820261

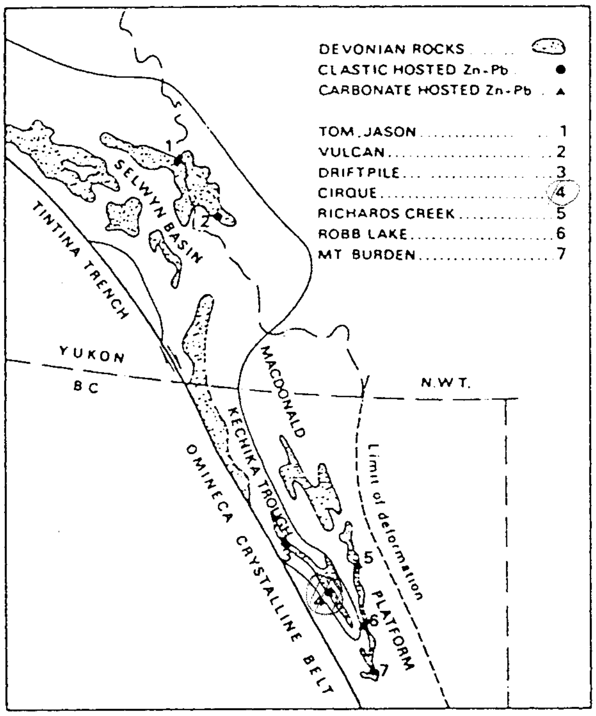
3.0 GEOLOGY

Introduction

The Cirque deposit is located within the western half of the Rocky Mountain Fold and Thrust belt of northeastern British Columbia, approximately 35 kilometres east of the Rocky Mountain trench. This part of the tectonic belt is underlain by folded and thrust faulted continental-margin sedimentary strata, within the west-trending Kechika trough (see Figure 1). The Cirque deposit is located within an area known as the Akie-River District.

The Akie-River District is underlain by sedimentary strata ranging in age from Proterozoic to early Triassic. The various formations are arranged in narrow discontinuous belts bound by northwest trending and southwest dipping thrust faults. The thrust plates are themselves completely faulted and folded, making regional correlation difficult.

The barite-lead-zinc deposits are stratiform banded to massive sulphide bodies enveloped in a sequence of late Devonian to Early Mississippian Age black shales.



Distribution of Devonian sedimentary rocks and location of major stratabound Zn-Pb deposits in northeast British Columbia and the Yukon.

FIGURE 1

(AFTER McINTYRE, 1982)

3.1 Area Geology

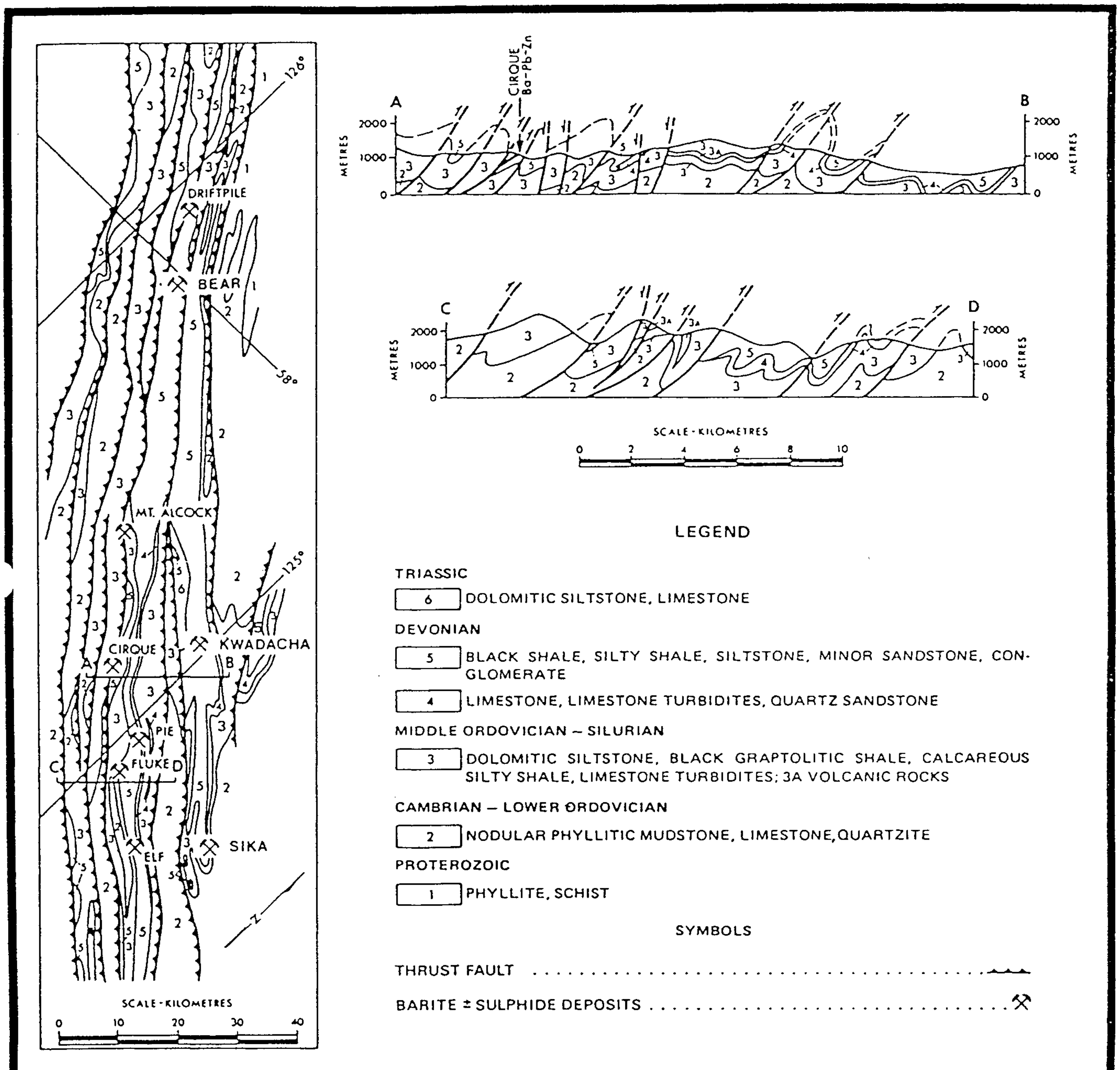
Regional Geology

The generalized geology of the Akie-River barite-zinc-lead district is shown in Figure 2. The district is underlain by basinal clastic and carbonate rocks which range in age from Early Cambrian to Triassic. The rocks outcrop in narrow northwest trending belts bounded by southwest dipping thrust faults. The more resistant formations are usually well exposed while the more recessive formations are poorly exposed. This is particularly true of the Devonian shales which host the Cirque deposit.

The oldest rocks in the district are Hadrynian to Early Cambrian and consist of structurally complex phyllites and schists. These rocks are unconformably overlain by a sequence of Early to Late Cambrian quartzite and limestone. These rocks have been well exposed at the base of several large thrust plates. The sequence is unconformably overlain by a succession of calcareous siltstone, shale, limestone, and volcanic rocks, which have been assigned to the Middle Ordovician to Upper Silurian, Road River formation. The Road River shales are unconformably overlain by a sequence of orange to brown weathering siltstone and limestone of Silurian Age.

The Silurian siltstone is conformably to disconformably overlain by Earn Group Strata of Devonian and Mississippian Age. The Earn Group (see Figure 3) consists of a series of shales, siltstones, and limestones belonging to the Akie formation, Gunsteel formation, and the Warneford formation. It is the Gunsteel formation shales which host the barite-lead-zinc deposits within the area.

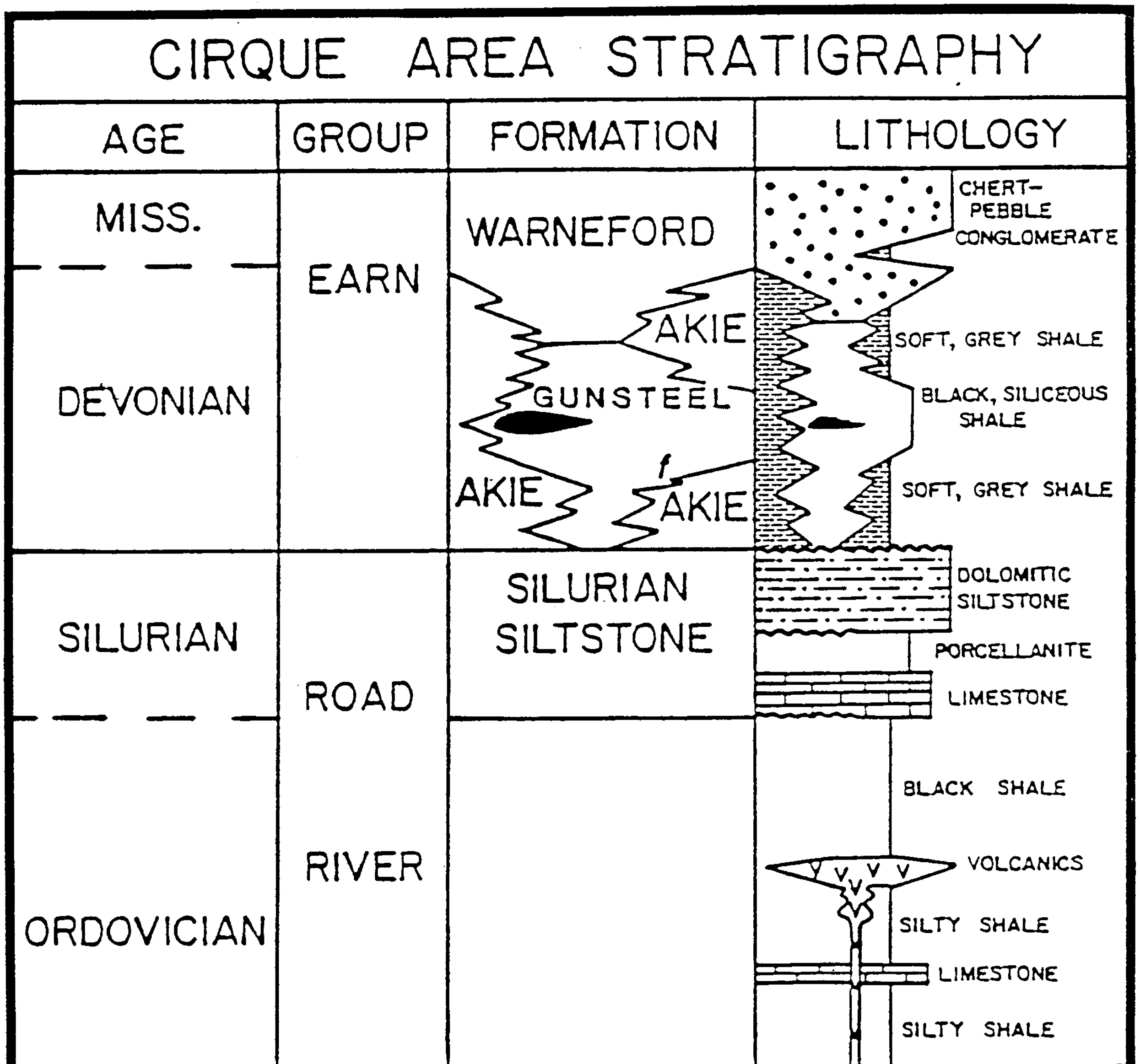
Structurally, the area is quite complex. The formations are arranged in narrow discontinuous belts bounded by northwest trending and southwest dipping thrust faults. Northeast directed compressional forces have resulted in a series of stacked thrust plates of detatched supracrustal sedimentary strata. These plates have also been folded during thrusting. In general, tight isoclinal folds with southwest dipping axial planes occur below major thrust surfaces. These folds are often offset by small scale imbricate trusts. Above the major thrust surfaces, assymetric overturned folds with northeast dipping axial planes are common. High angle normal and reverse faults are also common and appear to postdate the thrusting.



General geology of the Driftpile Creek - Akie River Ba-Zn-Pb mineral district.

FIGURE 2

(AFTER McINTYRE, 1982)



Stratigraphy, Cirque area; mineral deposit in black. Position of dated ammonoid samples shown by symbol "f".

FIGURE 3

(AFTER CYPRUS ANVIL, 1983)

Geology of the Cirque Deposit

The Cirque deposit is hosted by a series of Gunsteel shales located within a thrust plate of Earn Group strata. This thrust plate has been segmented by a series of southwest dipping imbricate thrust faults. The Earn Group strata are part of the bottom limb of an overturned syncline.

The basal part of the Devonian succession exposed on the Cirque property is the Akie formation, characterized by laminated silty shale with thin siltstone, sandstone, conglomerate, and limestone interbeds. Under the Cirque deposit, this unit is less than 50 metres thick. The overlying Gunsteel formation is more than 100 metres thick and is characterized by rhythmically bedded argillites, cherts, and carbonaceous black shales, which form the footwall of the deposit.

The Cirque sulphide body is overlain by late Devonian Gunsteel and Akie shales.

The Cirque deposit is best described as a tabular to lensoidal strataform baritic sulphide body which strikes northwest and dips from 20 - 30^{o} to the southwest. The Cirque deposit, as delineated by 42 diamond drill holes, is roughly 1000 metres long, 300 metres wide, and varies from 2 to 70 metres thick (Cyprus Anvil, 1983). The deposit is enveloped in black siliceous shales and lies along the southwest dipping limb of a northwest trending anticline. The deposit remains open both down dip and to the south along strike.

Cirque is a bedded barite deposit with a continium of facies from nearly 100 percent barite to 100 percent sulphides. The deposit contains several interbeds of shale, siliceous shale, and siliceous siltstone which can be locally correlated; however, these interbeds constitute less than 10 percent of the deposit. Contacts between the deposit and the boundary clastics is visually abrupt and depositional in nature.

Mineralization

The major minerals in decreasing order of abundance are barite, pyrite, sphalerite, and galena. Average mineral contents of the deposit can be seen in Table I.

TABLE I

Grades of mineral facies, Cirque deposit, from assay of three composite samples, from 29 drill intersections. Specific gravities (S.G.) were determined for selected typical samples.

FACIES	Pb(%)	Zn(%)	Ba(%)	Fe(%)	Ag(g/t)	S.G.
Baritic	1.3	6.2	40.7	6.2	33	4.2-4.4
Pyritic	3.3	11.1	19.4	17.9	73	4.4-4.7
Laminar Banded Pyrite	0.6	4.0	2.0	-	24	2.7-3.1

Since no silver minerals other than minor tetrahedrite have been located, it is assumed that most of the silver is present through substantial solid solution within the pyrite and sphalerite. Copper minerals have not been observed and very little carbonate has been identified (Cyprus Anvil, 1983).

Three major mineral facies have been recognized with the deposit. (See Table I.) The baritic and pyritic facies constitute the majority of the deposit while the laminar banded pyrite occurs as a dominantly fringe facies within the siliceous shale. Only the baritic and pyritic facies have been included within the orebody calculations.

The baritic facies consists of pale grey to white, diffusely laminated, fine to medium grained barite with less than 40 percent sulphides. Sulphides occur as discontinuous 1-5 mm thick, wavy, lenticular laminations of pyrite, sphalerite, and minor galena. Sulphide laminae are concentrations of framboidal pyrite in a matrix of interlocking barite and sphalerite grains. Framboids range from single grains to 25 micron clusters. Galena is remobilized and commonly occurs at grain boundaries and as fracture fillings in barite. The spatial associations of sphalerite with pyrite and galena with barite are consistent throughout the baritic facies.

The pyritic facies is distinguished from the baritic facies by its greater sulphide content (40%) and higher lead-zinc-silver grades (Table I). The pyritic facies ranges from diffusely interbedded sulphides and barite to nearly 100 percent sulphides. Mineralogy consists of pyrite, barite, sphalerite, and galena. Pyrite occurs as framboidal clusters with subhedral pyrite overgrowths. Fractures in the large pyrite grains are filled by sphalerite and galena. Irregular colloform aggregates of pyrite (20-50 microns in diameter) with galena and sphalterite interlayers and subspherical atoll structures with

Mineralization/Cont'd

pyrite-rimmed cores of galena or sphalerite are also seen. Sphalerite occurs as interlocking grains with pyrite and as fine-grained laminations within the pyrite beds. Cross-cutting sharp-edged veins to diffuse pods of coarsely crystalline barite with patches of coarsely crystalline galena are restricted to the pyritic facies.

In plan, the pyritic facies predominates in the north end of the deposit, and baritic facies predominates in the south. In cross section, the position of the pyritic facies trends from the top of the deposit in the west to the bottom in the east. The axis of the thickest barite and sulphides trends northerly and is near the western margin of the deposit. Thickness decreases more rapidly to the west than to the east. The axis of highest grade material lies just east of the thickest portion of the deposit and is skewed to the northwest. The axes of best grade and greatest thickness coincide in the northern end of the deposit where grades are uniformly high. The deposit contains a zinc-rich western margin with increasing lead content to the east. The highest grades and thickest portions of the deposit have a zinc/lead ratio of between 3 and 4. The highest silver grades occur in the northern end of the deposit where the pyritic facies predominates.

Genetic Model

The barite sulphide deposits of the Akie-River District apparently formed in a northwest trending trough which was bounded by carbonate reefs. It has been proposed by several authors that these deposits formed from metalliferous brines in strongly reduced isolated basins within this environment.

If the deposits did form by the exhalation of heated metalliferous brines into euxinic reduced basins, as implied by the model proposed by Finbow-Bates (1980), Sato (1972), and Robert et al. (1980), then the apparent westward increase in pyrite content suggests that exhalative vents were located in this direction. To date, no stockwork feeder zones of this type have been found; however, the extensive deep-seated faulting would have destroyed or offset much of the evidence. The nature of formation suggests that the mineralized horizon does offer significant potential for further deposits of a similar nature to the Cirque deposit. The existence of the South Cirque deposit within this horizon, although displaying facies differences, indicates that the horizon may indeed host further undetected mineralized zones along its length.

3.2 Ore Reserves

Two distinct ore reserve figures have been proposed for the Cirque deposit. Calculations have been completed by Cyprus Anvil staff; one based on a 1983 unfaulted geologic interpretation and one based on a 1981 faulted interpretation. Each of these reserve figures has been calculated, by block, on diamond drill cross sections. A total of 46 diamond drill holes totalling 19,000 metres have been drilled and of these, 29 holes cut mineralized sections.

CIRQUE DEPOSIT RESERVES

DRILL INDICATED RESERVES	TONNES	Pb%	Zn%	Pb+Zn%	Ag g/t
1981 Faulted Interpretat	38,500,000	2.2	7.97	10.17	47.2
1983 Unfaulted Interpretat	32,170,556	2.15	7.88	10.03	47.7

The reserves presented above are drill-indicated reserves, calculated using a minimum block width of 2 metres and setting the block limits to coincide with the inferred geological boundary between the mineral horizon and the enclosing shale.

Tonnages were calculated using blocks with vertical sides and rhombohedral plans centered on section lines. The sides of the rhombohedrons were constructed parallel to inferred isopach. All block areas were measured using a planimeter, and tonnages were calculated using the half distance to the adjacent section and the average specific gravity for the appropriate ore facies.

In the 1981 calculations, the abrupt thickness changes were explained using the substantial faulting pattern, mapped from surface exposures. In the 1983 calculations, these thickness changes were assumed to be due to depositional variation, and dip changes were assumed to be due to folding. Each of these interpretations is limited by the amount of information available, and Cyprus Anvil staff indicated that these interpretations represent a rather simplistic view of the Cirque deposit, which can only be improved upon with further exploration.

It is suggested that the final reserve figures will be calculated by incorporating both of these geologic interpretations. The genetic nature of the deposit suggests deposition variations; however, the complex regional structure also indicates several generations of fault-cutting mineralized zones.

3.2 Ore Reserves/Cont'd

A significant potential does exist to expand the reserves of the Cirque deposit. The limited drilling has tested only a small area of the horizon which hosts the Cirque deposit. Further potential exists both along strike to the south and down dip of the outlined deposit.

The existence of the South Cirque deposit indicates that the mineralized horizon does indeed hold promise for futher mineralized areas. The area between the Cirque deposit and the South Cirque mineralized body holds significant exploration potential and requires further drilling.

Mining Reserves

Mining reserves are based on geological drill indicated reserves, calculated by Cyprus Anvil.

Drill indicated reserves of 38,500,000 tonnes, containing 10.17% lead and zinc and 47 g/t silver were used.

Mining reserves were calculated as follows:

9,670,000 tonnes	containing	14.54% Pb + Zn at 12% cut off, or
12,810,000 tonnes		13.60% Pb + Zn at 10% cut off, or
– 22,170,000 tonnes		11.99% Pb + Zn at 8% cut off

Silver content was calculated at 57 g/tonne.

For the mining plan, the cut off grade of 8% Pb + Zn was considered, resulting in mining reserves of

– 22,170,000 tonnes	2.74% Pb
	9.25% Zn
	57 g/t Ag

The mathematical balance of 16,330,000 tonnes containing 8.17% Pb + Zn, includes fractions of higher grade ore by the hangingwall and footwall, considered not mineable at the present time.

Underground exploration work (excavation and drilling) is required to confirm the indicated reserves.

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

REGIONAL GEOLOGIC SETTING

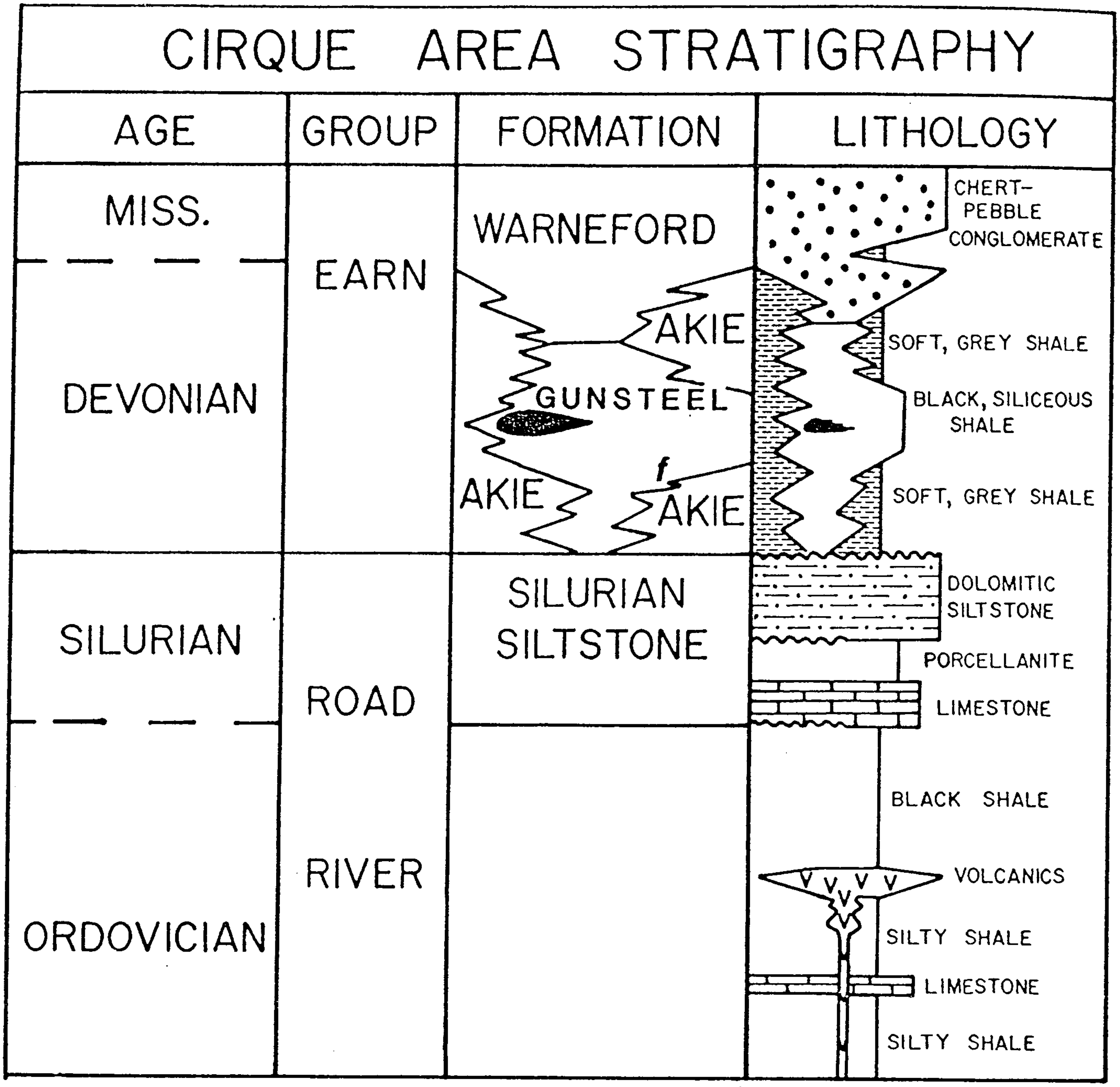
The Cirque deposit is located in the western portion of the Northern Rocky Mountain Fold and Thrust Belt (NRMFTB) (figure 2b). The NRMFTB is characterised by late Precambrian through early Mesozoic miogeoclinal strata. In the eastern part of the NRMFTB limestone, dolomite and orthoquartzite of a shallow water, platformal aspect dominate. In that part of the belt rugged bare rock ridges predominate. The ore deposit is located in the western portion of the NRMFTB, where the coeval strata are characterised by shale, chert and siltstone of deeper water, basinal aspect. In the western area folds are tighter and folding is more pervasive on all scales. The generally recessive strata of the western part of the belt form more rounded ridges with steep colluvial slopes covered by dense forest and alpine tundra. Strata of the NRMFTB are generally un-metamorphosed however in the western portion the products of the lowest grades of regional metamorphism are present. The area did not experience significant igneous intrusive activity. Strata of the belt are thrown into northeasterly verging folds and cut by northeast directed thrust faults (figure 2a). Post fold and thrust fault deformation is dominated by extensional faulting related to relaxation of the compressional strain which formed the fold and thrust belt. Approximately 10 km southwest of the ore deposit is the enigmatic Northern Rocky Mountain Trench. The Trench is thought to be the locus of significant strike slip faulting and in the region represents the boundary of the NRMFTB with the Omenica Crystalline Belt.

PROPERTY GEOLOGY

The Cirque claims are underlain by an Ordovician through Mississippian sequence (figure 3) exposed in three major thrust fault bounded panels (figure 4). For the purposes of discussion of the orebody and it's vicinity only two units of this sequence, the Devonian and Mississippian **Earn Group** and the Ordovician and Silurian **Road River Group,** are important. The structure and stratigraphy of the property and vicinity are well described in Pigage (1986). That paper provides information on the units not described below since they are more remote from the deposit as well as further detail on the units described herein.

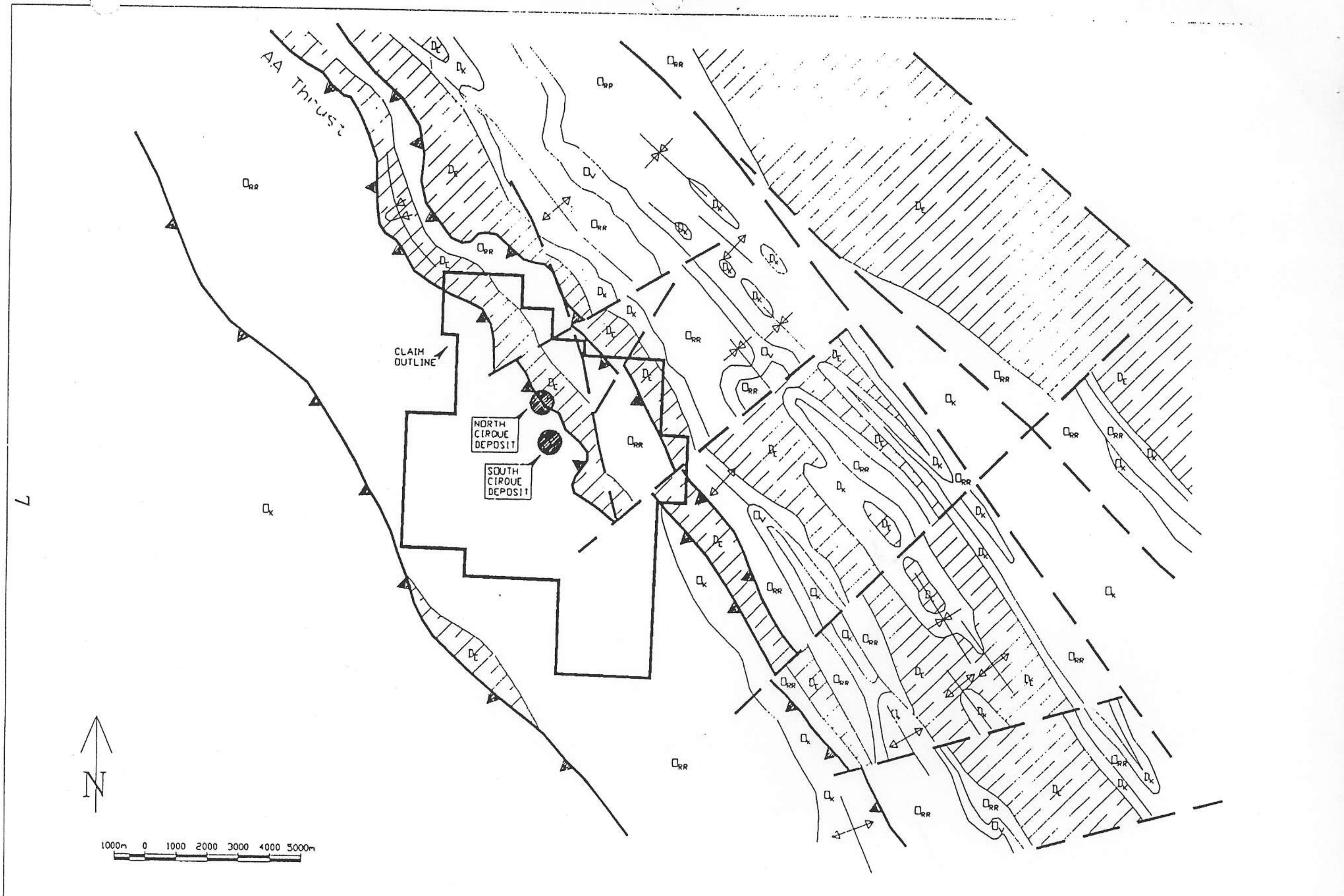
The Earn Group has been divided into several informally named formations in the area. The **Gunsteel formation**, which hosts the mineralization, is characterised by blue grey weathering, non-calcareous, black, siliceous shale and black banded porcellanite or chert. The member of the Gunsteel formation which is the immediate host of the ore body is termed the Pregnant shale. This is a non-calcareous, hard, siliceous shale; it contains local framboidal pyrite bands and bedded barite. Its distribution is well delineated by anomalous lead and zinc in soil geochemical surveys (figure 5). The Gunsteel formation in the area of the ore deposit is overlain and underlain by grey, soft, non-calcareous shale of the **Akie formation**.

