90CU170

9. Geotechnical Drill Hole Logs, Curragh Inc., October 1, 1991

10. Diamond Drill Holes - Assays

11. A Report Summarizing 1989-1991, Advanced Exploration Program and Geological Reserves for North Cirque, Stronsay Corporation, Report # WH9102, Volume I, May 1991.

WH9102	Volume I, Appendix A	- Drawings included separately
WH9102	Volume II, Appendix A	- Drawings included separately
WH9102	Volume III, Appendix A	- Drawings included separately

2 copies - Summary and detail - all Permits

Report List Continued

11. (cont'd)
    - a) WH9102 Volume III, Appendix B, C, D.
    - b) WH9102 Addendum - Appendix E, F, G.
    - c) WH9102 Addendum - Appendix H

12. Bulk Sample Report

    a) Comparison of samples in raises, Jim Paxton, April 1991.
    b) File Note - Stronsay Bulk Sample, G. Clow, August 6, 1992.

13. Rock Mechanics Design and Testwork

    a) Geotechnical Input, Feasibility Report on Cirque Project, John D. Smith and Associates, January 22, 1991.
    b) Stronsay Project Pillar Design, John D. Smith and Associates, March 25, 1991.
    c) Geotechnical Evaluation of Drill Core and Underground Mapping Data, Stronsay Project, John D. Smith and Associates, July 29, 1991.

14. Backfill Design and Testwork

    a) Backfill Schedule and Design Summary.
    b) Backfill Testing and Analysis, Stronsay Project, John D. Smith and Associates, March 15, 1991.
    c) Backfill Cemented Strength and Percolation Rates - 15 micron split, Stronsay Project, John D. Smith and Associates, May 7, 1991.
    d) 14 day Backfill Cemented Strength and Percolation Rate - 15 micron split, John D. Smith and Associates, May 7, 1991.
    e) 28 day Backfill Cemented Strength - 15 micron split, John D. Smith and Associates, May 31, 1991.
    f) 90 day Backfill Cemented Strength Testing, John D. Smith and Associates, May 31, 1991.
    g) Cirque Tailings Management, Kilborn, December 10, 1990.
    h) Backfill Recoveries, John D. Smith and Associates, March 21, 1991.
    i) Backfill Design, Kilborn, April 2, 1991.

15. Grinding System for Cirque Project, A.R. MacPherson Consultants, January 1991.

16. Autogenous and Semi-Autogenous Grinding, Progress Report No. 1, Lakefield Research, March 18, 1991.

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Report List Continued

17. Laboratory Development Testwork, Progress Report No. 1, Lakefield Research, July 12, 1991.

18. Laboratory and Pilot Plant Development Testwork, Volume I, Lakefield Research, March 13, 1991; Volume II, April 1, 1991; Volume III, April 22, 1991.

19. Sag Mill Testing of Cirque Project Ore Types, A. R. MacPherson Consultants, January 30, 1991.

20. Column Flotation Pilot Plant Work, Minnovex Technologies Inc., Progress Report 2, March 25, 1991.

21. Cirque Ore Flotation, Progress Reports 1, 2, 3, March, April, May 1990, Lakefield Research.

22. Development Plan for Finbow Aerodrome, Accuratus Engineering Ltd., December 1991.

23. Stronsay Project, Preliminary Equipment List, Curragh Inc., April 1992.

24. Stronsay Corporation, Marine Transportation Segment, Lake Williston Barge, Design Report, Volumes I and II, Finlay Navigation Ltd. and Polar Design Associates, August 28, 1991.

25. Cirque Project, Environmental Impact Statement, January 1991, Addendum Report, June 1991, Baseline Environmental Monitoring Report, January 1993, Rescan Consultants Inc.

26. Construction Drawings and Documents for Stronsay Tailings Storage Facility, Stage 1, Golder Associates, September 31, 1992. (Drawings Separate)

27. Stronsay Project, Preliminary Engineering Assessments and Cost Estimates for Upgrading the Access Road from the Stronsay Mine to Chowika Barge Loading Site, Kilborn Pacific, June 1991. (Drawings Separate)

28. Stage 1 Level Geotechnical Report, Stronsay Property, Golder Associates, July 1991.

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Report List Continued

29. Geotechnical Design of Water Supply Dam and Plantsite, Golder Associates, April 1991 (Drawings Separate)

30. Reclamation Plan, Stronsay Project, Rescan Consultants, October 1991.

31. Stronsay Project, Reclamation Plan, Cost and Funding Detail, Curragh Inc., October 1992.

32. Preliminary Avalanche Hazard Assessment, Cirque Mine Property, Chris Stethem and Associates, March 30, 1991.

33. Stronsay Project, Capital and Operating Costs, Detail, Curragh Inc., July 18, 1991.

STRONSAY PROJECT

Drawing List

1. CRI - Geology X-Sections
   303+60N to 298+50N
   24 Drawings

2. CRI
   Longitudinal Sections
   1:500
   197+90E to 202+10E, 30 m intervals
   14 Drawings

3. CRI
   Geological Level Plans
   1:500
   4 Drawings

4. CRI
   Geological Map in Plane of
   Underground Workings
   3 Drawings

5. CRI
   Underground Mapping
   7 Drawings

6. CRI
   Geological Reserve Blocks
   24 Drawings

7. CMD Geological Reserve
   S. Part of N. Cirque
   296+40 to 298+20
   1:1000
   14 Drawings

8. CMD Reserve Blocks
   1:250
   24 Drawings

STRONSAY PROJECT

Drawing List Continued

9. CRI
   Mining X-Section on CRI Reserve Blocks
   298+50 to 303+60
   1:250
   24 Drawings

10. CRI
    Plan Projection of N. Cirque Hanging Wall
    1:1000
    4 Drawings

11. CRI
    X-Sections - Geological Outline and
    Histogram of % Recovery and RQD
    3 Drawings

12. CMD Mine Plan
    11 Drawings

13. CMD General Mining Methods
    2 Drawings

14. CMD
    PPD Schedule
    2 Drawings

15. R & M Engineering
    Conceptual Proposal for Squamish Terminal
    28 Drawings

16. Golder Associates
    Water Supply Dam Geotechnical
    3 Drawings

17. Golder Associates
    Plant Site Investigation
    2 Drawings

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Drawing List Continued

18. Golder Associates
    Tailings Storage Dam Geotechnical
    10 Drawings

19. Kilborn Pacific
    Access Road Upgrade Design
    10 Drawings

STRONSAY PROJECT DRAWING LIST

CURRAGH INC.
GEOLOGY X-SECTIONS
1:250
24 DRAWINGS

STRONSAY PROJECT
VERTICAL SECTION
DRILL HOLES WITH
LITHOLOGIES + Pb + Zn%

298 + 50 N (W)
DRAWING # AK-CQ-91-045
FILENAME GL29850W.DWG

298 + 50 N (E)
DRAWING # AK-CQ-91-047
FILENAME GL29850W.DWG

298 + 80 N (W)
DRAWING # AK-CQ-91-041
FILENAME GL29880W.DWG

298 + 80 N (E)
DRAWING # AK-CQ-91-043
FILENAME GL29880E.DWG

299 + 10 N (W)
DRAWING # AK-CQ-91-037
FILENAME GL29910W.DWG

299 + 10 N (E)
DRAWING # AK-XQ-91-039
FILENAME GL29910E.DWG

299 + 40 N (W)
DRAWING # AK-CQ-91-033
FILENAME GL29940W.DWG

299 + 40 N (E)
DRAWING # AK-CQ-91-035
FILENAME GL29940E.DWG

299 + 70 N (W)
DRAWING # AK-CQ-91-035
FILENAME GL29970W.DWG

299 + 70 N (E)
DRAWING # AK-CQ-91-031
FILENAME GL29970W.DWG

300 + 00 N (W)
DRAWING # AK-CQ-91-025
FILENAME GL30000W.DWG

300 + 00 N (E)
DRAWING # AK-CQ-91-027
FILENAME GL30000E.DWG

300 + 30 N
DRAWING # AK-CQ-91-023
FILENAME GL30030.DWG

300 + 60 N
DRAWING # AK-CQ-91-021
FILENAME GL30060.DWG

300 + 90 N
DRAWING # AK-CQ-91-019
FILENAME GL30090.DWG

301 + 20 N
DRAWING # AK-CQ-91-017
FILENAME GL30120.DWG

STRONSAY PROJECT DRAWING LIST

301 + 50 N
DRAWING # AK-CQ-91-015
FILENAME GL30150.DWG

301 + 80 N
DRAWING # AK-CQ-91-013
FILENAME GL30180.DWG

302 + 10 N
DRAWING # AK-CQ-91-011
FILENAME GL30210.DWG

302 + 40 N
DRAWING # AK-CQ-91-009
FILENAME GL30240.DWG

302 + 70 N
DRAWING # AK-CQ-91-007
FILENAME GL30270.DWG

303 + 00 N
DRAWING # AK-CQ-91-005
FILENAME GL30300.DWG

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STRONSAY PROJECT DRAWING LIST

CURRAGH INC.
LONGITUDINAL SECTIONS
1:500
197+ 90E to 202 + 10E, 30m INTERVALS
15 DRAWINGS

VERTICAL LONG SECTION
197 + 90 E
DRAWING # AK-CQ-91-0

VERTICAL LONG SECTION
198 + 20 E
DRAWING # AK-CQ-91-0

VERTICAL LONG SECTION
198 + 50 E
DRAWING # AK-CQ-91-0

VERTICAL LONG SECTION
198 + 80 E
DRAWING # AK-CQ-91-0

VERTICAL LONG SECTION
199 + 10 E
DRAWING # AK-CQ-91-0

VERTICAL LONG SECTION
199 + 40 E
DRAWING # AK- CQ-91-0

VERTICAL LONG SECTION
199 + 70 E
DRAWING # AK-CQ-91-0

VERTICAL LONG SECTION
200 + 00 E
DRAWING # AK-CQ-91-0

VERTICAL LONG SECTION
200 + 30 E
DRAWING # AK-CQ-91-0

VERTICAL LONG SECTION
200 + 60 E
DRAWING # AK-CQ-91-0

VERTICAL LONG SECTION
200 + 90 E
DRAWING # AK-CQ-91-0

VERTICAL LONG SECTION
201 + 20 E
DRAWING # AK-CQ-91-0

VERTICAL LONG SECTION
201 + 50 E
DRAWING # AK-CQ-91-0

VERTICAL LONG SECTION
201 + 80 E
DRAWING # AK-CQ-91-0

VERTICAL LONG SECTION
202 + 10 E
DRAWING # AK-CQ-91-0

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STRONSAY PROJECT DRAWING LIST

CURRAGH INC.
GEOLOGICAL LEVEL PLANS
1:500
4 DRAWINGS

STRONSAY PROJECT
PLAN GEOLOGY
1650m ELEVATION
DRAWING # AK-CQ-91-0

STRONSAY PROJECT
PLAN GEOLOGY
1600m ELEVATION
DRAWING # AK-CQ-91-0

STRONSAY PROJECT
PLAN GEOLOGY
1550m ELEVATION
DRAWING # AK-CQ-91-0

STRONSAY PROJECT
PLAN GEOLOGY
1500m ELEVATION
DRAWING # AK-CQ-91-0

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STRONSAY PROJECT DRAWING LIST

CURRAGH INC.
GEOLOGICAL MAP IN PLANE OF
UNDERGROUND WORKINGS
3 DRAWINGS

CIRQUE STRUCTURAL GEOLOGY
1 OF 3 SHEET

CIRQUE STRUCTURAL GEOLOGY
2 OF 3 SHEET

CIRQUE STRUCTURAL GEOLOGY
3 OF 3 SHEET

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STRONSAY PROJECT DRAWING LIST

CURRAGH INC.
PLAN PROJECTION OF N. CIRQUE
HANGING WALL
1:1000
4 DRAWINGS

PLAN MAP NORTH CIRQUE
HANGING WALL ORE
% RECOVERY, % RQD BREAKAGE
(2 METRE THICKNESS)
DRAWING # AK-CQ-91-132
FILENAME HWOREGT.DWG

PLAN MAP
NORTH CIRQUE
HANGING WALL
Pb + Zn GRADES (%)
TOP 2 METRES OF DEPOSIT
DRAWING # AK-CQ-91-136

PLAN MAP
NORTH CIRQUE
HANGING WALL
STRUCTUAL CONTOURS
10 METRE CONTOUR INTERVAL
DRAWING # AK-CQ-91-137

PLAN MAP
NORTH CIRQUE
HANGING WALL SHALE
% RECOVERY, % RQD BREAKAGE
2 METRE THICKNESS
DRAWING # AK-CQ-91-131
FILENAME HWWASTGT.DWG

STRONSAY PROJECT DRAWING LIST

CRI
UNDERGROUND MAPPING
7 DRAWINGS

NORTH CIRQUE
GEOLOGY
UNDERGROUND WORKINGS
NORTHEAST WALL
DRAWING # AK-CQ-91-124
FILENAME: CQWALL1.DWG

NORTH CIRQUE
GEOLOGY
UNDERGROUND WORKINGS
NORTHEAST WALL
DRAWING # AK-CQ-91-125
FILENAME: CQWALL2.DWG

NORTH CIRQUE
GEOLOGY
UNDERGROUND WORKINGS
NORTHEAST WALL
DRAWING # AK-CQ-91-126

NORTH CIRQUE
GEOLOGY
UNDERGROUND WORKINGS
NORTHEAST WALL
DRAWING# AK-CQ-91-127
FILENAME: CQWALL4.DWG

NORTH CIRQUE
GEOLOGY
UNDERGROUND WORKINGS
SOUTHEAST WALL
DRAWING # AK-CQ-91-128

NORTH CIRQUE
GEOLOGY
UNDERGROUND WORKINGS
DECLINE
DRAWING # AK-CQ-91-129

NORTH CIRQUE
GEOLOGY OF THE RAISES
DRAWING # AK-CQ-91-130
FILENAME: CQRAISE.DWG

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STRONSAY PROJECT DRAWING LIST

CURRAGH INC.
X-SECTION - GEOLOGICAL OUTLINE AND HISTOGRAM OF % RECOVERY AND RQD
3 DRAWINGS

VERTICAL SECTION
303 + 00 N
HISTOGRAMS SHOWING
RECOVERY % & RQD%
DRAWING AK-CQ-91-133

VERTICAL SECTION
303 + 30 N
HISTOGRAMS SHOWING
RECOVERY % & RQD%
DRAWING AK-CQ-91-134

VERTICAL SECTION
303 + 60 N
HISTOGRAMS SHOWING
RECOVERY % & RQD%
DRAWING # AK-CQ-91-135

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STRONSAY PROJECT DRAWING LIST

CURRAGH INC.
GEOLOGICAL RESERVE BLOCKS
1:250
24 DRAWINGS

STRONSAY PROJECT
VERTICAL SECTION
ORE RESERVE
POLYGONS

298 + 50 N (W)
DRAWING # AK-CQ-91-046
FILENAME: RS29850W.DWG

298 + 50 N (E)
DRAWING # AK-CQ-91-048
FILENAME: RS29850E.DWG

298 + 80 N (W)
DRAWING # AK-CQ=91-042
FILENAME: RS29880W.DWG

298 + 80 N (E)
DRAWING # AK-CQ-91-044
FILENAME: RS29880E.DWG

299 + 10 N (W)
DRAWING # AK-CQ-91-038
FILENAME: RS29910W.DWG

299 + 10 N (E)
DRAWING # AK-CQ-91-040
FILENAME: 29910E.DWG

299 + 40 N (W)
DRAWING # AK-CQ-91-034
FILENAME: RS29940W.DWG

299 + 40 N (E)
DRAWING # AK-CQ-91-036
FILENAME: RS29940E.DWG

299 + 70 N (W)
DRAWING # AK-CQ-91-030
FILENAME: RS29970W.DWG

299 + 70 N (E)
DRAWING # AK-CQ-91-032
FILENAME: RS29970E.DWG

300 + 00 N (W)
DRAWING # AK-CQ-91-028
FILENAME: RS30000W.DWG

300 + 00 N (E)
DRAWING # AK-CQ-91-028
FILENAME: RS30000E.DWG

300 + 30 N
DRAWING # AK-CQ-91-024
FILENAME: RS30030.DWG

300 + 60 N
DRAWING # AK-CQ-91-022
FILE RES30060.DWG

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STRONSAY PROJECT DRAWING LIST

300 + 90 N
DRAWING # AK-CQ-91-020
FILENAME: RES30120.DWG

301 + 20 N
DRAWING # AK-CQ-91-018
FILENAME: RES30120.DWG

301 + 50 N
DRAWING # AK-CQ-91-016
FILENAME: RES30150.DWG

301 + 80 N
DRAWING # AK-CQ-91-014
FILENAME: RES30180.DWG

302 + 10 N
DRAWING # AK-CQ-91-012
FILENAME: RES30210.DWG

302 + 40 N
DRAWING # AK-CQ-91-010
FILENAME: RES30240.DWG

302 + 70 N
DRAWING # AK-CQ-91-008
FILENAME: RES30270.DWG

303 + 00 N
DRAWING # AK-CQ-91-006
FILENAME: RES30300.DWG

303 + 30 N
DRAWING # AK-CQ-91-004
FILENAME: RES30330.DWG

303 + 60 N
DRAWING # AK-CQ-91-002
FILENAME: RES30360.DWG

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STRONSAY PROJECT DRAWING LIST

