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Cyprus Gold Australia Corporation

PROJECT A-78-60
DEV349

FEASIBILITY STUDY

RETENTION LICENCE APPLICATION

OCEANA

ZEEHAN

TASMANIA

P INGHAM JUNE 1988 REPORT 574
PART 2

CYPRUS

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Executive Summary

The Oceana project is located approximately 2 kilometres south of the township of Zeehan on the west coast of Tasmania. Oceana is within an old lead-silver mining area and was most recently mined between 1954 and 1960. The mine closed due to high water inflows and low metal prices.

The deposit is found in two distinctive zones, epigenetic and stratiform within the Gordon Limestone. The epigenetic zone occurs between two major faults and is found at surface to a depth in excess of 200 metres. The stratiform zone occurs to the south of the epigenetic zone and at a depth in excess of 200 metres. The values of silver, lead and zinc are significant in both areas. A full description of the geology of the deposit is given in the accompanying geological report.

The geological inferred resource at Oceana is estimated as 2,465,000 tonnes at 9.4%Pb, 4.0%Zn and 75g/t Ag.

The epigenetic zone is proposed to be mined by open pit methods to a depth of 70 metres with a stripping ratio of 6.9:1. The remaining reserve is proposed to be accessed by decline from the open pit and mined by the underground method of mechanised cut and fill, although the ground conditions may prove too bad for this mining technique. The decline is

proposed to be extended south to allow access to the underground stratiform deposit where mechanised cut and fill methods would again be used for extraction. There is expected to be a large flow of water into the workings which will make mining difficult and expensive.

It is proposed to treat the ores at a plant on site and produce lead and zinc concentrates. This decision was based on preliminary testwork at Roseberry that showed the Oceana ore was not compatible with the standard Roseberry flotation circuit. The concentrates would be shipped to smelters for final treatment. The throughput would be at a rate of 1000 tonnes of ore per day or 365,000 tonnes per year. The recoveries assumed for lead, zinc and silver were 92%, 73%, and 85% respectively. These recoveries are based on industry standards, as little metallurgical testwork was conducted, and are regarded as being on the optimistic side.

The future outlook of the prices of lead, zinc and silver are that the prices are unlikely to improve significantly over the next five years. The prices used in the study were lead AUS$950/tonne, zinc AUS$1350/tonne and silver AUS$8.00/ounce.

Economic modelling of the project shows that a discounted cashflow rate of return of zero is obtained at the above metal prices. This result is obtained even with what is regarded as optimistic mining and metallurgical recoveries.

The revenue from the sales would need to increase by in excess of 30% for the project to become economically viable with a rate of return of 15%.

Without the high grade section of the underground ore in the epigenetic zone, (which may be impossible to mine due to intense alteration and weathering of the ground), the revenue from sales would need to increase by in excess of 90% for the project to become viable.

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OCEANA FEASIBILITY STUDY

C O N T E N T S

FIGURES

PLATES

ENCLOSURES

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Project Location

5 cm

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1.0 Introduction

1.1 Location

The Oceana project area is located approximately 2 kilometres south of the small township of Zeehan on the west coast of Tasmania. The Emu Bay Railway and a sealed road connect Zeehan with the Port of Burnie, located 140 kilometres to the north.

The two lane bitumen road between Zeehan and Strahan passes through the project area. Access to the Oceana deposit is by a small unpaved track which is only accessible with four-wheel drive vehicles.

Figure 1 illustrates the property location.

1.2 Tenure

The current tenure status is illustrated in Figure 2 and the Retention Licence application area is shown. The total area applied for is 7.93 km^2. Within the application area lies Mining Leases, 60M/77 (held by Electroytic Zinc) 39M/77 and 38M/77 (held by Enraght-Moony).

1.3 Scope of Study

The scope of this study is to provide a feasibility study of Oceana lead-silver-zinc deposit based on the results from the exploration work done on Exploration Licence 4/78. The study is part of the Retention Licence Application by Cyprus Gold Australia Corporation. The geology of the area is covered in detail in a report accompanying this study.