On the claims Earn Group strata are overthrust by Ordovician and Silurian strata of the Road River Group. These strata are well represented in the upper parts of the access decline at Cirque and crop out widely on the ridges surrounding the upper Cache Creek basin. The Road River Group has been divided into several informal, unnamed formations including: Ordovician black



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.	O_V Ospika volcanics basalt
D_K Kwadacho reef limestone	O_K Kechika Group calc. shale
O_{RR} Road [illegible]er Group calc. [illegible] shale, limestone, chert, siltstone	

REVISIONS

Curragh Resources Inc.

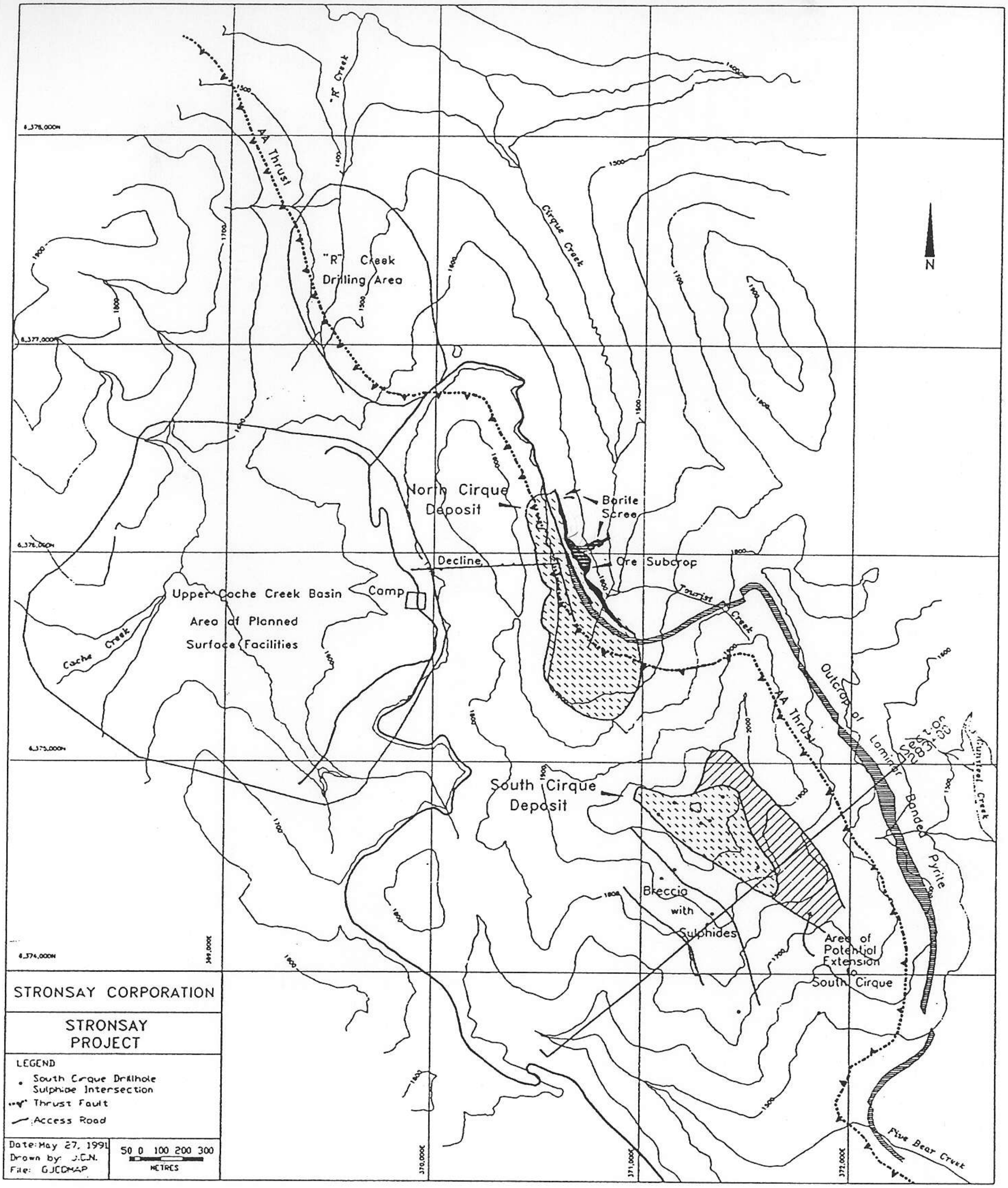
Figure: 4

CIRQUE PROJECT

Date 31/12/1990 Geologist L Pigage

Geology of Cirque Claims

Drawing No:



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

calcareous shale, Ordovician or Silurian limestone and chert and Ordovician and Silurian tan weathering, dolomitic siltstone. The siltstone is quite thick and is one of the more resistant units of the area, supporting most of the ridges.

The predominant compressional structure on the claims is the AA thrust (figure 5, 6). This northwest trending, southwest dipping thrust fault places southwesterly dipping Road River Group strata on southwest dipping Earn Group strata. In the hanging wall of the AA thrust the Silurian siltstone outlines several significant northeast verging folds. More steeply dipping reverse faults splay off the AA thrust into its hanging wall and shear out the limbs of these folds. A slaty cleavage is axial planar to these folds and sub-parallel to the reverse faults. Surface mapping has revealed several extensional faults. These faults typically trend north-south to northwest-southeast and dip westerly with the west side down thrown. Many of the extensional faults are listric, rooting in the older thrust fault planes. Also common in the area are east - northeast trending steeply dipping faults which appear to have strike slip displacements.

DEPOSIT GEOLOGY

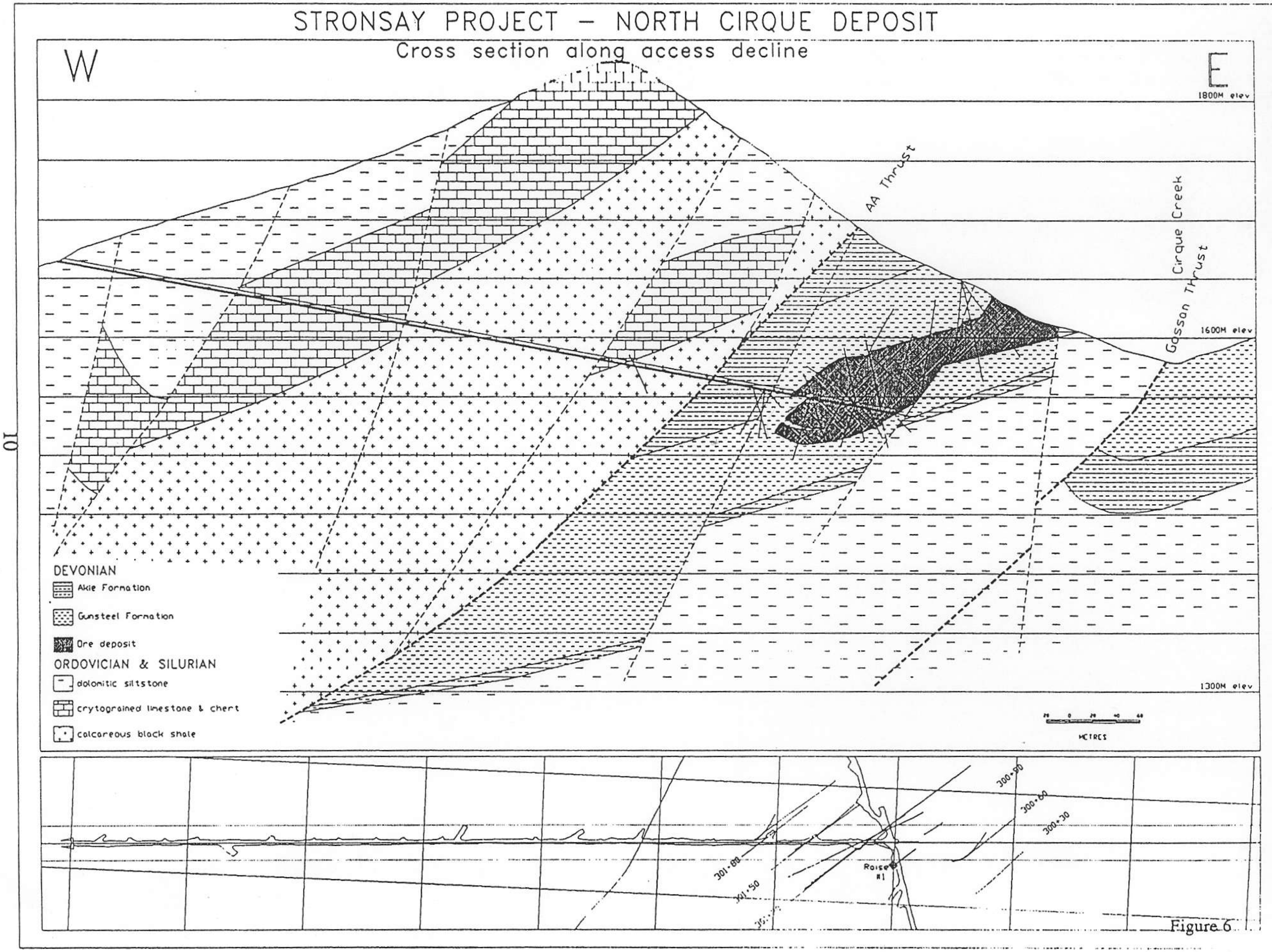
The Cirque deposit is a stratiform syn-sedimentary massive sulphide-barite deposit hosted by black, siliceous, Devonian shale and chert of the Earn Group. The deposit is a tabular body with the overall shape of an elongate asymmetric lens approximately parallel to layering in the host sedimentary strata. The lens is wedge shaped in cross section. The southwest termination of the wedge is rounded on most sections. The thin edge of the wedge tapers up-dip toward the erosion surface. The deposit thus strikes northwest and dips south west approximately 30°. In plan view the ore lens is elongate in the north south direction; its southward plunge is due to the approximately 45° rake of the lens in the bedding plane (figure 7).

Measured in the horizonal plane the long dimension of the lens is 1,100 m. (N-S) and the lens is 400 m. across (E-W); maximum thickness of the lens is 65 to 75 m. (on sections 299+10N and 300+90N); the outer limit is arbitrarily placed at a minimum thickness is 2 m. in the southern part of the deposit (figure 7). The structure of the deposit as revealed by the results of the advanced exploration programme is discussed in more detail below.

There are three major lithologies that comprise the North Cirque orebody. These lithologies not only are logging units in individual drill holes but also show good continuity along sections and form units that appear to be traceable throughout the investigated length of the deposit. The lithologies may be separable to some extent by selective mining and are of significance from a metallurgical point of view as they vary in performance. There are also significant grade differences between the three lithologies.

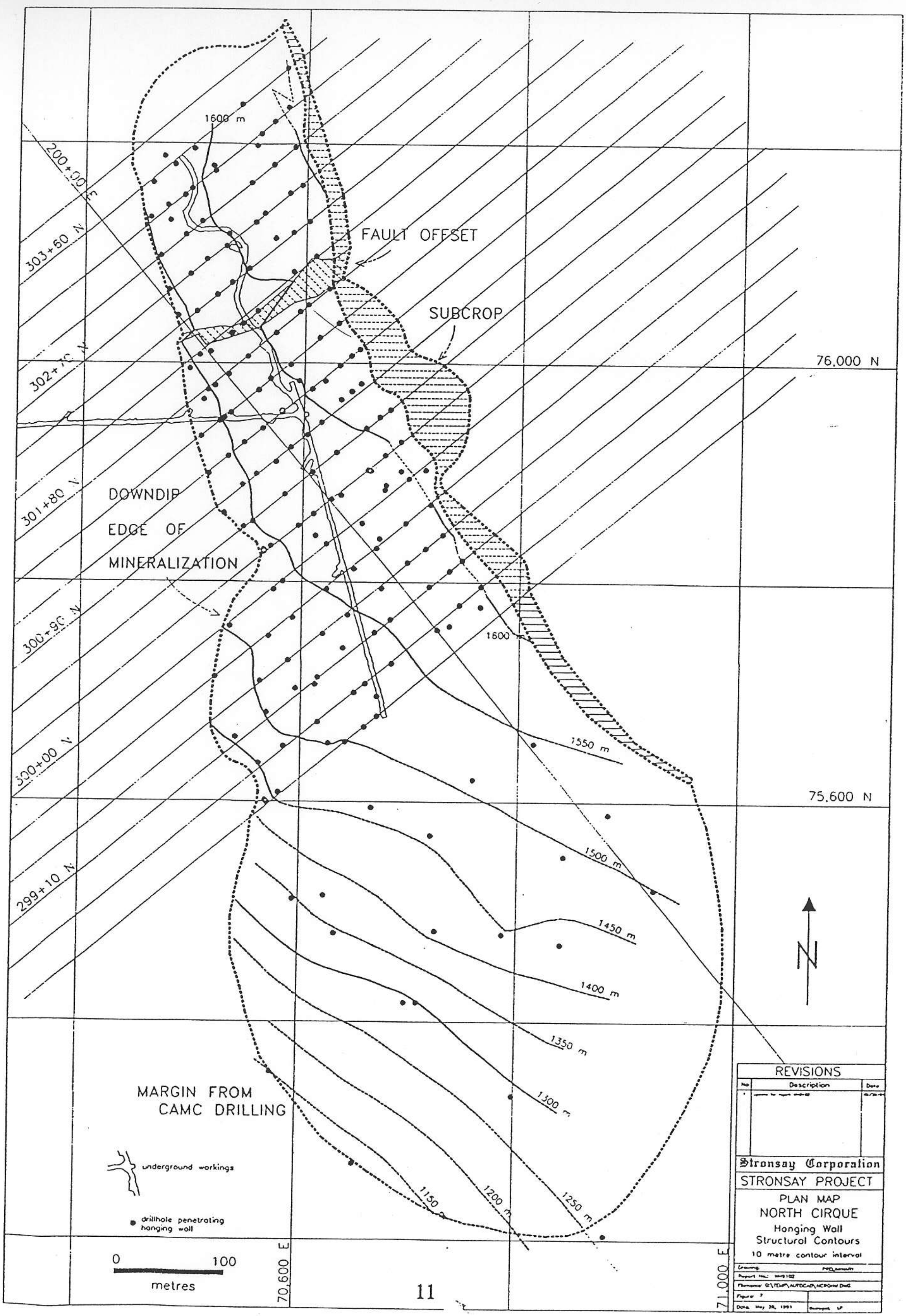
Lithology 1 -Barite with sulphides:

This material consists of medium grey fine to medium grained barite containing fine disseminated pyrite, sphalerite and galena which is locally in well developed bands up to 3 cm. thick. This lithology contains a strong foliated or gneissic



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
dolonitic siltstone
crytograined limestone & chert
calcareous black shale
METRES
300+90
300+60
300+30
301+80
301+50
Raise
#1

Figure 6



1600 m
200+00 E
303+60 N
FAULT OFFSET
SUBCROP
76,000 N
301+80 N
DOWNDIP
EDGE OF
MINERALIZATION
300+90 N
1600 m
300+00 N
1550 m
75,600 N
1500 m
299+10 N
1450 m
1400 m
1350 m
1300 m
MARGIN FROM
CAMC DRILLING
underground workings
drillhole penetrating hanging wall
1150
1200 m
1250 m
0
100
metres
70,600 E
71,000 E
N
REVISIONS
No
Description
Date
Stronsay Corporation
STRONSAY PROJECT
PLAN MAP
NORTH CIRQUE
Hanging Wall
Structural Contours
10 metre contour interval

texture. The defining characteristic of this lithology is that it contain greater than 60% barite. This material is normally low in contained lead-zinc; the arithmetic mean of 2079 samples of this material is 7.37 % Pb+Zn. The most important traceable unit composed of this material occurs in the southwest part of the ore lens. It appears to be stratigraphically the lowest unit of the lens and occurs to some extent from section 298+50N in the south part of the deposit to 302+70N in the north. This unit is discussed in more detail below. The lithology is also found with lesser continuity near the subcrop of the ore lens locally and is interlayered with the upper unit of lithology 4 (see below). Lithology 1 has also been referred to as the "barite facies" in the text of chapter 2 of the Stage I EIS, "low grade baritic ore" in figures 2-3 and 2-4 and "DB(BS)" or "sample group 12" on table 5-8 and in the text of Chapter 5 of that report. Lithology 1 is the baritic ore mentioned in the metallurgical section of the Stage I EIS.

Lithology 4 - Sulphides with barite:

This material is defined as containing 20% to 60% barite visually. It is transitional between lithologies 1 and 5 and grades imperceptibly into those types. The material shows characteristics of both lithologies 1 and 5 depending on how much barite is present and commonly is simply an interbanding of the two lithologies on a scale too fine to log separately. Two stratigraphic units are recognized. The lesser of these units is at the transition between the footwall baritic unit described above and the overlying massive sulphides where the unit is texturally like lithology 1 but contains a greater sulphide content. The other unit is thicker and overlies the lithology 5 high grade zone (see below), there it resembles lithology 5 but contains increased interstitial and patchy or blebby coarse white barite. Lithology 4 material is transitional in lead-zinc content as well as its other characteristics; the arithmetic mean of 1735 samples is 10.72 % Pb+Zn. This lithology is included in the "pyritic facies" of the text of Chapter 2 of the Stage I EIS: is referred to as "mixed baritic and pyritic ore" in figures 2-3 and 2-4 and as "DB(SB)" or "sample group 14" on table 5-8 and in the text of Chapter 5 of that report. Lithology 4 and 5 (see below) collectively form the pyritic ore of the metallurgical section of the Stage I EIS.

Lithology 5 - Massive Sulphides:

The massive sulphides consist of very fine to medium grained brassy yellow pyrite with bands of laminated fine pyrite and fine tan sphalerite. The sphalerite and galena are typically very difficult to distinguish in hand specimen until one's eye becomes accustomed. Barite occurs as coarse irregular white patches or to a lesser extent as medium crystalline, grey interstitial grains. The barite content of this lithology is by definition less than 10 to 15% visually. The massive sulphides form a traceable unit of great importance due to its very high grade. This unit occurs above and to the west and down dip of the north and south headings. The arithmetic average % Pb+Zn of 2452 samples of this lithology is 16.85. Lithology 5 comprises the remainder of the "pyritic facies" of the text of Chapter 2 of the Stage I EIS. Lithology 5 is also referred to as "high grade pyritic sulphides" on figures 2-3 and 2-4 and is unit "DB(MS)" or "sample group 13" on table 5-8 and the text of Chapter 5 of the EIS.

Table II summarizes the metal content of individual drill core samples and composited samples of each of these materials, figures 8 through 25 show histograms of various metals for each lithology.

There are a few other lithologies of very local importance in the ore lens. A minor sulphide type is lithology 6, a massive sulphide characterised by fine framboidal pyrite laminae. This lithology is generally included in one of the other material types for purposes of reserve blocks. There are many isolated occurrences of lithology 3 logged, this rock type is a coarse white barite vein with a characteristic comb texture and locally important coarse galena and or sphalerite. Spectacular examples of isoclinaly folded veins of lithology 3 occur in the access decline. This material occurs throughout the ore lens but is volumetrically minor and has little continuity thus it is combined with its host sulphide - barite lithology for purposes of reserve blocks. Lithology 3 is of historic importance in that is appears to be the source of much of the barite scree at the discovery showing in Cirque Creek (figure 26). Neither of these rock types was mentioned in the Stage I report however, the "barite veins" of figures 2-3 and 2-4 of that report are part of lithology 3.

Outside the ore lens are siliceous shales with laminae of dark brassy yellow framboidal pyrite. This rock type is termed "laminar banded pyrite" (lithology 50 on cross sections), it is locally quite rich in zinc but is nowhere broken out as an ore lithology. The laminar banded pyrite may be a distal facies of the ore lens although it presently tends to occur dominantly in the hanging wall shales near the subcrop of the orebody. In the Stage I EIS reference was made to this material in Chapter 2 where it was called the "laminar banded pyrite facies" and "DG(LB)" or "sample group 15" in table 5-8 and the text of Chapter 5.

Additional rock types occurring within the ore lens and worthy of note are thin black siltstone, siliceous shale or chert layers which range in thickness from centimetres to ten meters. Some of the thicker units form markers which can be traced through the south part of the area studied.

PREVIOUS EXPLORATION WORK

From 1978 through 1982 gridded geochemical soil sampling, detailed geological mapping, and exploration surface diamond drilling programs were conducted on the Cirque claims.

Surface diamond drilling conducted during that time totals 48,092.9 m. in 113 diamond drill holes. Drilling on the property has been in three main areas (figure 5):

TABLE II

STRONSAY PROJECT
NORTH CIRQUE DEPOSIT
UNIVARIATE STATISTICS

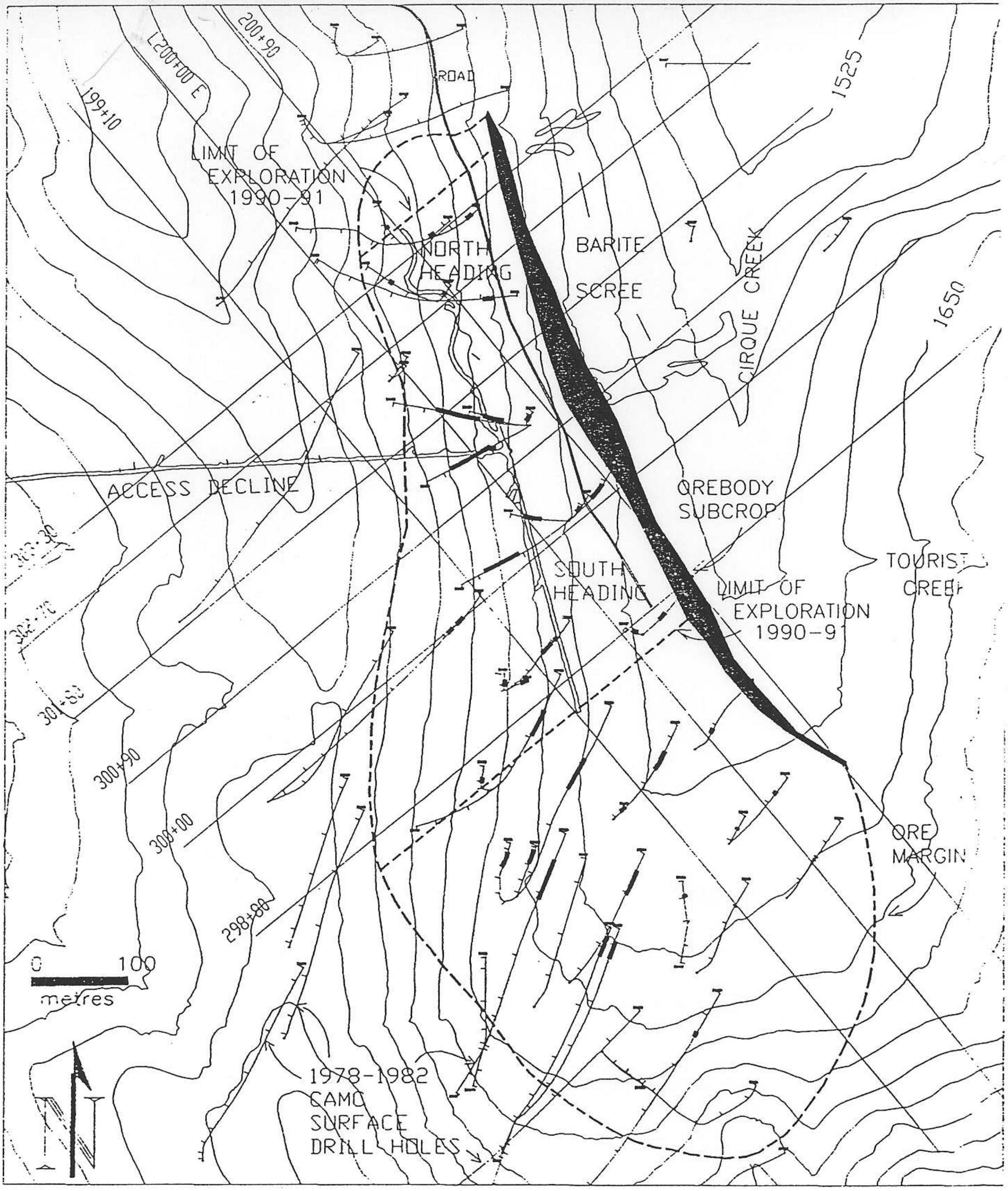
INDIVIDUAL ASSAYS

	LITHOLOGY 1 Barite Sulphide							LITHOLOGY 4 Sulphide Barite							LITHOLOGY 5 Massive Sulphide						
	Pb+Zn%	Pb%	Zn%	Ag g/t	Fe Total%	Ba%	SG-Pulp gm/cc	Pb+Zn%	Pb%	Zn%	Ag g/t	Fe Total%	Ba%	SG-Pulp gm/cc	Pb+Zn%	Pb%	Zn%	Ag g/t	Fe Total%	Ba%	SG-Pulp gm/cc
Total Population	2042	2042	2042	2042	1721	2008	1721	1702	1702	1702	1702	1529	1679	1529	2427	2427	2427	2427	2223	2416	2223
Minimum Value	0.56	0.01	0.13	6.4	0.27	2.82	3.13	0.44	0.02	0.21	5.8	0.47	0.45	3.03	0.35	0.01	0.30	4.5	1.57	0.08	2.88
Maximum Value	29.11	28.50	17.95	430	29.00	57.75	4.98	33.76	13.40	28.38	147.2	60.3	56.10	5.61	47.82	36.20	30.06	230.0	34.18	55.40	5.69
5th Percentile	4.06	0.01	3.27	6.4	0.33	22.95	3.88	5.21	0.02	3.70	19.8	1.87	9.95	3.91	10.26	0.01	7.17	46.3	10.31	1.00	3.88
95th Percentile	11.17	3.40	8.92	48.9	7.98	50.51	4.47	17.18	4.81	13.56	84.9	17.57	43.64	4.56	26.25	7.30	20.39	131.1	28.45	29.77	4.67
Arithmetic Mean	7.39	1.51	5.88	29.1	3.39	37.90	4.27	10.75	2.43	8.31	49.1	9.33	28.16	4.34	16.84	3.60	13.24	80.8	20.62	11.71	4.41
Standard Deviation	2.30	1.43	1.84	15.8	2.63	8.43	0.20	3.53	1.43	2.97	19.7	4.98	9.91	0.22	5.03	2.19	4.02	25.8	5.47	9.10	0.25

5 METRE ASSAY COMPOSITES

	LITHOLOGY 1 Barite Sulphide							LITHOLOGY 4 Sulphide Barite							LITHOLOGY 5 Massive Sulphide						
	Pb+Zn%	Pb%	Zn%	Ag g/t	Fe Total%	Ba%	SG-Pulp gm/cc	Pb+Zn%	Pb%	Zn%	Ag g/t	Fe Total%	Ba%	SG-Pulp gm/cc	Pb+Zn%	Pb%	Zn%	Ag g/t	Fe Total%	Ba%	SG-Pulp gm/cc
Total Population	649	649	649	649	522	628	522	516	516	516	516	451	508	45	723	723	723	723	651	719	651
Minimum Value	0.00	0.00	0.00	0.0	0.55	8.63	3.39	2.92	0.05	1.74	12.7	0.96	0.44	3.10	4.07	0.13	3.18	28.0	6.30	0.15	3.31
Maximum Value	18.71	6.22	16.60	119.4	17.73	54.43	4.63	20.80	7.39	16.59	101.7	21.49	53.43	4.66	29.61	8.85	23.23	197.2	32.25	45.12	4.72
5th Percentile	4.58	0.17	3.80	17.6	1.30	20.91	3.75	6.55	0.24	4.73	23.1	3.56	13.40	3.78	5.58	0.31	7.52	42.4	9.50	1.49	3.75
95th Percentile	11.46	3.21	8.81	52.8	9.65	48.94	4.44	15.38	4.18	12.26	77.0	16.29	43.18	4.53	23.32	6.24	18.45	112.0	27.46	28.35	4.65
Arithmetic Mean	7.40	1.36	6.04	30.4	3.99	35.90	4.21	10.52	2.33	8.19	48.2	9.37	27.05	4.30	15.70	3.45	12.25	74.8	18.79	13.32	4.35
Standard Deviation	2.23	1.11	1.76	12.9	2.93	8.81	0.21	2.82	1.13	2.28	16.0	4.07	8.95	0.23	4.31	1.48	3.45	22.4	5.24	8.54	0.27

includes all Cyrpus Anvil Mining Corporation and Curragh Resources Inc. diamond drill holes for North Stronsay deposit.