CMD
RESERVE BLOCKS
1:250
24 DRAWINGS

STRONSAY PROJECT
VERTICAL SECTION
DRILL HOLES WITH
LITHOLOGIES & PB + ZN%

298 + 50 N (W)
DRAWING # AK-CQ-91-045
FILENAME GL29850W.DWG

298 + 50 N (E)
DRAWING # AK-CQ-91-047
FILENAME GL29850E.DWG

298 + 80 N (W)
DRAWING # AK-CQ-91-041
FILENAME GL29880W.DWG

298 + 80 N (E)
DRAWING # AK-CQ-91-043
FILENAME GL29880E.DWG

299 + 10 N (W)
DRAWING # AK-CQ-91-037
FILENAME GL29910W.DWG

299 + 10 N (E)
DRAWING # AK-CQ-91-039
FILENAME GL29910E.DWG

299 + 40 N (W)
DRAWING # AK-CQ-91-033
FILENAME GL29940W.DWG

299 + 40 N (E)
DRAWING # AK-CQ-91-035
FILENAME GL29940E.DWG

299 + 70 N (W)
DRAWING # AK-CQ-91-029
FILENAME GL29970W.DWG

299 + 70 N (E)
DRAWING # AK-CQ-91-031
FILENAME GL29970E.DWG

300 + 00 N (W)
DRAWING # AK-CQ-91-025
FILENAME 30000W.DWG

300 + 00 N (E)
DRAWING # AK-CQ-91-027
FILENAME 30000E.DWG

300 + 30 N
DRAWING # AK-CQ-91-023
FILENAME 30030.DWG

300 + 60 N
DRAWING # AK-CQ-91-021
FILENAME GL30060.DWG

300 + 90 N
DRAWING # AK-CQ-91-019
FILENAME GL30090.DWG

301 + 20 N
DRAWING # AK-CQ-91-017
FILENAME GL30120.DWG

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STRONSAY PROJECT DRAWING LIST

301 + 50 N
DRAWING # AK-CQ-91-015
FILENAME GL30150.DWG

301 + 80 N
DRAWING # AK-CQ-91-013
FILENAME GL30180.DWG

302 + 10 N
DRAWING # AK-CQ-91-011
FILENAME GL30210.DWG

302 + 40 N
DRAWING # AK-CQ-91-009
FILENAME GL30240.DWG

302 + 70 N
DRAWING # AK-CQ-91-007
FILENAME GL30270.DWG

303 + 00 N
DRAWING # AK-CQ-91-005
FILENAME GL30300.DWG

303 + 30 N
DRAWING # AK-CQ-91-003
FILENAME GL30330.DWG

303 + 60 N
DRAWING # AK-CQ-91-001
FILENAME GL30360.DWG

STRONSAY PROJECT DRAWING LIST

CMD
GEOLOGICAL RESERVE
S. PART OF N. CIRQUE
296 + 40 to 298 + 20
1:1000
15 DRAWINGS

MINING BLOCK @ 8% Pb +Zn
SECTION 296 + 40 N
DRAWING # 90-07-027

MINING BLOCK @ 8% Pb +Zn
SECTION 296 + 70 N
DRAWING # 90-07-026

MINING BLOCK @ 8% Pb +Zn
SECTION 297 + 00 N
DRAWING # 90-07-025

MINING BLOCK @ 8% Pb +Zn
SECTION 297 + 30 N
DRAWING # 90-07-024

MINING BLOCK @ 8% Pb +Zn
SECTION 297 + 60 N
DRAWING # 90-07-023

MINING BLOCK @ 8% Pb +Zn
SECTION 297 + 90 N
DRAWING # 90-07-022

MINING BLOCK @ 8% Pb +Zn
SECTION 298 + 20 N
DRAWING # 90-07-021

MINING BLOCK @ 8% Pb +Zn
SECTION 298 + 40 N
DRAWING # 90-07-020

MINING BLOCKS @ VARIOUS CUTOFFS
SECTION 296 + 40 N

MINING BLOCKS @ VARIOUS CUTOFFS
SECTION 296 + 70 N

MINING BLOCKS @ VARIOUS CUTOFFS
SECTION 297 + 00 N

MINING BLOCKS @ VARIOUS CUTOFFS
SECTION 297 + 30 N

MINING BLOCKS @ VARIOUS CUTOFFS
SECTION 297 + 60 N

MINING BLOCKS @ VARIOUS CUTOFFS
SECTION 297 + 90 N

MINING BLOCKS @ VARIOUS CUTOFFS
SECTION 298 + 20 N

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STRONSAY PROJECT DRAWING LIST

CURRAGH INC./CMD
MINING X-SECTION ON CRI RESERVE BLOCKS
1:250
24 DRAWINGS

STRONSAY PROJECT
VERTICAL SECTION
CMD MINING BLOCK

298 + 80 N (W)
DRAWING # AK-CQ-91-042
CMD DWG REF. # 90-7-197
FILENAME RS29880W.DWG

298 + 80 N (E)
DRAWING # AK-CQ-91-044
CMD DWG REF. # 907-196
FILENAME RS29880E.DWG

298 + 50 N (W)
DRAWING # AK-CQ-91-046
CMD DWG REF. # 90-7-199
FILENAME RS29850W.DWG

298 + 50 N (E)
DRAWING # AK-CQ-91-048
CMD DWG REF. # 90-7-198
FILENAME RS29850E.DWG

299 + 10 N (W)
DRAWING # AK-CQ-91-038
CMD DWG REF. # 90-7-195
FILENAME RS29910W.DWG

299 + 10 N (E)
DRAWING # AK-CQ-91-040
CMD DWG REF. # 90-7-194
FILENAME RS29910E.DWG

299 + 40 N (E)
DRAWING # AK-CQ-91-036
CMD DWG REF. # 90-70-192
FILENAME RS29940E.DWG

299 + 40 N (W)
DRAWING # AK-CQ-91-034
CMD DWG REF. # 90-7-193
FILENAME RS29940W.DWG

299 + 70 N (W)
DRAWING # AK-CQ-91-030
CMD DWG REF. # 90-7-191
FILENAME RS29970W.DWG

299 + 70 N (E)
DRAWING # AK-CQ-91-032
CMD DWG REF. # 90-7-190
FILENAME RS29970E.DWG

300 + 00 N (W)
DRAWING # AK-CQ-91-026
CMD DWG REF. # 90-7-189
FILENAME RS30000W.DWG

300 + 00 N (E)
DRAWING # AK-CQ-91-028
CMD DWG REF. # 90-7-188
FILENAME RS30000E.DWG

300 + 30 N
DRAWING # AK-CQ-91-024
CMD DWG REF. # 90-7-187
FILENAME RES30030.DWG

300 + 60 N
DRAWING # AK-CQ-91-022
CMD DWG. REF. # 90-7-186
FILENAME RES30060.DWG

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STRONSAY PROJECT DRAWING LIST

300 + 90 N
DRAWING # AK-CQ-91-020
CMD DWG. REF. # 90-7-185
FILENAME RES30090.DWG

301 + 20 N
DRAWING # AK-CQ-91-018
CMD DWG. REF.# 90-7-184
FILENAME RES30130.DWG

301 + 50 N
DRAWING # AK-CQ-91-016
CMD DWG. REF.# 90-7-183
FILENAME RES30150.DWG

301 + 80 N
DRAWING # AK-CQ-91-014
CMD DWG. REF.# 90-7-182
FILENAME RES30180.DWG

302 + 10 N
DRAWING # AK-CQ-91-012
CMD DWG REF. # 90-7-181
FILENAME RES30210.DWG

302 + 40 N
DRAWING # AK-CQ-91-010
CMD DWG REF. # 90-7-180
FILENAME RES30240.DWG

302 + 70 N
DRAWING # AK-CQ-91-008
CMD DWG REF. # 90-7-179
FILENAME RES30270.DWG

303 + 00 N
DRAWING # AK-CQ-91-006
CMD DWG REF. # 90-7-178
FILENAME RES3O300.DWG

303 + 30 N
DRAWING # AK-CQ-91-004
CMD DWG REF. # 90-7-177
FILENAME RES30330.DWG

303 + 60 N
DRAWING # AK-CQ-91-002
CMD DWG REF. # 90-7-176
FILENAME RES30360.DWG

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STRONSAY PROJECT DRAWING LIST

CMD
GENERAL MINING METHODS
2 DRAWINGS

CANADIAN MINE DEVELOPMENT
TYPICAL PANEL AND FILL MINING METHOD
DRAWING # 90-7-175

CANADIAN MINE DEVELOPMENT
TYPICAL LONGHOLE MINING
DRAWING # 90-7-170

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STRONSAY PROJECT DRAWING LIST

CMD
MINE PLAN
11 DRAWINGS

NORTH ACCESS RAMP
1:500
DRAWING # 90-7-160

NORTH ACCESS RAMP BELOW 1542 EL.
1:500
DRAWING # 90-7-161

NORTH ACCESS RAMP 1543-1586 EL.
1:500
DRAWING # 90-7-162

NORTH ACCESS RAMP 1586-1605 EL.
1:500
DRAWING # 90-7-163

NORTH ACCESS RAMP 1605-1617 EL
1:500
DRAWING # 90-7-164

NORTH ZONE ACCESS RAMP
LONGITUDINAL SECTION
DRAWING # 90-7-165

NORTH ACCESS RAMP
LONGITUDINAL SECTION
1:500
DRAWING # 90-7-166

GENERAL ARRANGEMENT
COLOURED COMPOSITE
1:1000
DRAWING # 90-7-167

PREPRODUCTION COMPOSITE PLAN
1:500
DRAWING # 90-7-169

PREPODUCTION COMPOSITE PLAN
1:500
DRAWING # 90-7-169

PREPRODUCTION COMPOSITE PLAN
1:500
DRAWING # 90-7-169

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STRONSAY PROJECT DRAWING LIST

CMD
PPD SCHEDULE AND MISCELLANEOUS
2 DRAWINGS

PREPRODUCTION WORK,
MANPOWER & EQUIPMENT SCHEDULE
DRAWING # 90-7-168
REV. 2

CRUSHER STATION G/A
DRAWING # 90-7-172

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STRONSAY PROJECT DRAWING LIST

R & M ENGINEERING
CONCEPTUAL PROPOSAL FOR SQUAMISH TERMINAL
30 DRAWINGS

CONCEPUAL PROPOSAL FOR
MINERAL CONCENTRATES
TERMINAL
SQUAMISH
1 OF 28 SHEETS
PROJECT # 911336

SITE PLAN
2 OF 28 SHEETS

SITE PLAN
2A OF 28 SHEETS

SUITE IMPROVEMENT SECTIONS
3 0F 28 SHEETS

SITE PLAN & ELEVATION
LEAD/ZINC TERMINAL
SQUAMISH INDUSTRIES SITE
10A OF 29 SHEETS

SITE IMPROVEMENT SECTIONS
4 0F 28 SHEETS

SITE IMPOVEMENT SECTION
5 OF 28 SHEETS

SITE IMPROVEMENT SECTIONS
6 0F 28 SHEETS

SITE IMPROVEMENT & INLET
STRUCTURE
7 OF 28 SHEETS

EQUIPMENT WASHDOWN AREA
8 OF 28 SHEETS

CREW CHANGE FACILITY
9 0F 28 SHEETS

SITE PLAN & ELEVATION
10 OF 28 SHEETS

MARINE FACILITY GENERAL PLAN
AND SECTIONS
11 OF 28

MARINE FACILITY
SERVICE PIER
SECTIONS & DETAILS
12 OF 28 SHEETS

SHIPLOADER ELEVATION
13 OF 28 SHEETS

SHIPLOADER GENERAL
ARRANGEMENT
ELEVATION
14 OF 28 SHEETS

SHIPLOADER GENERAL
ARRANGEMENT REAR
ELEVATION
15 OF 28 SHEETS

ORE STORAGE BUILDING
ELECTRICAL PLANS
ELEVATION
16 OF 28 SHEETS

ORE STORAGE BUILDING
ELECTRICAL SCHMATICS
DETAILS
17 OF 28 SHEETS

ORE STORAGE BUILDING
AIR VENTILATION &
FILTRATION PLANT &
ELEVATION
18 OF 28 SHEETS

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STRONSAY PROJECT DRAWING LIST

AIR VENTILATION &
FILTRATION SECTIONS
& DETAILS
19 OF 28 SHEETS

ORE STORAGE BUILDING
AIR VENTILATION &
FILTRATION SPECIFCATIONS
& DETAILS
20 OF 28 SHEETS

MECHANICAL SYSTEM
SUPPORT PLATFORM
21 OF 28 SHEETS

CONVEYOR SYSTEM
22 OF 28 SHEETS

CONVEYOR # 1 (48" BELT)
MECHANICAL GENERAL
ARRANGEMENT
23 OF 28 SHEETS

CONVEYOR #1
STRUCTURAL
GENERAL ARRANGEMENT
24 OF 28 SHEETS

STRUCTURAL CONVEYOR
#2 ARRANGEMENT OF
TAIL END
25 OF 28 SHEETS

CONVEYOR #2 STRUCTURE
GENERAL ARRANGEMENT
26 OF 28 SHEETS

CONVEYOR #1 STRUCTURE
ARRANGEMENT OF
TRANSFER PLATFORM
27 OF 28 SHEETS

STRONSAY PROJECT DRAWING LIST

GOLDER ASSOCIATTES
TAILINGS STORAGE DAM GEOTECHNICAL
7 DRAWINGS

TAILINGS STORAGE FACILITY
GEOLOGICAL INTERPRETATION
CROSS SECTION
PROJECT # 892-1159
DRAWING # 1001
REV. 2

TAILINGS STORAGE FACILITY
GEOLOGICAL INTERPRETATION
CROSS SECTIONS
PROJECT # 892-1159
DRAWING # 1002
REV. 2

TAILINGS DAM AND ROCK QUARRY
DESIGN
PLAN AND SECTIONS
PROJECT 892-1159
DRAWING # 1003
REV. 2

TAILINGS DAM DESIGN
CROSS SECTIONS
PROJECT # 892-1159
DRAWING # 1004
REV. 2

TAILINGS DAM DESIGN DETAILS
PROJECT # 892-1159
DRAWING # 1005
REV. 2

TAILINGS DAM DESIGN
STAGED CONSTRUCTION AND SPILLWAY
DETAILS
1 OF 2 SHEETS
PROJECT # 892-1159
DRAWING # 1006
REV. 2

TAILINGS DAM DESIGN
STAGED CONSTRUCTION AND
SPILLWAY DETAILS
2 0F 2 SHEET
PROJECT # 892-1159
DRAWING # 1007

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STRONSAY PROJECT DRAWING LIST

GOLDER ASSOCIATES
WATER SUPPLY DAM
3 DRAWINGS

WATER SUPPLY DAM
SITE INVESTIGATION
PLAN VIEW
PROJEC # 912-1021
DRAWING # 3001
REV. 0

WATER SUPPLY DAM
GEOLOGICAL INTERPRETATION
CROSS SECTIONS
PROJECT # 912-1021
DRAWING # 3002
REV. 0

WATER SUPPLY DAM
CROSS SECTION & DETAILS
PROJECT # 912-1021
DRAWING # 3003
REV. 0

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STRONSAY PROJECT DRAWING LIST

GOLDER ASSOCIATES
PLANT SITE INVESTIGATION
2 DRAWINGS

CIRQUE PROJECT
PLANTSITE LOCATION
SITE INVESTIGATION
PLAN VIEW
PROJECT # 912-1021
DRAWING # 2001
REV. 0

CIRQUE PROJECT
PLANTSITE LOCATION
GEOLOGICAL INTERPRETATION
CROSS SECTIONS
PROJECT # 912-1021
DRAWING # 2002
REV. 0

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STRONSAY PROJECT DRAWING LIST

KILBORN PACIFIC
ACCESS ROAD UPGRADE DESIGN
10 DRAWINGS

ACCESS ROAD
REGIONAL PLAN
DEVELOPMET PHASE - COLOURED
DRAWING # 100-30-FO1
REV.A

ACCESS ROAD
REGIONAL PLAN
DRAWING # 100-30-FO1
REV.B

GENERAL AREA PLAN
DRAWING # 100-30-FO2

OVERALL PLAN
DRAWING # 100-30-001

ACCESS ROAD
PAUL RIVER VALLEY
DRAWOMG # 100-30-002

ACCESS ROAD
CHOWIKA
DRAWING # 100-30-003

ACCESS ROAD
CHOWIKA
DRAWING # 100-30-004

ACCESS ROAD
CHOWIKA
DRAWING # 100-30-005

ROAD DETAILS
DRAWING # 100-30-006

BRIDGE DETAILS
DRAWING # 100-30-007

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STRONSAY PROJECT DRAWING LIST