With the metallurgical work from Roseberry giving poor results and the high volumes of water expected in the mine, it was decided to evaluate a stand-alone treatment plant with a throughput of 1000 tonnes per day. This rate was chosen as the time period that water requires to be pumped is

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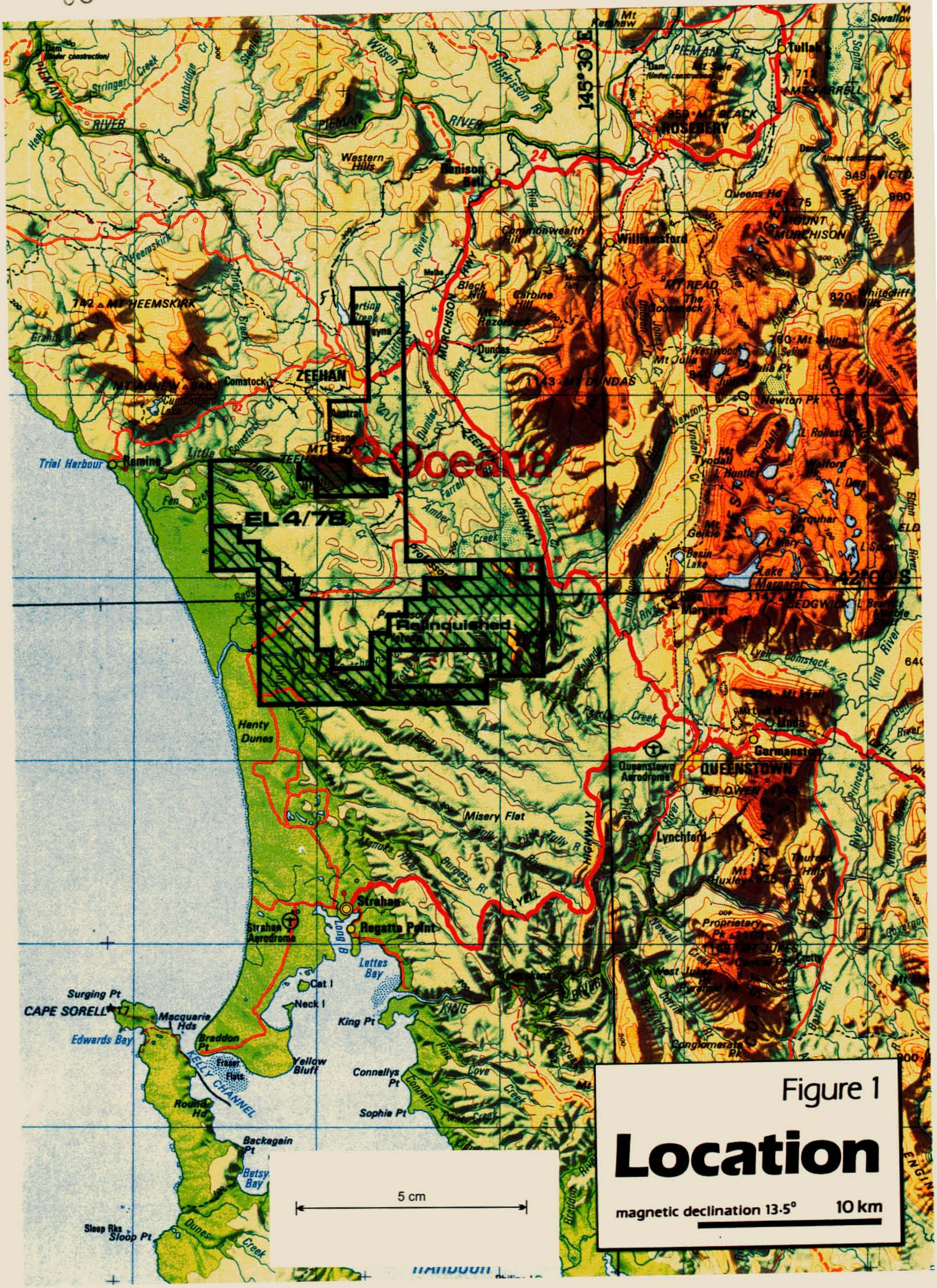