Vertical projection of surface holes drilled by CAMC from 1978 to 1982. The deposit outline is the 2m isopach as defined by CAMC. The portion of the deposit examined by Stronsay is indicated as is the location of Stronsay's underground workings and pad. The cross and longitudinal sections are shown on Figures 27 through 35.

Figure 26

AREA	# OF DRILL HOLES	METRES
R. Creek	11	3,453.1
North Cirque	74	23,388.5
South Cirque	28	21,251.3
	113	48092.9

The North Cirque deposit was delineated by 42 diamond drill hole intersections. Collar locations of the drill holes were carefully surveyed and downhole deviations were measured at regular intervals down the holes using a Sperry-Sun single shot camera instrument. This provided an excellent and reliable information base which has stood up well to close examination. Core for the mineralized drill intersections is stored at B.C. Government facilities at Charlie Lake near Fort St. John. All unmineralized drill core is stored in racks constructed on the Cirque claims. Surplus assay sample material for analyzed intersections is still stored at Kamloops Research and Assay Laboratory in Kamloops, British Columbia.

Eleven geological cross-sections, irregularly spaced from 50 to 130 m. apart were completed (figure 49). Of the 42 Cyprus Anvil drill holes, 27 are located within that part of the deposit, sections 298+50N to 303+60N, within which Stronsay carried out surface and underground exploration in 1991-1991 (figure 26).

1989-1991 ADVANCED EXPLORATION PROGRAMME

An underground programme was carried out at the Cirque property in two phases, first from August 1989 to February 1990 and second from August 1990 to February 1991. The underground excavation included a 681.8 m. access decline (3.5 m. x 4.0 m.) driven mainly in the first phase and two headings (also 3.5 m. x 4.0 m.) driven along the footwall of the mineralization during the second phase (figure 26). The two headings were driven along a 580 m. length of the ore lens. The thick portion of the lens extends at least 900 m. in the north south direction and sulphides are known over a 1,100 m. length. The north heading is an incline driven at +15 to +20 percent 298.4 m. from the bottom of the access decline. The south heading is a -10 to -20 percent decline driven 296.5 m. to the south from the bottom on the access decline. Two raises were excavated to collect bulk samples representative of the early ore feed. These 1.3 m. by 1.3 m. raises were put in on sections 300+90N and 302+70N at +50° to the southwest to intersect the complete thickness of the ore lens.

At the close of the first phase, four short holes were drilled from the end of the decline to locate the footwall of the ore lens.

Drilling was carried out both from surface and underground during the advanced exploration programme. Most of the drilling was completed during the second phase of the programme. During September through November of 1990, 3312 m. in 45 holes were drilled from a road built into the Cirque Creek valley approximately 50 m. above the ore subcrop (figure 26). This

Surf 3312 in 45
Ug 9738 in 170
13050 215

position allowed definition of the up dip edge of the ore lens better than would have been possible from underground.

From October 1990 to February 1991, 170 holes totalling 9738 m. were drilled from the two headings (from the north heading for sections 301+20N to 303+60N and from the south heading for section 300+00N to 298+50N) and the access decline (sections 301+20N to 301+80N). Both surface and underground drilling was carried out on 30 m. spaced cross sections oriented perpendicular to the strike of the mineralized lens, though at an angle to its elongation. The location and line of the cross sections was laid out in the field and underground with careful survey control.

13044 99.

4 holes N.L 99 m. In total there has been 12,950 m. drilled in 211 holes during the advanced exploration programme. Additionally there were 27 holes put down by Cyprus Anvil from 1978 to 1982 in the area of the 1990-1991 work (figure 26).

All surface holes recovered NQ size diamond drill core. Country rock shales were typically very broken and poor recovery was common. With the exception of the holes close to the ore subcrop (angle holes drilled the northeast) the ore was moderately broken but recovery was very good. Average recovery in ore was 94%.

All underground holes were also diamond core holes but recovered BQ size core. Typically core recovery was excellent in mineralization (averaging 96%) and acceptable in host shales.

Underground drill hole collars were located by tying into survey layout points and by later survey pickup but surface holes were only tied into nearby (few metres) markers surveyed in when the section lines were laid out along the drill road. Down hole deviation was determined for all holes more than approximately 30 m. long by Sperry-Sun single shot survey.

All holes were logged for lithology, structure and faults; assay sample intervals were laid out to correspond as closely as possible to the logged intervals. In thick units a 1.0 to 1.5 m. interval was used. All core was photographed and a log of geotechnical characteristics including RQD, degree of breakage, core recovery and joint/fracture frequency was compiled. All logging data has been entered to a PCXPLOR computer data base.

Mineralized core was split longitudinally by hammer and chisel type core splitter, half was stored in covered core racks on site and half was shipped to the North Vancouver laboratory of Mineral Environment Laboratories where it was assayed for lead, zinc, silver, iron and barium. Pulp S.G. was also determined on a regular basis. Min-En's description of their procedures follows:

SAMPLE PREPARATION

"Samples are dried @ 95°C and when dry are crushed on a jaw crusher. The -1/4 inch

output of the jaw crusher is put through a secondary roll crusher to reduce it to - 1/8 inch. The whole sample is then riffled on a Jones Riffle down to a statistically representative 400-500 gram sub-sample (in accordance with Gy's statistical rules). This sub-sample is then pulverized in a ring pulverizer to 95% minus 120 mesh, rolled and bagged for analysis. The remaining reject from the Jones riffle bagged and stored."

LEAD, ZINC, SILVER, IRON

"A 2.000 gram sub-sample is weighed from the pulp bag for analysis. Each batch of 70 assays has a natural standard and a reagent blank included. The assays are digested using a HNO_3 - $KClO_4$ mixture and when reaction subsides, HCl is added to assay before it is placed on a hot plate to digest. After digestion is complete the assays are cooled, diluted to volume and mixed.

The assays are analyzed on atomic absorption spectrophotometers using the appropriate standard sets. The natural standard digested along with this set must be within 3 standard deviations of its known or the whole set is re-assayed. If any of the assays are >1% they are re-assayed at a lower weight."

BARIUM

"A representative sub-sample is weighed from the pulp bag for analysis. Each batch of 24 assays has a natural standard and a reagent blank included. The assays are fused using lithium metaborate in a graphite crucible @ 100°C to obtain a melt. The melt is poured into dilute nitric acid and stirred until dissolved, then diluted to volume in a volumetric flask and mixed.

The assays are analyzed on a inductively coupled plasma (ICP) using the appropriate analytical control table (ACT). This ACT contains all interelement corrections. The natural standard fused along with this set must be within 3 standard deviations of its known or the whole set is re-assayed."

SG PROCEDURE

"A 200 ml flask is filled with boiled deionized water (@ 20°C) and weighed. Some of the water is poured off and a 50.000 gram sub-sample is weighed and added to the flask. The flask is boiled, cooled and filled to volume and weighed again.

The specific gravity is calculated and reported in grams per cubic centimetre."

A programme of check assaying has been undertaken however the programme is not yet complete and the results are not yet fully available. The first phase of this programme involved re-preparation of the assay pulp for 5% of the samples and re-assay by Min-En. In the second phase pulps will also be re-analyzed at another laboratory. Preliminary results of the first phase show very good reproducibility for lead, zinc, silver and iron with correlation coefficients of 0.945 or better and regression line slopes close to one. Barium and Pulp SG results show more scatter with correlation coefficients of 0.89 and 0.88 respectively. The results show that the assays have good precision as have previous checks on this laboratory for other projects. Accuracy checks are not complete yet but previous checks do not indicate cause for concern.

GEOLOGICAL INTERPRETATION

Geological sections were plotted and interpreted at 30 m. intervals along the portion of the deposit drilled during the advanced exploration programme. Drill holes were projected orthogonally onto the sections using trajectories calculated by PCXPLOR software. Drill holes were projected from 15 m. on either side on the section. The geology was first interpreted on cross section and then transferred to longitudinal section where some changes were required to rationalise section to section differences. The longitudinal sections helped to clarify the faults considerably however there was not sufficient time to grapple with every problem, this limitation notwithstanding the interpretation should adequately constrain the volume of the deposit and most significant (from the perspective of offsetting the deposit outline) faults appear to have been detected. Several level plans were also interpreted from the cross and longitudinal section interpretations to further tie the fault interpretation together. Additionally a plan in the approximate plane of the underground workings was complied and interpreted. This plan helped considerably to clarify the geology, particularly faults, mapped underground and it's relation to the sections.

The resulting geological interpretation is illustrated by a number of typical cross sections (figures 27 to 32), longitudinal sections (figure 33 to 35) and level plans (figure 36 and 37) in the following pages and 1:250 geological plus reserve block sections in pockets at the near of the report in Appendix A (figures A01 to A24).

The deposit is a southwest dipping lens concordant with large scale lithologic layering in the enclosing shales. The lens is also nearly concordant to the sheet dip of the foliation in the enclosing shales. The overall ore lens dips 30° southwest and in much of the area investigated the strike of the lens in parallel to the longitudinal sections. In the southern portion of the area drilled the strike appears to turn to the southeast causing an apparent plunge to the south on the longitudinal sections. Similarly in the north the layering appears to swing to the north east causing an apparent north plunge on longitudinal section.

Within the ore lens the lithological layering is quite distinct on a large scale and it is clear from the detailed drilling that the layering is not generally parallel to the external contacts of the ore lens. Most commonly the layering is flat, dips gently toward the northeast or more shallowly toward the southwest than foliation and the overall ore lens. This layering, which appears to represent bedding, is locally thrown into significant northeast verging folds. North east of and above the south heading, layering is near vertical in the steep limbs of these folds. This

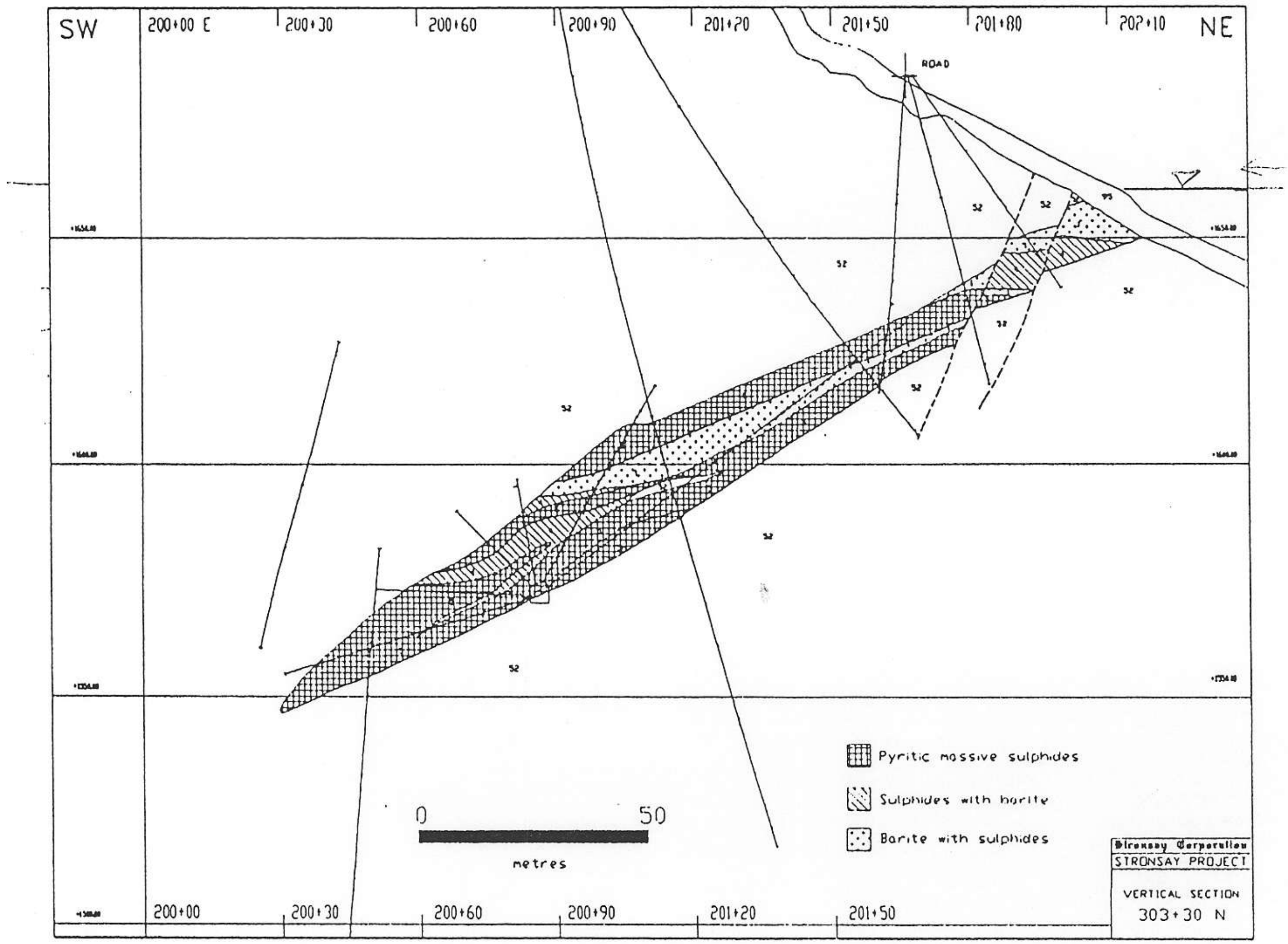

SW
NE
200+00 E
200+30
200+60
200+90
201+20
201+50
201+80
202+10
ROAD
Pyritic massive sulphides
Sulphides with barite
Barite with sulphides
0
50
metres
Stronsay Corporation
STRONSAY PROJECT
VERTICAL SECTION
303+30 N
200+00
200+30
200+60
200+90
201+20
201+50

Figure 27

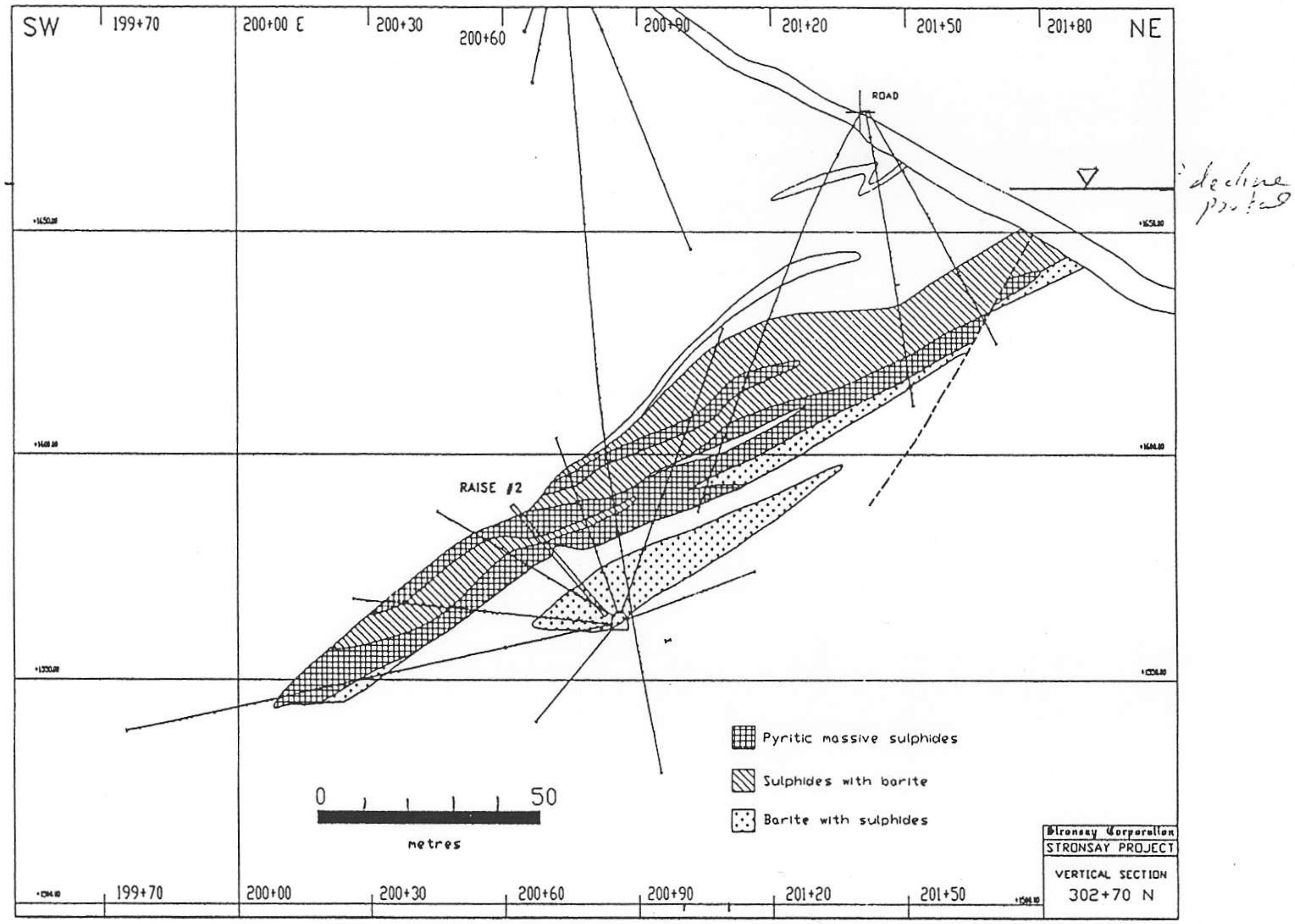

SW
199+70
200+00 E
200+30
200+60
200+90
201+20
201+50
201+80
NE
ROAD
decline portal
RAISE #2
0
50
metres
Pyritic massive sulphides
Sulphides with barite
Barite with sulphides
Stronsay Corporation
STRONSAY PROJECT
VERTICAL SECTION
302+70 N
199+70
200+00
200+30
200+60
200+90
201+20
201+50

Figure 28

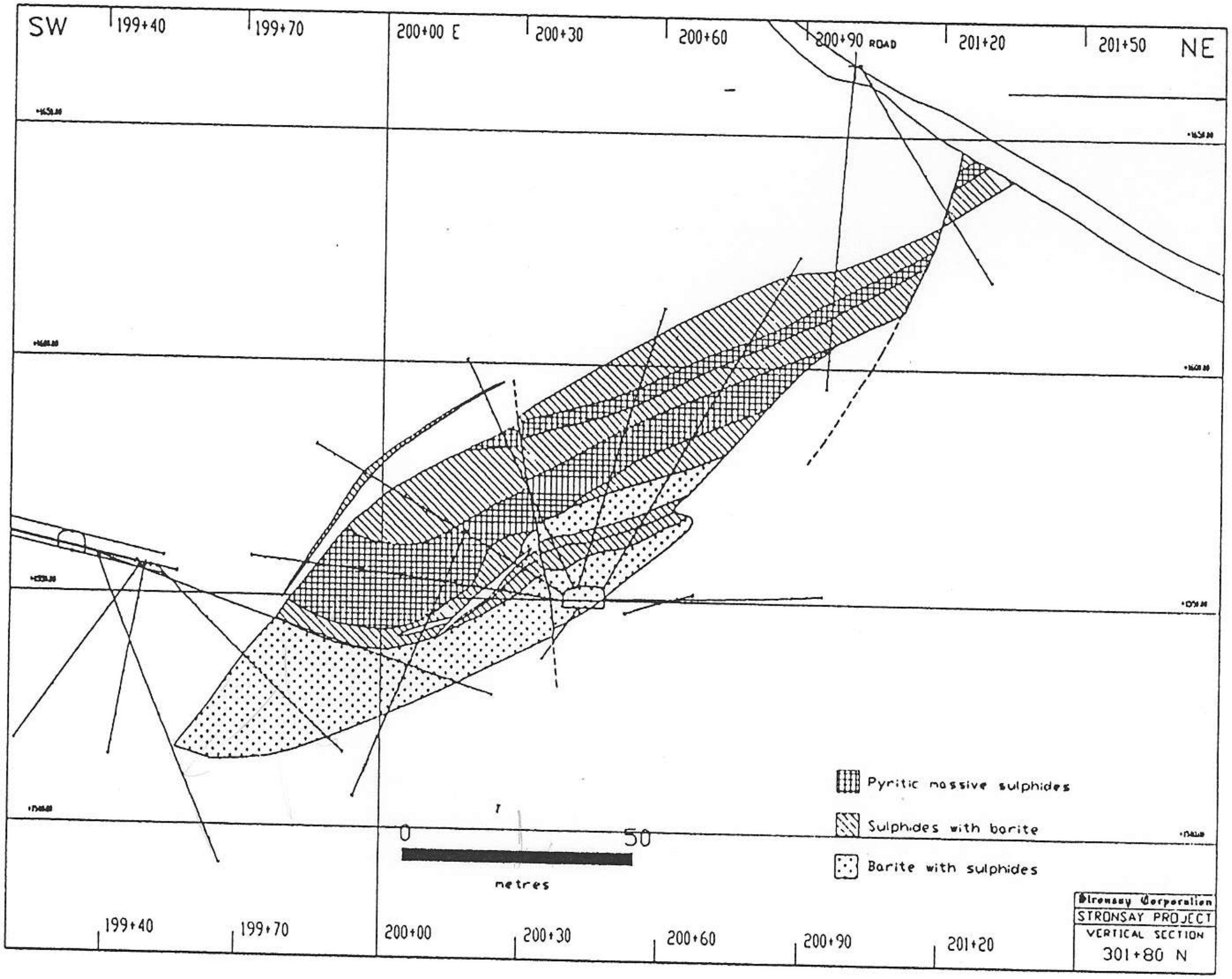
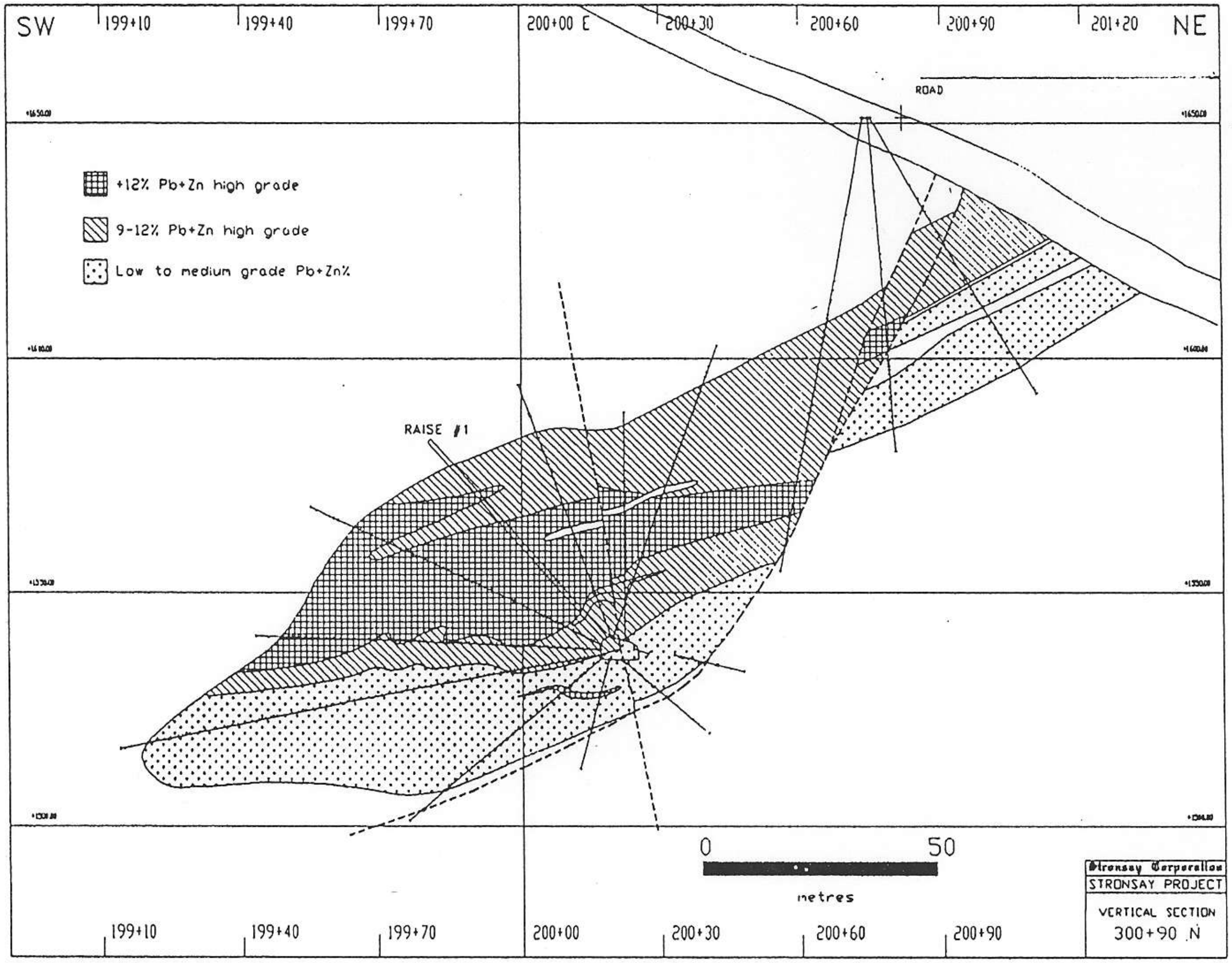

SW
199+40
199+70
200+00 E
200+30
200+60
200+90 ROAD
201+20
201+50
NE
Pyritic massive sulphides
Sulphides with barite
Barite with sulphides
0
50
metres
200+00
Stronsay Corporation
STRONSAY PROJECT
VERTICAL SECTION
301+80 N

Figure 29



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

Figure 30

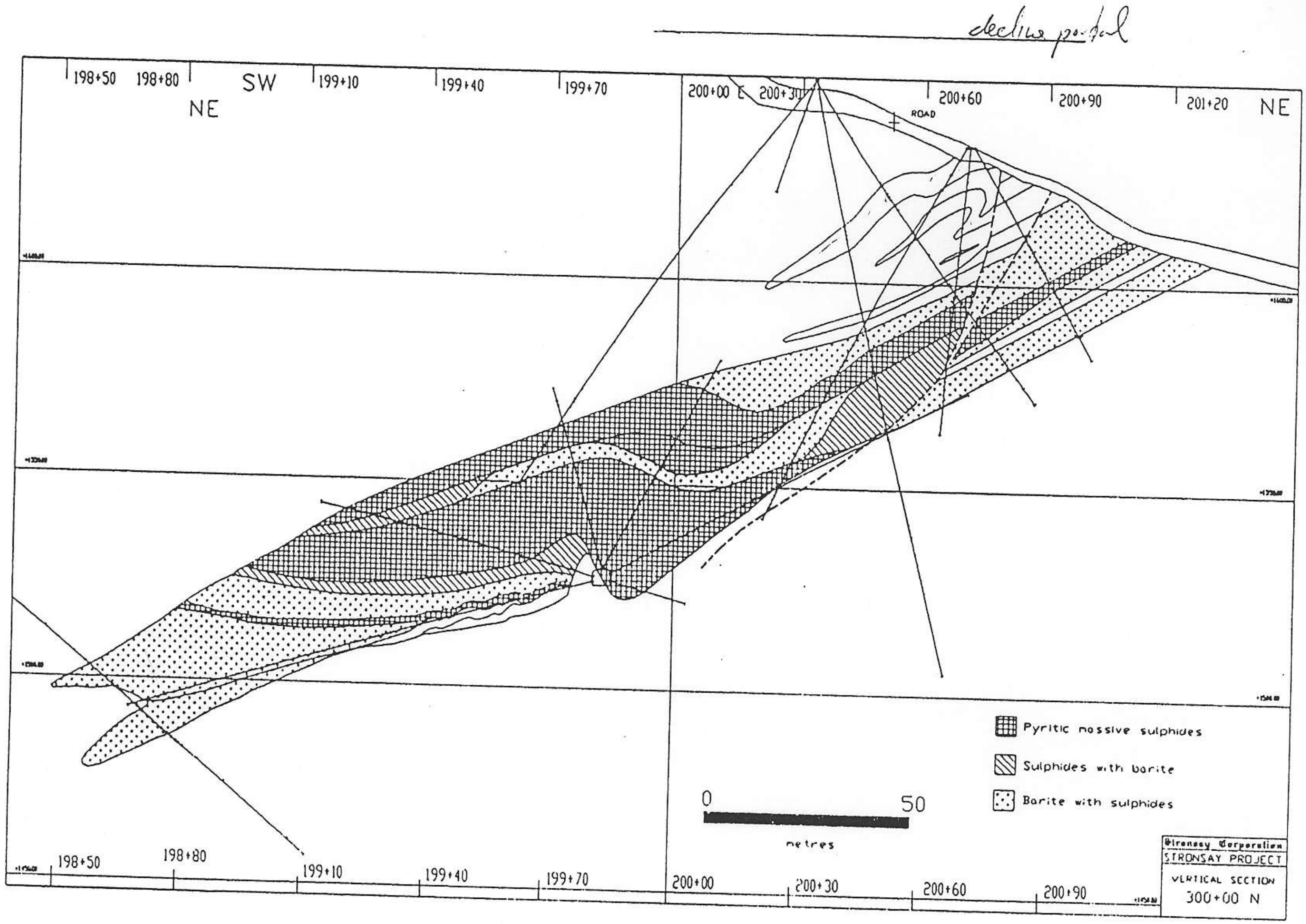

decline portal
198+50
198+80
NE
SW
199+10
199+40
199+70
200+00
200+30
200+60
ROAD
200+90
201+20
NE
Pyritic massive sulphides
Sulphides with barite
Barite with sulphides
0
50
metres
Stronsay Corporation
STRONSAY PROJECT
VERTICAL SECTION
300+00 N
198+50
198+80
199+10
199+40
199+70
200+00
200+30
200+60
200+90