KILBORN
PLANT FLOWSHEETS AND GA'S
11 DRAWINGS

3920 T/D LEAD/ZINC CONCENTRATOR
GENERAL
COARSE ORE HANDLING & GRINDING
PROCESS FLOWSHEET
PROJECT # 3547
DIVISION # 20
DRAWING # 100-10-FA1
REV. B

3920 T/D LEAD/INC CONCENTRATOR
GENERAL
BULK FLOTATION & LEAD FLOTATION
PROCESS FLOWSHEET
PROJECT # 3547
DIVISION # 20
DRAWING # 100-10-FA2
REV.B

3920 F/D LEAD/ZINC CONCENTRATOR
GENERAL
ZINC FLOTATION
PROCESS FLOWSHEET
PROJECT # 3547
DIVISION # 100-10-FA3
REV.B

3920 T/D LEAD/ZINC CONCENTRATOR
GENERAL
LEAD DEWATERING
PROCESS FLOWSHEET
PROJECT # 3547
DIVISION # 20
DRAWING # 100-10-FA4

3920 T/D LEAD/ZINC CONCENTRATOR
GENERAL
ZINC DEWATERING PROCESS FLOWSHEET
PROJECT # 3547
DIVISION # 20
DRAWING # 100-10-FA5
REV. B

3920 T/D LEAD/ZINC CONCENTRATOR
GENERAL
BACKFILL PLANT
PROCESS FLOWSHEET
PROJECT # 3547
DIVISION # 20
DRAWING # 100-10-FA6
REV. B

3920 T/D LEAD/ZINC CONCENTRATOR
GENERAL
REAGENTS - SHEET 1
PROCESS FLOWSHEET
PROJECT # 3547
DIVISION # 20
DRAWING # 100-10-FA7
REV. B

3920 T/D LEAD/ZINC CONCENTRATOR
GENERAL
REAGENT AND UTILITIES - SHEET 2
PROCESS FLOWSHEET
PROJECT # 3547
DIVISION # 20
DRAWING # 100-10-FA8
REV. B

STRONSAY PROJECT DRAWING LIST

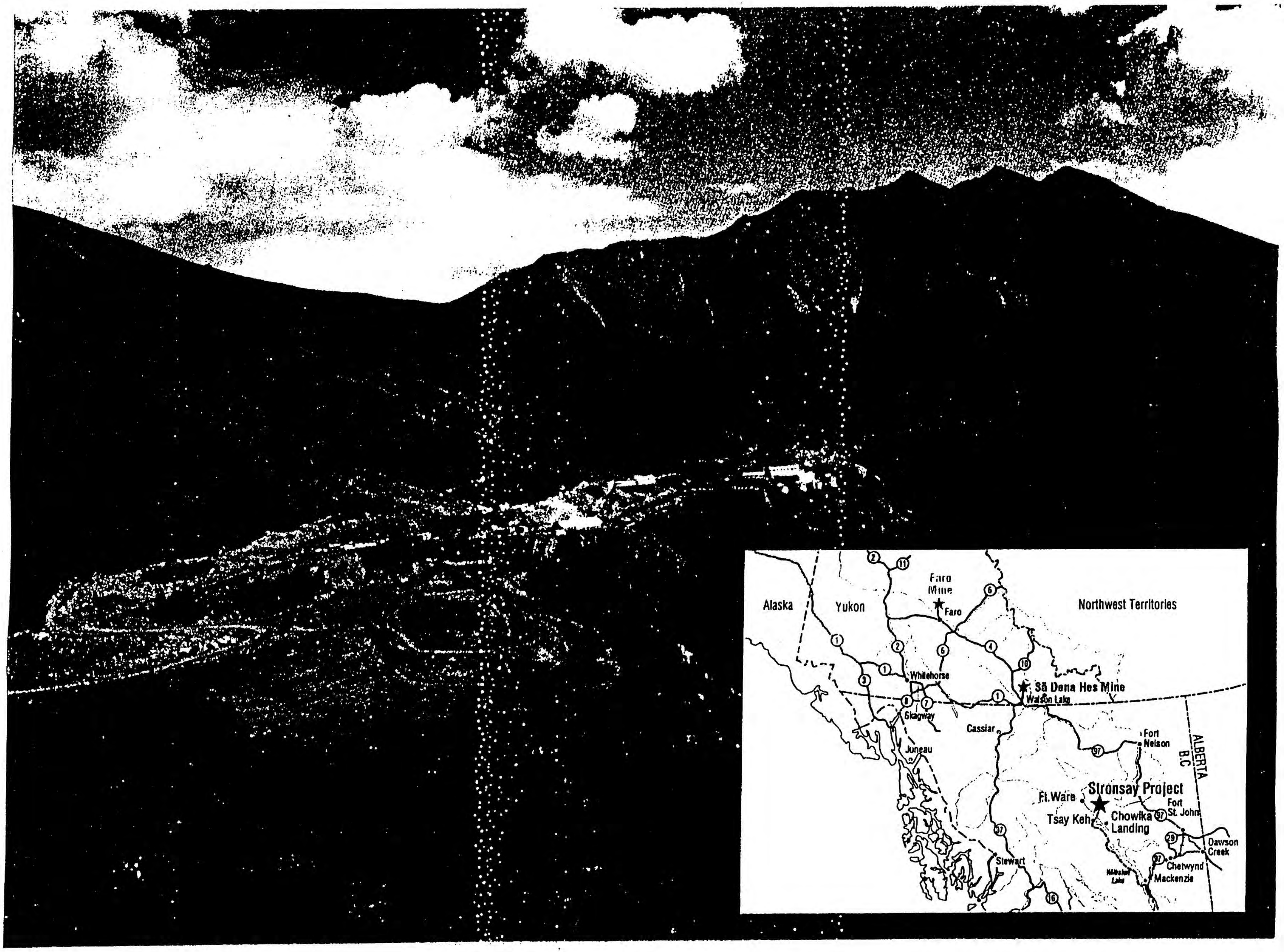
3920 T/D LEAD/ZINC CONCENTRATOR
GENERAL
FRESH WATER SUPPLY, STORAGE,
 TREATMENT, AND DISTRIBUTION
PROCESS FLOWSHEET
PROJECT # 3547
DIVISION # 20
DRAWING NUMBER 100-10-FA9
REV. B

3920 T/D LEAD/ZINC CONCENTRATOR
GENERAL
SITE WATER BALANCE
PROCESS FLOWSHEET
PROJECT # 3547
DIVISION # 20
DRAWING NUMBER 100-10-FA10
REV. B

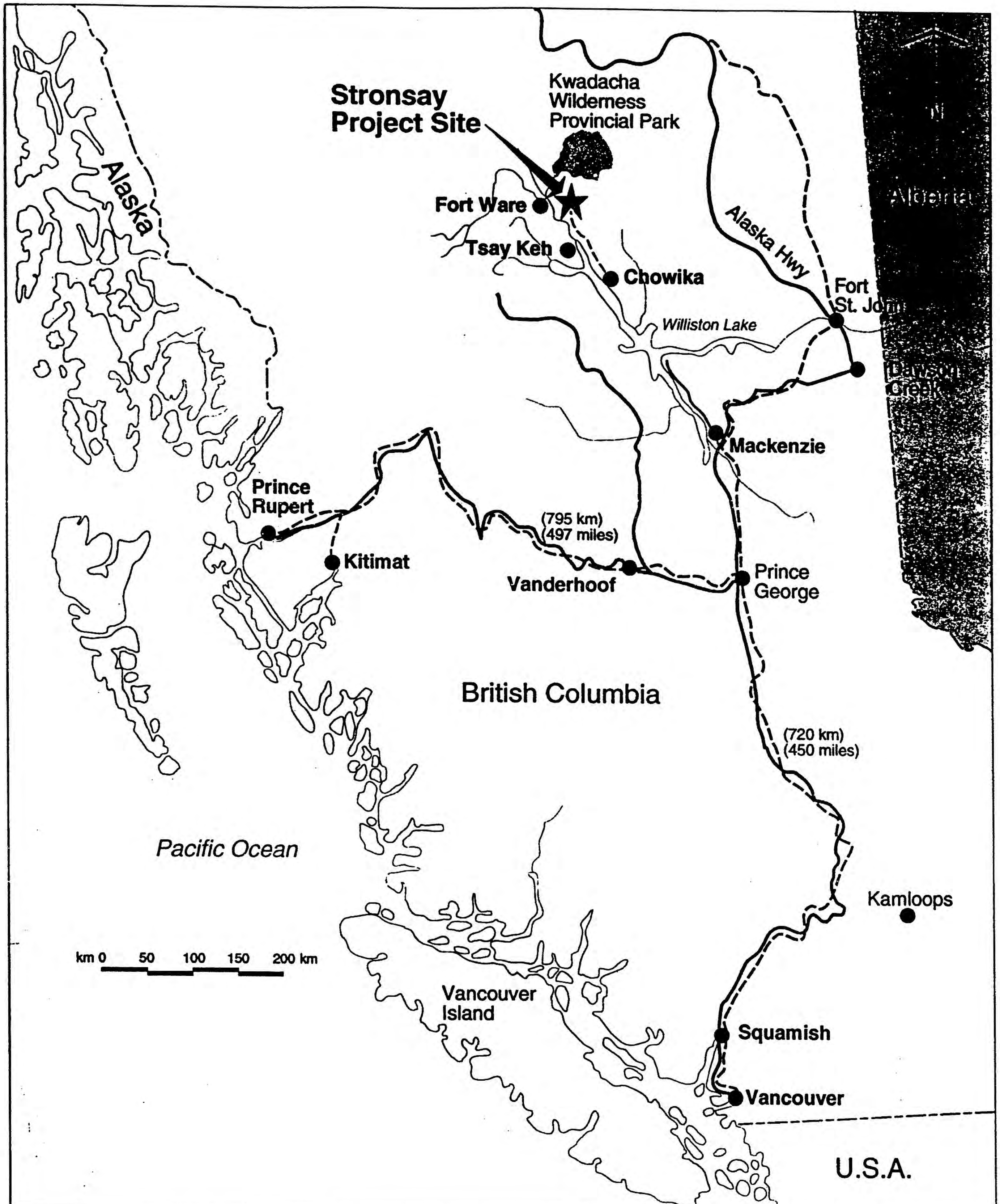
3920 TPD LEAD/ZIND CONCENTRATOR
CIRQUE DEPOSIT
SITE DEVELOPMENT
COMBINED SERVICE PLAN
PROJECT # 3547
DIVISION # 19
DRAWING # 100-30-FO5
REV. B

Curragh Inc.

Stronsay Project Briefing Paper



Stronsay Project

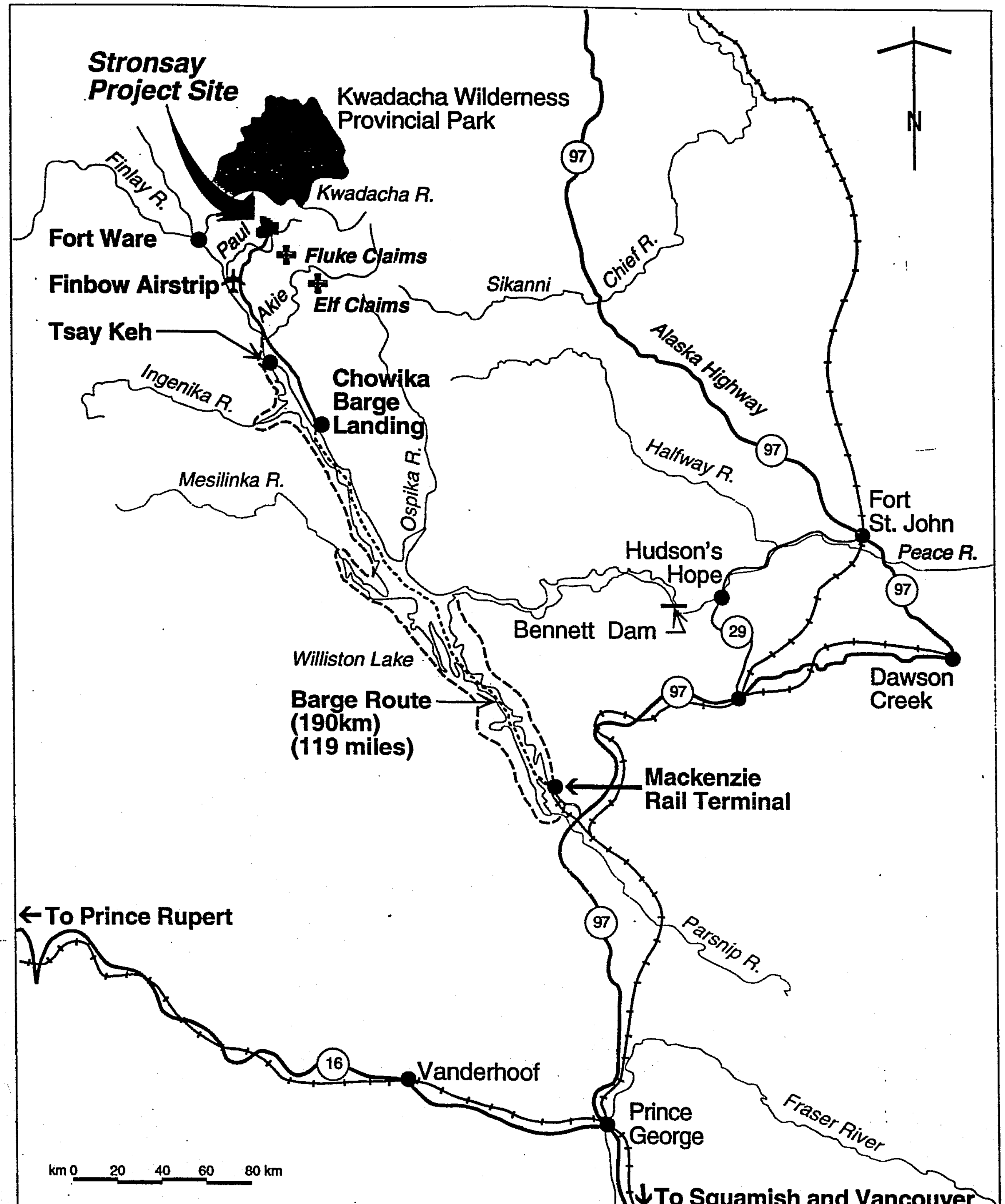


Stronsay Project

Project Location Map

STRONSAY PROJECT
SUMMARY FACTS SHEET

Project	Underground lead and zinc sulphide mine including surface facilities to produce 250,000 tonnes of saleable lead and zinc sulphide concentrates per year.
Project Location	280 air kilometres north of town of MacKenzie, British Columbia, Canada.
Ownership	Stronsay Corporation owned by Curragh Inc.
Manager	Curragh Inc.
Diamond Drilling To Date	
- 1978 - 1982	48,000 m in 113 holes from surface.
- 1989 - 1991	13,000 m in 215 holes from surface and underground 30m sections.
Underground Development	
- Adit	700 m at -18%.
- Drifts	300 m in North Drift; 300 m in South Drift.
- Raises (2)	100 m.
Expenditures to Date ($1992)	$Cdn 60 million.
Reserves	Two deposits, North Cirque and South Cirque. Geological reserves: 50 million tonnes of mineralization, 2% Pb, 8% Zn plus silver and cadmium. Initial mineable reserves (at 6% combined cutoff): 24,678,000 tonnes, 2.3% Pb, 8.5% Zn, 50.8 g/t Ag. High grade ore mineable in early years: 11,807,000 tonnes, 2.6% Pb, 9.0% Zn, 55.2 g/t Ag.
Production Rate	1,431,000 tpa; 3920 tpd.
Mine Life	35+ years reasonable, 17+ years from present mineable reserves.
Production Start-up	18 months from decision to proceed.



Stronsay Project

Site Location Map

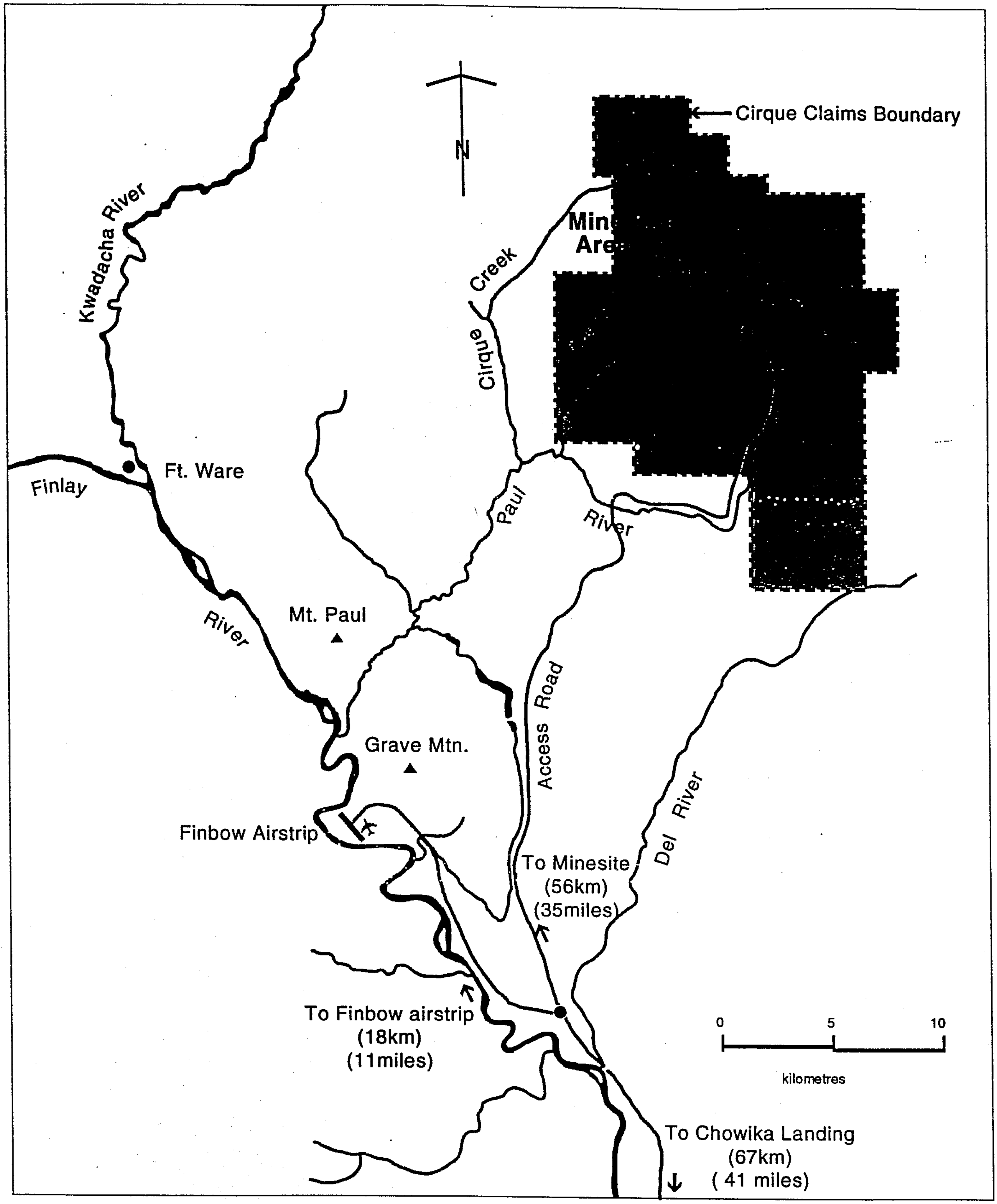
STRONSAY PROJECT
SUMMARY FACTS SHEET

GEOLOGY

North Cirque	Continuous wedge shaped, bedded barite sulphide lens approximately concordant with Devonian siliceous black shale host sediments; 1000 m n-s, 400 m e-w, 2-70 m thick; dips 30° to 55° southwest; contains all presently known mineable reserves.
South Cirque	500 m south of North Cirque, extent has not been fully defined. Not included in mineable reserves.
Mineralization	Sphalerite, galena, pyrite and barite; banded, varying from thin laminations to broad layering.
Ore Types (3)	Based on % barite: Massive sulphide (barite <20%); sulphide with barite (20%<barite<60%); barite with sulphide (barite >60%).
Specific Gravity	Based on pulps with allowance for porosity: Massive sulphide 4.41 Sulphide with barite 4.35 Barite with sulphide 4.20

MINING

Rate	3920 tonnes per day, 3 x 8hr shifts, 7 days per week; expansion can be readily accomplished.
Methods	Longitudinal longhole stoping (12m stopes and pillars); panel and fill stoping (7m sequential panels).
Backfill	Cemented tailings with crushed rock fill of primary stopes to allow for pillar recovery; crushed rock content 15%; cement ratio 1:23 (4.25%).



Stronsay Project
Site Map

STRONSAY PROJECT
SUMMARY FACTS SHEET

MINING (Cont'd)

Recovery	By area: Longhole Stopes 98% to 100% Longhole Pillars 90% Panel and Fill 98%
Dilution	0.5m hangingwall cuts in stopes; 1.0 m hanging wall cuts in pillars; footwall dilution calculated for each stope; 0.3 m fill in pillar mining; 7.5% by volume mined for fill in panel stoping. All dilution at zero grade.
Dewatering	30 l/sec (400 Igpm).
Ventilation	142 m3/sec (300,000 cfm).
Equipment	3 development jumbos; 2 longhole jumbos; 2 rockbolt jumbos; 4 x 8 cuyd scooptrams; 4 x 26 tonne haulage trucks; misc service equipment.

PROCESSING

Design Rate	3920 tonnes per day, at 2.3% Pb, 8.5% Zn; 2 x 12 hr shifts, 7 days per week; 90% availability.
Crushing	Underground 1220 x 1067 jaw crusher to - 200 mm; conveyed to 3500 tonne coarse ore bin located on surface; conveyed to 200 tonne surge bin.
Grinding	6400 x 2340 fully autogenous grinding mill, with 3460 x 5490 pebble mill for secondary; 85% minus 200 mesh; design primary Bond Work Index 7.5; autogenous mill powered to allow for media if necessary.

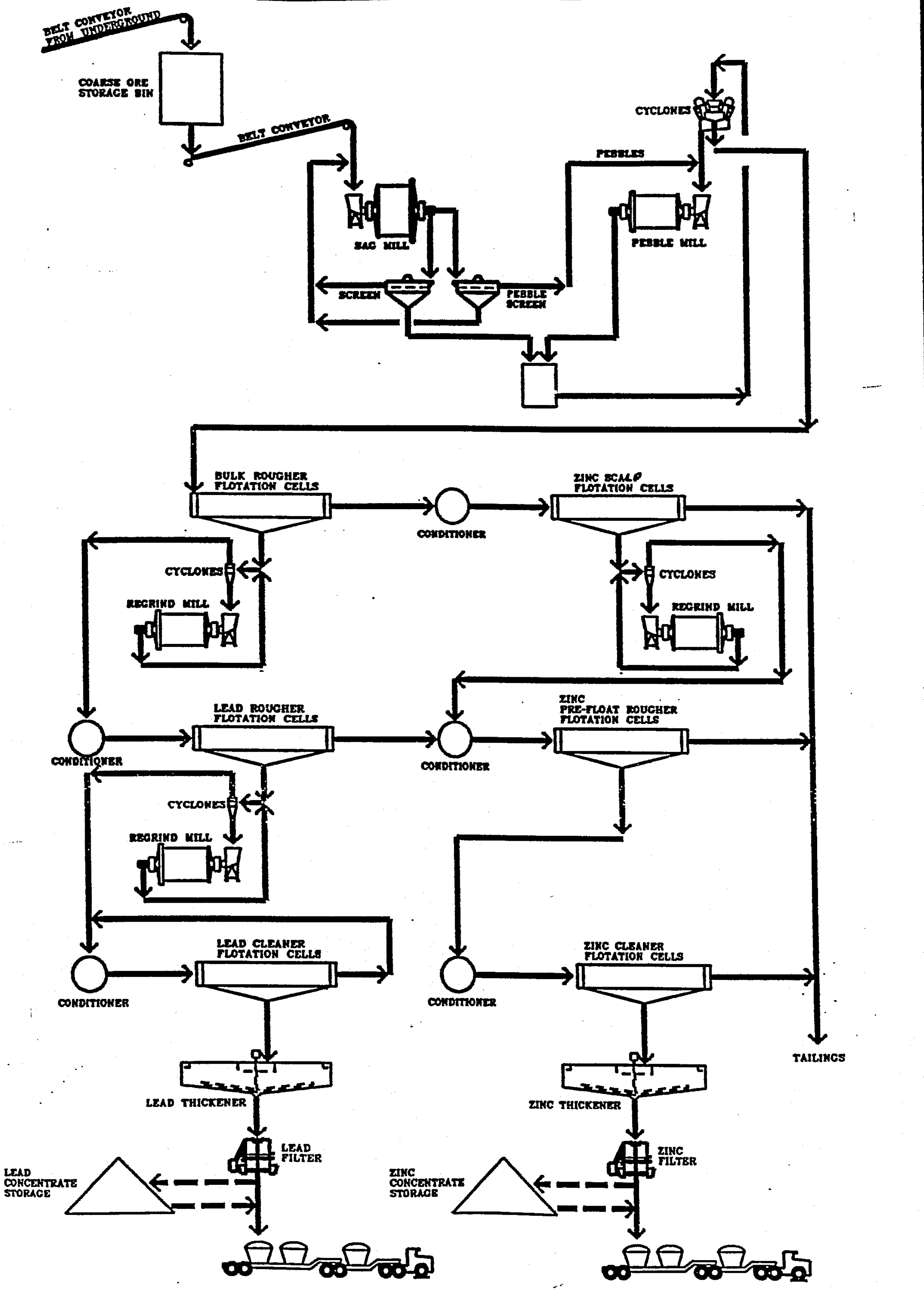
PROCESSING (Cont'd)

Flotation	Semi bulk flotation to remove carbonaceous material, followed by regrind; lead rougher conc floated followed by regrind; four cleaning stages; zinc rougher conc floated from semi bulk tails, followed by regrind; four cleaning stages.
Metallurgy	Lead Conc 71.0% Pb; 75.0% recovery. Zinc Conc 56.6% Zn; 88.0% recovery. Silver 291 g/t Ag in lead conc; 14% recovery.
Reagents	Soda ash, lime, copper sulphate, Iso Butyl Xanthate/thiourea/M2030/CA830, MIBC/DF1012, SD200/sodium cyanide, flocculant.
Dewatering	High capacity thickener (Pb 7000mm, Zn 12000mm); Pressure filters designed for 7% lead conc moisture, 8% zinc conc moisture.
Instrumentation	Full coverage with on-stream analyzer; distributed control system.
Cyanide Destruction	Inco SO_2 process.
Loadout	Concentrates conveyed to separate 100 tonne loadout bins or to 1000 tonne lead conc and 4000 tonne zinc conc storage sheds; reclaimed by front end loader.
Tailings	Portion not used for backfill (<12 microns) stored in manmade impoundment in Cache Creek Valley; all tailings deposited sub-aqueously.

TRANSPORTATION

Method/Route	Custom-designed truck (Faro design) over road system from site to Chowika (East of Williston Lake); barge from Chowika to Mackenzie; rail system from Mackenzie to port; then by ocean vessels to Asian and European markets.

STRONSAY PROJECT



STRONSAY PROJECT
SUMMARY FACTS SHEET

CONCENTRATES ANALYSES (% unless otherwise indicated)

Element		Lead Concentrate	Zinc Concentrate
Lead	Pb	71.0	1.24 - 2.84
Zinc	Zn	3.48 - 5.10	57.0
Copper	Cu	0.007 - 0.016	0.065 - 0.079
Iron	Fe	2.20 - 4.55	5.87 - 6.00
Nickel	Ni	<0.002	<0.002 - 0.002
Bismuth	Bi	<0.002	<0.002
Cadmium	Cd	0.025 - 0.047	0.33 - 0.34
Cobalt	Co	<0.002	<0.002
Chromium	Cr	<0.002	<0.002
Arsenic	As	<0.001	<0.001
Antimony	Sb	0.008 - 0.011	<0.002
Tin	Sn	<0.001	<0.001
Gallium	Ga	<0.0001 - 0.0001	0.0003
Germanium	Ge	<0.0010	0.0043 - 0.0107
Indium	In	<0.002	<0.002
Manganese	Mn	0.003	0.024 - 0.035
Mercury	Hg	0.0010 - 0.0013	0.013 - 0.0058
Molybdenum	Mo	<0.002	<0.002
Thallium	Tl	0.004 - 0.005	0.007 - 0.010
Thorium	Th	<0.001	<0.001
Selenium	Se	<0.0003 - 0.0004	<0.0003
Tellurium	Te	<0.0003	0.0006 - 0.0014
Uranium	U	<0.001	<0.001
Gold	Au g/t	0.05 - 0.13	<0.02 - 0.03
Silver	Ag g/t	171 - 263	157 - 166
Titanium	$Ti0_2$	<0.10	<0.10
Silicon	$Si0_2$	0.20 - 0.49	0.64 - 0.94
Aluminum	Al_20	<0.10	<0.10 - 0.13
Calcium	Ca0	0.03 - 0.07	0.08 - 0.10
Magnesium	Mg0	<0.01	<0.01
Sodium	Na_20	<0.002 - 0.002	0.003 - 0.005
Potassium	K_20	<0.002 - 0.013	0.009 - 0.012
Fluorine	F	<0.01 - 0.04	0.02 - 0.03
Chlorine	Cl	0.0038 - 0.0081	0.013 - 0.014
Sulphur	S	16.9 - 18.9	34.2 - 35.6
Phosphorus	P	0.0015 - 0.0064	0.0029 - 0.0048
Carbon	C	0.20 - 1.29	0.28 - 0.44
Insoluble		0.13 - 0.70	2.36 - 2.47
L.O.I. (1000○C)		8.64 - 10.3	18.4 - 18.5
Recovery		75	88

STRONSAY PROJECT
SUMMARY FACTS SHEET

TRANSPORTATION (Cont'd)

Containers	Products containerized at site in 2.8 m high x 3.2 m diameter sealed steel containers and transported all the way to an environmentally safe port (Skagway design); no transloading of bulk concentrates along the transportation route.
- Required number	750 (±).
- Tare weight	1,600 kg.
- Payload	22.5 tonnes.
- Moving	Containers to be moved\transferred\dumped by 30,000 kg capacity forklifts.
Haulage Trucks	
- Unit	104,000 kg GVW, 67.5 tonne payload B-train\tandem\tridem configuration; 23 m in length.
- Fleet	5 units.
- Round trip from Site to Chowika	6 hours.
- Frequency	10 - 12 truck trips per day.
Road from Site to Chowika	123 km gravel allweather.
- Current status	Existing road is basic access only; classed as forestry access road; requires upgrade.
- Ownership	Province of British Columbia.
- Uses	Multi-use; common.
Chowika Landing	
- Required Staging Area	3 hectares.
- Required Facilities	Loading ramp to road to staging area; 2 @ 30,000 kg capacity forklifts; storage/maintenance building; 450,000 l fuel oil storage facility.