Figure 31

decline portal

43

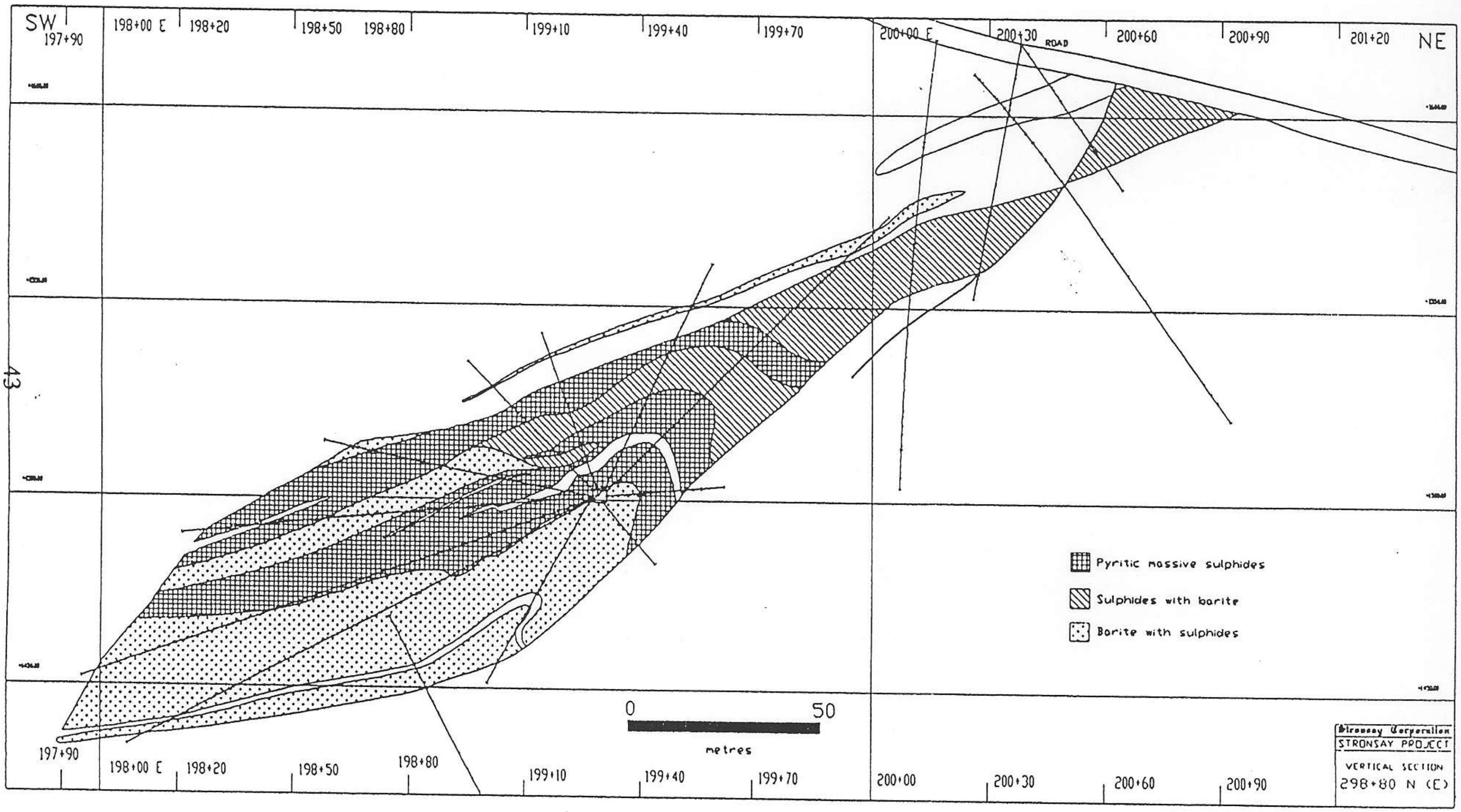


Figure 32

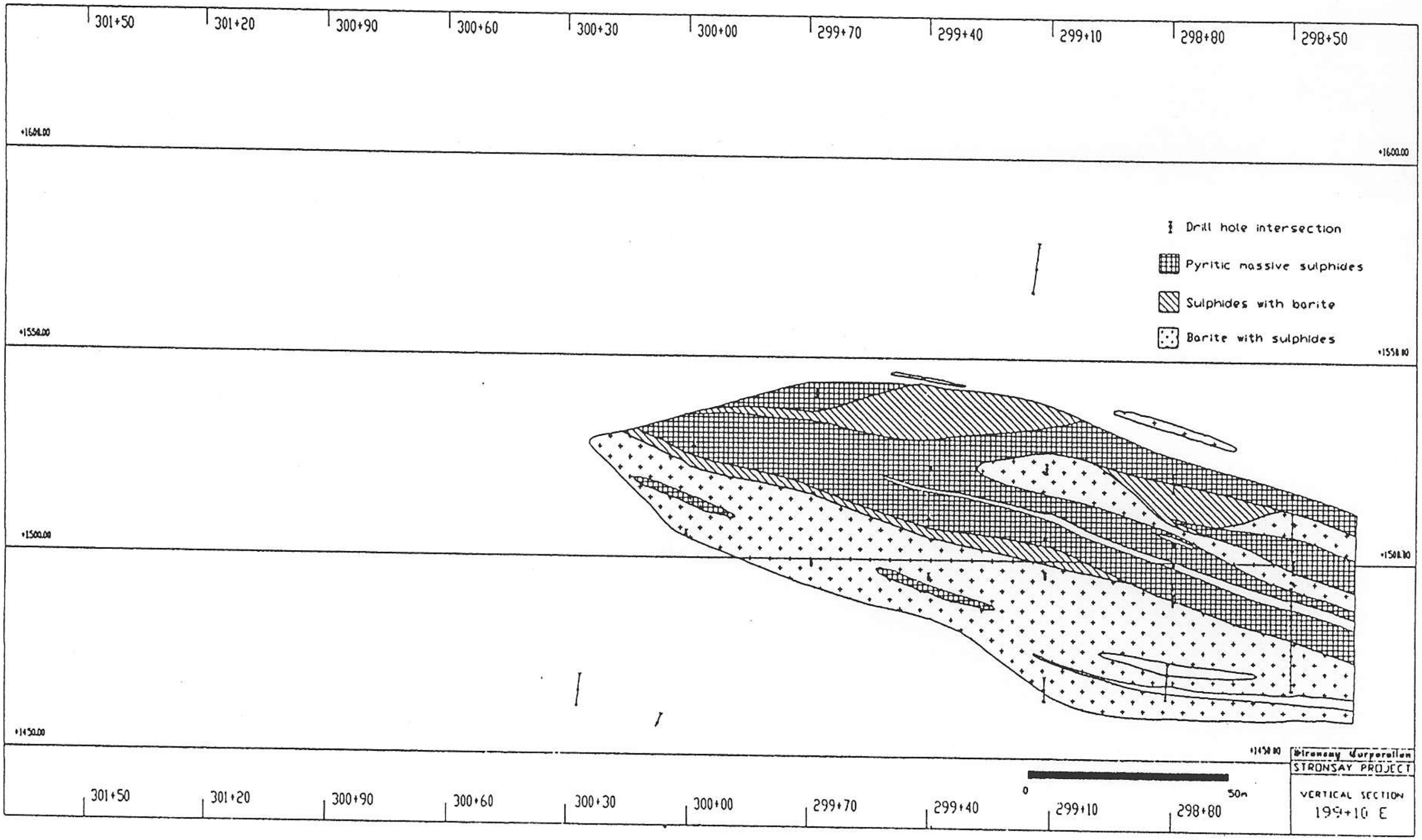

301+50
301+20
300+90
300+60
300+30
300+00
299+70
299+40
299+10
298+80
298+50
Drill hole intersection
Pyritic massive sulphides
Sulphides with barite
Barite with sulphides
0
50m
STRONSAY PROJECT
VERTICAL SECTION
199+10 E

Figure 33

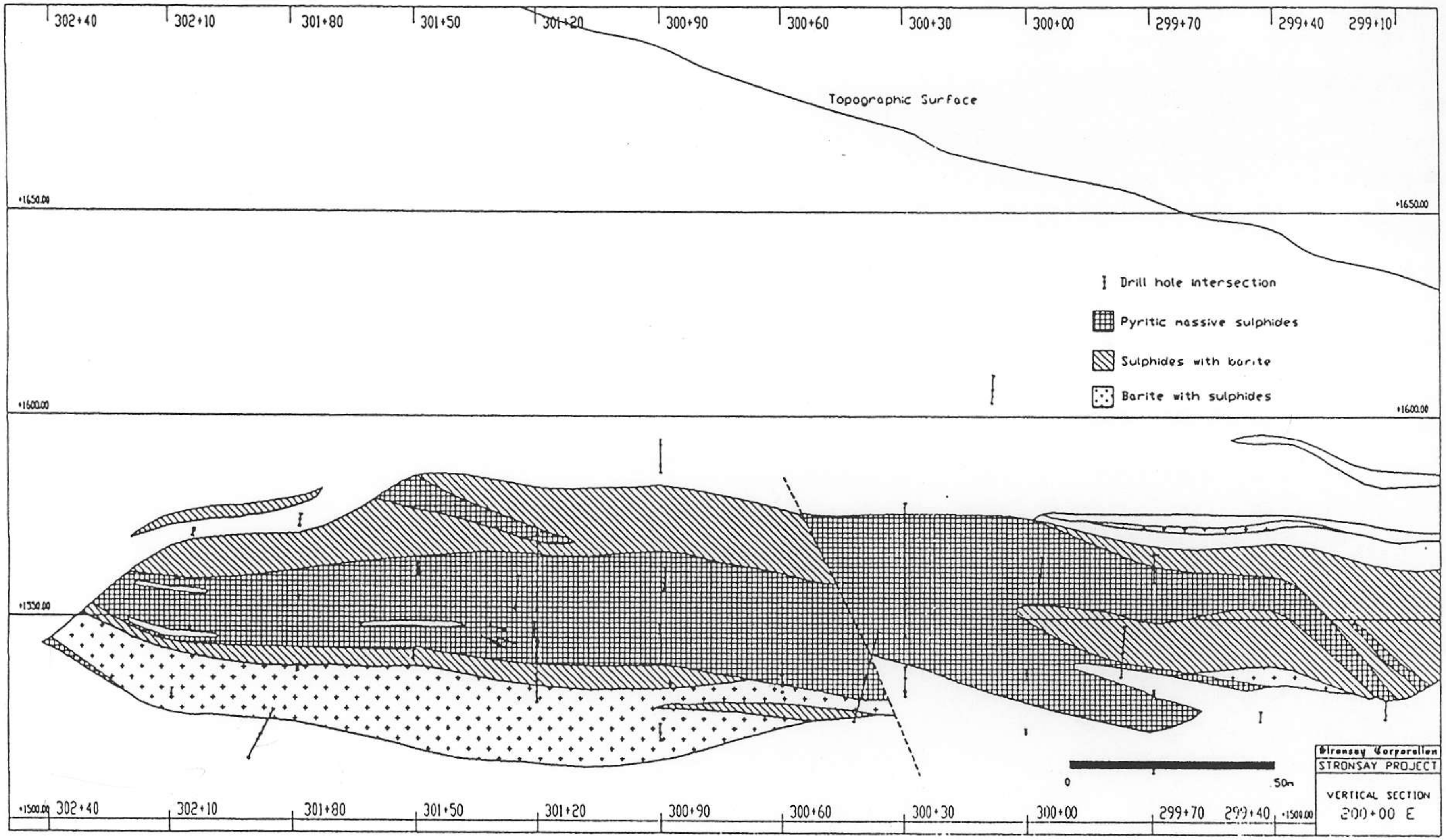

302+40
302+10
301+80
301+50
301+20
300+90
300+60
300+30
300+00
299+70
299+40
299+10
Topographic Surface
+1650.00
+1600.00
+1550.00
+1500.00
Drill hole intersection
Pyritic massive sulphides
Sulphides with barite
Barite with sulphides
0
50m
Stronsay Corporation
STRONSAY PROJECT
VERTICAL SECTION
200+00 E

Figure 34

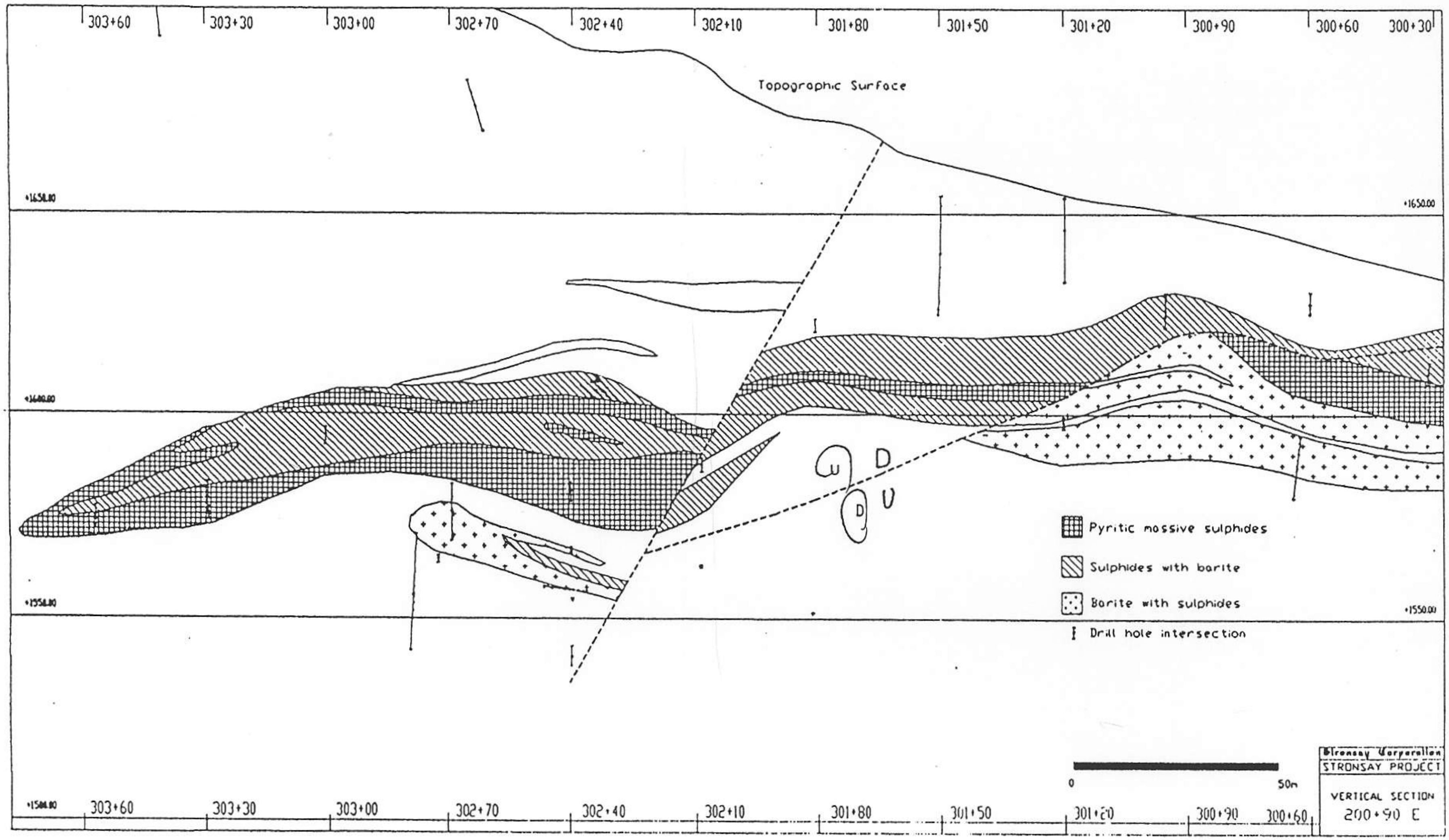
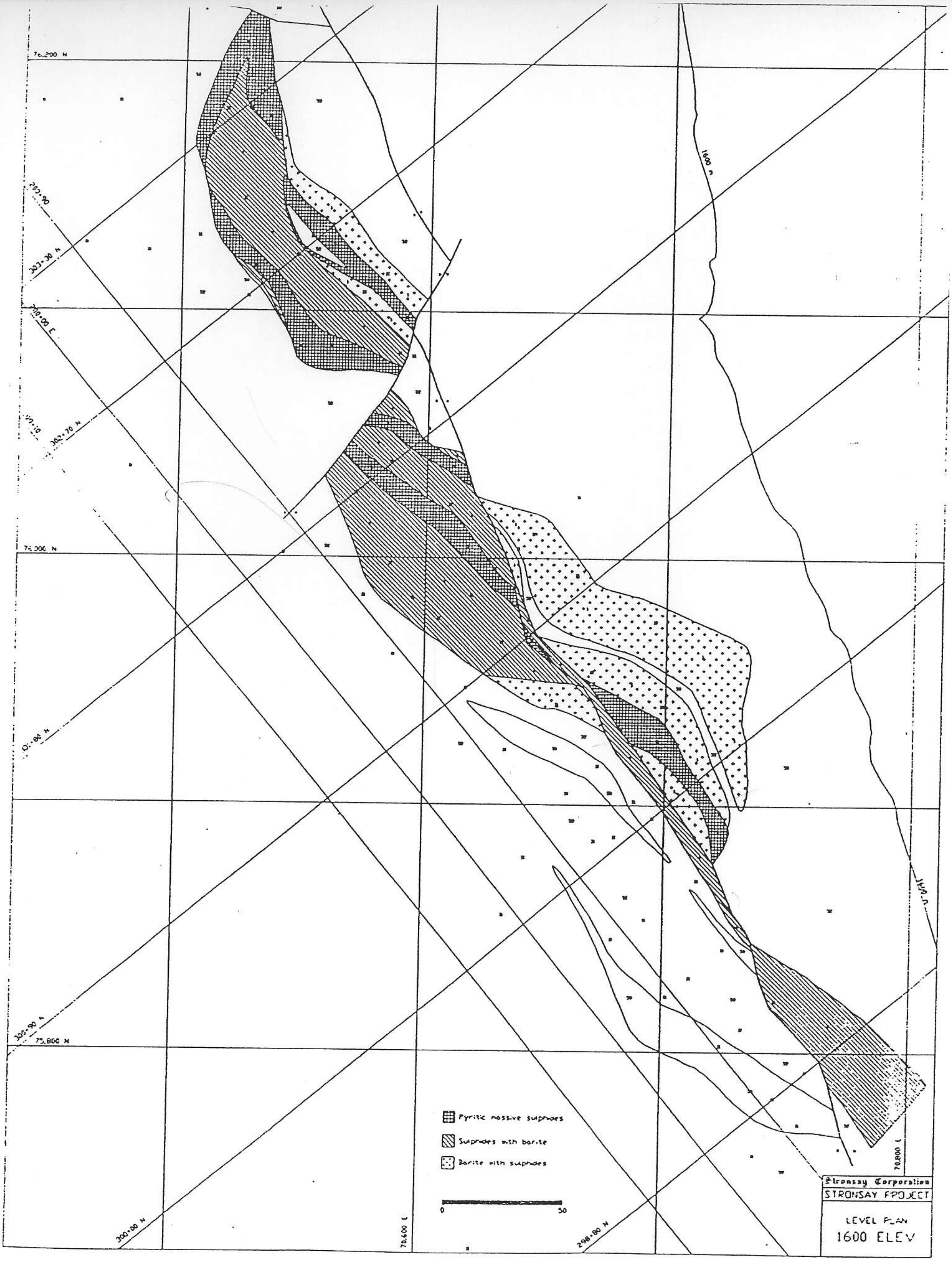

303+60
303+30
303+00
302+70
302+40
302+10
301+80
301+50
301+20
300+90
300+60
300+30
Topographic Surface
D
U
D
U
Pyritic massive sulphides
Sulphides with barite
Barite with sulphides
Drill hole intersection
0
50m
Stronsay Corporation
STRONSAY PROJECT
VERTICAL SECTION
200+90 E

Figure 35



Pyritic massive sulphides
Sulphides with barite
Barite with sulphides
0
50
Stronsay Corporation
STRONSAY PROJECT
LEVEL PLAN
1600 ELEV
298+80 N
300+00 N
70,600 E
70,800 E
75,800 N

Figure 36


Pyritic massive sulphides
Sulphides with barite
Barite with sulphides
Stronsay Corporation
STRONSAY PROJ
LEVEL PLAN
1550 ELEV
76,000 N
75,800 N
70,500 E
70,700 E
300+00 N
298+00 N
0
50

Figure 37

arrangement results in a characteristic tendency for lithologic, and hence grade, units to pass from a hanging wall position to a footwall position moving Northeast along a given cross section. The folds and this tendency are well illustrated on sections 299+70N, 299+10N and 298+80N (figure 32). The implication of this arrangement is that there are significant faults along the hanging wall and possibly the footwall contacts of the ore lens. There is clear evidence of such a faulted upper contact where it is exposed in the access decline and some evidence that the basal contact could, as well, be defined by a fault zone in places though clearly not everywhere locally. The truncation of the footwall low grade barite (lithology 1) and overlying high grade ore (lithology 5) by this fold-fault couple is particularly well displayed on section 300+00N (figure 31) and the four sections on either side of it. The orebody thus has the appearance of having been deformed under the influence of a clockwise shear couple (viewed to northwest) as might be expected in the NRMFTB. In the northern part of the deposit where it is thinner,and definition drilling from the footwall is more difficult, internal layering may be parallel to the external contacts. The fold and bounding fault arrangement creates a bulbous down-dip termination on many sections (good examples are 300+00N (figure 31) to 301+50N) where the basal low grade baritic unit appears to steepen and truncates against the hanging wall fault. This structure made drilling the down-dip portion of the orebody from the footwall heading more effective on some sections than was originally anticipated.

Despite the appearance of an important fault along the upper contact of the ore lens there is a thin and apparently continuous laminated barite-sulphide layer 1 to 3 m. thick approximately 5 to 10 m. above the hanging wall contact. This layer is associated with siltstone which is either at or within a few meters of the shale- ore contact. this siltstone is significant since it is relatively competent and should bolt well. On a few sections there is a suggestion that this upper barite wraps around the southwest termination of the ore lens.

Half way up the north heading the ore lens thins rapidly from several 10's of meters to 10 m. or less. This thinning corresponds to a separation of the lower low grade baritic material from the massive sulphide and the development of a shale parting of locally over 10 m. thickness. The lower barite disappears between section 302+70N (figure 28) and 303+00N. The termination may simply be stratigraphic however it is possible that there is an unrecognised longitudinal low angle fault which has stacked ore on itself locally. This structure would be difficult to distinguish from a transposed sedimentary contact and would have the same implication for reserve calculations. The tongue of shale in the massive sulphide in the access decline may be related to the shale parting near the lower barite pinch out. North of the pinch out (section 303+00N and north) the ore body is more planar and generally more uniformly high grade.

Stratigraphically above the ore lens near its up dip edge (just below the erosion surface) the siliceous shales are interlayered with laminar banded pyrite (lithology 50) in discontinuous units 5 to 10 m. thick. Locally this material contains up to 10% zinc however none has been included in the reserve blocks since it is unlikely to be mineable from either access, dilution, continuity, rock mechanical or metallurgical points of view. This material has been interpreted to be the distal facies equivalent of the ore deposit opening the possibility that this laminar banded pyrite could be the down dip distal edge of ore offset to the northeast by a hanging wall thrust fault and since eroded.

Several notable faults have been delineated through underground mapping and section interpretation of drill hole data. The most continuous of these faults strikes north 35° west and dips 70° west. The structure is only well constrained within the ore lens but is interpreted to flatten downward to 30 to 45° westerly dip in the shales below the ore lens. This fault down-drops the ore lens 20 to 40 m. on its west side and appears to be an extensional structure. The displacement of this fault is difficult to determine since the ore lens thins rapidly near the fault. It is thus likely that there are further complexities to be explained. A second significant fault strikes north 25 to 50° east and dips north approximately 60 degrees. This fault occurs between sections 302+10N and 302+40N and appears to offset the ore lens about 30 to 40 m. in a right lateral fashion however the apparent displacement could be due to about 15 to 20 m. of normal displacement.

There is an abrupt and as yet only partly delineated apparent truncation of the ore lens on the extreme southwest and down dip edge of sections 298+50N and 298+80N (figure 32). This could be caused by a fault striking northwest and dipping steeply to the west. This could be the fault intersected in the access decline near section 301+80N close to the AA Fault. Also in the extreme south end of the drill grid the ore lens begins to thicken, this could be apparent due to an unknown flexure parallel to the cross section or more likely is an indication of considerable ore to the south of the area explored in 1990-1991 as indicated by the several 40 to 60 m. thick intersections in that direction in holes drilled by Cyprus Anvil (figure 26).

The geological interpretation was completed by L.C. Pigage and J. Paxton in March, 1991.

RESERVE CALCULATION

Reserve blocks were laid out on cross sections to correspond to lithologic contacts where possible. An attempt was also made to respect grade breaks in the drillholes however this was not always possible since not all assay information was available when the interpretation was being done. Consequently there is no hard and fast rule applied to grade for internal reserve block boundaries although a 9% and 12% Pb+Zn assay sample cutoff was loosely observed partly because some lithologic contacts approximately correspond to those grades. The reserve blocks are laid out to encompass all barite plus sulphide mineralization within the distinct shale contacts at the footwall and hanging wall of the ore lens. Where 0% Pb+Zn cutoff reserve is quoted the entire lens is thus included. In most cases the grade of ore at the external contact is high thus there are relatively few places where the limit of ore at cutoffs higher than 0% Pb+Zn was not obviously the shale contact. The major exceptions are:

1) the contact between lithology 1 baritic facies and overlying high grade lithology 5 west of and below the south heading and;
2) the up dip portion of the ore lens.

Both these areas have been dealt with when defining the reserve blocks as indicated below.

In the first case the lithology 1 - lithology 5 contact closely follows the break from very high grade to very low grade mineralization thus its location is not very sensitive to high cutoff grades. The boundaries chosen approximated a 9% cutoff grade since the contact between lithologies tends to be the locus of a very rapid grade change which typically jumps from about 6 to 7% Pb+Zn to 10 to 12% Pb+Zn or higher over a 3 to 5 m. thickness. Unfortunately many holes in the vicinity of this contact were parallel to it which made the distinction of these two important lithology difficult.

The lower baritic unit (lithology 1) is quite low grade and generally decreases in grade with stratigraphic depth. This unit's internal grade subdivision contacts would become quite sensitive to cutoff grade if sample cutoffs as low as 6% Pb+Zn were used. In the south part of the deposit this unit develops thin high grade massive sulphide layers which are difficult to define with the available flat drill holes. The massive sulphide was differentiated on section 299+10N but not on 298+80N or 298+50N thus the lower unit appears to increase in grade on those sections.

In the second case noted above the boundaries are more transitional and adjustments were made to the block outlines so that peripheral mineralization below a 9% Pb+Zn sample cutoff grade was separated to be potentially excluded if desired. These contacts are relatively sensitive to cutoff grade.

There is another natural internal lithologic contact which corresponds to an important grade break. This is the contact of the main body of lithology 5 in the area above and west of the north and south headings. This natural break was reflected in the layout of the reserve blocks; the contact tends to be nearly equivalent to a 12 % Pb+Zn cutoff. The lower contact of this unit is the same contact noted two paragraphs above.

To summarise, the reserve blocks as laid out observe three cutoff grades in a loose sense as a result of the natural grade breaks in the ore deposit these are: 0%, 9% and 12% Pb+Zn however these are not true assay sample cutoffs.

In general reserve blocks were laid out to be approximately 5 m. or more thick. In some cases very thick blocks were arbitrarily broken into two or more thinner blocks. Internal shale bands were included with the reserve blocks unless they were thick enough to be separable (i.e. 5 m. as above).