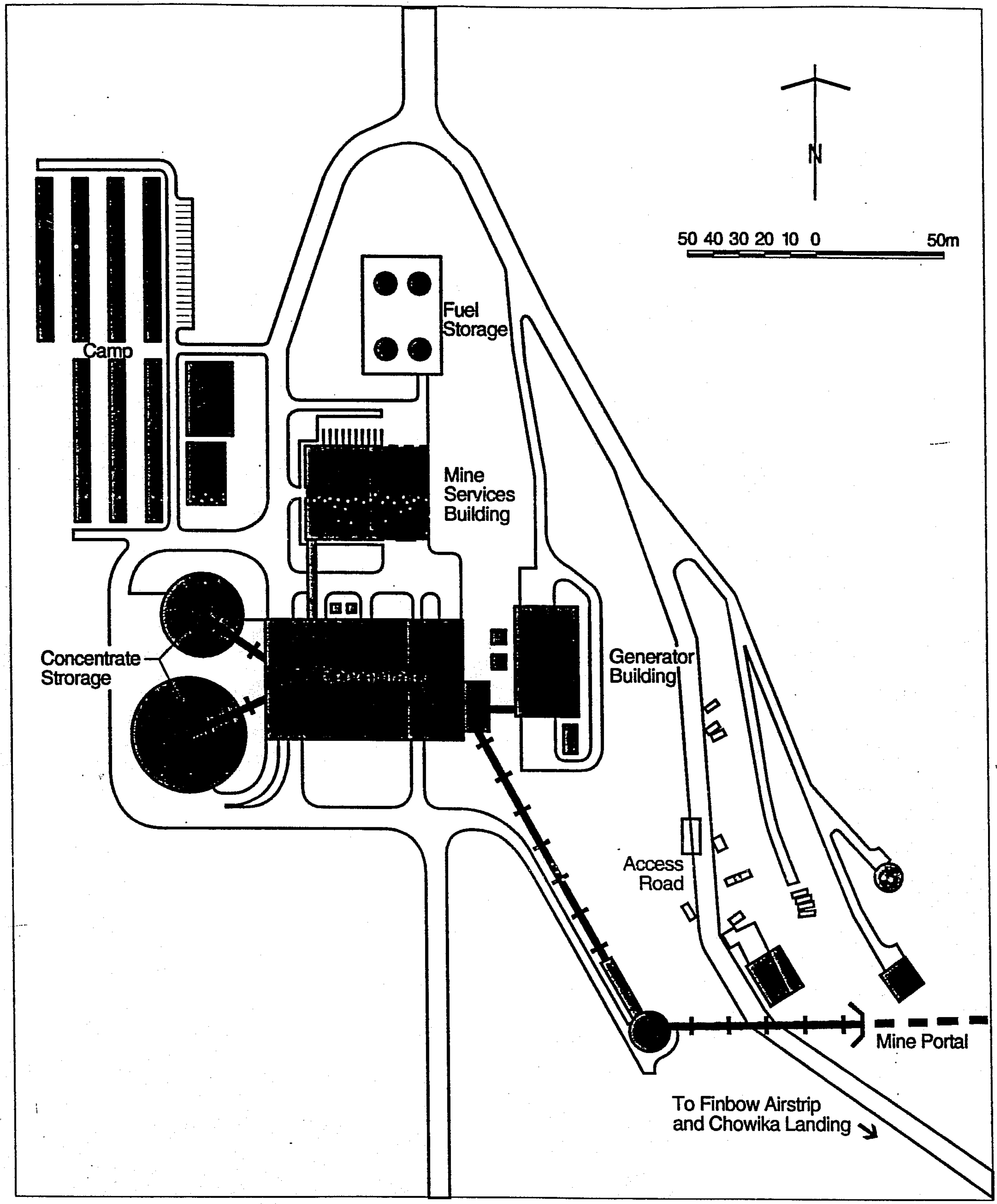
STRONSAY PROJECT
SUMMARY FACTS SHEET

TRANSPORTATION (Cont'd)

Williston Lake from Chowika to Mackenzie	190 km.
- Open Water	June to November.
- Ice	December to May.
- Ice Thickness	Less than 1 m.
- Depths	Greater than 6 m.
Barge	5,600 DWT (tonnes) self-propelled, 9000 Bhp, ice breaking, steel hulled, compartment barge; 29 m x 119 m.
- Design	By Polar Design Associates Ltd.; Reviewed by consultant, Arno Keinonen; Ice Class 1A.
- Draft	3.4 m.
- Fleet	1 unit.
- Speed	
- Summer	10 knots.
- Winter	7 knots.
- Round Trip from Chowika to Mackenzie	36 hours.
- Service Frequency	2 - 3 trips per week.
- Ownership	Private operator or Province of British Columbia.
- Operator	Private operator or Stronsay Corporation
- Uses	Multi use, common.
- Usage Fees	On a per tonne basis; 100% (less operating and maintenance costs) credited to amortization costs.
- Operating and Maintenance Costs	By operator.
Mackenzie	
- Required Staging Area	3 hectares.
- Required Facilities	Loading ramp and road to staging area; rail siding for 12 rail cars at staging area; 2 @ 30,000 kg capacity forklifts; storage\maintenance, office building; 450,000 l fuel oil storage facility.

TRANSPORTATION (Cont'd)

B.C. Rail from Mackenzie to Squamish	720 km.
- Forehaul Traffic Volume	270,000 tonnes per year; 750 tonnes per day; 9 - 10 rail cars per day; all containerized; may be expanded.
- Backhaul Traffic Volume	48,000 tonnes per year; 140 tonnes per day all containerized; may be expanded.
- Fuel Oil	16,000 tonnes per year.
- Supplies	10,000 tonnes per year.
- Cement	22,000 tonnes per year.
- Required Rolling Stock	Approximately 70 standard flatdeck rail cars, decking removed; 119,000 kg GVW and max 24,000 kg tare.
- Round trip from Mackenzie to Squamish	7 days, daily service
Port at Squamish (proposed)	
- Location	B.C. Rail site B, Squamish.
- Required Area	3 hectares including the laydown storage area for general freight.
- Required Facilities	450 m dock with 13 m minimum water depth for 40,000 DWT class ships; 60,000 tonne capacity storage building; 2,000 tonne per hour single point ship loader; 2,000 tonne per hour conveying system; rail siding for 12 cars; silos for cement, soda ash and lime on rail line,; all environmentally safe; 2 @ 30 000 kg capacity forklifts; 3 @ Cat 980 FEL.
- Ownership	B.C. Rail.
- Operator	Stronsay Corporation.
- Usage and Fees	Multi use, multi product, fees on a per tonne basis; 100% (less operating and maintenance costs) credited to amortization costs (above).
- Operating and Maintenance Costs	By operator.



Stronsay Project
Plant Site

STRONSAY PROJECT
SUMMARY FACTS SHEET

SERVICES

Site	Completely self contained operation with all facilities on the site.
Power Requirements	Peak demand 11 Mw; average load 9.2 Mw; annual consumption 73 Gwh.
Power Supply	5 units at 2.7 Mw each; installed capacity 13.6 Mw with allowance in the plant for a sixth engine; full heat recovery; distribution at 4160v primary.
Hydro	Study underway in conjunction with Native bands and Department of Public Works to review potential sites at capacities up to 20 Mw on Finlay River; most promising site is Deserters' Canyon near Tsay Keh.
Site Facilities	Concentrator building containing process facilities and power plant; adjacent building with shop warehouse dry, offices, lab etc.; 300 bed camp; kitchen; and recreational facilities.
Building Services	Heat and power from power plant; water supply from dam in Cache Creek Valley; sewage treated in treatment plant; full fire protection system.
Air Access	Finbow Airstrip 1400 m; requires upgrade of navigation aids.

- To Prince George	- 440 km.
- To Ft. St. John	- 310 km.
- Road from Finbow to mine	- 74 km.

STRONSAY PROJECT
SUMMARY FACTS SHEET

ENVIRONMENT, TAILINGS, WATER

Mine permitting in the Province of British Columbia follows the Mine Development Review Process (MDRP).

Under auspices of the Mine Development Steering Committee (MDSC). Umbrella group composed of all provincial departments which may have some jurisdiction over a project. MDSC chaired by the Ministry of Energy, Mines, and Petroleum Resources.

The federal government has input through the Department of Fisheries and Oceans and the Inland Waters Directorate of Environment Canada to the MDRP or the Environmental Assessment and Review Process (EARP).

The MDRP follows three stages:

Stage 1	- Submission of Environmental Impact Statement
Stage 2	- At the discretion of the MDSC, more detailed follow-up work to Stage 1
Stage 3	- Issuance of Mine Development Certificate and various operating permits.

The Stronsay Project has been reviewed three times with the MDSC and numerous times with other (including federal) agencies. Local Native Bands (Fort Ware and Tsay Keh) have reviewed the Stronsay Project and support the Environmental Impact Statement.

The following submissions have been made to the MDSC:

Prospectus	July 1989
Status Report	November 1989
Status Report	May 1990
Status Report	September 1990
Environmental Impact Statement	February 1991 - Stage 1
Addendum to EIS	June 1991
Ongoing Correspondence	To Present

Approvals and Permits received:

Mine Development Certificate - overall authority to develop the mine.
Effluent Discharge Permit - discharge from tailings pond.
Reclamation Permit - specifies closure, reclamation, monitoring requirements.
A Refuse Permit, an Air Emissions Permit, and a Water License (to use water) have been applied for and are expected to be issued shortly. Other minor permits have been applied for and will be issued as required.

STRONSAY PROJECT
SUMMARY FACTS SHEET

ENVIRONMENT, TAILINGS, WATER (Cont'd)

Two main environmental concerns in the project design:
The ore and waste rock are potentially acid generating; and,
the metals content of the effluent from the tailings pond.

54% of tailings and 100% of post-development waste rock will be stored underground. All remaining tailings will be deposited underwater from the very beginning of the mine operations in a specially-built tailings impoundment. The tailings will be covered by water. 92% of water going into the tailings impoundment will be recycled back to the mill for reuse during operations.

Waters will be released from the tailings pond using flow paced discharge which will allow present applicable CCRM criteria for drinking water and aquatic life to be met at 6.7 km in Cache Creek, upstream of the Paul River.

Fresh water source will be a reservoir in the Cache Creek catchment area, near the plant site. Total usage will average 62 m^3/hr (230 Igpm).

The mine concentrates will be transported in sealed containers all the way to an environmentally safe port (Skagway design).

Stronsay will monitor the Project site after closing and will ensure necessary adjustments, if any, are made for long term closure. Stronsay will establish from the beginning of mine operations a trusteed fund, growing up to $5.16 million ($Cdn 1992) at time of closure, paid in annual instalments over the life of the mine as security for long term closure and monitoring needs, if any.

NATIVES

Property lies in Kaska Dena (Fort Ware Band) traditional lands.Transportation route is through Carrier-Sekani (Tsay Keh Band) traditional lands.

Project representatives have had numerous meetings with both local Native groups including public meetings at the communities of Fort Ware and Tsay Keh.

Local Natives have reviewed and support the Environmental Impact Statement.

Agreements have been signed with both groups covering business, employment, and training opportunities.

Working with Natives on the feasibility of local hydro development in the area to meet needs of the Project and local communities; eventual native ownership.

STRONSAY PROJECT
SUMMARY FACTS SHEET

CONSTRUCTION

Schedule Construction period eighteen months.

Capital Cost ($Cdn '000 1991):

Site:	Mine Development	15,934
	Site Costs	8,483
	Concentrator	52,272
	Administration & Services	16,064
	Indirects	31,879
	Contingency	4,850
	TOTAL	129,482

Site Leased Equipment (included in operating costs)	25,228

Infrastructure

Direct Government	
Chowika Road Upgrade	10,000
Finbow Airstrip Upgrade	2,000
	12,000
Private or Government	
Barge	10,000
BC Rail (Provincial Crown Corporation)	
Port	15,000
Total	37,000

Site Manpower

	(Peak Month)
Mine	80
Surface	251
Total	331
Total Man-years (Site)	344

STRONSAY PROJECT
SUMMARY FACTS SHEET

OPERATIONS

Average Operating Cost ($Cdn 1991)

	Average Over 17 Years			Average Over 5 Years
	Operating Costs ($/tonne milled	($/tonne dry concentrate)	($/lb. Pay Metal	($/lb Pay Metal
Site Costs:				
Mine	16.88	107.36	0.0952	0.0893
Mill	14.50	92.19	0.0818	0.0743
Concen. Handling	11.67	74.22	0.0658	0.0637
Freight Handling	0.71	4.51	0.0040	0.0034
Stronsay G & A	3.89	24.79	0.0220	0.0202
Off-Site G & A	0.77	4.84	0.0043	0.0040
TOTAL (C$)	48.43	307.92	0.2731	0.2548
Total (US$ @ 0.83 US$/C$ Exchange	40.19	255.57	0.2267	0.2115

STRONSAY PROJECT
SUMMARY FACTS SHEET

OPERATIONS (Cont'd)

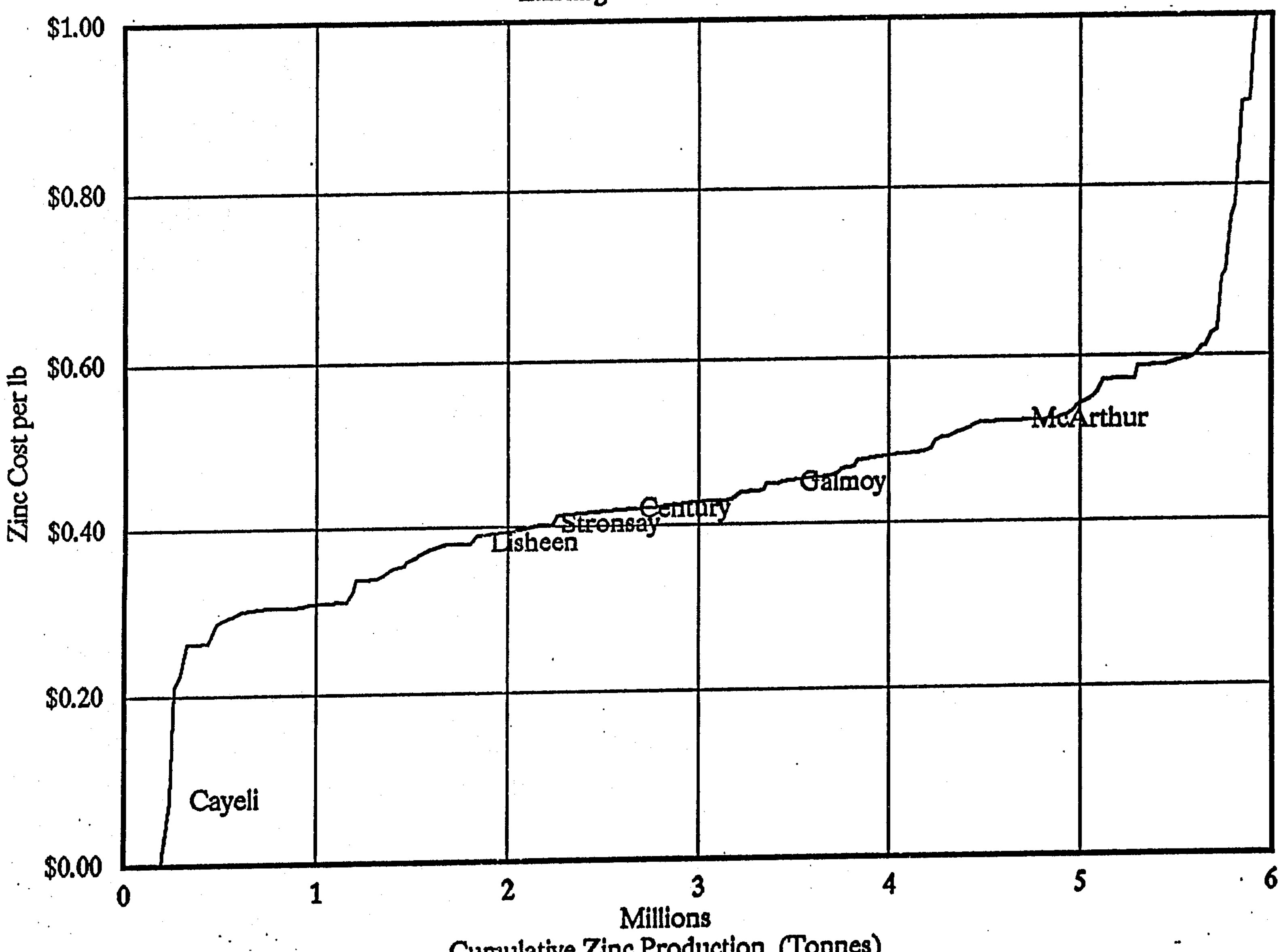
Manpower — All labour will be transported by bus and air on a rotational fly in/fly out basis between the site and Ft. Ware, Tsay Keh, Mackenzie, Ft. St. John, and Prince George.

Average for a typical year: (including Contractors):

	Working		Leave	Total on Payroll
	On Site	Off Site		
Mine	79	-	21	100
Processing	50	-	50	100
Administration	11	-	10	21
Ft. St. John Office	-	10	-	10
Trucking	24	-	7	31
Freight B/H	2	-	2	4
Camp	18	-	18	36
SUBTOTAL	184	10	108	302
Barge	-	6	-	6
Air	-	2	-	2
Rail	-	4	4	8
Port	-	13	-	13
SUBTOTAL	-	25	4	29
TOTAL B.C.	184	35	110	331
Toronto	-	2	-	2
Whitehorse	-	1	-	1
GRAND TOTAL	184	38	110	334

Zinc Production Costs – Byproduct Basis

Existing and Potential Producers



Includes all operating costs, transportation costs and smelter charges.
Long Term Average Smelter Charges. Prices: Pb–$.30/lb; Cu–$1.00/lb; Ag–$4/oz; Au–$350/0z

STRONSAY PROJECT BUDGET – CCAA Budget – Minimum Case

STRONSAY PROJECT – Budget by Month SUMMARY

Area	Code	Description	May/93	Jun/93	Jul/93	Aug/93	Sep/93	Oct/93	Nov/93	Dec/93	Jan/94	Feb/94	Mar/94	Apr/94	Total
Site Work	401	Camp Costs	4 300	1 700	1 700	1 700	1 700	3 200	0	0	0	0	0	0	14 300
	402	Site Supervision	11 800	3 800	3 800	3 800	3 800	6 800	0	0	6 000	0	0	0	39 800
	403	Vehicle Rentals	475	475	475	475	475	475	0	0	0	0	0	0	2 850
	404	Owner's Supplies	1 000	500	500	500	500	500	0	0	0	0	0	0	3 500
	405	Communication Charges	375	375	375	375	375	375	0	0	0	0	0	0	2 250
	454	Surveying	0	0	0	0	0	0	0	0	0	0	0	0	0
	475	Power Generation	200	200	200	200	200	5 100	0	0	0	0	0	0	6 100
	471	Roads	0	0	0	0	0	0	0	0	0	0	0	0	0
	472	Barge	0	0	0	0	0	0	0	0	0	0	0	0	0
	485	Shutdown Care & Maintenance	0	0	0	0	0	0	0	0	0	0	0	0	0
Sub total			18 150	7 050	7 050	7 050	7 050	16 450	0	0	6 000	0	0	0	68 800
Diamond Drilling															
	420	Surface Drilling	0	0	0	0	0	0	0	0	0	0	0	0	0
	431	Underground Drilling	0	0	0	0	0	0	0	0	0	0	0	0	0
	441	Core Handling	0	0	0	0	0	0	0	0	0	0	0	0	0
	442	Interpretation	0	0	0	0	0	0	0	0	0	0	0	0	0
	482	Fluke Claims	0	0	0	0	0	0	0	0	0	0	0	0	0
Sub total			0	0	0	0	0	0	0	0	0	0	0	0	0
Mining	490	Contractor	0	0	0	0	0	0	0	0	0	0	0	0	0
Testing															
	443	Metallurgy	0	0	0	0	0	0	0	0	0	0	0	0	0
	451	Rock Mechanics	0	0	0	0	0	0	0	0	0	0	0	0	0
	452	Surface Geotechnical	0	0	0	0	0	0	0	0	0	0	0	0	0
	453	Mill Site Geotechnical	0	0	0	0	0	0	0	0	0	0	0	0	0
Sub total			0	0	0	0	0	0	0	0	0	0	0	0	0
Environmental															
	461	Soils	0	0	0	0	0	0	0	0	0	0	0	0	0
	462	Aquatic Life	0	0	0	0	0	0	0	0	0	0	0	0	0
	463	Water Quality	1 000	1 000	1 000	1 000	1 000	1 000	0	0	0	0	0	0	6 000
	464	Air Quality	0	0	0	0	0	0	0	0	0	0	0	0	0
	465	Wildlife	0	0	0	0	0	0	0	0	0	0	0	0	0
	466	Acid Mine Drainage	0	0	0	0	0	2 000	0	0	0	0	0	0	2 000
	467	Settling Ponds	0	0	0	0	0	0	0	0	0	0	0	0	0
	468	Waste Handling	0	0	0	0	0	0	0	0	0	0	0	0	0
	502	Permitting	100	0	700	0	800	0	600	1 575	0	0	0	0	3 775
Sub total			1 100	1 000	1 700	1 000	1 800	3 000	600	1 575	0	0	0	0	11 775
Administration															
	501	Consultants	0	0	0	0	0	0	0	0	0	0	0	0	0
	503	Financial	8 600	8 600	14 625	8 600	8 600	8 600	8 600	8 600	13 600	8 600	8 600	8 600	114 225
	504	Legal Fees	0	0	0	0	0	0	0	0	0	0	0	0	0
	505	Head Office	5 000	0	0	5 000	0	0	5 000	0	2 000	5 000	0	0	22 000
	506	Partner's Reviews	0	0	0	0	0	0	0	0	0	0	0	0	0
	507	Management Fees	0	0	0	0	0	0	0	0	0	0	0	0	0
	508	Interest	0	0	0	0	0	0	0	0	0	0	0	0	0
Sub total			13 600	8 600	14 625	13 600	8 600	8 600	13 600	8 600	15 600	13 600	8 600	8 600	136 225
Total			32 850	16 650	23 375	21 650	17 450	28 050	14 200	10 175	21 600	13 600	8 600	8 600	216 800

STRONSAY PROJECT BUDGET – CCAA Budget – Minimum Case

STRONSAY PROJECT – Budget by Month SITE COST ELEMENTS															
Description	Account Code		May/93	Jun/93	Jul/93	Aug/93	Sep/93	Oct/93	Nov/93	Dec/93	Jan/94	Feb/94	Mar/94	Apr/94	Total
	Account	Sub													
Freight – Finlay Nav	39031	401	1 500	0	0	0	0	1 500	0	0	0	0	0	0	3 000
Freight – Air	39031	401	500	500	500	500	500	500	0	0	0	0	0	0	3 000
Freight – Truck	39031	401	600	0	0	0	0	0	0	0	0	0	0	0	600
Expediting	39031	401	200	200	200	200	200	200	0	0	0	0	0	0	1 200
Food	39031	401	1 500	1 000	1 000	1 000	1 000	1 000	0	0	0	0	0	0	6 500
Helicopter	39016	402	5 000	0	0	0	0	0	0	0	3 000	0	0	0	8 000
Travel – Watchman	39591	402	3 000	0	0	0	0	3 000	0	0	0	0	0	0	6 000
Salary – Regular	39632	402	3 300	3 300	3 300	3 300	3 300	3 300	0	0	0	0	0	0	19 800
Salary – Security	39632	402	0	0	0	0	0	0	0	0	0	0	0	0	0
Benefits – Regular	39632	402	500	500	500	500	500	500	0	0	0	0	0	0	3 000
Truck Rental	39013	403	400	400	400	400	400	400	0	0	0	0	0	0	2 400
Truck Parts	39013	403	75	75	75	75	75	75	0	0	0	0	0	0	450
Snowmobile – Parts	39013	403	0	0	0	0	0	0	0	0	0	0	0	0	0
Misc. supplies	39013	404	1 000	500	500	500	500	500	0	0	0	0	0	0	3 500
Communications – Fixed	33711	405	125	125	125	125	125	125	0	0	0	0	0	0	750
Communications – LD	33711	405	250	250	250	250	250	250	0	0	0	0	0	0	1 500
Water Samples	39516	463	1 000	1 000	1 000	1 000	1 000	1 000	0	0	0	0	0	0	6 000
Lime	39013	466	0	0	0	0	0	2 000	0	0	0	0	0	0	2 000
Dozer Rental	39013	471	0	0	0	0	0	0	0	0	0	0	0	0	0
Dozer Misc	39013	471	0	0	0	0	0	0	0	0	0	0	0	0	0
Fuel & Lubes	39031	475	100	100	100	100	100	5 000	0	0	0	0	0	0	5 500
Genset Purchase	39032	475	0	0	0	0	0	0	0	0	0	0	0	0	0
Genset Parts	39032	475	100	100	100	100	100	100	0	0	0	0	0	0	600
Fluke Assessment	39518	482	0	0	0	0	0	0	0	0	0	0	0	0	0
PERMITS TOTAL	39619	502	100	0	700	0	800	0	600	1 575	0	0	0	0	3 775
LO702682–Finbow			0	0	0	0	0	0	600	0	0	0	0	0	600
Free Miner's Certificate			0	0	0	0	0	0	0	1 575	0	0	0	0	1 575
SUP 17444–Access road			25	0	0	0	0	0	0	0	0	0	0	0	25
SUP 17445– Access Road			25	0	0	0	0	0	0	0	0	0	0	0	25
Water Lic. – Anvil Brook			50	0	0	0	0	0	0	0	0	0	0	0	50
WMP10601–Settling Pond			0	0	0	0	400	0	0	0	0	0	0	0	400
WMP10605–PAG Pad			0	0	0	0	400	0	0	0	0	0	0	0	400
WMP10939–Incinerator			0	0	700	0	0	0	0	0	0	0	0	0	700
PROPERTY TAXES TOTAL	39900	503	0	0	6 025	0	0	0	0	0	0	0	0	0	6 025
Finbow			0	0	125	0	0	0	0	0	0	0	0	0	125
Camp Site			0	0	5 000	0	0	0	0	0	0	0	0	0	5 000
Access Road			0	0	600	0	0	0	0	0	0	0	0	0	600
Access Road			0	0	300	0	0	0	0	0	0	0	0	0	300
TOTAL			19 250	8 050	14 775	8 050	8 850	19 450	600	1 575	3 000	0	0	0	83 600

STRONSAY PROJECT – Budget by Month WHITEHORSE ELEMENTS															
Description	Account Code		May/93	Jun/93	Jul/93	Aug/93	Sep/93	Oct/93	Nov/93	Dec/93	Jan/94	Feb/94	Mar/94	Apr/94	Total
Salary	39611	505	0	0	0	0	0	0	0	0	2 000	0	0	0	2 000
Travel	39611	402	0	0	0	0	0	0	0	0	3 000	0	0	0	3 000
TOTAL			0	0	0	0	0	0	0	0	5 000	0	0	0	5 000

STRONSAY PROJECT BUDGET – CCAA Budget – Minimum Case

STRONSAY PROJECT – Budget by Month CONSULTANTS AND CONTRACTORS ELEMENTS															
Description	Account Code		May/93	Jun/93	Jul/93	Aug/93	Sep/93	Oct/93	Nov/93	Dec/93	Jan/94	Feb/94	Mar/94	Apr/94	Total
	Account	Sub													
Kilborn	39520	501	0	0	0	0	0	0	0	0	0	0	0	0	0
Rescan	39520	502	0	0	0	0	0	0	0	0	0	0	0	0	0
Lakefield	39511	443	0	0	0	0	0	0	0	0	0	0	0	0	0
Cassels Brock	39590	504	0	0	0	0	0	0	0	0	0	0	0	0	0
Sultan	39520	501	0	0	0	0	0	0	0	0	0	0	0	0	0
Anderson	39520	501	0	0	0	0	0	0	0	0	0	0	0	0	0
Price Waterhouse	39520	503	0	0	0	0	0	0	0	0	5 000	0	0	0	5 000
CMD	39530	490	0	0	0	0	0	0	0	0	0	0	0	0	0
Joe Martin	39519	471	0	0	0	0	0	0	0	0	0	0	0	0	0
TOTAL			0	0	0	0	0	0	0	0	5 000	0	0	0	5 000

STRONSAY PROJECT – Budget by Month HEAD OFFICE ELEMENTS															
Description	Account Code		May/93	Jun/93	Jul/93	Aug/93	Sep/93	Oct/93	Nov/93	Dec/93	Jan/94	Feb/94	Mar/94	Apr/94	Total
	Account	Sub													
															0
CKB/GGC Travel & Expenses	39591	505	5 000	0	0	5 000	0	0	5 000	0	0	5 000	0	0	20 000
Other Travel & Expenses	39591	505	0	0	0	0	0	0	0	0	0	0	0	0	0
Misc Charges	39619	505	0	0	0	0	0	0	0	0	0	0	0	0	0
Toronto Overhead	39612	505	0	0	0	0	0	0	0	0	0	0	0	0	0
Capital Taxes	39901	503	8 600	8 600	8 600	8 600	8 600	8 600	8 600	8 600	8 600	8 600	8 600	8 600	103 200
Management Fees	52000	507	0	0	0	0	0	0	0	0	0	0	0	0	0
TOTAL			13 600	8 600	8 600	13 600	8 600	8 600	13 600	8 600	8 600	13 600	8 600	8 600	123 200

STRONSAY PROJECT – RESERVES AND INVENTORY

MINERAL INVENTORY – 0% Cutoff													
Source	Area	Proven + Probable				Possible				Total			
		Tonnes	Pb%	Zn%	Ag g/t	Tonnes	Pb%	Zn%	Ag g/t	Tonnes	Pb%	Zn%	Ag g/t
CMD	North Cirque 298+35N to 303+75N	17 263 730	2.37	8.81	51.7					17 263 730	2.37	8.81	51.7
CMD	North Cirque 296+25N to 298+35N	11 848 483	2.07	7.84	47.3					11 848 483	2.07	7.84	47.3
										0	0.00	0.00	0.0
	Total North Cirque	29 112 213	2.25	8.41	49.9	0	0	0	0	29 112 213	2.25	8.41	49.9
CRI	Total South Cirque					20 013 083	1.94	6.41	29.6	20 013 083	1.94	6.41	29.6
	Total	29 112 213	2.25	8.41	49.9	20 013 083	1.94	6.41	29.6	49 125 296	2.12	7.60	41.6

MINEABLE RESERVES – 6% Cutoff													
Source	Area	Proven + Probable				Possible				Total			
		Tonnes	Pb%	Zn%	Ag g/t	Tonnes	Pb%	Zn%	Ag g/t	Tonnes	Pb%	Zn%	Ag g/t
CMD	North Cirque 298+35N to 303+75N	14 859 672	2.42	8.84	51.9					14 859 672	2.42	8.84	51.9
CMD	North Cirque 296+25N to 298+35N	9 818 344	2.20	8.05	49.1					9 818 344	2.20	8.05	49.1
										0	0.00	0.00	0.0
	Total North Cirque	24 678 016	2.33	8.53	50.8	0	0	0	0	24 678 016	2.33	8.53	50.8
	Total South Cirque									0	0.00	0.00	0.0
	Total	24 678 016	2.33	8.53	50.8	0	0	0	0	24 678 016	2.33	8.53	50.8

STRONSAY PROJECT

PROJECT EXPENDITURE SUMMARY

The work, and costs thereof, conducted to date on the Stronsay Property are summarized annually as follows:

1977	Staked Cirque claims, mapping and geochemistry.	
	Exploration Expenditures (1977$)	$ 84,000
1978	Staked Elf and Fluke claims, mapping, geochemistry, geophysics.	
	Diamond drilling 6 holes/882 m.	
	Exploration Expenditures (1978$)	$ 333,000
1979	Mapping, geology, geochemistry.	
	Diamond drilling 29 holes/9,018 m.	
	Exploration expenditures (1979$)	$1,566,000
1980	Mapping, geology, geochemistry, geophysics.	
	Diamond drilling 43 holes/16,408 m.	
	Exploration expenditures (1980$)	$4,370,000
	Finbow airstrip construction (1980$)	$ 190,000
1981	Geology.	
	Diamond drilling 53 holes/21,816 m	
	Exploration expenditures (1981$)	$5,235,000
	Finbow airstrip (1981$)	$ 116,000
	Akie/Paul Valley Road (1981$)	$4,990,000
	Capital Equipment (1981$)	$1,600,000
1982	Geology.	
	Diamond drilling 14 holes/13,088 m.	
	Exploration Expenditures (1982$)	$2,726,000
	Road Construction (1982$)	$ 93,000
1983 - 1988	No activity	
1989	Camp and equipment mobilization, access decline, mapping, environmental studies.	
	Diamond drilling 4 holes/99 m.	
	Exploration Expenditures (1989$)	$8,045,000
1990	Underground exploration development, surface and underground diamond drilling, environmental studies, engineering studies.	
	Diamond drilling 125 holes/7,978 m.	
	Exploration Expenditures (1990$)	$8,868,000
1991	Underground exploration, mapping, bulk sample, environmental studies, engineering studies.	
	Diamond drilling 86 holes/4,973 m	
	Exploration Expenditures (1991$)	$3,826,000
1992	Permitting, environmental studies.	
	Exploration Expenditures (1992$)	$ 940,000

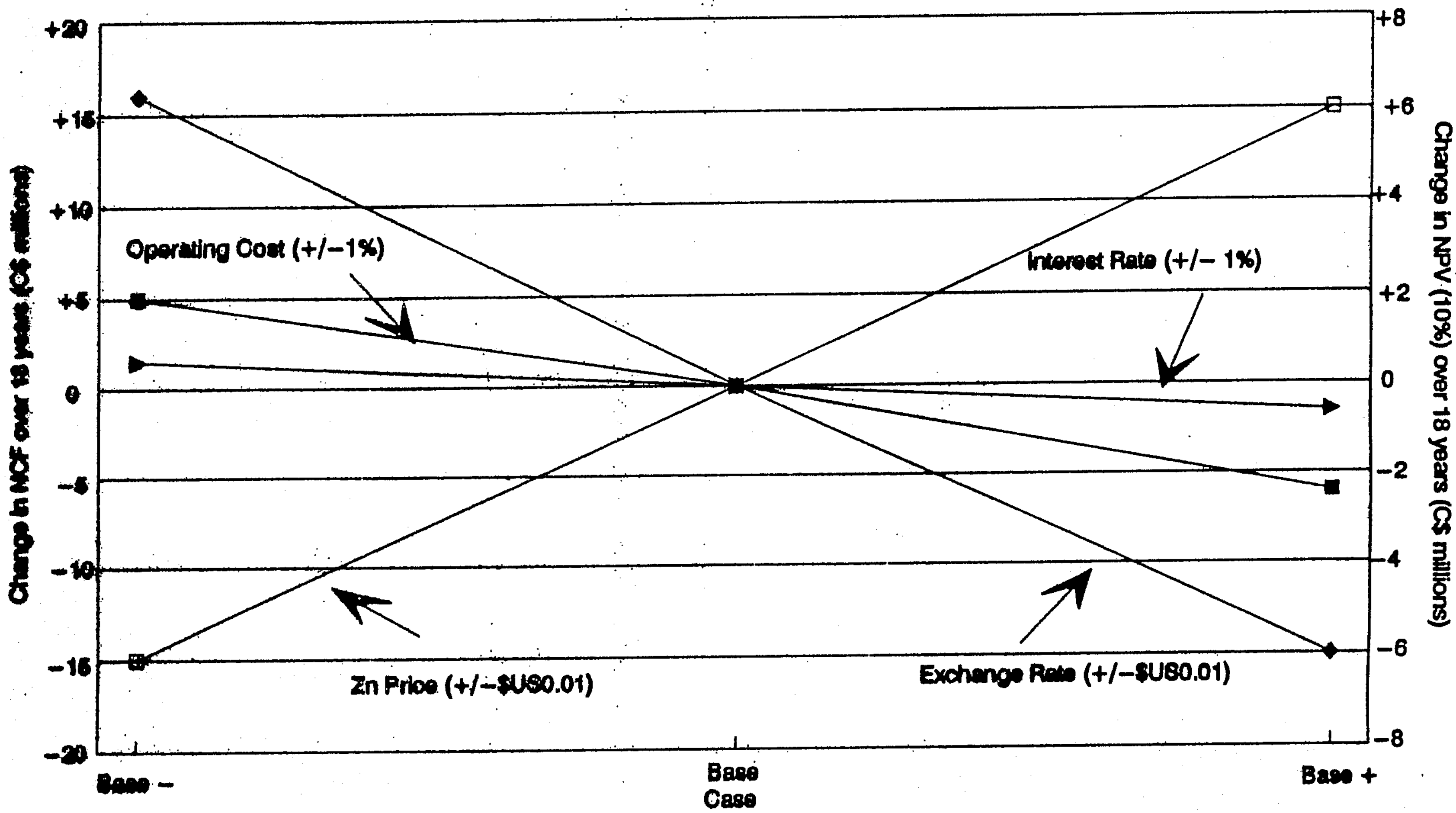
In dollars of the day, the total expenditures on the Stronsay Property are:

1977 - 1982	$21,303,000
1989 - 1992	$21,679,000
TOTAL	$42,982,000

In 1992 dollars, the total spent from 1977 through 1982 is $37,457,000, and from 1989 through 1992 is $24,104,000 for total expenditures of $61,561,000 (1992$).