The cross sectional area of each block was determined by digitizing the outline and calculating the area using two different software packages (AUTOCAD v10 and GEOMODEL). Selected blocks were checked by hand and all methods gave comparable results. Cross sectional area was converted to volume by using a 15 m. length of influence on either side of the section or 30 m. total (half way to the adjacent section if there was one). No special correction was made in the vicinity of faults. Each reserve block was given a lithology determined by the dominant logged drill hole lithology. There were three ore lithologies applied to the blocks (lithologies 1, 4, or 5). Volume was converted to tonnage (metric tonnes) using an average density for each of the three ore types. This average was derived from the mean of the pulp specific gravities reduced by 2% to make an allowance for porosity. This approach has worked well in estimating tonnage for Curragh Resources' stratiform lead-zinc massive sulphide deposits of similar visible porosity range. At the time the estimate of the mean was done only a portion of the pulp S.G. data was available. The average data is summarised in Table III. Also given is a comparison to limited whole rock S.G. data from Cyprus Anvil. It should be noted that this approach yields lower densities than those used previously by Cyprus Anvil.

Drill hole assays were composited using intervals that corresponded to the cross sectional reserve block outlines. The average composite interval was approximately 10 m. with a range of 0.6 to 94 m. The overwhelming majority of the composites were in the range of 5 to 15 m. Metal statistics of these composite are not provided in this report, however Table III gives statistics for 5 m. equal length composites calculated from the drill hole collar. As would be expected compositing assays into longer intervals tends to reduce variance and for this data set tends to lower the population mean. Statistical and geostatistical analyses of these data are currently underway. Assays were averaged by weighting for their length but no weighting was used for S.G.. The composites were calculated by PCXPLOR software. Selected manual checks gave full agreement on the composite values. The lack of S.G. weighting is not likely to have a significant impact on average grade as it is not clear that there is a systematic grade-S.G. relationship. The use of this weighting was precluded by the fact that data were not universally available at the time the compositing was done. Each drill hole composite was assigned to a block and all composites within a block were length weighted to arrive at the average grade for a block.

Some blocks contained no drill hole intercepts. These blocks were given the average of the remaining blocks of the same lithology. The values used are presented in Table IV. These are the blocks considered less reliable ore and are indicated with a probability score of 0 on sectional block tallies in Appendices B through D.

The reserve blocks were laid out by J. Paxton and L.C. Pigage; assay composite and area calculation runs were completed by D. Brownlee. The final reserve calculation was by a Symphony spreadsheet database designed and run by G. Jilson.

GEOLOGICAL RESERVES

The tonnage and average grade of all material in the sulphide-barite lens has been calculated using the methods outlined above. The result is presented in Table V. Also presented in Table V are two other estimates for smaller parts of the barite-sulphide lens that meet certain grade and continuity criteria. As noted above these estimates are not strictly above a specific sample cutoff grade. The portion of the lens estimated has been tailored to be a set of contiguous reserve blocks that might be a logical target for a mining plan. Most blocks are above the stated

TABLE III

BLOCK ROCK TYPE	DESCRIPTION	PURE ROCK	NUMBER OF SAMPLES	ARITHMETIC MEAN	MAXIMUM VALUE	MINIMUM VALUE	GEOMETRIC MEAN	STD. DEV	MEDIAN	5% BELOW THIS SG	5% ABOVE THIS SG
SUMMARY OF CURRAGH PULP S.G. DATA											
1	barite sulphide	unit 1	201	4.23	4.80	2.52	4.20	0.45	4.40	3.17	4.70
1	barite breccia	unit 2	11	3.82	4.30	3.52	3.81	0.26	3.80	n/a	n/a
1	barite vein	unit 3	111	4.38	4.87	3.36	4.37	0.25	4.43	3.81	approx. 4.70
4	sulphide with barite	unit 4	1194	4.36	5.61	approx. 3.0	4.35	0.22	4.40	4.04	approx. 4.70
5	massive sulphide	unit 5	1473	4.43	5.66	2.88	4.42	0.25	4.48	4.02	approx. 4.70
NU*	laminated pyrite	unit 6	24	3.90	4.43	approx. 3.5	3.89	0.21	3.88	n/a	n/a
NU	laminar banded pyrite	unit 50	52	3.43	4.14	2.73	3.41	0.33	3.35	n/a	n/a

*NU = not used in blocks

ROCK TYPE	PULP SG	RECOMMENDED DENSITY FROM REDUCING PULP SG BY 2%	USED PREVIOUSLY BY CAMC
RECOMMENDED DENSITY BASED ON PULP S.G. DATA			
1	4.20	4.12	4.30
4	4.35	4.26	4.50
5	4.42	4.33	4.50

ROCK TYPE	AVERAGE	STD. DEV.	MAXIMUM	MINIMUM
SUMMARY OF CAMC WHOLE ROCK S.G. DATA FROM SOUTH CIRQUE				
1	4.32	0.08	4.45	4.45
4 & 5	4.16	0.31	4.55	3.42
4 & 5 with no carbonate	4.29	0.28	4.55	3.42

TABLE IV

Average grade applied to blocks with no holes in them - based on average of other blocks with same lithology with no cutoff applied.

Average Grades

	Pb+Zn	Pb	Zn	Ag
Unit 1	7.25	1.25	6.00	30
Unit 4	10.25	2.50	8.00	50
Unit 5	15.00	3.00	12.00	75

TABLE V

Stronsay Corporation

Stronsay Project, British Columbia

Undiluted, In-situ, Geological Reserve 298+50N to 303+60N, North Cirque Deposit

rock code	volume (cu. m.)	tonnage (tonnes)	lead + zinc (%)	lead (%)	zinc (%)	silver (g/t)	proportion of rock type
Entire ore lens, no cutoff applied, i.e. 0% cutoff							
1	1,455,882	5,992,410	7.28	1.25	6.04	30.4	35%
4	1,375,917	5,865,534	10.61	2.49	8.12	51.3	34%
5	1,242,855	5,383,551	16.01	3.53	12.48	75.6	31%
total	4,074,654	17,241,495	11.14	2.38	8.76	51.6	100%
Portion of ore lens above 9% Pb+Zn sample cutoff, in contiguous blocks							
1	204,597	842,121	8.48	2.06	6.42	38.3	8%
4	1,098,450	4,682,692	10.99	2.52	8.47	52.6	43%
5	1,216,170	5,267,962	16.07	3.54	12.53	75.7	49%
total	2,519,217	10,792,775	13.27	2.98	10.29	62.8	100%
High Grade massive sulphide unit, mainly +12% Pb+Zn, in contiguous blocks							
1	51,768	213,077	7.98	2.28	5.71	37.4	3%
4	394,515	1,681,817	11.90	2.83	9.06	56.7	25%
5	1,145,787	4,963,091	16.24	3.60	12.64	76.4	72%
total	1,592,070	6,857,985	14.92	3.37	11.55	70.3	100%

G. Jilson 29-May-91

tssec386\D:\data\cirque\reserves\contres.wr1 cutoff grade however some lower grade blocks are included and some isolated high grade blocks are excluded. The listing of blocks for each section corresponding to each of these summaries is provided in Appendix B to D. Figures 38 through 42 show the blocks used for the reserve calculation in each cased noted above. Figures 44 to 49 show the distribution of grade in the barite-sulphide lens as estimated by this calculation. A full set of sections is included in Appendix A.

Figure 50 shows the tonnage above cutoff grade for a variety of grades. The tonnage for each cutoff is based on simple block exclusion regardless of location. The figure shows that although the geological reserve is quoted at a 0% cutoff there is in fact very little material below 6% and virtually none below 4% Pb+Zn content included.

Table VI and VII compare the results of this calculation to the previous work of Cyprus Anvil Mining Corporation. The Stronsay reserve estimates in those tables are made on a the basis of block exclusion. This is appropriate since this is how the CAMC result above a 9% Pb+Zn cutoff was calculated. As most of CAMC's sections were in a different location that Stronsay's (figure 51) it is not possible to compare on a section by section basis.

All figures quoted are undiluted, in-situ geological material; there have been no adjustments made to the numbers to reflect mining loss or dilution.

As was expected from the preliminary drill results there has been a loss of volume of the barite-sulphide lens but the mineralization is higher grade than previously indicated. A total reserve for the north part of the barite sulphide lens of 17.2 million tonnes based on the old and new drilling results is compared to 21.6 million tonnes for the same strike length per CAMC. This represents a decrease in tonnage of 20%. CAMC used a higher density than that being used currently. If compared on the basis of the same density it is revealed that half of the tonnage loss is due to lost volume and the remainder is due to lower density. At a higher block exclusion cutoff grade the two calculations compare more favourably however there is still a 15% lower tonnage, 1/3 of which is due to the use of a lower density. Since the average grade is higher in the new calculation, the change in total contained metal is not as large as the tonnage change, decreasing by 15% at the 0% cutoff or 12% at the 9% cutoff. Much of the volume of ore lost is in the northern portion of the portion of the deposit investigated, the more southerly sections compare better. For example CAMC's section 298+50 is recast for a 30 m. rather than 85 m. strike length would amount to 1.75 million tonnes averaging 10.25% combined Pb+Zn. The current reserve shows 1.80 million tonnes averaging 9.93% Pb+Zn for the same section.

A yardstick used to assess proven ore in other lead-zinc deposits is having sufficient drilling to define less than 1000 tonnes of ore per meter of drilling in ore. Curragh Resources experience with other stratiform lead-zinc deposits has been that once drill density is adequate to provide 1 m. for each 2000 to 2500 tonnes or ore there are few geological or reserve surprises at the mining stage and thus from an open pit perspective the ore is essentially proven ore. Since Cirque would not be mined by open pit and some holes are subparallel to layering the requirements for proven ore will be more stringent. At Cirque the total length of drilling

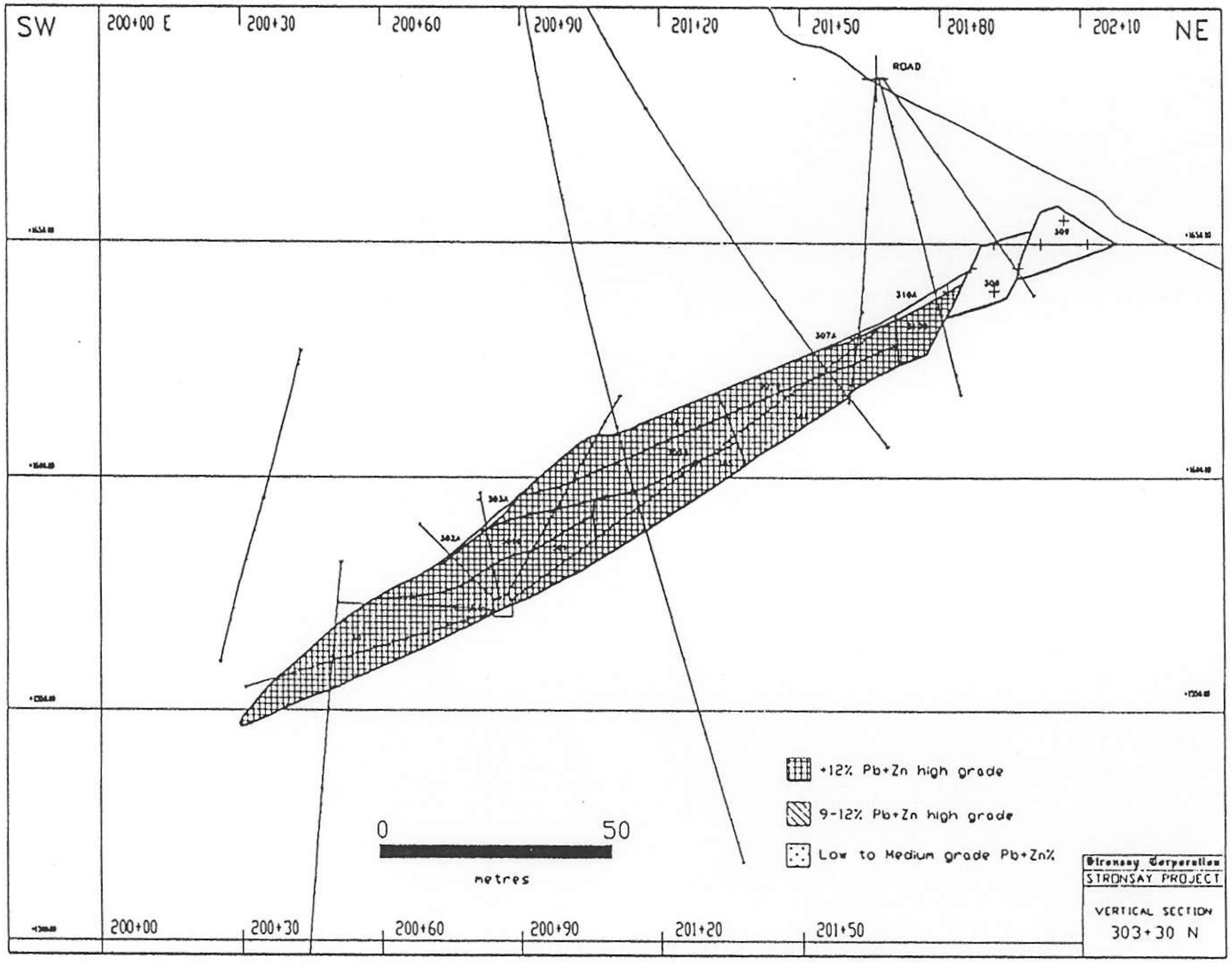

SW
NE
200+00 E
200+30
200+60
200+90
201+20
201+50
201+80
202+10
ROAD
0
50
metres
+12% Pb+Zn high grade
9-12% Pb+Zn high grade
Low to Medium grade Pb+Zn%
Stronsay Corporation
STRONSAY PROJECT
VERTICAL SECTION
303+30 N
200+00
200+30
200+60
200+90
201+20
201+50

Figure 38

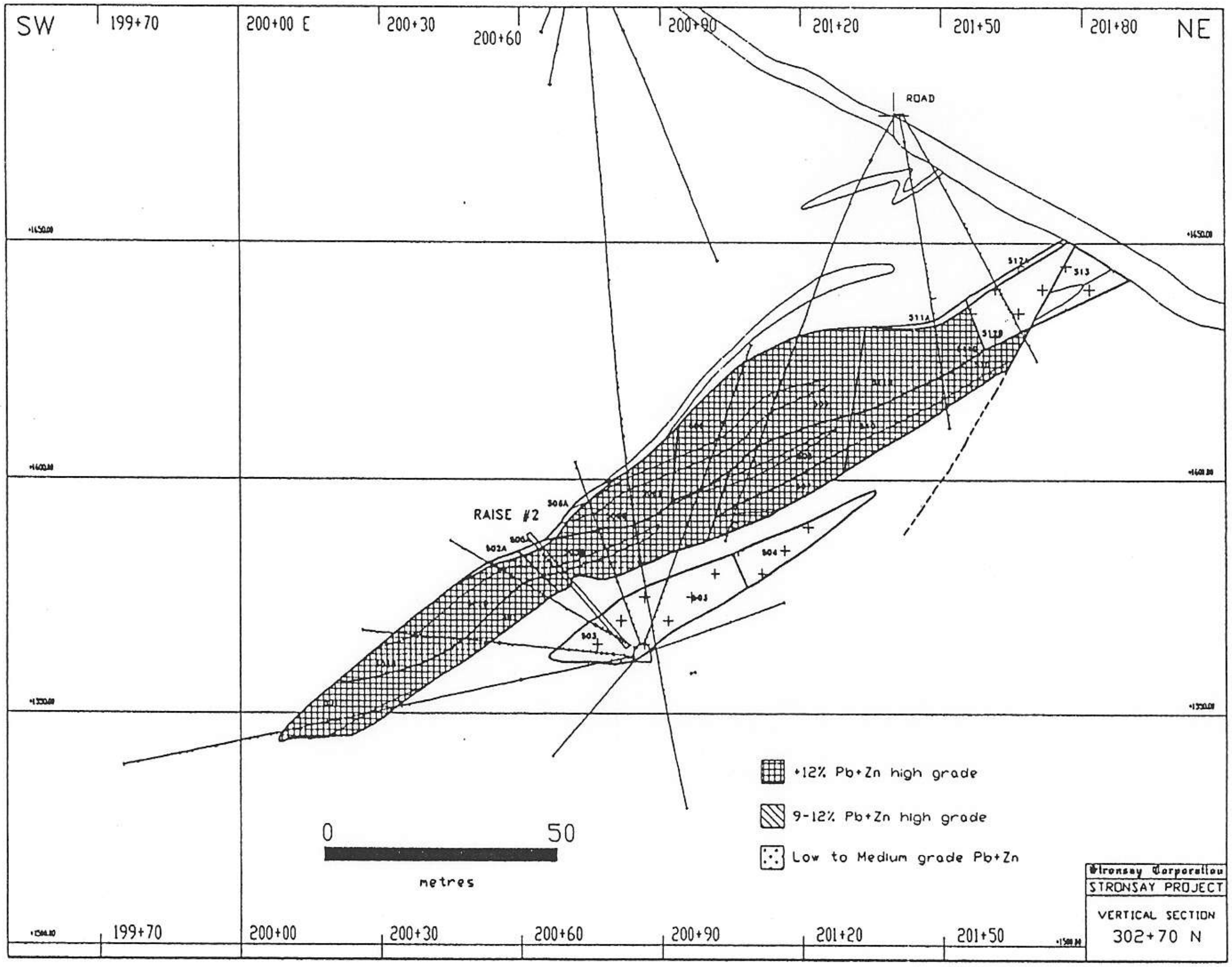

SW
NE
199+70
200+00 E
200+30
200+60
200+90
201+20
201+50
201+80
ROAD
RAISE #2
+12% Pb+Zn high grade
9-12% Pb+Zn high grade
Low to Medium grade Pb+Zn
0
50
metres
Stronsay Corporation
STRONSAY PROJECT
VERTICAL SECTION
302+70 N
199+70
200+00
200+30
200+60
200+90
201+20
201+50

Figure 39

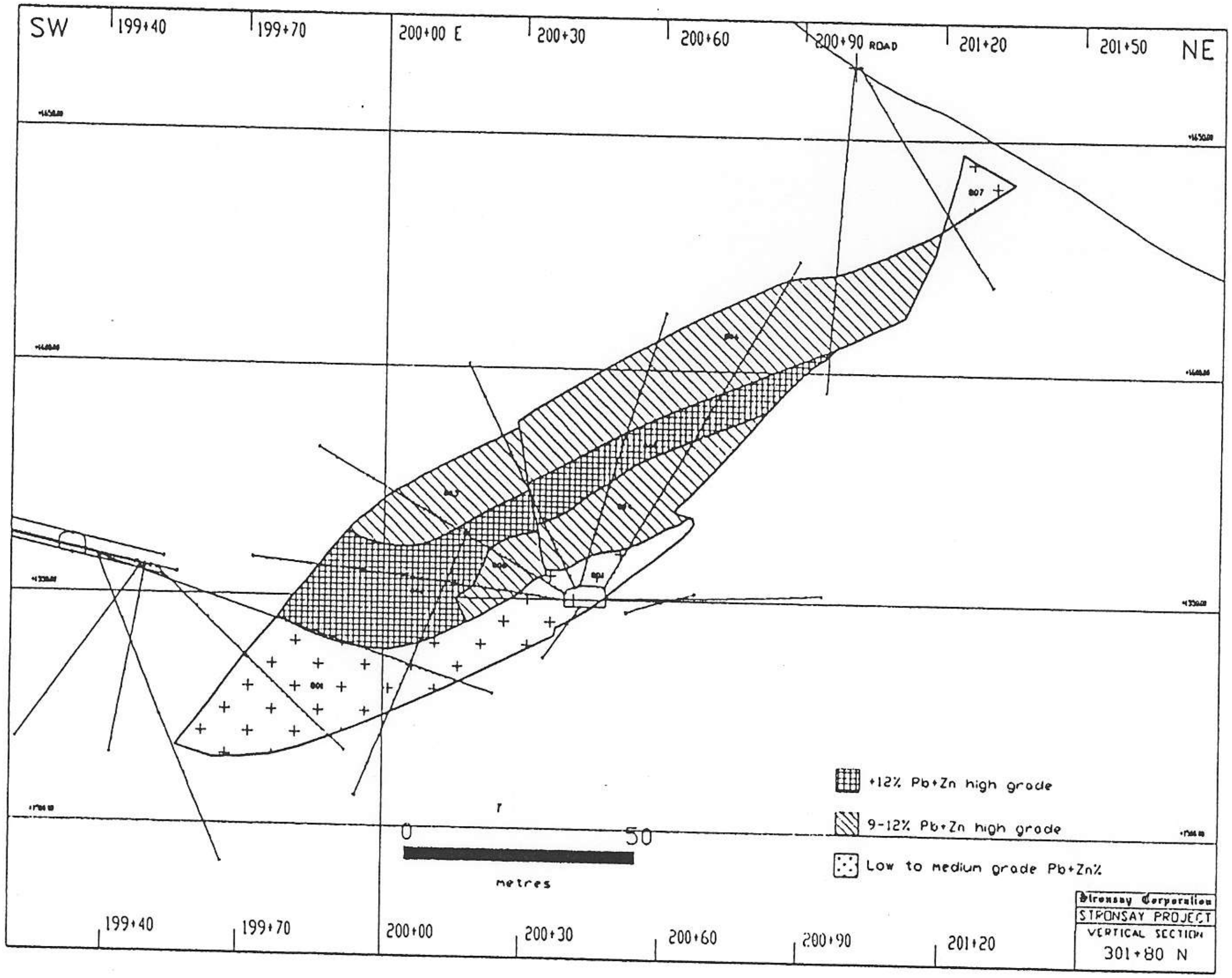

SW
NE
199+40
199+70
200+00 E
200+30
200+60
200+90 ROAD
201+20
201+50
0
50
metres
+12% Pb+Zn high grade
9-12% Pb+Zn high grade
Low to medium grade Pb+Zn%
Stronsay Corporation
STRONSAY PROJECT
VERTICAL SECTION
301+80 N
199+40
199+70
200+00
200+30
200+60
200+90
201+20

Figure 40

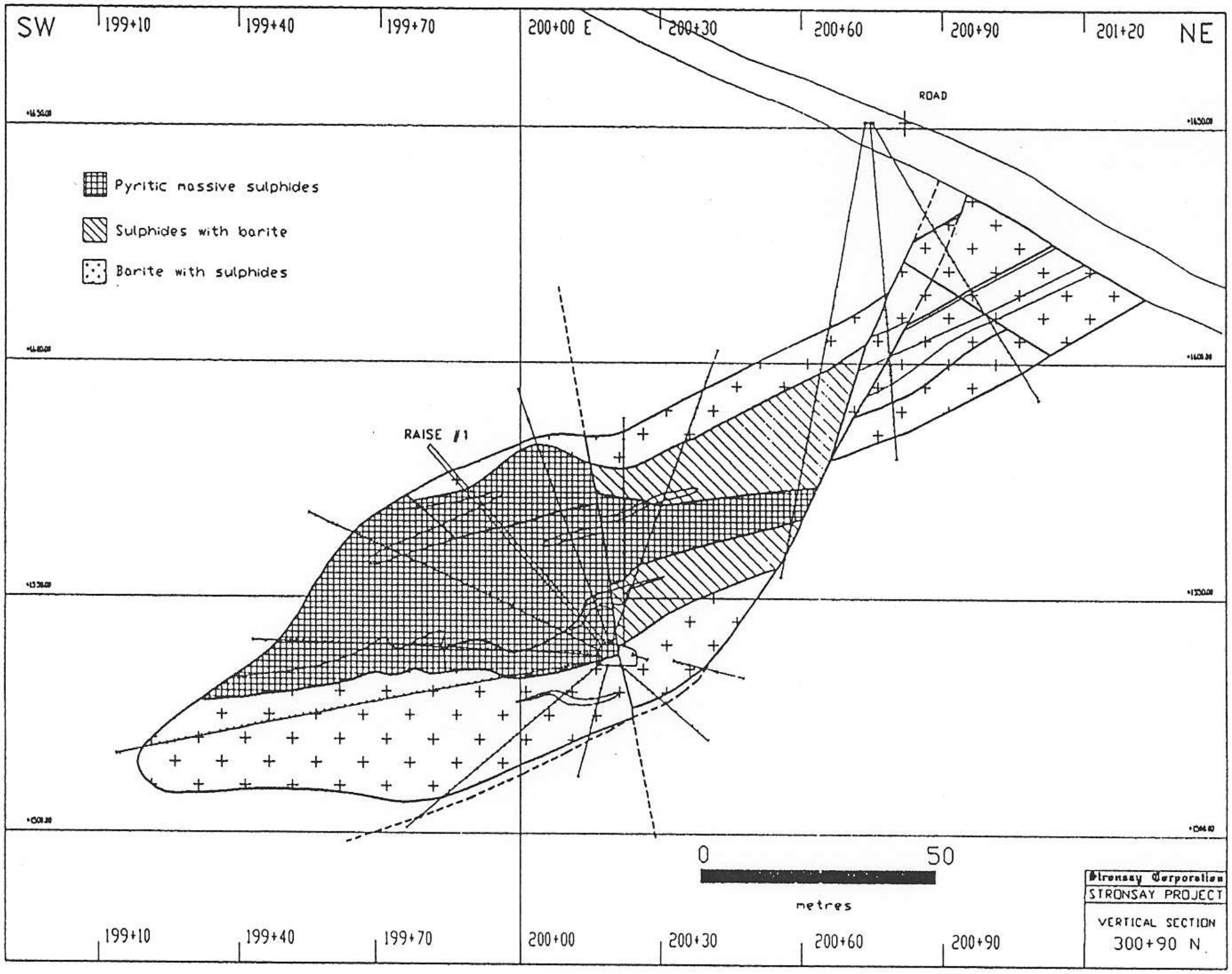

SW
199+10
199+40
199+70
200+00 E
200+30
200+60
200+90
201+20
NE
ROAD
Pyritic massive sulphides
Sulphides with barite
Barite with sulphides
RAISE #1
0
50
metres
200+00
200+30
200+60
200+90
Stronsay Corporation
STRONSAY PROJECT
VERTICAL SECTION
300+90 N

Figure 41

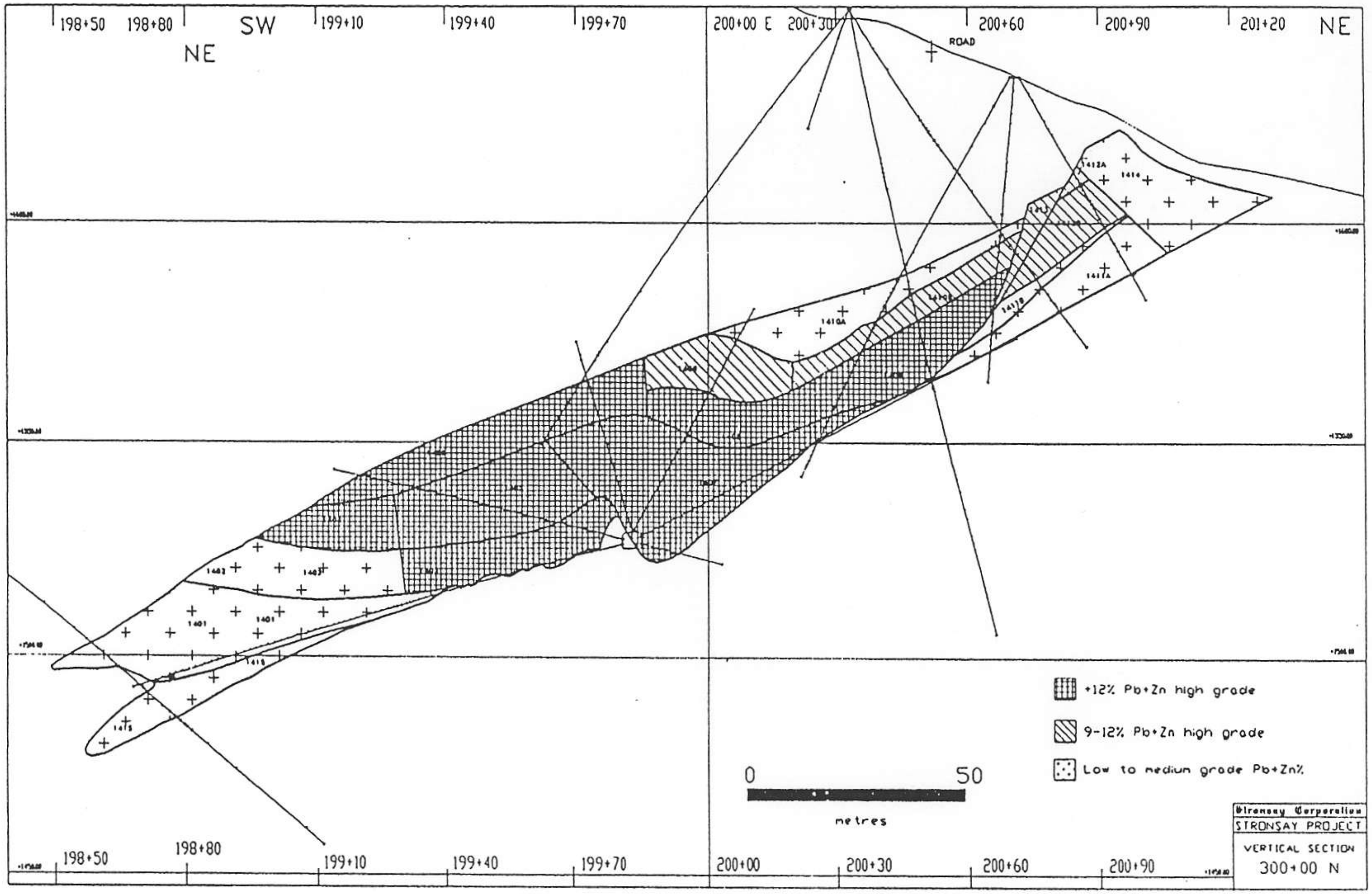

198+50
198+80
SW
NE
199+10
199+40
199+70
200+00 E
200+30
200+60
200+90
201+20
NE
ROAD
+12% Pb+Zn high grade
9-12% Pb+Zn high grade
Low to medium grade Pb+Zn%
0
50
metres
Stronsay Corporation
STRONSAY PROJECT
VERTICAL SECTION
300+00 N
198+50
198+80
199+10
199+40
199+70
200+00
200+30
200+60
200+90