STRONSAY PROJECT

Sensitivity Analysis



Base Case Assumptions:

Zinc Price:	GBSDT4
Exchange Rate:	C$1.00 = US$0.78
Interest Rate:	8%
Debt:	C$100 million

Transportation

SA DENA HES
LOADOUT AND ROAD COSTS

		DRY 12340	WET 13142
CONTRACTORS	4,819.00	.39	.37
CONSULTANTS	1,274.00	.10	.10
LEASE-TRAILERS	83,383.00	6.76	6.34
WAYBILLS	91.00	.0075	.0075
POWER	6,110.00	.50	.46
TRUCKING		55.63	52.01
TAXES & INSURANCE	16,338.00	1.32	1.24
	TOTAL	64.71	60.53

ROAD TRANSPORT

CONTRACTORS -TRAILER RAIL MAINTENANCE	6,667.00	.16	.15
CONSULTANTS-TECHNICAL TRAILER INSPECTION- BACON DONALDSON.	1,357.00	.03	.03
CONCENTRATE TRUCKING		43.99	41.07
FUEL (REBATE) EXCESS COST	10,000.00	.24	.22
CONCENTRATE PROFIT (REBATE)	(115,850.00)	(2.73)	(2.56)
OVERWEIGHT CHARGES	4,667.00	.11	.10
DELAY CHARGES	3,583.00	.08	.08
INSURANCE ON TRAILERS	12,000.00	.28	.27
ROYALATIES		1.07	1.10
PUBLIC RELATIONS PRINTING & DISTRIBUTION OF INFO. PAMPHLETS.	500.00	.01	.01
TAXES	1,050.00	.02	.02
	TOTAL	43.26	40.49

LOADOUT

TYPICAL EXAMPLE - MOST COSTS ARE FIXED

		DMT 42500	WMT 45263
CONTRACTORS - TECHNICAL NORTHLAND FLEET - MONTHLY CLEAN-UP -	$ 1,000.00		
CONTRACTORS - OPERATIONS NORTHLAND FLEET LOADING OF TRUCKS.	68,069.00		
WAYBILLS/MANIFESTS	300.00		
	69,369.00	1.63	1.53
GLYCOL (STILLBOTTOMS)	8,572.00	.20	.19
	TOTAL	1.83	1.72

LOADOUT - FARO

ORIGIN	1994 VOLUMES LEAD/ZINC	LOCATION OF ACTIVITY	DEFINITION/SIZE OF TERMINAL, EQUIPMENT, CREW & WORK	COSTS: DRY M. TONNE	COSTS: WET M. TONNE
				AVERAGE MONTHLY TONNAGE	
FARO	DRY M. TONNES 510,000 WET M. TONNES 543,150	MILL-FARO LOADOUT	INSIDE STORAGE: 5,000 TONNES ZN, 3,000 TONNES PB. DIRECT FEED -MILL TO WAREHOUSE BY CONVEYOR. EQUIPMENT: 1 X 980 FRONT END LOADER PB - HOPPER AND CONVEYOR ZN - HOPPER AND CONVEYOR PB - RETRACTABLE CHUTE WITH VACUUM SYSTEM ZN - RETRACTABLE CHUTE WITH VACUUM SYSTEM GLYCOL STORAGE AND DISPENSER SCALE: 1 X 114' LONG X 11' WIDE (5 PLATES) CONCRETE PAD - 184' LONG SCALE HOUSE WITH COMPUTER PACKAGE CREW: 1 SCALE OPERATOR PER 12 HOUR SHIFT 1 LOADER OPERATOR PER 12 HOUR SHIFT MAINTENANCE: PROVIDED BY MILL PERSONEL. OP/PERSONNEL: PROVIDED BY CONTRACTOR ADMIN/SUPERVISOR: TRANSPORTATION DEPARTMENT WORK: OPERATION OF THE LOADOUT FACILITY AT THE MILL IN FARO INCLUDES LOADING OF CONCENTRATE HAULAGE TRUCKS TO MAXIMUM CAPACITY UTILIZING A LOADER, HOPPER, CONVEYOR, CHUTE SYSTEM IN CONJUNCTION WITH A COMPUTERIZED SCALE SYSTEM. LOADOUT OPERATES TWENTY FOUR HOURS PER DAY, SEVEN DAYS PER WEEK.	42,500 DMT LOADOUT COSTS $1.83	45,263 WMT LOADOUT COSTS $1.72

LOADOUT

TYPICAL EXAMPLE - MOST COSTS ARE FIXED

		DMT 42500	WMT 45263
CONTRACTORS - TECHNICAL	$ 1,000.00		
NORTHLAND FLEET - MONTHLY CLEAN-UP -			
CONTRACTORS - OPERATIONS	68,069.00		
NORTHLAND FLEET LOADING OF TRUCKS.			
WAYBILLS/MANIFESTS	300.00		
	69,369.00	1.63	1.53
GLYCOL (STILLBOTTOMS)	8,572.00	.20	.19
TOTAL		1.83	1.72

ROAD TRANSPORT - FARO

ORIGIN	1994 VOLUMES LEAD/ZINC	LOCATION OF ACTIVITY	DEFINITION/SIZE OF TERMINAL, EQUIPMENT, CREW & WORK	COSTS DRY M. TONNE	COSTS WET M. TONNE
				AVERAGE MONTHLY TONNAGE	
FARO	DRY M. TONNES 510000 WET M. TONNES 543150	WHITEHORSE	TERMINAL: 5 TRUCK BAYS, DISPATCH ROOM AND OFFICE - OWNED BY CURRAGH EQUIPMENT: 44 TRACTORS - TANDEM DRIVE, 425 H.P./1500 LBS. TORQUE, CAT ENGINES, TRACTORS SUPPLIED BY CONTRACTOR 44 B-TRAIN TRAILER SETS, EQUIPPED WITH 4 STAINLESS STEEL CONTAINERS PER TRAILER. PAYLOAD CARRYING CAPACITY 47.0 WET M. TONNES. TRAILERS AND CONTAINERS OWNED BY CURRAGH INC. VEHICLE LENGTH 85 FEET, GVW 160,000 LBS. OPERATION AND MAINTENANCE PROVIDED BY CONTRACTOR CREW: 115 DRIVERS, 33 MAINTENANCE AND OPERATIONS SUPPORT PEOPLE AND 6 ADMINISTRATIVE SUPPORT PEOPLE SYSTEM CAPACITY: CAPABLE OF TRANSPORTING 1900 WET METRIC TONNES PER DAY OF CONCENTRATE. CAPABLE OF TRANSPORTING A MINIMUM OF 500-20 FT. CONTAINERS ON BACKHAUL. CAPABLE OF TRANSPORTING 90,000 LITRES OF FUEL PER DAY ON BACKHAUL.	42500 DMT ROAD TRANSPORT COSTS $43.26	45263 WMT ROAD TRANSPORT COSTS $40.49

ROAD TRANSPORT - FARO

ORIGIN	1994 VOLUMES LEAD/ZINC	LOCATION OF ACTIVITY	DEFINITION/SIZE OF TERMINAL, EQUIPMENT, CREW & WORK	COSTS	
				DRY M. TONNE	WET M. TONNE
				AVERAGE MONTHLY TONNAGE	
FARO	51000 DMT 543150 WMT	WHITEHORSE	WORK: THE ROAD TRANSPORTATION SYSTEM FOR THE FARO MINE INCLUDES THE CONTRACT ROAD HAULAGE OF CONCENTRATES AND EMPTY CONTAINERS FROM THE FARO MILL TO THE ORE TERMINAL IN SKAGWAY, AND THE CARRIAGE OF CONTAINERS LOADED WITH LIME, SODA ASH, GRINDING MEDIA, OTHER CHEMICALS FROM THE PORT OF SKAGWAY TO THE FARO MILL.		

ROAD TRANSPORT

CONTRACTORS -TRAILER RAIL MAINTENANCE	6,667.00	.16	.15
CONSULTANTS-TECHNICAL TRAILER INSPECTION - BACON DONALDSON.	1,357.00	.03	.03
CONCENTRATE TRUCKING		43.99	41.07
FUEL (REBATE) EXCESS COST	10,000.00	.24	.22
CONCENTRATE PROFIT (REBATE)	(115,850.00)	(2.73)	(2.56)
OVERWEIGHT CHARGES	4,667.00	.11	.10
DELAY CHARGES	3,583.00	.08	.08
INSURANCE ON TRAILERS	12,000.00	.28	.27
ROYALATIES		1.07	1.10
PUBLIC RELATIONS PRINTING & DISTRIBUTION OF INFO. PAMPHLETS.	500.00	.01	.01
TAXES	1,050.00	.02	.02
	TOTAL	43.26	40.49

LOADOUT AND ROAD TRANSPORT

ORIGIN	1994 VOLUMES LEAD/ZINC	LOCATION OF ACTIVITY	DEFINITION/SIZE OF TERMINAL, EQUIPMENT, CREW & WORK	COSTS DRY M. TONNE	COSTS WET M. TONNE
				AVERAGE MONTHLY TONNAGE	
SA DENA HES	DRY M. TONNES 148080 WET M. TONNES 157705	WATSON LAKE	TERMINAL: 3 TRUCK BAYS, DISPATCH ROOM AND OFFICE OWNED BY CURRAGH. EQUIPMENT: 15 TRACTORS - TRIDEM POWER DRIVE; 425 H.P., 1500 LB. TORQUE, CAT ENGINE. TRACTORS SUPPLIED BY CONTRACTOR. 15 B-TRAIN TRAILERS, EQUIPPED WITH 4 STAINLESS STEEL CONTAINERS PER TRAILER. VEHICLE LENGTH IS 82.0 FEET, GVW 160,000 LBS. TRAILERS AND CONTAINERS ARE OWNED BY CURRAGH. OPERATOR AND MAINTENANCE PROVIDED BY THE CONTRACTOR CREW: 30 DRIVERS, 14 MAINTENANCE AND OPERATIONS PEOPLE AND 4 ADMINISTRATIVE SUPPORT PEOPLE. SYSTEM CAPACITY: CAPABLE OF CARRYING 500 WET METRIC TONNES OF CONCENTRATE PER DAY. CAPABLE OF TRANSPORTING FUEL IN CONTAINERS. WORK: THE ROAD TRANSPORTATION SYSTEM FOR THE SA DENA HES MINE INCLUDES THE OPERATION OF LOADOUT AT THE MINE, HAULAGE OF CONCENTRATES FROM THE SA DENA HES MINE TO THE ORE TERMINAL IN SKAGWAY AND THE CARRIAGE OF FUEL IN CONTAINERS FROM THE PORT OF SKAGWAY TO THE SA DENA HES MINE.	12340 DMT LOADOUT COSTS $64.71	13142 WMT LOADOUT COSTS $60.53

SA DENA HES
LOADOUT AND ROAD COSTS

		DRY 12340	WET 13142
CONTRACTORS	4,819.00	.39	.37
CONSULTANTS	1,274.00	.10	.10
LEASE-TRAILERS	83,383.00	6.76	6.34
WAYBILLS	91.00	.0075	.0075
POWER	6,110.00	.50	.46
TRUCKING		55.63	52.01
TAXES & INSURANCE	16,338.00	1.32	1.24
	TOTAL	64.71	60.53

SKAGWAY - TERMINAL, SHIPLOADER AND FUEL FACILITY

ORIGIN	1994 VOLUMES LEAD/ZINC	LOCATION OF ACTIVITY	DEFINITION/SIZE OF TERMINAL, EQUIPMENT, CREW & WORK		COSTS DRY M. TONNE AVERAGE MONTHLY TONNAGE	 WET M. TONNE
FARO & SDH	543150 WMT 157704 WMT 700854 WMT	SKAGWAY, ALASKA	TERMINAL & SHIPLOADER	THE TERMINAL IS 720' LONG AND 150' WIDE; STORAGE CAPACITY OF 96000 WMT. TWO RECEIVING DOORS, 6 BELT FEEDERS AND CONVEYOR SYSTEM TO THE SHIPLOADER. SYSTEM IS CAPABLE OF LOADING 15000 WMT PER DAY. THE TERMINAL WAS RENOVATED IN 1990/91 TO MEET EPA AND OSHA STANDARDS. SHIPLOADERS FOUNDATION WAS REPLACED. EXPECTED LIFE OF THE FACILITY IS 30 YEARS. COST OF RENOVATION US$ 24 MILLION. FACILITY IS OWNED BY ALASKA INDUSTRIAL DEVELOPMENT AGENCY - AN ALASKA STATE AGENCY. CURRAGH HAS A 20 YEAR LEASE ON THE FACILITY. COST OF PURCHASE AND RENOVATIONS HAS BEEN AMORTIZED OVER 15 YEARS.	FARO = 42500 SDH = 12340 TOTAL: 54840 SKAGWAY TML & FUEL FACILITY OPERATING COSTS $13.35 ALLOCATION OF COSTS FARO $11.52 SDH $ 4.98	FARO = 45263 SDH = 13142 TOTAL: 58405 SKAGWAY TML & FUEL FACILITY OPERATING COSTS $12.52 ALLOCATION OF COSTS FARO $10.81 SDH $4.67
			EQUIPMENT:	2 FORKLIFTS WITH ROTATOR CAPACITY OF 32 S/TON PER LIFT, 1 CAT DH-6 RUBBER TRACKED WITH BLADE. EQUIPMENT IS SUPPLIED BY CURRAGH 2 FRONT END LOADERS (980 CAT). EQUIPMENT IS SUPPLIED BY CONTRACTOR.		

SKAGWAY - TERMINAL, SHIPLOADER AND FUEL FACILITY

ORIGIN	1994 VOLUMES LEAD/ZINC	LOCATION OF ACTIVITY	DEFINITION/SIZE OF TERMINAL, EQUIPMENT, CREW & WORK		COSTS: DRY M. TONNE	COSTS: WET M. TONNE
					AVERAGE MONTHLY TONNAGE	
			FUEL FACILITY:	FUEL FACILITY WAS BUILT IN 1992 AND MEETS E.P.A. FEDERAL AND ALASKA'S STATE ENVIRONMENTAL LAWS AND THE ALASKA STATE FIRE MARSHALL'S REGULATIONS. THE FACILITY WAS BUILT AND IS OWNED BY THE ALASKA INDUSTRIAL DEVELOPMENT AGENCY. THE FACILITY SERVES BOTH THE FARO AND SA DENA HES ROAD TRANSPORT CARRIERS. IT IS A "SELF SERVE" SYSTEM - DRIVERS FUEL THEIR OWN VEHICLES. EXPECTED LIFE OF THE FACILITY IS 30 YEARS AND FORMS PART OF THE 20 YEAR LEASE OF THE TERMINAL FACILITY. THIS PROJECT IS AMORTIZED OVER 7 YEARS.		
			STORAGE CAPACITY:	160,000 US GALLONS		
			CREW:	TERMINAL OPEATIONS - 10 PERMANENT STAFF SHIPLOADING - 8 TEMPORARY STAFF TOTAL: 18		

	DRY	WET
SDH	12340	13142
FARO	42500	45263
TOTAL	54840	58405

PORT OF SKAGWAY		DRY	WET
ELECTRICAL	$ 21,250.00	.36	.36
CONTRACTORS-TECHNICAL YEAR END SURVEYORS, FLOW MOISTURE CERTIFICATION.	5,200.00	.09	.09
O.S.H.A. SHIP LOADER INSPECTION.		.01	.01
CONTRACTORS-EPA AIR QUALITY ANALYSIS.	4,078.00	.09	.09
CONTRACTORS-BHEP	5,150.00	.09	.09
CONTRACTORS - SPM	6,667.00	.12	.11
TML & FUEL FACILITY	528,054.00	9.63	9.04
ASSAY FEES HUNZ + HUNZ.	2,511.00	.05	.04
CONCENTRATE PORT HANDLING (INVOICED DIRECT TO FARO/SDH AT THIS RATE)		3.15	2.96
SLIP	4,402.00	.08	.08
CONCENTRATE PROFIT REBATE		(.35)	(.37)
US CUSTOMS DUTY		.02	.02
	TOTAL COSTS	13.35	12.52
ALLOCATION TO SDH ($100,107) INCLUDES WHITEHORSE SUPPORT & ADMINISTRATION		(1.83)	(1.71)
COST TO FARO		11.52	10.81
COST TO SDH		4.98	4.67

ESTIMATED 1994 OPERATING AND MAINTENANCE EXPENSES OF SKAGWAY ORE TERMINAL AND FUEL FACILITY

1.