Figure 42

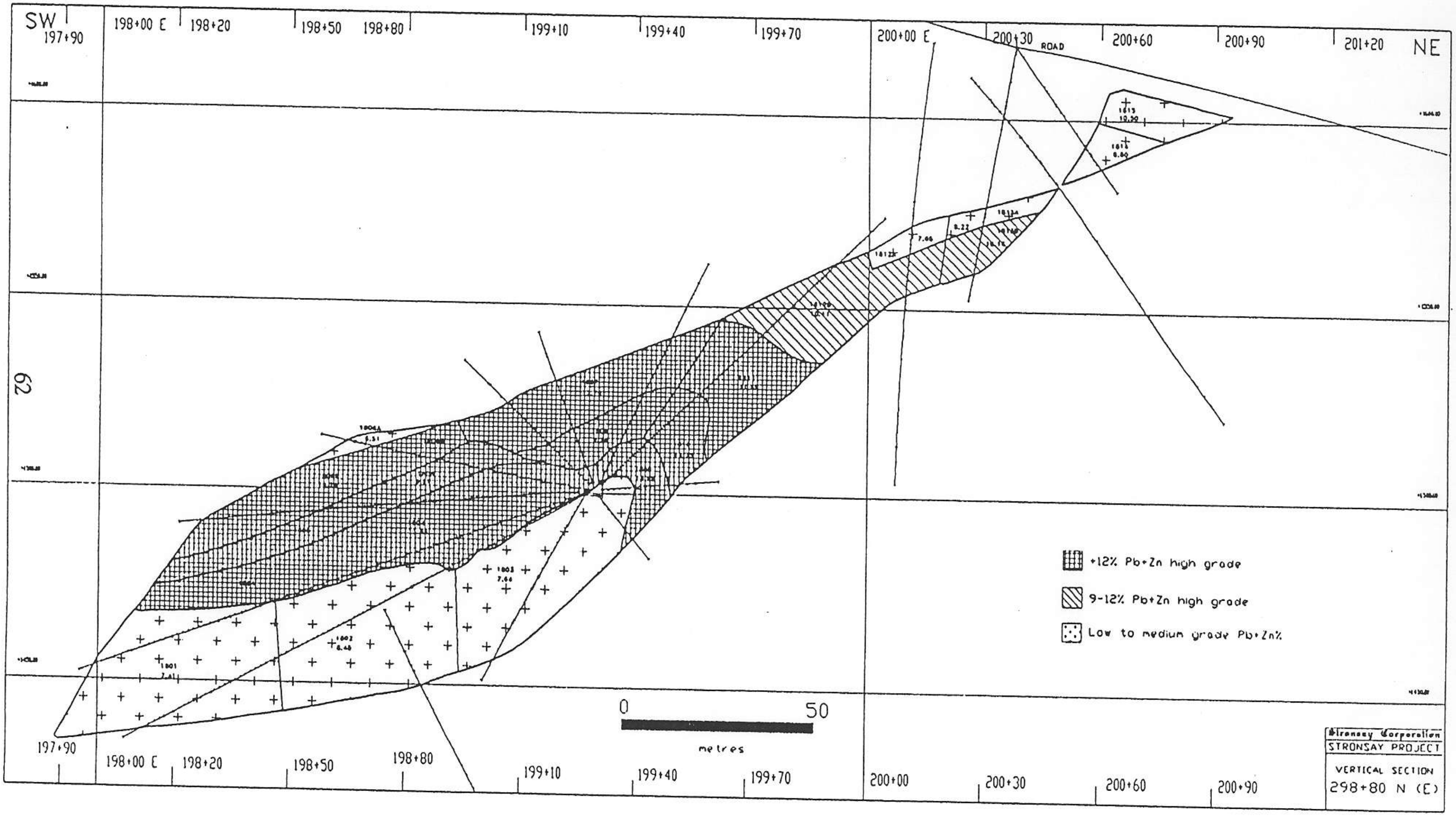

SW
NE
197+90
198+00 E
198+20
198+50
198+80
199+10
199+40
199+70
200+00 E
200+30
200+60
200+90
201+20
ROAD
+12% Pb+Zn high grade
9-12% Pb+Zn high grade
Low to medium grade Pb+Zn%
0
50
metres
Stronsay Corporation
STRONSAY PROJECT
VERTICAL SECTION
298+80 N (E)

Figure 43

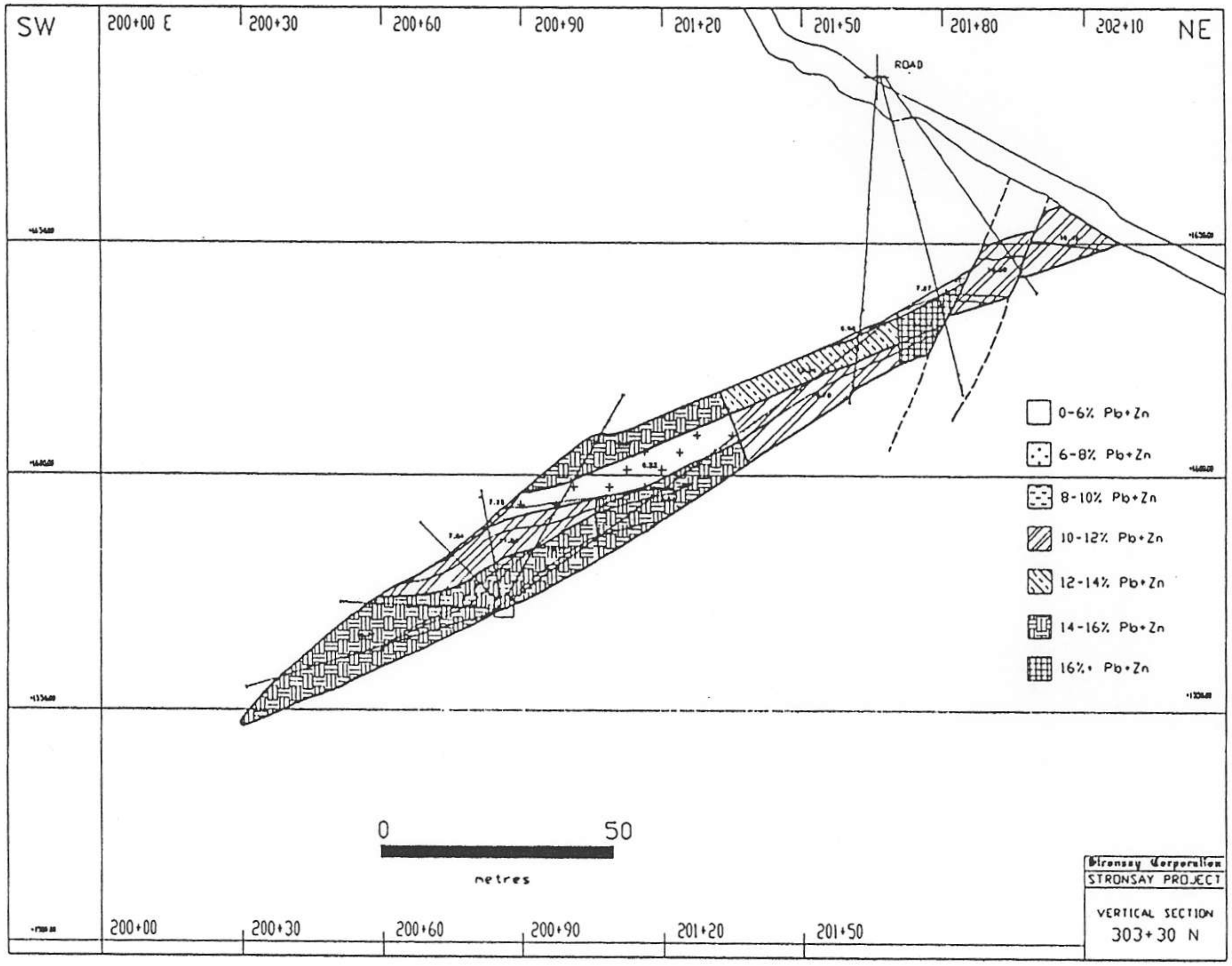
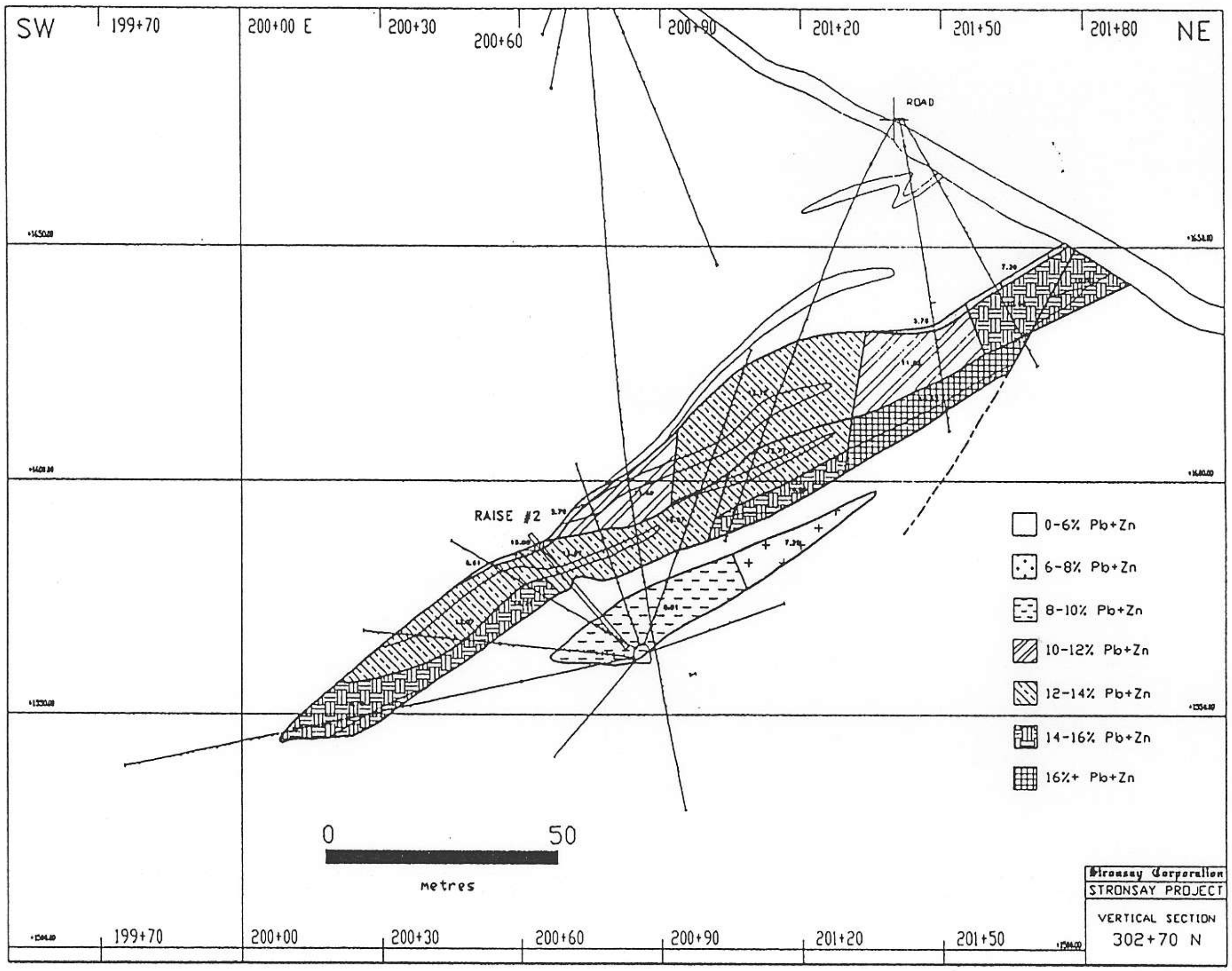

SW
200+00 E
200+30
200+60
200+90
201+20
201+50
201+80
202+10
NE
ROAD
0-6% Pb+Zn
6-8% Pb+Zn
8-10% Pb+Zn
10-12% Pb+Zn
12-14% Pb+Zn
14-16% Pb+Zn
16%+ Pb+Zn
0
50
metres
Stronsay Corporation
STRONSAY PROJECT
VERTICAL SECTION
303+30 N
200+00
200+30
200+60
200+90
201+20
201+50

Figure 44



SW
199+70
200+00 E
200+30
200+60
200+90
201+20
201+50
201+80
NE
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

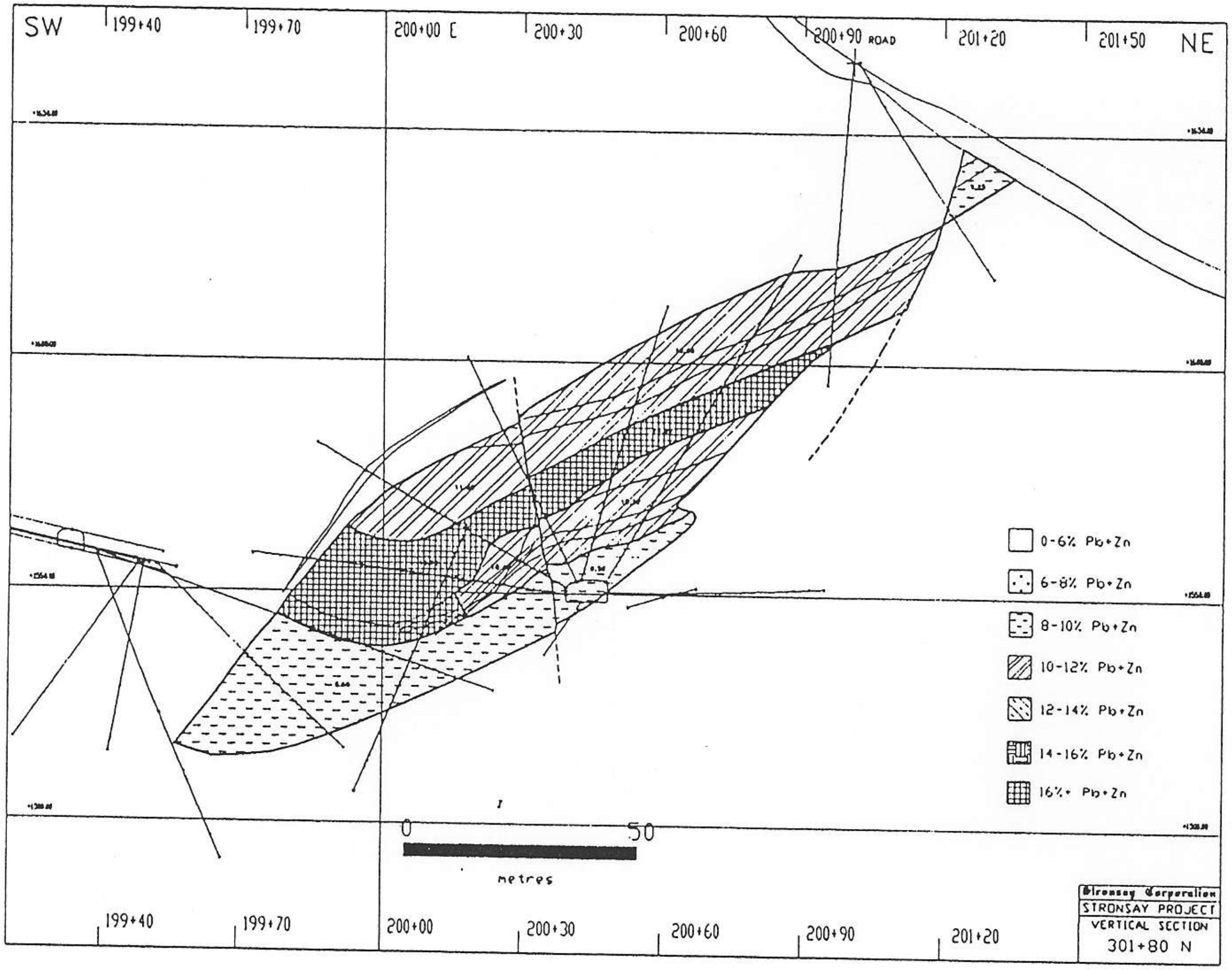

SW
NE
199+40
199+70
200+00 E
200+30
200+60
200+90
ROAD
201+20
201+50
0-6% Pb+Zn
6-8% Pb+Zn
8-10% Pb+Zn
10-12% Pb+Zn
12-14% Pb+Zn
14-16% Pb+Zn
16%+ Pb+Zn
0
50
metres
199+40
199+70
200+00
200+30
200+60
200+90
201+20
Stronsay Corporation
STRONSAY PROJECT
VERTICAL SECTION
301+80 N

Figure 46

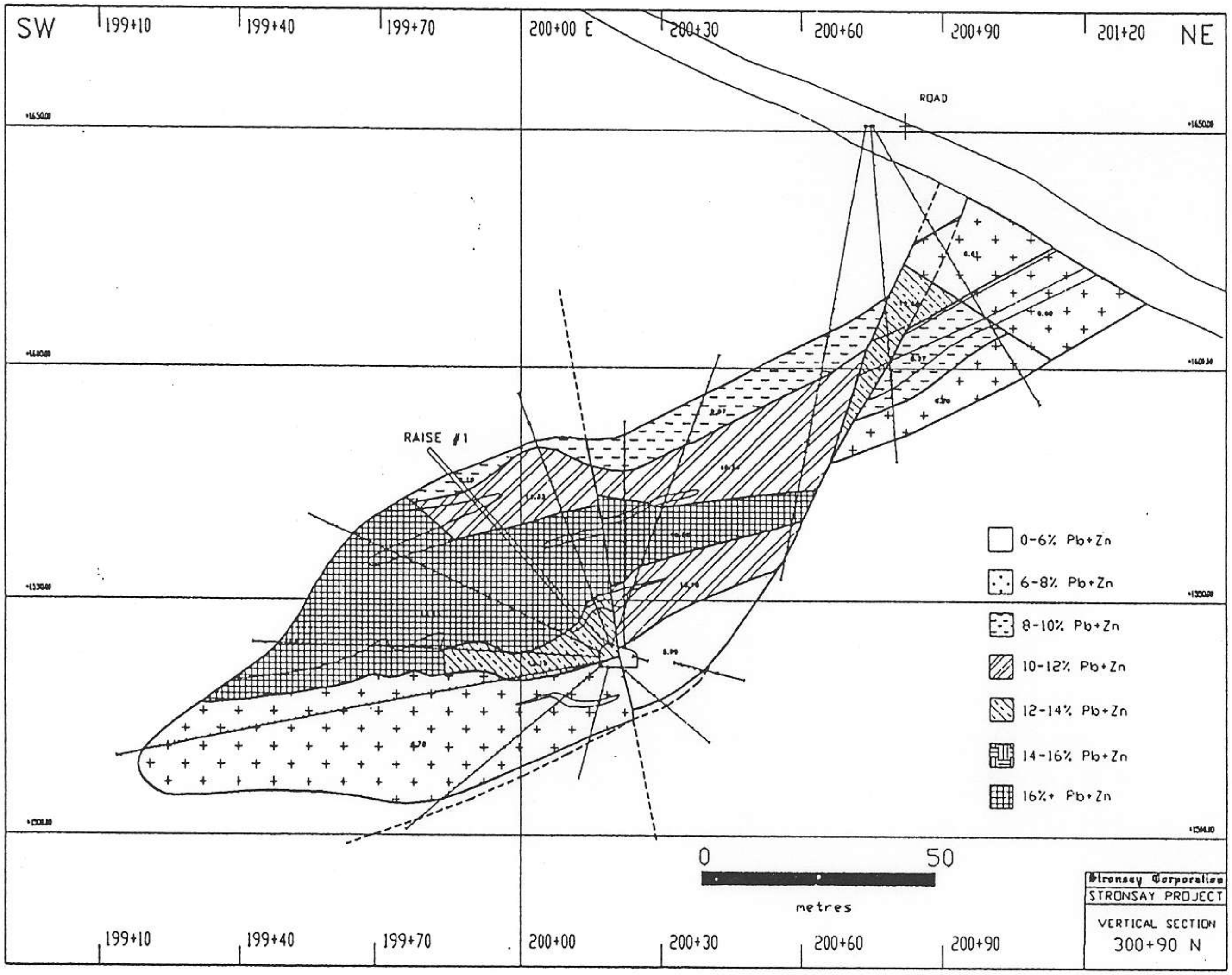
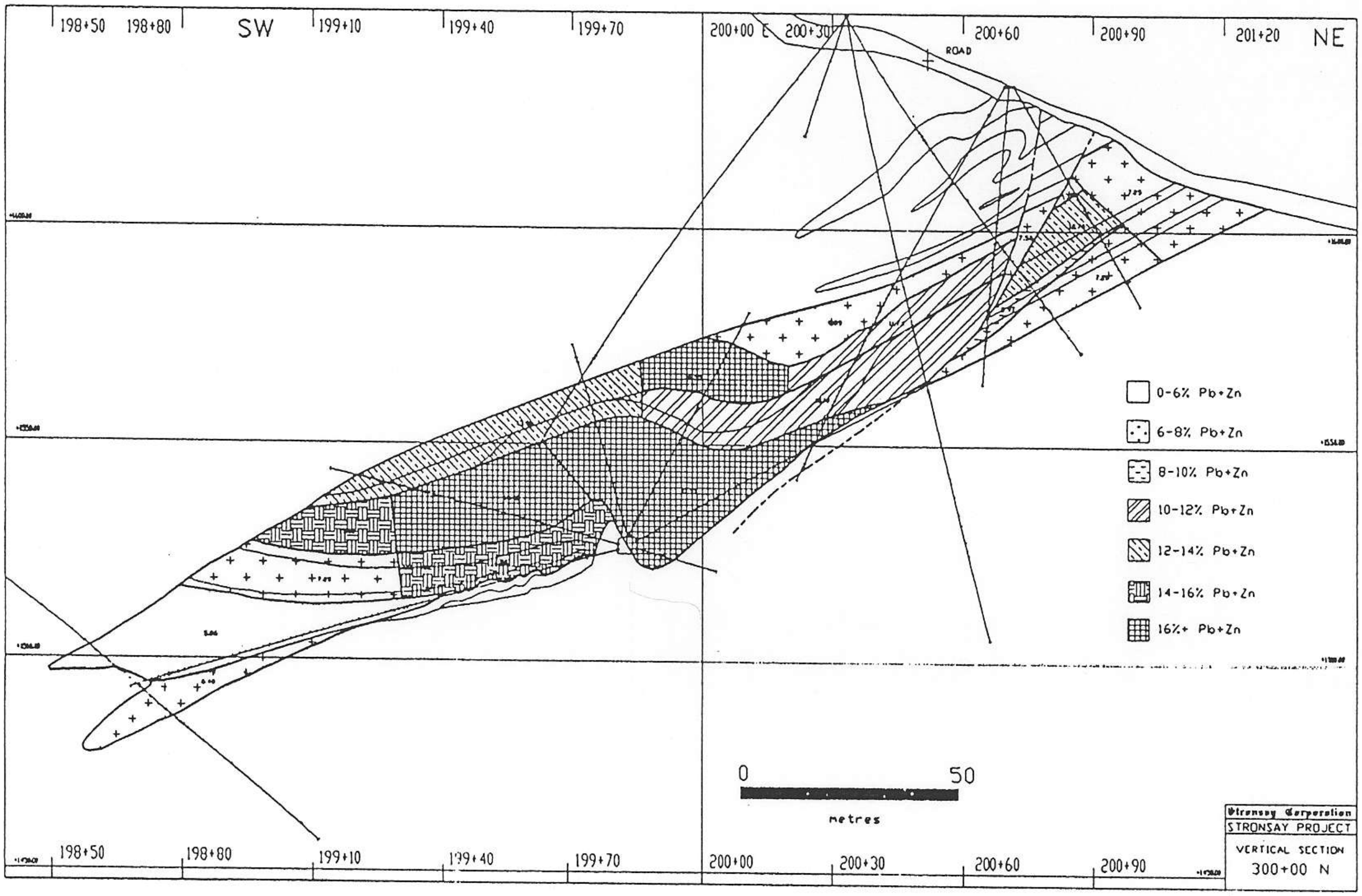

SW
199+10
199+40
199+70
200+00 E
200+30
200+60
200+90
201+20
NE
ROAD
RAISE #1
0-6% Pb+Zn
6-8% Pb+Zn
8-10% Pb+Zn
10-12% Pb+Zn
12-14% Pb+Zn
14-16% Pb+Zn
16%+ Pb+Zn
0
50
metres
Stronsay Corporation
STRONSAY PROJECT
VERTICAL SECTION
300+90 N
199+10
199+40
199+70
200+00
200+30
200+60
200+90

Figure 47



198+50
198+80
SW
199+10
199+40
199+70
200+00 E
200+30
200+60
200+90
201+20
NE
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
300+00 N
198+50
198+80
199+10
199+40
199+70
200+00
200+30
200+60
200+90

Figure 48

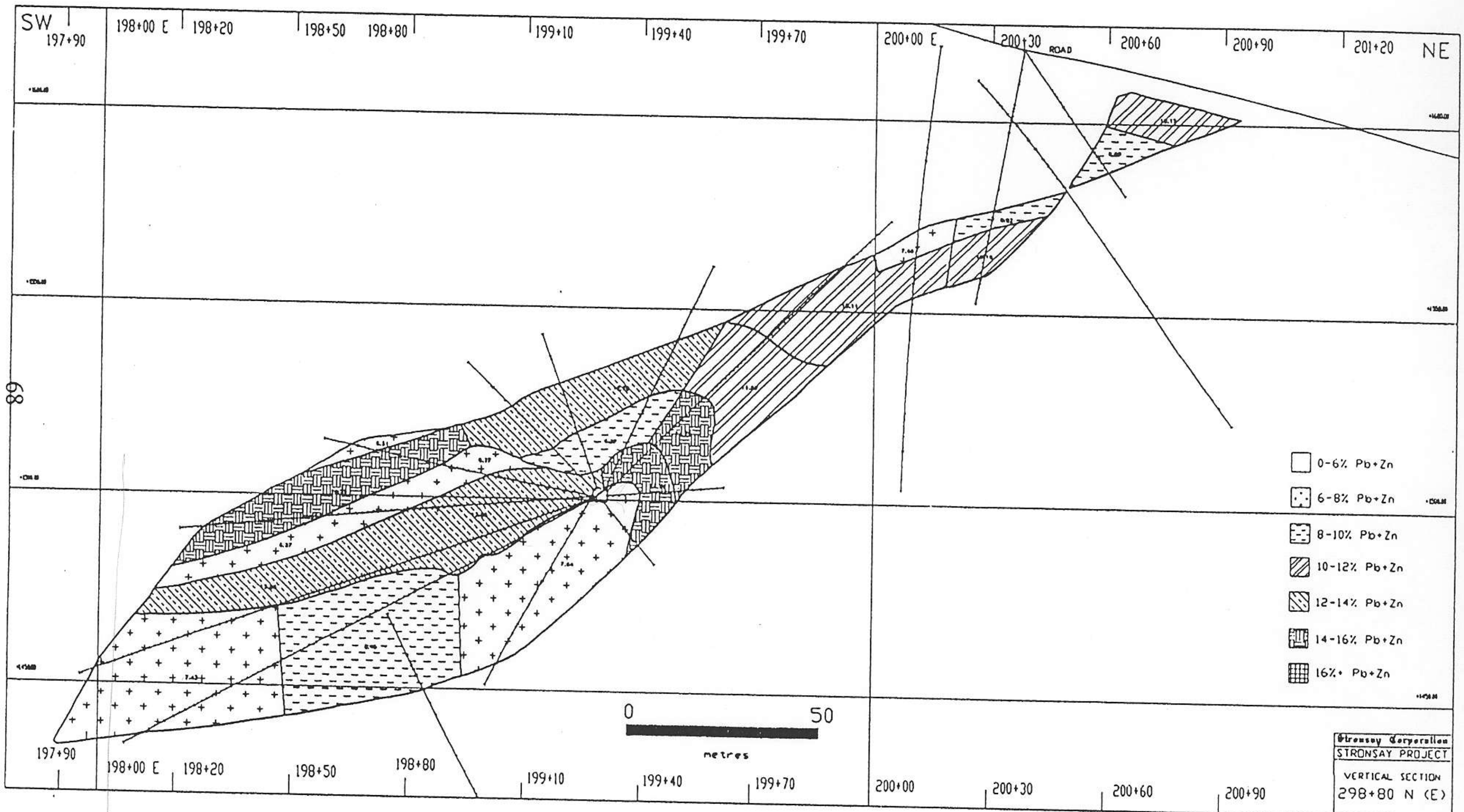
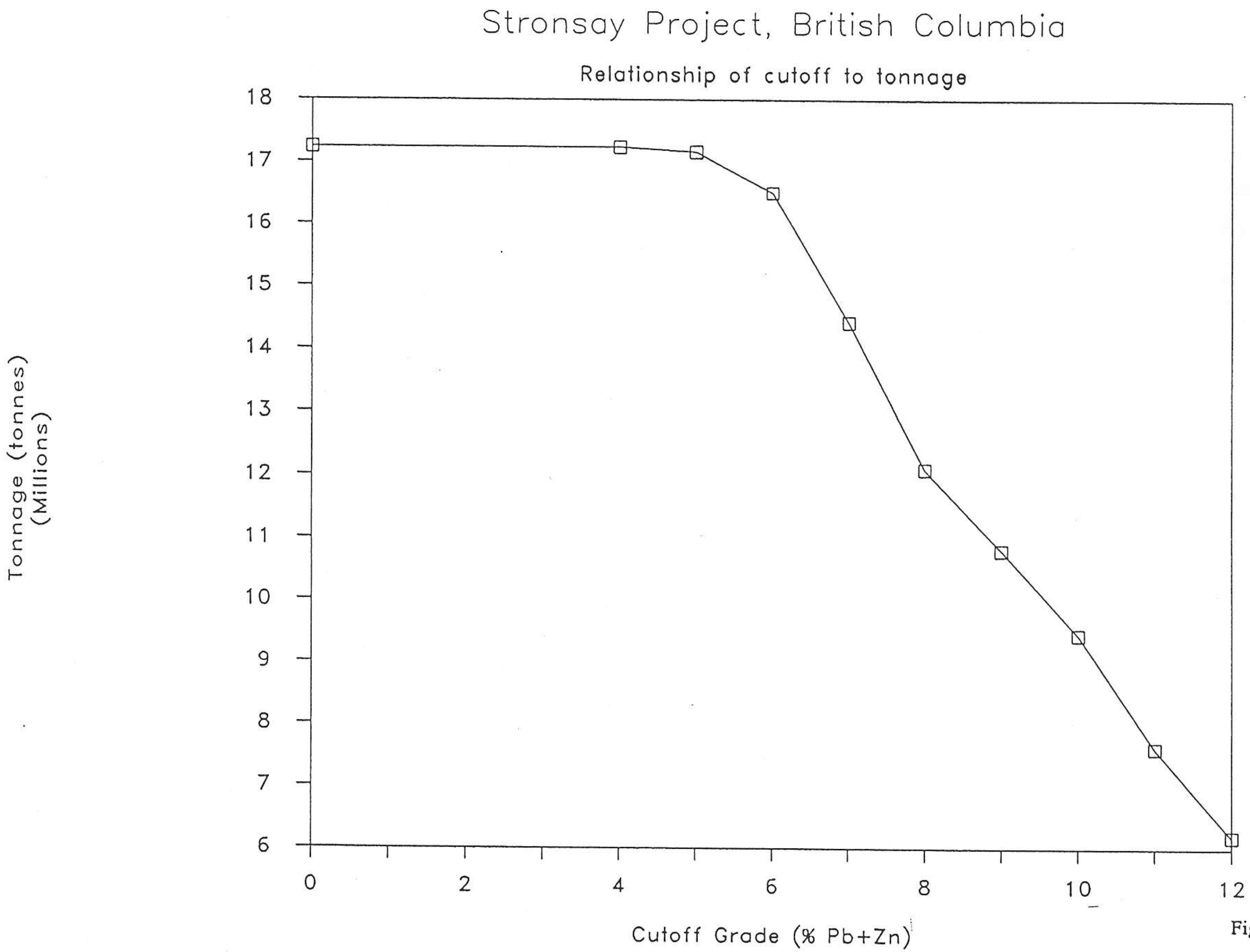

SW
197+90
198+00 E
198+20
198+50
198+80
199+10
199+40
199+70
200+00 E
200+30
ROAD
200+60
200+90
201+20
NE
0-6% Pb+Zn
6-8% Pb+Zn
8-10% Pb+Zn
10-12% Pb+Zn
12-14% Pb+Zn
14-16% Pb+Zn
16%+ Pb+Zn
0
50
metres
Stronsay Corporation
STRONSAY PROJECT
VERTICAL SECTION
298+80 N (E)

Figure 49



Stronsay Project, British Columbia
Relationship of cutoff to tonnage
Tonnage (tonnes)
(Millions)
18
17
16
15
14
13
12
11
10
9
8
7
6
0
2
4
6
8
10
12
Cutoff Grade (% Pb+Zn)

Figure 50

TABLE VI

STRONSAY CORPORATION

STRONSAY PROJECT, BRITISH COLUMBIA

Comparison of new reserve to old reserve, 0% Pb+Zn cutoff
Undiluted geological reserve, from Cyprus Anvil, "1983 unfaulted" interpretation

Entire Deposit (304+00N in north to 295+50N in south) as defined by Cyprus Anvil

Cross Section	Pb %	Zn %	Ag g/t	Pb+Zn %	ABOVE CUTOFF tonnes	Pb tonnes	Zn tonnes	AG grams	(Pb+Zn) tonnes
295+50N	2.08	7.21	38	9.29	1,792,068	37,275	129,208	68,098,584	166,483
296+40N	1.49	6.57	34	8.06	1,923,767	28,741	126,341	64,516,904	155,082
297+50N	2.07	6.90	43	8.97	6,837,904	141,401	471,786	293,931,389	613,187
298+50N	1.97	8.28	50	10.25	4,954,170	97,812	409,992	247,309,414	507,805
299+20N	1.90	8.06	47	9.95	5,804,112	110,079	467,609	273,554,395	577,688
300+00N	2.13	8.46	49	10.59	4,389,880	93,706	371,372	215,895,020	465,078
301+30N	2.19	7.58	47	9.77	6,119,327	134,111	463,819	285,084,133	597,930
302+50N	3.37	9.16	65	12.53	1,383,477	46,570	126,792	90,197,577	173,362
303+00N	3.73	10.73	74	14.46	656,395	24,491	70,421	48,256,495	94,912
303+50N	3.61	8.31	50	11.92	360,934	13,028	30,006	18,123,532	43,034
304+00N	4.41	10.72	71	15.13	327,076	14,415	35,056	23,279,608	49,471
TOTAL OLD	2.15	7.82	47.1	9.97	34,549,110	741,630	2,702,401	1,628,247,051	3,444,031

Portion within strike length drilled by Stronsay (303+60N to 298+50N) but as calculated by C.A.M.C

Cross Section	part	Pb %	Zn %	Ag g/t	Pb+Zn %	ABOVE CUTOFF tonnes	Pb tonnes	Zn tonnes	AG grams	(Pb+Zn) tonnes
295+50N	0.0	2.08	7.21	38.0	9.29	0	0	0	0	0
296+40N	0.0	1.49	6.57	33.5	8.06	0	0	0	0	0
297+50N	0.0	2.07	6.90	43.0	8.97	0	0	0	0	0
298+50N	0.6	1.97	8.28	49.9	10.25	2,914,218	57,537	241,172	145,476,126	298,709
299+20N	1.0	1.90	8.06	47.1	9.95	5,804,112	110,079	467,609	273,554,395	577,688
300+00N	1.0	2.13	8.46	49.2	10.59	4,389,880	93,706	371,372	215,895,020	465,078
301+30N	1.0	2.19	7.58	46.6	9.77	6,119,327	134,111	463,819	285,084,133	597,930
302+50N	1.0	3.37	9.16	65.2	12.53	1,383,477	46,570	126,792	90,197,577	173,362
303+00N	1.0	3.73	10.73	73.5	14.46	656,395	24,491	70,421	48,256,495	94,912
303+50N	1.0	3.61	8.31	50.2	11.92	360,934	13,028	30,006	18,123,532	43,034
304+00N	0.0	4.41	10.72	71.2	15.13	0	0	0	0	0
TOTAL OLD		2.22	8.19	49.8	10.41	21,628,343	479,522	1,771,190	1,076,587,279	2,250,712
TOTAL NEW*		2.38	8.75	51.5	11.13	18,044,767	429,465	1,578,917	929,305,501	2,008,383

** The new calculation has been totaled on the basis of ignoring weathering, pillars etc. in order to be comparable to the old calculation as much as possible*

TABLE VII

STRONSAY CORPORATION

STRONSAY PROJECT, BRITISH COLUMBIA

Comparison of new reserve to old reserve, 9% Pb+Zn cutoff
Undiluted geological reserve, from Cyprus Anvil, "1983 unfaulted" interpretation

Entire Deposit (304+00N in north to 295+00N in south) as defined by Cyprus Anvil

Cross Section	Pb %	Zn %	Ag g/t	Pb+Zn %	ABOVE CUTOFF tonnes	Pb tonnes	Zn tonnes	Ag grams	(Pb+Zn) tonnes
295+50N	2.08	7.21	38	9.29	1,792,068	37,275	129,208	68,098,584	166,483
296+40N	3.12	9.10	64	12.22	185,400	5,783	16,876	11,943,900	22,658
297+50N	2.95	8.37	57	11.32	3,712,451	109,440	310,673	210,045,441	420,113
298+50N	2.91	11.32	70	14.23	2,202,120	64,032	249,369	153,468,450	313,401
299+20N	2.72	9.74	60	12.46	3,084,534	83,990	300,280	186,237,477	384,270
300+00N	3.08	11.30	68	14.38	2,267,650	69,876	256,214	153,676,600	326,091
301+30N	2.75	8.98	57	11.73	4,060,315	111,793	364,529	229,750,830	476,322
302+50N	3.62	10.68	74	14.31	1,014,800	36,756	108,419	75,444,759	145,175
303+00N	3.84	11.05	76	14.90	637,090	24,491	70,421	48,256,495	94,912
303+50N	3.90	8.97	54	12.87	334,474	13,028	30,006	18,123,532	43,034
304+00N	4.41	10.72	71	15.13	327,076	14,415	35,056	23,279,608	49,471
TOTAL OLD	2.91	9.54	60.1	12.45	19,617,978	570,880	1,871,050	1,178,325,676	2,441,930

Portion within strike length drilled by Stronsay (303+60N to 298+50N) but as calculated by C.A.M.C

Cross Section	part	Pb %	Zn %	Ag g/t	Pb+Zn %	ABOVE CUTOFF tonnes	Pb tonnes	Zn tonnes	Ag grams	(Pb+Zn) tonnes
295+50N	0.0	2.08	7.21	38.0	9.29	0	0	0	0	0
296+40N	0.0	3.12	9.10	64.4	12.22	0	0	0	0	0
297+50N	0.0	2.95	8.37	56.6	11.32	0	0	0	0	0
298+50N	0.6	2.91	11.32	69.7	14.23	1,295,365	37,666	146,688	90,275,559	184,354
299+20N	1.0	2.72	9.74	60.4	12.46	3,084,534	83,990	300,280	186,237,477	384,270
300+00N	1.0	3.08	11.30	67.8	14.38	2,267,650	69,876	256,214	153,676,600	326,091
301+30N	1.0	2.75	8.98	56.6	11.73	4,060,315	111,793	364,529	229,750,830	476,322
302+50N	1.0	3.62	10.68	74.3	14.31	1,014,800	36,756	108,419	75,444,759	145,175
303+00N	1.0	3.84	11.05	75.7	14.90	637,090	24,491	70,421	48,256,495	94,912
303+50N	1.0	3.90	8.97	54.2	12.87	334,474	13,028	30,006	18,123,532	43,034
304+00N	0.0	4.41	10.72	71.2	15.13	0	0	0	0	0
TOTAL OLD		2.97	10.06	63.2	13.03	12,694,227	377,601	1,276,557	801,765,252	1,654,158
TOTAL NEW*		3.07	10.45	64.3	13.52	10,798,886	331,526	1,128,484	694,368,370	1,460,009

** The new calculation has been totaled on the basis of ignoring weathering pillars etc. in order to be comparable to the old calculation as much as possible*

05-Jun-91 G. Jilson

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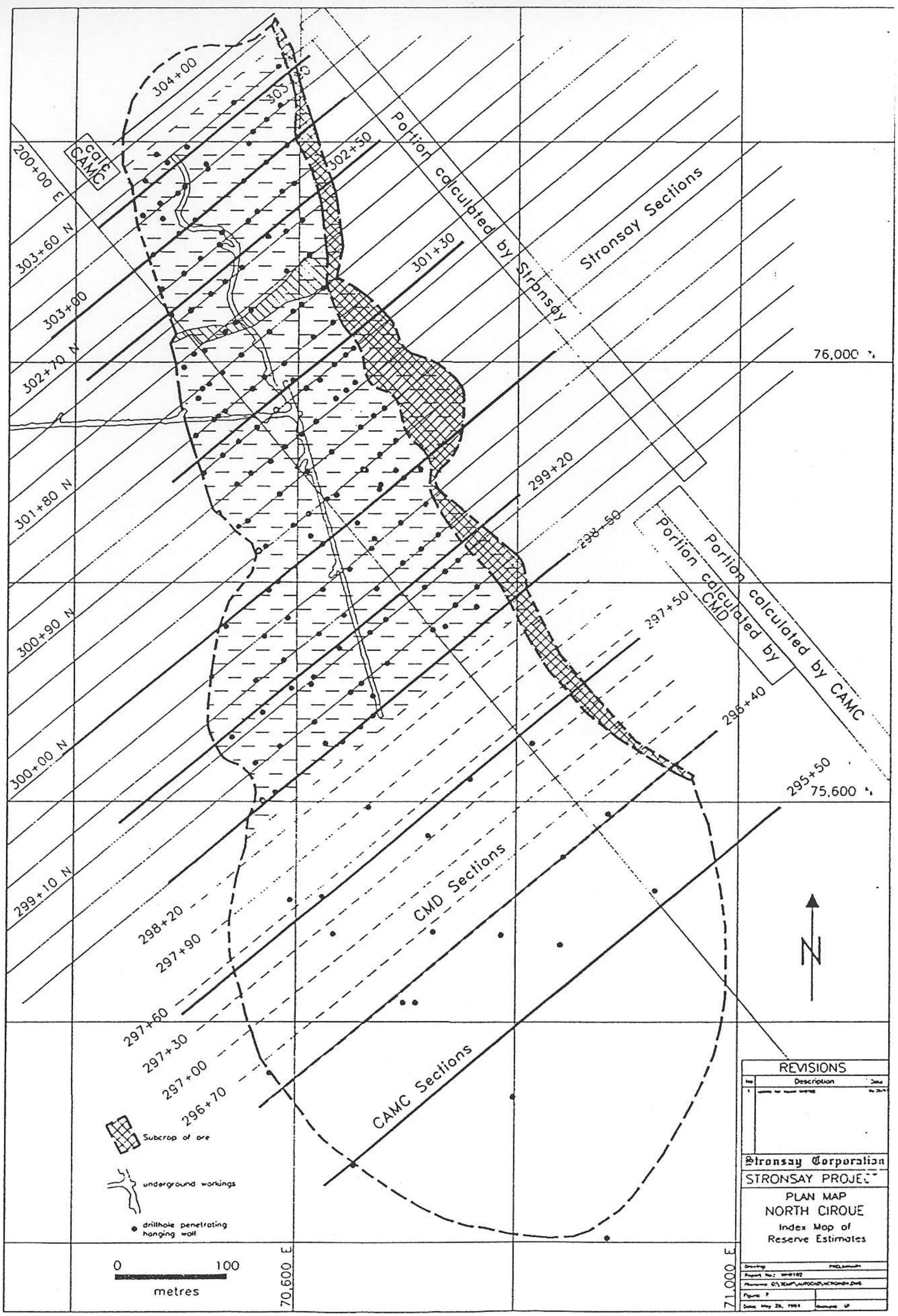


304+00
303+50
302+50
301+30
299+20
298+50
297+50
296+40
295+50
calc CAMC
200+00 E
303+60 N
303+00
302+70 N
301+80 N
300+90 N
300+00 N
299+10 N
Portion calculated by Stronsay
Stronsay Sections
76,000 N
75,600 N
Portion calculated by CMD
Portion calculated by CAMC
CMD Sections
CAMC Sections
298+20
297+90
297+60
297+30
297+00
296+70
Subcrop of ore
underground workings
drillhole penetrating hanging wall
0
100
metres
70,600 E
71,000 E
REVISIONS
Description
Stronsay Corporation
STRONSAY PROJECT
PLAN MAP
NORTH CIRQUE
Index Map of
Reserve Estimates

Figure 51

in mineralization is 8,479 m. This amount of drilling represents approximately 1 m. of drilling in mineralization to define each 2,000 tonnes of ore. Nearly half of the ore is included within reserve blocks with less than 2,000 tonnes inferred from 1 m. of drilling in mineralization and approximately 1,000,000 tonnes in blocks with less than 1000 tonnes inferred from 1 m. of drilling in mineralization. Much of this material represents the ore within 20 m. of the North and South headings which is generally very well constrained and closely sampled. The overall drill density has provided very good definition of the ore lens structure and of the grade distribution within the lens. It is reasonable to conclude that the bulk of the ore in the part of the North Cirque deposit investigated by the advanoed exploration program falls collectively into the proven plus probable category. A small component of the ore near the edges in the weak probable or possible category. In total 1.06 million tonnes falls into this later category at a 0% cutoff, 25% of that is tied up in the likely crown pillar; at a 9% cutoff 500,000 tonnes in total is assigned to this category of which 313,000 tonnes is not tied up in the likely crown pillar.

The new calculation of the tonnage and grade of the entire barite-sulphide body from 298+50N to 303+60N is thought to be the best available estimate using all the new data and should be adopted as the geological reserve for this part of the North Cirque deposit.

The geological reserves of the southern portion of the North Cirque deposit, that is, south of 298+35N, can be obtained by subtracting the CAMC reserves of the northern portion from the total CAMC reserves, as reassessed by Curragh in 1986, as shown in Table VIII.

TABLE VIII
SOUTHERN PORTION OF THE NORTH CIRQUE DEPOSIT

		Tonnes	% Pb	% Zn	g/t Ag
Total	CAMC reserves	34,549,110	2.15	7.82	47
Less:	298+35-303+75N	21,628,343	2.22	8.19	40
	Section 304+00N	327,076	4.41	10.72	71
	Balance	12,593,691	1.97	7.12	42

Canadian Mine Development (CMD) has done an independent assessment of the geological reserve of the south part of the North Cirque deposit (as delineated in figure 51). The CMD work covered the 210 m. strike length from section 298+20 to 296+40. Table IX compares this calculation to the reserve of CAMC recalculated for the same 210 m. strike length

TABLE IX
COMPARISON OF CMD AND CAMC ESTIMATES FOR
PART OF THE SOUTH PORTION OF THE NORTH CIRQUE DEPOSIT

	Tonnes	% Pb	% Zn	% Pb+Zn	g/t Ag
CMD	11,849,000	2.05	7.84	9.89	47
CAMC	10,225,000	1.98	7.13	9.11	44

These results and the comparison of Stronsay's and CAMC's results for section 298+50N show that CAMC's estimate of the southern portion may be conservative and are the best available estimate for the remainder of the deposit.

The geological reserves of the southern portion of North Cirque are likely close to that estimated by CAMC, based on current knowledge, but may be reduced by underground exploration as has been demonstrated in the northern part of the deposit. These reserves, although less well-defined than those in the northern part of the deposit, can be classified as Probable Reserves.

The total geological reserve for the North Cirque deposit is estimated to 30.2 million tonnes averaging 10.33% Pb+Zn as shown in Table X below.

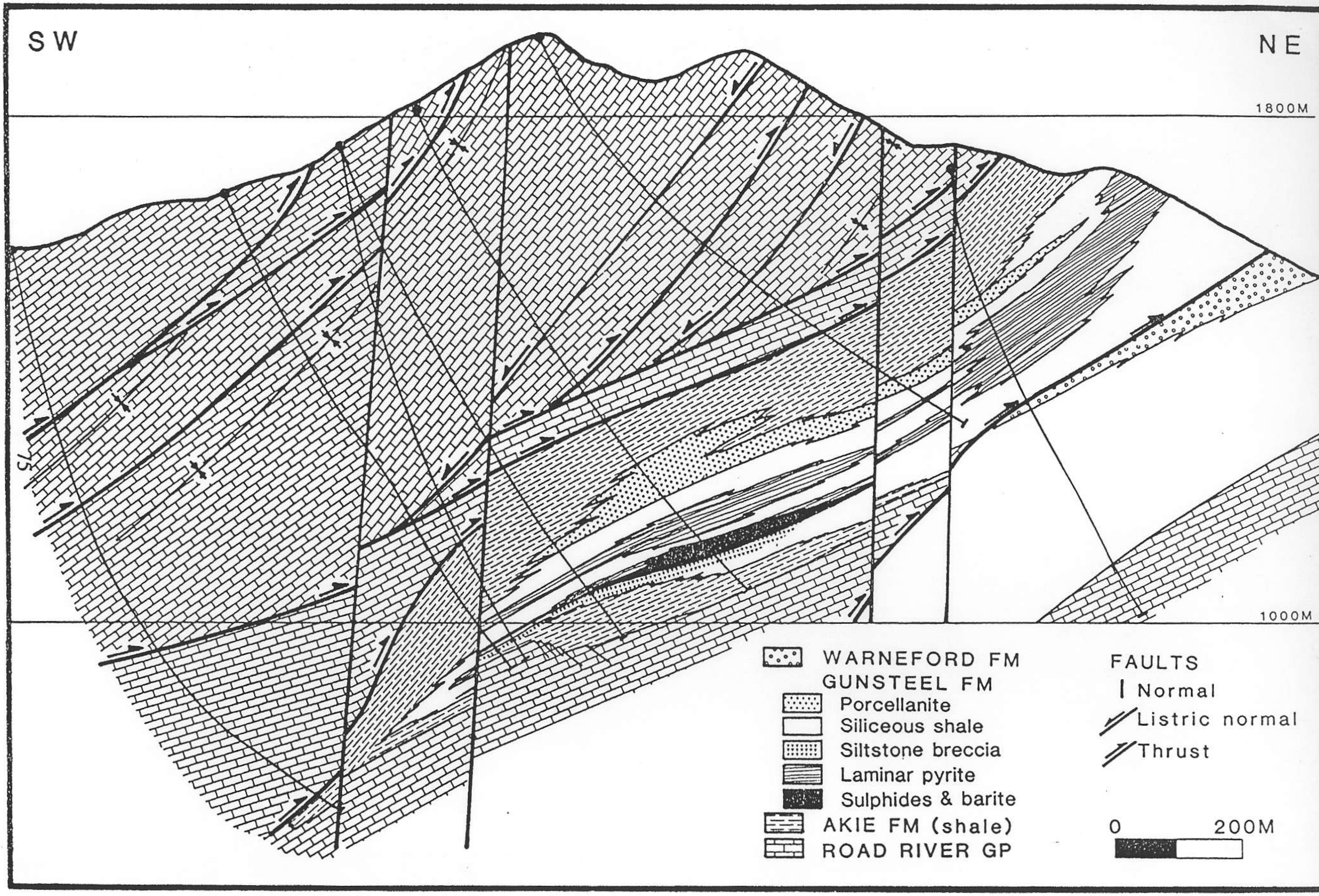
TABLE X
TOTAL GEOLOGICAL RESERVE NORTH CIRQUE DEPOSIT

Source	Portion	Tonnes	% Pb	% Zn	% Pb+Zn	g/t Ag
Stronsay	298+35 to 303+75	17,244,496	2.38	8.76	11.14	52
CAMC	north of 303+75	327,076	4.41	10.72	15.13	71
CAMC	south of 298+35	12,593,691	1.97	7.12	9.09	42
	Total Deposit	30,162,263	2.23	8.10	10.33	47.8

30165263

SOUTH CIRQUE

The South Cirque deposit occurs 1 km south of North Cirque (figure 6). The deposit is not well known as it has only been intersected in six drill holes. The ore types and host rocks are similar to those of the North Cirque deposit but the ores tend to be more calcareous than those of North Cirque. The stratigraphic and structural position of South Cirque is similar to North Cirque (figure 52). The sulphide barite mineralization is interpreted to occur in an area of approximately 195,000 m² in the bedding plane. True thicknesses of mineralization varies from



rtical cross section 283+00 of the South Cirque Deposit

4 to 29 m. averaging approximately 18m. Due to the fact that drill spacings are very large, and undoubtedly beyond any reasonable range of influence, a proper polygonal or sectional reserve calculation has not been carried out for South Cirque. The reserve had been previously estimated at 10 million tonnes with potential for an additional 10 million tonnes updip. Tables XIa and XIb show the known drillhole information for South Cirque for all mineralization (types 1 to 5 inclusive), and for pyritic mineralization (types 4 and 5) respectively. The quantity of mineralization can be estimated by calculating an average thickness, multiplying by the area of sulphide mineralization in the bedding plane and by a density of 4.3 tonnes/m^3. The grade can be estimated by length weighting all the drill intercepts. The results show a total tonnage of mineralization of 15 million tonnes of which 8.5 million tonnes is higher grade pyritic mineralization.

The area up dip from the known mineralization is untested by drilling and has potential for further barite sulphide mineralization as indicated on figure 6. Down dip from the deposit are scattered, thin, high grade intersections associated with a siltstone breccia unit. These scattered intersections have not been included in the above calculations. One notable intersection in that area returned over 45% Pb+Zn over 1.8m.

The South Cirque mineralization should be considered possible ore. Currently there are no development plans for this area. Further exploration from surface is not likely thus future exploration would be by an extension of the lower North Cirque workings. Haulage of the ore during production would most likely be through North Cirque to the mill location currently in the development plan.

CONCLUSIONS

The north portion of the North Cirque deposit has been well defined by drilling carried out in 1990 and 1991. This program has defined 17.2 million tonnes of mineralization averaging 2.38% lead, 8.76% zinc and 52 g/t silver. Most of this material can be considered to be in the undifferentiated proven plus probable category and a small component in the possible category. Additional less well defined mineralization to the south brings the total in-situ geological reserve for North Cirque to 30.2 million tonnes averaging 2.23% lead, 8.10% zinc and 47.8 g/t silver. Well defined high grade zones have been traced through the deposit.

RECOMMENDATIONS

The reserve quoted herein is based on a preliminary geological interpretation completed within one month of the completion of drilling and is itself a first pass quantification or ore tonnage and grade.

TABLE XIa

Stronsay Corporation
South Cirque, all sulphides and barite

section	hole	drilled thickness	true thickness	lead	zinc	silver
296+50	82C01	18.4	15.1	2.09	7.86	37.4
296+50	82C09	30.1	25.4	0.82	5.71	23.51
296+50	82C11	30.3	28.6	4.23	5.19	26.28
288+60	82C03	11.1	10.0	1.07	6.29	30.99
288+60	82C12	4.2	4.0	1.55	4.41	21
283+00	82C10	25.1	24.0	1.21	8.72	39.64
AVERAGE TRUE THICKNESS AND GRADE			17.8	1.94	6.41	29.6
TOTAL AREA OF ORE IN BEDDING			195,601	SQ. M		
TOTAL VOLUME			3,491,415	CU. M		
TOTAL TONNAGE			15,013,083	TONNES		

TABLE XIb

South Cirque drill results, pyritic ore

section	hole	drilled thickness	true thickness	lead	zinc	silver
296+50	82C01	7.9	6.5	3.27	13.69	62.9
296+50	82C09	7.7	6.5	1.16	11.29	46.2
296+50	82C11	15.9	15.0	3.33	5.34	15.9
288+60	82C03	11.1	10.0	1.07	6.29	30.99
288+60	82C12	4.2	4.0	1.55	4.41	21
283+00	82C10	19.9	19.0	1.10	10.16	45.55
AVERAGE TRUE THICKNESS AND GRADE			10.2	1.92	8.55	36.6
TOTAL AREA OF ORE IN BEDDING			195,601	SQ. M		
TOTAL VOLUME			1,988,610	CU. M		
TOTAL TONNAGE			8,551,024	TONNES		

Further work should begin with a careful reinterpretation of the geology of the deposit with particular emphasis in faulting. This interpretation should take advantage of recent detailed remapping in the headings and raises.

Further statistical and geostatistical analyses of the assay data and equal length composites should be carried out. Preliminary geostatistical analyses have been encouraging and it is considered likely that the data will prove amenable to kriging. A gridded point type of model should be calculated using kriging or another relevant grade interpolation methodology and the reserves requantified.

Now that the assay data base is complete barium and iron should be added to the suite of elements interpolated.

Further work should be carried out to characterize ore density.