DESCRIPTION	AMOUNT	FREQUENCY	TOTAL
LEASE COSTS	373.70	12	4,484.40
INSURANCE	40,832.45	1	40,832.45
			45,316.85
7% GENERAL AND ADMINISTRATIVE EXPENSES			3,172.18
TOTAL ESTIMATED 1993 O & M EXPENSES			48,489.03

PAYABLE:

JANUARY 1	(PAID)	12,122.25
APRIL 1	(PAID)	12,122.25
JULY 1		12,122.27
OCTOBER 1		12,122.26
		48,489.03

2. MINIMUM AMOUNT DUE FOR QUARTERLY PERIOD.

A. LEVEL DEBT SERVICE ON THE SERIES 1990A BONDS USING A 15 YEAR AMORTIZATION SCHEDULE PRODUCES TOTAL DEBT SERVICE OVER THE PERIOD OF $43,349,807, WITH AVERAGE ANNUAL DEBT SERVICE OF	2,889,987
B. COST OF ISSUANCE IN EXCESS OF 500,000	-
C. INITIAL ANNUAL TRUSTEE FEE	5,465
ANNUAL MINIMUM AMOUNT	2,895,452
QUARTERLY PAYMENT	723,863

THROUGHPUT FEE:

WET METRIC TONNES	FEE PER WET METRIC TONNE	YTD TONNES SHIPPED THROUGH QUARTEREND	THROUGHPUT FEE
1 - 200,000	$ -	132,605	-
200,001 - 400,000	1.13	-	-
400,001 - 600,000	.84	-	-
600,001 AND OVER	.38	-	-
TOTAL		132,605	
PAID YEAR TO DATE			-
DUE FOR THIS QUARTER'S THROUGHPUT			-
TOTAL DUE			723,863

PLEASE WIRE TRANSFER THE BALANCE DUE NO LATER THAN 10 DAYS AFTER THE END OF THE QUARTER. (ADJUSTMENTS ARE MADE AT YEAR END).

SKAGWAY FUEL FACILITY

ESTIMATED 1994 OPERATING AND MAINTENANCE EXPENSES OF TANK FARM:

DESCRIPTION	AMOUNT	FREQUENCY	TOTAL
LEASE COSTS	$176.28	12	$2,115.36
INSURANCE	965.00	1	965.00
			3,080.36
7% GENERAL AND ADMINISTRATIVE EXPENSES			215.62
TOTAL ESTIMATED 1994 O & M EXPENSES			3,295.98

PAYABLE:

JANUARY 1	(PAID)	823.98
APRIL 1	(PAID)	824.00
JULY 1		824.00
OCTOBER 1		824.00
		$3,295.98

PLUS

USER FEE FOR THE FACILITY IS US$ $0.03 PER GALLON

AVERAGE MONTHLY CONSUMPTION:

YUKON ALASKA TRANSPORT (FARO) 200,000 GALLONS
GALLON X 3 MONTHS 600,000 GALLONS X .03 = $18,000.00

GATEWAY TRANSPORT (SDH) 53,000 GALLONS
GALLONS X 3 MONTHS 159,000 GALLONS X .03 = $4,770.00

PAYMENT DUE ON OR BEFORE THE LAST DAY IN THE QUARTER
(ADJUSTMENTS ARE MADE AT YEAR END)

PRODUCT: BULK FREIGHT CONTAINERS

ORIGIN	1994 VOLUMES LEAD/ZINC	LOCATION OF ACTIVITY	DEFINITION/SIZE OF TERMINAL, EQUIPMENT, CREW & WORK	COSTS	
				DRY M. TONNE	WET M. TONNE
				AVERAGE MONTHLY TONNAGE	
VARIOUS LOCATIONS IN THE U.S.A. VIA PORT OF SKAGWAY	268 CONTAINERS	WHITEHORSE	CURRAGH FARO AND SA DENA HES PURCHASE LIME, SODA ASH AND GRINDING BALLS IN BULK AND OTHER CHEMICALS SUCH AS CYANIDE AND XANTHATE IN DRUMS IN THE UNITED STATES AND CANADA. SUPPLIERS SUBMIT BID PRICES FOB ORIGIN TO THE PROCUREMENT DEPARTMENT AND THE PRICES REMAIN IN EFFECT FOR ONE YEAR OR FOR LONGER PERIODS. TO DETERMINE THE LANDED COSTS (FOB MINE) THE TRANSPORTATION DEPARTMENT PROVIDES PROCUREMENT WITH TRANSPORTATION ROUTINGS, FREIGHT RATES AND ASSSOCIATED SERVICE FACTORS SUCH AS TRANSIT TIME AND CUSTOMS SERVICE ON FOREIGN GOODS ENTERING CANADA. PRESENTLY FARO'S GRINDING BALLS ORIGINATE IN MONT JOLI QUEBEC AND ARE SOLD FOB MINE SITE. IN THE PAST THEY MANUFACTURED GRINDING BALLS IN DULUTH, MINNESOTA AND KANSAS CITY, MISSOURI. PRESENTLY SODA ASH ORIGINATES IN GREENRIVER, WYOMING AND LIME ORIGINATES IN TACOMA, WASHINGTON. PRODUCTS ORIGINATING IN DULUTH AND KANSAS CITY WERE SHIPPED BY RAIL IN GONDALA CARS TO ALASKA MARINE LINES (AML) SEATTLE DOCK, TRANSFERED FROM RAILCAR INTO TWENTY FOOT HALF-HIGH CONTAINERS. SODA ASH FROM GREEN RIVER IS LOADED DIRECTLY INTO CURRAGH'S CONTAINERS WHICH TRAVEL ON RAIL CONTAINER CARS. ON ARRIVAL AT AML'S DOCK IN SEATTLE, THEY ARE TRANSFERRED DIRECTLY TO THE BARGE. ALL FREIGHT IN CONTAINERS IS LOADED ABOARD THE BARGE AND CARRIED TO SKAGWAY, ALASKA, TRANSFERRED FROM BARGE TO TRUCK AND CARRIED TO THE MINE. UPON ARRIVAL AT THE MINE AN ON-SITE CONTRACTOR UNLOADS AND STORES THE CONTAINERS INTO STORAGE AREA AND LOADS OUT THE EMPTIES ON CONCENTRATE TRAILERS FOR FURTHERANCE TO SUPPLIERS. THE ON-SITE CONTRACTOR DISTRIBUTES THESE BULK AND PACKAGED FREIGHT TO THE MINE SITE AND USERS.	42500 DMT BACKHAUL COSTS $2.29 DMT	45263 WMT BACKHAUL COSTS $2.14 WMT

PRODUCT: BULK FREIGHT CONTAINERS

ORIGIN	1994 VOLUMES LEAD/ZINC	LOCATION OF ACTIVITY	DEFINITION/SIZE OF TERMINAL, EQUIPMENT, CREW & WORK	COSTS	
				DRY M. TONNE	WET M. TONNE
				AVERAGE MONTHLY TONNAGE	
			EQUIPMENT: CURRAGH OWNS AND OERATES 76 CONTAINERS (TWENTY-EIGHT, EIGHT FOOT) THAT ARE DESIGNED FOR LOADING THROUGH THE ROOF. THEY ARE COMMITTED TO CARRYING LIME AND SODA ASH. WORK: ALL WORK IS PERFORMED BY CONTRACTORS.		

FREIGHT BACKHAUL (FARO) (20 CONTAINERS)		DRY 42500	WET 45263
FREIGHT-CONSOLIDATION/ DRAYAGE	$ 2,885.00	.07	.06
FREIGHT-BARGE/TRUCK BACKHAUL	30,485.00	.72	.67
FREIGHT-OFFLOAD-SITE DISTRIBUTION	58,072.00	1.37	1.28
CONTRACTORS-TECHNICAL	4,324.00	.10	.10
CONTRACTORS-REPAIRS	1,422.00	.03	.03
	TOTAL	2.29	2.14

SUMMARY OF COSTS F.O.B. SHIP SKAGWAY

	FARO		SDH	
	DRY	WET	DRY	WET
LOADOUT	1.83	1.72		
ROAD TRANSPORT	43.26	40.49	64.71	60.53
STORAGE SHIPLOADING	11.52	10.81	4.98	4.67
BACKHAUL	2.29	2.14	2.29	2.14
	58.90	55.16	71.98	67.34

Human Resources

Revised June 15, 1993

Curragh Inc.

HUMAN RESOURCES INFORMATION

NORMAL MANPOWER LEVELS

	Faro	Sä Dena Hes	Whitehorse
Hourly	360	44	
Salaried	80	48	9
Sub-total	440	92	9
On-Site Contractors	168	98	
Sub-total	608	190	9
Concentrate Trucking Contractors	156	51	
TOTAL	**764**	**241**	**9**

Total - 1,014

CURRENT MANPOWER STATUS (June 1, 1993)

	Faro	Sä Dena Hes	Whitehorse
Hourly	6		
Salaried	17	7	6
TOTAL	**23**	**7**	**6**

Total - 36

LAYOFF STATUS

Faro: 32 Salaried employees on temporary layoff
200 Hourly employees on temporary layoff

Sä Dena Hes: 5 Salaried employees on temporary layoff
1 Hourly employee on temporary layoff

Remaining employees are on indefinite layoff.

Unionized employees have recall rights for up to two years.

LIABILITIES

	Hourly	**Salaried**
Faro		
Vacation Pay	763,304.58	116,985.03
Severance	989,913.52*	118,188.82
P.I.L.	541,222.27	18,957.66
* Payable only in event of permanent shutdown of Company operations		
Sä Dena Hes		
Vacation Pay	2,000	13,600
Severance	1,500	9,950
P.I.L.	–	–
Housing Buy–back	–	260,000

Total Liabilities – 2,575,621.88

LABOUR RELATIONS

In Yukon employees who are unionized are regulated by the Canada Labour Code.

All other employees are regulated by the Yukon Employment Standards Act.

At present all operating and maintenance employees (hourly) at Faro are covered by a collective agreement with the United Steelworkers of America, Local 1051.

Operating and maintenance employees at Sä Dena Hes have recently become certified by U.S.W.A., but no agreement is yet in place.

All other employees, i.e. office, clerical, engineering and technical, supervisory and above, are non–union.

SUMMARY OF EXISTING COLLECTIVE AGREEMENT

Scope: Agreement covers all maintenance and operating employees below the rank of supervisor (foreman) and excludes office and clerical and technical employees.

Terms of Agreement: Three years – November 1, 1990 to October 31, 1993

Rates of Pay:

Weighted average prior to November 1, 1990	$17.07
Weighted average end of Year 1	18.27 – 7%
Weighted average end of Year 2	19.02 – 4%
Weighted average end of Year 3	19.92 – 4.7%

Present rates range from Labourer at	$15.78
to Journeyman/Tradesman at	22.53

Curragh's rates of pay compare favourably with mining industry rates in B.C. and the Yukon.

Specific rates of pay are found on Page 54 of the Collective Agreement.

Hours of Work: Employees work either 8 hours per day – 5 days per week (5 x 2 schedule), or rotating shifts of four 12–hour shifts on, 4 days off, rotating from days to nights (4 x 4 schedule).

Overtime is paid at time and one half for all hours worked in excess of 8 per day or 40 per week for 5 x 2 scheduled employees, and in excess of 12 hours per day, or on scheduled days off, for 4 x 4 scheduled employees.

We do not pay double time overtime.

Lunch Breaks: 8– hour shift employees receive one 30–minute paid lunch break per shift.

12–hour shift employees receive two 30–minute paid lunch breaks per shift.

Rest Breaks: In addition to lunch breaks, maintenance employees receive two 15–minute rest breaks per shift. Operations employees take rest breaks only as conditions permit.

Safety & Health: A joint Union/Management Occupational Health and Safety Committee is in place and deals with all health and safety matters.

Job Postings & Lines of Progression: A system of job postings and lines of progression provides for on–the–job training and promotion from within.

Vacation: Vacation entitlements are three (3) weeks after one completed calendar year, and 4 weeks after three completed calendar years.

Vacation pay is based on previous years earnings as follows:

4% less than one year service
5% one or two years service
7% three or more years service

Paid Holidays: We observe ten (10) paid holidays per year, as well as three (3) floaters (2 for 12–hour shift employees.

Shift Premiums: 60¢ per hour for night shifts
75¢ per hour for work on Saturday and Sunday

Health & Welfare Benefits:	Employees are covered for the following group insurance benefits: Life Insurance, Accidental Death and Dismemberment, Dental, Major Medical, Hospital, Vision Care, and Weekly Indemnity
R.R.S.P.	Although there is no Pension Plan, the Company will contribute to an employee's R.R.S.P. in accordance with the schedule outlined on Page 81 of the Collective Agreement.
Transportation Subsidy:	The Company provides a subsidy to the bus transportation company to ensure the return fare does not exceed $3.50 per day (current level of subsidy is $2.00 per passenger per day).
Protective Clothing & Tool Allowance:	The Company provides two pairs of coveralls per year to each employee. Employees are responsible for laundering their own coveralls. In addition, an allowance of $150.00 per annum is paid to operations employees and $200.00 per annum is paid to maintenance employees.

RECOMMENDATIONS FOR NEW COLLECTIVE AGREEMENT

Term:	Three years, expiring October 31, 1996.
Lunch Breaks:	Eliminate one lunch break for 12–hour shift employees
Hot Change:	Hot change to be introduced for mine operations
Wages:	Roll back of $1.00 per hour in Year 1, and maintain present wages for remaining 2 years
Monetary Items:	No change to any other monetary items.
Productivity:	Set up joint committee to study productivity. We may have to devise some method of compensation for productivity improvements.

SUMMARY OF HEALTH & WELFARE BENEFITS

	Salaried	Hourly
Life Insurance	3 x Annual Salary	1 x Annual Salary
A.D.D.	3 x Annual Salary	1 x Annual Salary
Weekly Indemnity	67% of weekly earnings to max. $1,000 for 26 weeks	$450.00 per week for 52 weeks
L.T.D.	67% of first $2,500 monthly earnings + 50% of next $2,500 monthly earnings + 40% of remainder of monthly earnings MAX benefit $10,000 Commences on expiration of W.I. Benefit	N/A
Hospital	Covers difference between private ward allowance and semi-private room rate	Same
Medical	Provides coverage for health care beyond Yukon Health Insurance, incl. prescription drugs, physician's fees, ambulance services, etc.	Same
Eye Care	Up to $200.00 per two-year period, for employees and dependents	Same
Pension Plan	None	None
R.R.S.P.	None	Faro Only - RRSP contributions by Company per C/A

In addition to the above, there is a statutory require that all employees in Canada be covered by Unemployment Insurance, Workers' Compensation insurance, and Canada Pension Plan.

AVERAGE ANNUAL COST PER EMPLOYEE FOR ABOVE BENEFITS

	Salaried	Hourly
Group Insurance	2,500	1,700
Unemployment Insurance	1,500	1,500
Canadian Pension Plan	750	750
Workers' Compensation	1,100	1,100
TOTAL	**5,850**	5,050

R.R.S.P. costs depend on level of participation, approximately $1,000 per employee participating.

SALARY ADMINISTRATION

All salaried positions are assigned a salary grade from one through 10.

Each grade has a starting point, mid-point, and maximum salary level.

Employees are placed in the scale initially at a point reflecting the experience they are bringing to the job.

Progression through the scale is based on an annual performance appraisal.

The complete salary scale is changed periodically to reflect changing conditions (e.g. consumer price index change, external industry comparisons, etc.)

Curragh salaried employees received a general increase of 4.2% on January 1, 1991, and 4% on September 1, 1992.

AVAILABILITY OF MANPOWER

Currently the majority of workers on layoff have remained in the area. Approximately 150 have left town.

We anticipate the longer the shutdown, more and more will relocate to other areas.

The end of June (school closing) and August (prior to winter) will be crucial dates.

Some positions will be difficult to recruit.

We recruit key positions all across Canada.

June 7, 1993

H. Ted Perry