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Jefferson, C.W., Kilby, D.B., Pigage, L.C., Roberts, W.J. (1983) The Cirque Barite-Lead-Zinc Deposits, Northeastern British Columbia, In *Short Course in Sediment-hosted Stratiform Lead-Zinc Deposits Mineralogical Association of Canada*, Short Course Handbook Volume 8, D.F. Sangster (ed), p. 121-140

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Pigage, L.C. (1986) Geology of the Cirque Barite-Zinc-Lead-Silver Deposits, northeastern British Columbia In J.A. Morin (ed.), *Mineral Deposits of Northern Cordillera*, Canadian Instutite of Mining and Metallurgy, Special Paper 37, p. 71-86

STRONSAY PROJECT

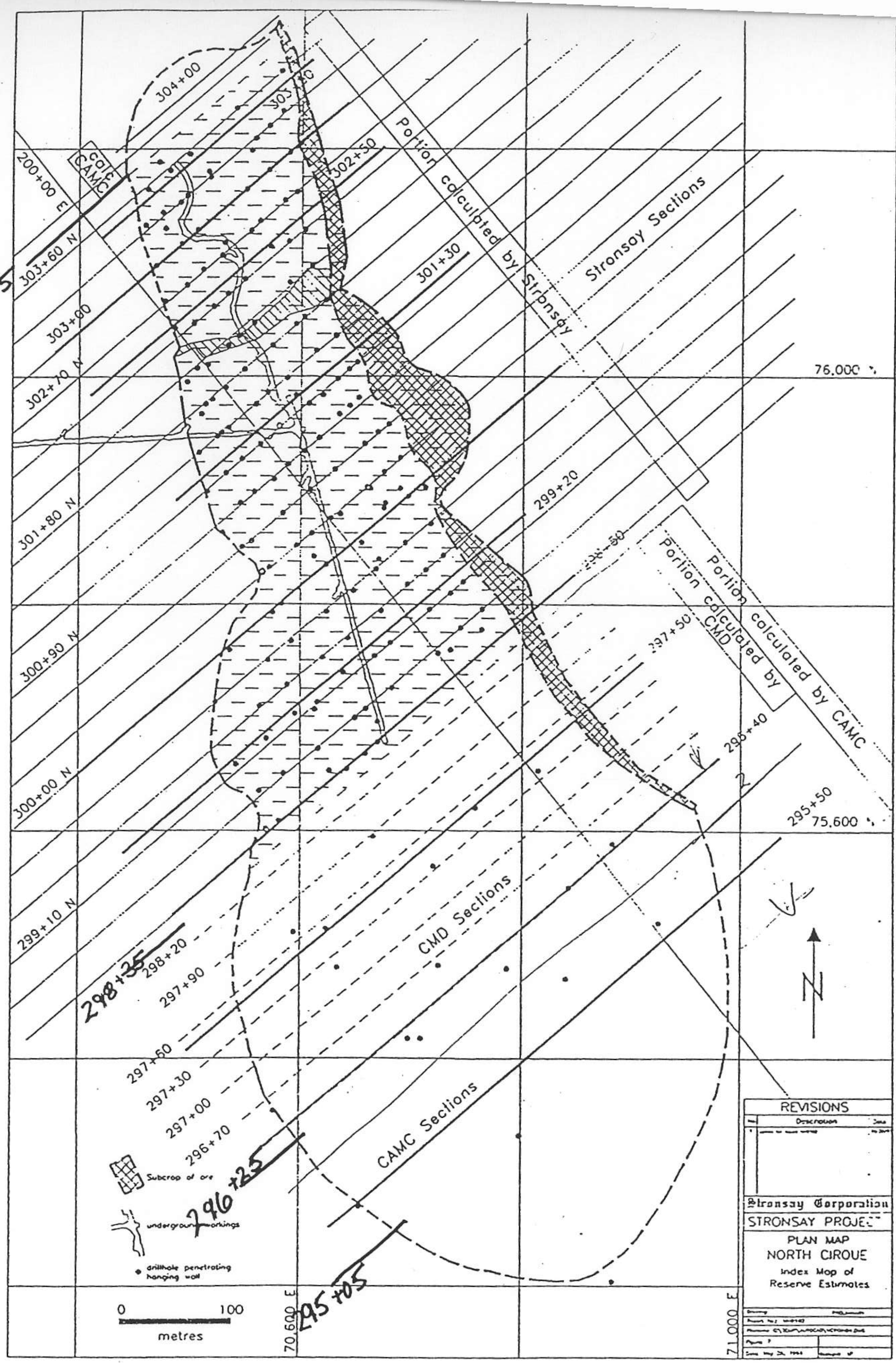
GEOLOGICAL RESERVES – 0% Cutoff														
Source	Area	Proven + Probable				Possible				Total				REFERENCE
		Tonnes	Pb%	Zn%	Ag g/t	Tonnes	Pb%	Zn%	Ag g/t	Tonnes	Pb%	Zn%	Ag g/t	
CMD	North Cirque 298+35N to 303+75N	17 263 730	2.37	8.81	51.7					17 263 730	2.37	8.81	51.7	MDP Vol1 pg 2–23 & CMD Jul 3/91 App1 pg 1
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	MDP Vol1 pg 2–24 & CMD Jul 3/91 App1 pg66
										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	CRI WH9102 pg 77 & GJ memo
	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				REFERENCE
		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	MDP Vol1 pg 2–24 & CMD Jul 3/91 App1 pg40
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	MDP Vol1 pg 2–24 & CMD Jul 3/91 App1 pg66
										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	

GEOLOGICAL RESERVES – 0% Cutoff														
Source	Area	Proven + Probable				Possible				Total				REFERENCE
		Tonnes	Pb%	Zn%	Ag g/t	Tonnes	Pb%	Zn%	Ag g/t	Tonnes	Pb%	Zn%	Ag g/t	
CRI	North Cirque 298+35N to 303+75N	17 241 496	2.38	8.76	51.6					17 241 496	2.38	8.76	51.6	MDP Vol1 pg 2–20 & CRI WH9102 pg 74
CAMC	North Cirque 295+05N to 298+35N	12 593 691	1.97	7.12	42.0					12 593 691	1.97	7.12	42.0	MDP Vol1 pg 2–21 & CRI WH9102 pg 74
										0	0.00	0.00	0.0	
	Total North Cirque	29 835 187	2.21	8.07	47.5	0	0	0	0	29 835 187	2.21	8.07	47.5	
CRI	Total South Cirque					20 013 083	1.94	6.41	29.6	20 013 083	1.94	6.41	29.6	CRI WH9102 pg 77 & GJ memo
	Total	29 835 187	2.21	8.07	47.5	20 013 083	1.94	6.41	29.6	49 848 270	2.10	7.40	40.3	

GEOLOGICAL RESERVES – 9% Cutoff														
Source	Area	Proven + Probable				Possible				Total				REFERENCE
		Tonnes	Pb%	Zn%	Ag g/t	Tonnes	Pb%	Zn%	Ag g/t	Tonnes	Pb%	Zn%	Ag g/t	
CRI	North Cirque 298+35N to 303+75N	10 792 775	2.98	10.29	62.8					10 792 775	2.98	10.29	62.8	MDP Vol1 pg 2–21 & CRI WH9102 pg 55
	North Cirque 295+05N to 298+35N									0	0.00	0.00	0.0	
										0	0.00	0.00	0.0	
	Total North Cirque	10 792 775	2.98	10.29	62.8	0	0	0	0	10 792 775	2.98	10.29	62.8	
	Total South Cirque									0	0.00	0.00	0.0	
	Total	10 792 775	2.98	10.29	62.8	0	0	0	0	10 792 775	2.98	10.29	62.8	

GEOLOGICAL RESERVES – 12% Cutoff														
Source	Area	Proven + Probable				Possible				Total				REFERENCE
		Tonnes	Pb%	Zn%	Ag g/t	Tonnes	Pb%	Zn%	Ag g/t	Tonnes	Pb%	Zn%	Ag g/t	
CRI	North Cirque 298+35N to 303+75N	6 857 985	3.37	11.55	70.3					6 857 985	3.37	11.55	70.3	MDP Vol1 pg 2–21 & CRI WH9102 pg 55
	North Cirque 295+05N to 298+35N									0	0.00	0.00	0.0	
										0	0.00	0.00	0.0	
	Total North Cirque	6 857 985	3.37	11.55	70.3	0	0	0	0	6 857 985	3.37	11.55	70.3	
	Total South Cirque									0	0.00	0.00	0.0	
	Total	6 857 985	3.37	11.55	70.3	0	0	0	0	6 857 985	3.37	11.55	70.3	



304+00
303+50
302+50
301+30
299+20
297+50
295+40
295+50
200+00 E
303+60 N
303+75
303+00
302+70 N
301+80 N
300+90 N
300+00 N
299+10 N
298+35
298+20
297+90
297+60
297+30
297+00
296+70
296+25
295+05
calc CAMC
Portion calculated by Stronsay
Stronsay Sections
Portion calculated by CAMC
Portion calculated by CMD
CMD Sections
CAMC Sections
76,000 N
75,600 N
70,600 E
71,000 E
Subcrop of ore
underground workings
drillhole penetrating hanging wall
0
100
metres
REVISIONS
Stronsay Corporation
STRONSAY PROJECT
PLAN MAP
NORTH CIRQUE
Index Map of Reserve Estimates

Figure 51

STRONSAY CORPORATION

STRONSAY PROJECT, BRITISH COLUMBIA

UNDILUTED GEOLOGICAL RESERVES - 30 m BLOCKS

19-Apr-91

SECTION	Pb+Zn	Pb	Zn	Ag	VOLUME	TONNAGE
298+50	10.00	1.72	8.27	45.28	423,144	1,789,899
298+80	10.30	2.06	8.24	47.82	334,704	1,415,798
299+10	10.29	1.97	8.32	46.17	394,746	1,669,776
299+40	10.54	1.84	8.70	48.88	312,213	1,320,661
299+70	11.23	2.03	9.19	50.08	258,279	1,092,520
300+00	12.00	2.55	9.45	56.38	254,670	1,077,254
300+30	12.29	2.27	10.02	56.66	242,631	1,026,329
300+60	12.40	2.62	9.79	54.10	259,410	1,097,304
300+90	10.70	2.22	8.49	50.04	278,154	1,176,591
301+20	10.38	2.25	8.12	48.36	280,623	1,187,035
301+50	11.53	2.62	8.90	50.25	228,942	968,425
301+80	11.33	2.69	8.64	49.70	170,700	722,061
302+10	10.95	2.48	8.47	52.93	135,378	572,649
302+40	12.24	3.08	9.16	60.70	160,608	679,372
302+70	12.03	3.56	8.48	60.66	125,895	532,536
303+00	11.69	3.68	8.02	66.87	97,122	410,826
303+30	14.70	4.58	10.13	68.59	86,490	365,853
303+60	16.27	4.13	12.14	73.44	37,551	158,841
TOTAL	11.18	2.37	8.81	51.69	4,081,260	17,263,730

SG: 4.23

STRONSAY CORPORATION

STRONSAY PROJECT, BRITISH COLUMBIA

IN-SITU RESERVES

SUMMARY

SECTIONS 303+60 to 298+50

	TONNAGE	Pb	Zn	Ag
O/C	1,014,715	2.73	8.46	54.95
U/C	241,526	1.94	7.86	41.94
U/C-CENTRE	96,300	3.42	11.48	63.10
STOPE	4,979,122	2.51	9.54	55.27
C & F	2,193,058	3.06	8.87	58.01
CROWN PILLAR	755,203	2.53	6.88	50.91
PILLARS	6,463,504	2.32	9.31	52.41
REMNANT	1,520,307	0.79	5.35	26.91
TOTAL	**17,263,736**	**2.37**	**8.81**	**51.70**

STRONSAY CORPORATION

STRONSAY PROJECT, BRITISH COLUMBIA

MINEABLE RESERVES @ 6% cut off.

SUMMARY

SECTIONS 303+60 to 298+50

Covers 303+75 to 298+35

	TONNAGE	Pb	Zn	Ag
O/C	1,067,903	2.60	8.04	52.22
U/C	241,526	1.94	7.86	41.94
U/C-CENTRE	96,300	3.42	11.48	63.10
STOPE	4,916,276	2.49	9.47	54.85
C & F	2,335,475	2.82	8.16	53.38
CROWN PILLAR	0	0.00	0.00	0.00
PILLARS	6,202,192	2.18	8.74	49.16
REMNANT	0	0.00	0.00	0.00
TOTAL	**14,859,672**	**2.42**	**8.84**	**51.90**

STRONSAY CORPORATION

STRONSAY PROJECT, BRITISH COLUMBIA

South Part of N Cirque.

SUMMARY OF MINEABLE RESERVES 6% cutoff

SECTION 298 + 20 TO 296 + 40

Cours 298+35 to 296+25

	TONNES		Pb %		Zn %		Ag g/t
298 + 20	1,437,343	@	1.92	:	8.96	:	52.9
297 + 90	1,620,016	@	2.13	:	9.60	:	53.4
297 + 60	1,687,835	@	2.34	:	8.36	:	51.7
297 + 30	1,706,019	@	2.50	:	6.73	:	44.8
297 + 00	1,249,897	@	2.48	:	7.18	:	48.3
296 + 70	1,300,593	@	1.71	:	7.24	:	41.2
296 + 40	816,641	@	2.31	:	8.06	:	51.0
TOTAL	**9,818,344**	**@**	**2.20**	**:**	**8.05**	**:**	**49.1**

Geological - 0% cutoff

		Pb	Zn	Ag
298+20	1647484	2.03	9.38	55.4
297+90	2103784	1.81	8.86	48.9
297+60	2031337	2.11	8.12	49.5
297+30	1995166	2.33	6.66	43.7
297+00	1761005	2.18	6.44	41.7
296+70	1405236	1.74	7.34	41.8
296+40	904471	2.40	8.12	51.8
	11848483	2.07	7.84	47.3

STRONSAY CORPORATION

STRONSAY PROJECT, BRITISH COLUMBIA

UNDILUTED GEOLOGICAL RESERVE, 0% PB+ZN CUTOFF

CROSS SECTION	NUMBER F BLOCKS	VOLUME (cu m)	TONNAGE (tonnes)	LEAD+ ZINC	LEAD (%)	ZINC (%)	SILVER (g/t)
30360N	6	36,687	158,217	16.20	3.86	12.34	70.5
30330N	14	86,607	371,232	13.31	4.30	9.00	65.8
30300N	7	95,970	408,362	11.93	3.47	8.45	61.9
30270N	18	126,387	538,362	12.21	3.59	8.62	61.1
30240N	12	159,054	675,539	12.01	3.04	8.97	59.2
30210N	13	136,362	578,133	10.97	2.51	8.46	51.8
30180N	8	171,252	726,041	11.68	2.67	9.02	51.9
30150N	10	224,763	949,315	11.23	2.47	8.76	51.2
30120N	17	279,732	1,180,742	10.35	2.31	8.04	48.0
30090N	15	276,114	1,166,235	10.97	2.31	8.66	51.1
30060N	13	260,325	1,106,504	12.48	2.64	9.84	54.5
30030N	17	242,877	1,029,823	11.94	2.30	9.64	55.4
30000N	18	252,837	1,069,241	12.25	2.66	9.59	57.9
29970N	13	254,148	1,071,112	11.29	2.08	9.20	50.4
29940N	20	309,279	1,304,059	10.56	1.97	8.59	49.6
29910N	26	399,978	1,690,327	10.01	1.87	8.14	45.4
29880N	18	335,916	1,417,514	10.33	2.07	8.26	47.4
29850N	22	426,366	1,800,738	9.93	1.82	8.11	45.3
TOTAL	267	4,074,654	17,241,496	11.14	2.38	8.76	51.6

using densities based on pulp S.G. averages by rock type
pulp averages reduced by 2% for porosity

08-Mar-91 G. Jilson

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STRONSAY CORPORATION
STRONSAY PROJECT, BRITISH COLUMBIA

UNDILUTED GEOLOGICAL RESERVE, 9% PB+ZN CUTOFF, CONTIGUOUS BLOCKS

CROSS SECTION	NUMBER OF BLOCKS	VOLUME (cu m)	TONNAGE (tonnes)	LEAD+ ZINC	LEAD (%)	ZINC (%)	SILVER (g/t)
30360N	3	31,188	135,094	17.22	4.00	13.22	72.1
30330N	8	75,420	324,203	13.79	4.06	9.73	63.3
30300N	5	81,843	350,215	12.87	3.60	9.27	62.5
30270N	9	95,409	408,846	13.22	3.88	9.34	66.9
30240N	6	114,594	489,764	13.34	3.37	9.97	68.2
30210N	6	78,786	339,757	12.97	3.22	9.75	63.7
30180N	6	121,386	520,323	13.64	3.36	10.27	62.7
30150N	7	150,009	640,182	13.26	2.99	10.27	62.8
30120N	10	169,998	726,352	12.51	2.85	9.66	59.0
30090N	6	132,522	569,883	14.77	3.18	11.59	68.6
30060N	9	196,983	844,086	13.65	2.93	10.71	60.6
30030N	10	170,238	728,204	13.76	2.82	10.94	65.1
30000N	11	156,459	672,549	15.48	3.51	11.96	75.2
29970N	8	157,869	674,029	13.98	2.93	11.05	63.2
29940N	10	167,832	720,198	13.06	2.79	10.28	65.7
29910N	15	237,039	1,011,540	11.71	2.33	9.39	54.2
29880N	10	189,882	813,269	12.05	2.47	9.59	57.9
29850N	11	191,760	824,282	12.25	2.39	9.86	57.1
TOTAL	150	2,519,217	10,792,776	13.27	2.98	10.29	62.8

using densities based on pulp S.G. averages by rock type
pulp averages reduced by 2% for porosity

STRONSAY CORPORATION

STRONSAY PROJECT, BRITISH COLUMBIA

UNDILUTED GEOLOGICAL RESERVE, 12% PB+ZN CUTOFF, CONTIGUOUS BLOCKS

CROSS SECTION	NUMBER OF BLOCKS	VOLUME (cu m)	TONNAGE (tonnes)	LEAD+ ZINC	LEAD (%)	ZINC (%)	SILVER (g/t)
30360N	3	31,188	135,094	17.22	4.00	13.22	72.1
30330N	8	75,420	324,203	13.79	4.06	9.73	63.3
30300N	4	72,306	310,961	13.25	3.65	9.59	63.9
30270N	9	95,409	408,846	13.22	3.88	9.34	66.9
30240N	6	114,594	489,764	13.34	3.37	9.97	68.2
30210N	6	78,786	339,757	12.97	3.22	9.75	63.7
30180N	2	41,616	180,264	18.72	4.35	14.37	86.3
30150N	4	98,982	424,805	14.75	3.32	11.44	69.7
30120N	2	54,015	232,977	16.53	4.27	12.25	78.2
30090N	4	93,264	402,526	16.65	3.66	13.00	77.2
30060N	3	105,885	458,651	17.08	3.39	13.69	73.6
30030N	4	93,360	403,432	16.97	3.36	13.61	77.6
30000N	7	135,762	585,291	16.08	3.68	12.40	79.0
29970N	3	63,087	272,906	18.16	3.56	14.60	80.3
29940N	4	78,375	339,060	15.88	3.12	12.76	75.9
29910N	4	70,797	306,431	15.66	3.15	12.51	74.6
29880N	7	151,770	650,241	12.28	2.45	9.83	57.9
29850N	9	137,454	592,775	12.91	2.53	10.37	60.5
TOTAL	89	1,592,070	6,857,984	14.92	3.37	11.55	70.3

using densities based on pulp S.G. averages by rock type
pulp averages reduced by 2% for porosity

TABLE 3-1
MINING RESERVE SUMMARY

Mining Method	Tonnes	Pb%	Zn%	Ag g/t
Sections 298 + 50 to 303 + 60				
Longhole Stoping				
Overcuts	1,067,903	2.60	8.04	52.2
Undercuts	241,526	1.94	7.86	41.9
Centre Drill Drifts	96,300	3.42	11.48	63.1
LH Blasting	4,916,276	2.49	9.47	54.9
Total Longhole Stoping	6,322,005	2.50	9.20	54.1
Pillars	6,202,192	2.18	8.74	49.2
Panel and Fill	2,335,475	2.82	8.16	53.4
Sub-Total	14,859,672	2.42	8.84	51.9
Sections 296+40 to 298+20				
Longhole Stoping	4,235,462	2.27	8.47	52.3
Pillars	4,097,782	2.16	8.04	49.6
Panel and Fill	1,485,100	2.14	6.86	38.4
Sub-Total	9,818,344	2.20	8.05	49.1
Total Mineable	24,678,016	2.33	8.53	50.8

3.2.3 Recovery

Recoveries were calculated for each mining area based on the method of extraction used. Losses due to the orebody geometry were already calculated as remnants in the in-situ reserve and, therefore, did not have to be taken into account in the recovery calculation.

The longhole stope overcut and undercut headings can be recovered completely. The recovery of the ore in longhole benching will be limited only by the ability of remote controlled Load-Haul-Dump (LHD) units to clean the stope floors. Technology is available to keep this loss to a minimum so a recovery factor of 98% was used.

Longhole pillar recovery will be more difficult but all stopes will be filled with a cemented backfill, which should permit a reasonably high recovery. The recovery assumed was 90%.

The panel and fill method is a variation of cut and fill. The only ore losses will be from irregular hangingwall or footwall contacts. A recovery of 98% was assumed.

3.2.4 Dilution

Dilution is expected from three sources: exposure to hangingwall shale, intentionally mined footwall waste that forms part of the stope, and from backfill. Internal waste has been included in the mineable tonnes and grade.

The top slice in the longhole stopes will be supported by an extensive rockbolting and screening program. Due to the nature of the hangingwall material, some dilution will occur and has been calculated at 0.5 metres for all stope hangingwall contacts. All foot wall waste that forms part of the stope has also been included as dilution.

The top slice in the pillars will be more difficult to control so 1.0 metres of waste was included as dilution for all exposed hangingwall contacts. Some backfill dilution from the previously filled stopes will occur. This has been included as 0.3 metres for all walls, at a specific gravity of 3.5 tonnes per cubic metre.

Dilution from the hangingwall in the panel and fill areas should be minimal and cut widths will be adjusted with experience. In this reserve 0.5 metres from all hangingwall contacts have been included. Fill dilution from floors and walls will be experienced in order to recover all the ore. A dilution of 7.5%

of the volume mined has been included at a specific gravity of 3.50 tonnes per cubic metre. The dilution in all cases is calculated at zero grade.

3.3 GEOTECHNICAL

3.3.1 General

John D. Smith prepared a report for Stronsay Corporation in January 1991 entitled Geotechnical Input for Feasibility Report on the Cirque Project. The report, which is reproduced in Appendix D, included the following conclusions and recommendations:

- The rock quality of the hangingwall and footwall formations adjacent to the ore contacts is quite poor, largely due to the thin bedding/shearing/foliation/cleavage planes and the amount of movement along them. Continuous support systems are needed.

- The ore is much more competent than the rocks but will also require spot artificial support in production and access openings.

- The orebody is located in a very low stress environment.

- Stope span estimates were obtained using three different beam design criteria. A span of 15 m is recommended for the feasibility report blast-hole stoping blocks and one of 7m is suggested for the one pass panel method. A sulphide roof beam is considered necessary in each case, varying from 3 m for the long-hole stopes to 2 m for the panel method. Artificial support of these beams is also considered necessary. Continuity of beam thickness could be a problem.

- Pillar mining methods should be reviewed with respect to undercutting ore blocks and fill strength requirements.

- Pillar stability is influenced by orientation relative to structure as well as dips of the contacts. Both of these must be taken into account at Stronsay.

- The weight recovery of tailings to backfill the production openings is estimated to be 54 tonnes per 100 tonnes of ore mined. This very high recovery requirement suggests that conventional, hydraulically placed backfill will prove to be unacceptable from the cost and operational standpoint.

- It might be possible to place close to 100% of the tailing underground, depending on compaction and porosity achieved by the placement method. Such dewatered fills require the least quantity of cementing agent to achieve the design strength.

3.3.2 Stope and Pillar Design

In an attempt to maximize the resource recovery, it was decided to reduce the longhole stope span from 15 m to 12m and eliminate the sulphide roof beam. The back will be supported with rebar bolts and screen, augmented where necessary by cable bolting. A comprehensive program of geotechhical investigation, including monitoring, is recommended particularly for the stope development and early production period in order to establish the most efficient roof support system, consistent with safety considerations.

Following unconfined compressive strength testing of whole core samples, John D. Smith examined CMD's revised mining layout. The results were set out in a letter report dated March 25, 1991, from which the following summary paragraph is extracted:

"Variable strength ore pillars with large variations in physical properties are anticipated at Stronsay, based on the laboratory testing done to date. The high barite content, low grade material, and the porous, weathered sulphide material will control the design of the pillars. The "worst case" situation would be to have pillar accesses driven in these much weaker materials, resulting in two small rib pillars composed of the poorest material.

In spite of this, plus taking the highest values for pillar loading, the calculated safety factors for 12m stopes and 12m pillars are quite acceptable up to 75% extraction, with the exception of small rib pillars in porous, weathered sulphides at 75% extraction where stability is expected to be marginal. In practice, only one or two pillars in a given panel would be subjected to the "worst case" loading situation and overall mine stability should be acceptable during pillar recovery."

3.3.3 Backfill Design

Deslimed mill tailings have become the commonest source of mine backfill material. At Stronsay, the combination of fine grinding and the need to fill all openings results in a shortage of coarse tailings. Possible solutions are either to supplement the coarse tailings with crushed waste rock for hydraulic placement, or to dry whole tailings to less than 15% moisture (paste) then either pump or haul them to the stopes. In all cases, cement must be added to backfill placed in primary stopes in order to achieve sufficient strength to permit mining of the intervening pillars.

Backfill testing and stability analysis was carried out by John D. Smith and reports were submitted to CRI on March 15 and 28, 1991 for 14 and 28 day strength tests, respectively. Tests were run at various cement ratios and pulp densities on four materials:

- whole tailings
- plus 12u tailings
- 85% > 12u tailings with 15% limestone crushed to minus 20 mm (3/4")
- 60% > 20u tailings with 40% < 20 mm limestone

Results showed that any of these materials could be used to achieve the required design strengths, that percolation rates were poor in every cemented mixture, and that cement requirements increased dramatically at higher moisture contents.

Options were reviewed in terms of practicality and costs. It was decided that paste fill was desirable due to minimal cement requirement, minimal surface tailings storage requirement and minimal cleanup underground (no surplus slurry water).

However, the cost of dewatering the tailings with pressure filters was very high, the cost of the paste pumping system was believed to be high, and trucking was impractical. Pending more detailed testwork and engineering on filtration and pumping of paste and fill, it was decided to base the development plan estimate on conventional hydraulic backfill.

Design parameters are as follows:

- 320 t/d (15%) crushed rock
- 1,810 t/d (85%) > 12u tailings
- 2,130 t/d backfill at 75% solids density
- 1:23 (4.25%) cement:fill ratio in primary stopes for a 28 day strength of 1,100 kPa

CALCULATIONS BY BELA BOGNAR

GATAGA-4.XLS

STRONSAY - GATAGA EVALUATION			gataga-4.xls		
	STRONSAY		GATAGA	LISHEEN 4,100 tpa	
	Case 1	Case 2	Case 1	MMC	Fluor
	No Leasing	No Leasing	No Leasing		Daniel
		Custom Milling	Custom Milling		
CONCENTRATOR C$	52,272,000	52,272,000	31,069,000		41,250,000
TOTAL CAPITAL INVESTMENT C$					
Preproduction	189,186,000	166,730,000	105,956,000		
Ongoing	8,000,000	8,000,000	8,000,000		
OPERATING COSTS C$/t:					
Mine	16.88	16.88	16.88	14.90	9.50
Mill	14.50	12.33	14.18	10.50	8.56
Custom Milling Fee		(1.85)			
Concentrate Handling	9.50	9.50	11.50		
Freight Backhaul	0.71	0.71	0.71		
General & Administration	4.66	4.66	4.66		1.29
Total	46.25	42.23	47.93		
NPV @15% Pre-Tax	(31,807,000)	10,481,000	7,890,000		
IRR Pre-Tax	11.3%	16.3%	16.4%		
NPV @15% After Tax	(50,426,000)	(21,817,000)	(15,856,000)		
IRR After Tax	8.2%	11.8%	11.3%		
Zn ~~RAISE~~ grade REQUIRED FOR 15% IRR	10.6%	9.5%	9.6%	after tax	
Zn Cash Breakeven Price US$/lb	0.47	0.45	0.48		
Zn Cash Breakeven Price + CAPEX	0.52	0.49	0.52		
ASSUMPTIONS COMMON TO ALL CASES					
Minable Reserves mt		24,327,000			
Grade Zn		8.5%			
Pb		2.3%			
Ag		51			
Recovery Zn		88.0%			
Milling Rate		4,000 tpd	1.431M tpa		
Treatment Charge Zinc US$/DMT		190	basis US$ 1,000/mt price participn 0.13 up, 0.10 down		
			Payment 85%		
Zn Price US$/lb		0.59			
Pb Price US$/lb		0.25			
Ag Price US$/oz		5.00			
Exchange Rate US$/C$		1.25			
Equity Financing		100%			

There is clear advantage in sharing facilities for both projects: capital investment for concentrate transport facilities are halved and operating costs for running the mill are reduced. For Stronsay, there is the added benefit of the custom milling fee. Stronsay Case 1 and Gataga Case 1 are very similar in terms of economic results. However, due to the custom milling fee Gataga cash unit operating costs are higher.