Figure 1
Location
magnetic declination 13·5° 10 km

Cyprus Gold Australia Corporation

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OCEANA

Retention Licence Application Area

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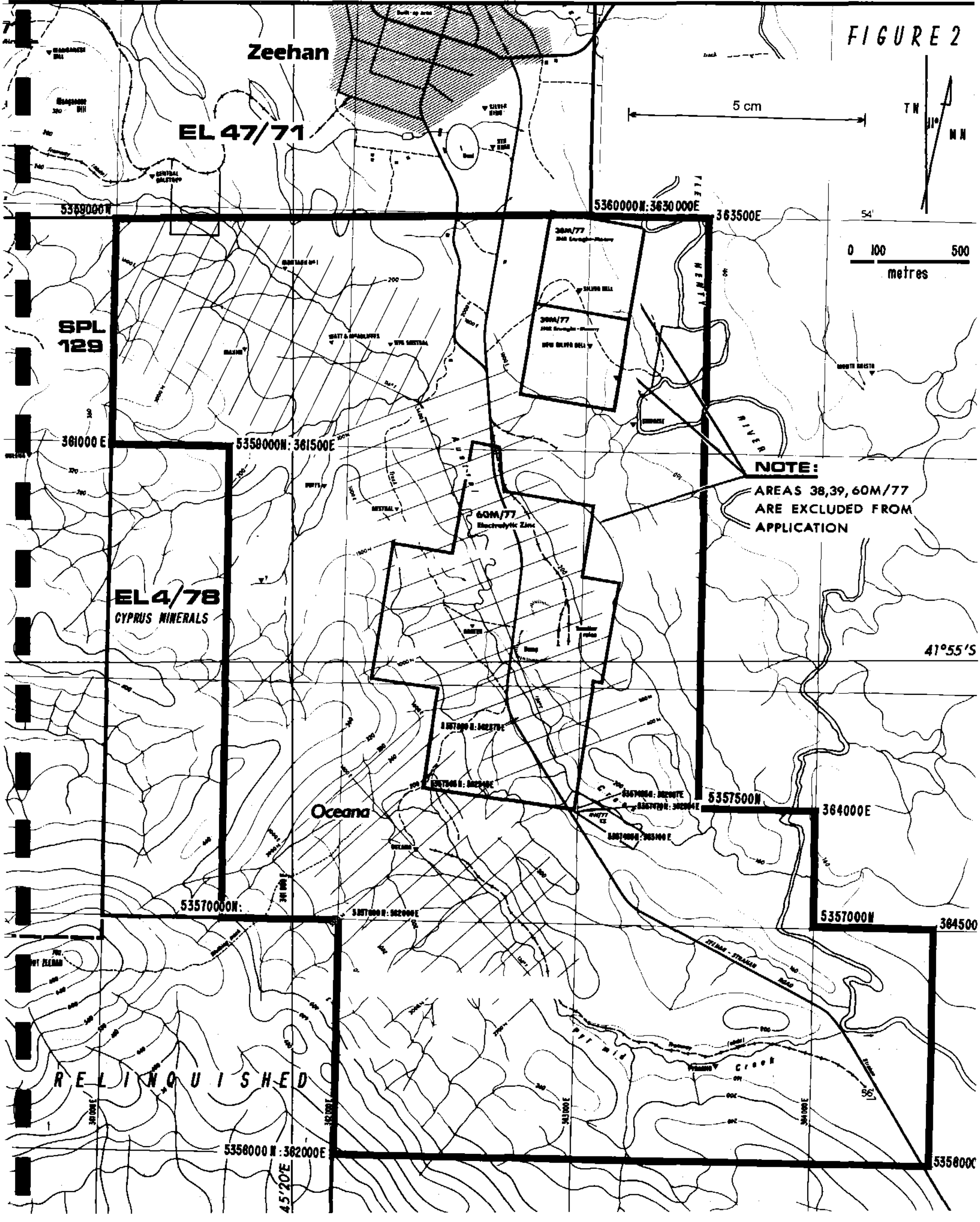

minimised.

2.0 History

The first discovery of argentiferous galena in the area was in 1882 and subsequently led to the discovery of lead mineralisation at the Mount Zeehan mine site. The main period of mining activity in the Zeehan district was between 1890 and 1918 when 278,000 tonnes of silver-lead ore were mined from a total of 31 different mines (King and Blissett, 1968) containing 193,000 tonnes of lead and 26,000,000 ounces of silver. The following figure details the production by year. King and Blissett give the reason for the abandoning of many of the larger mines in this period, was a decline in the ore shoots rather than depressed ore values, water problems, or increased costs of progressively deeper mines.

Within the proposed Retention Licence area there are five notable mines that were in operation at the end of the last century. A brief history of each mine follows.

2.1 Montagu No. 1 Mine

Exploration on the property commenced in 1887 and in 1897 an adit was driven on the course of O'Rourke's lode, yielding 200 tonnes of first class ore from 3 parallel veins of galena to a depth of 6.5 metres (Blissett, 1962).

In 1896 a shaft was sunk by Montagu No. 1 Silver Mining Co. N.L. and at 30 metres level a cross cut was driven to the orebody with unsatisfactory results. No significant work has been done on the lease area since. It is estimated that 235 tonnes of galena ore were produced containing 115 tonnes of lead and 1500 ounces of silver.

2.2 Watt and McAuliffe's and North Austral Mines

Prospecting started in 1887 and in 1895 ore was mined from a shallow adit and shaft (Blissett, 1962), but water affected

Production of silver-lead ore

from the Zeehan district 1890-1918

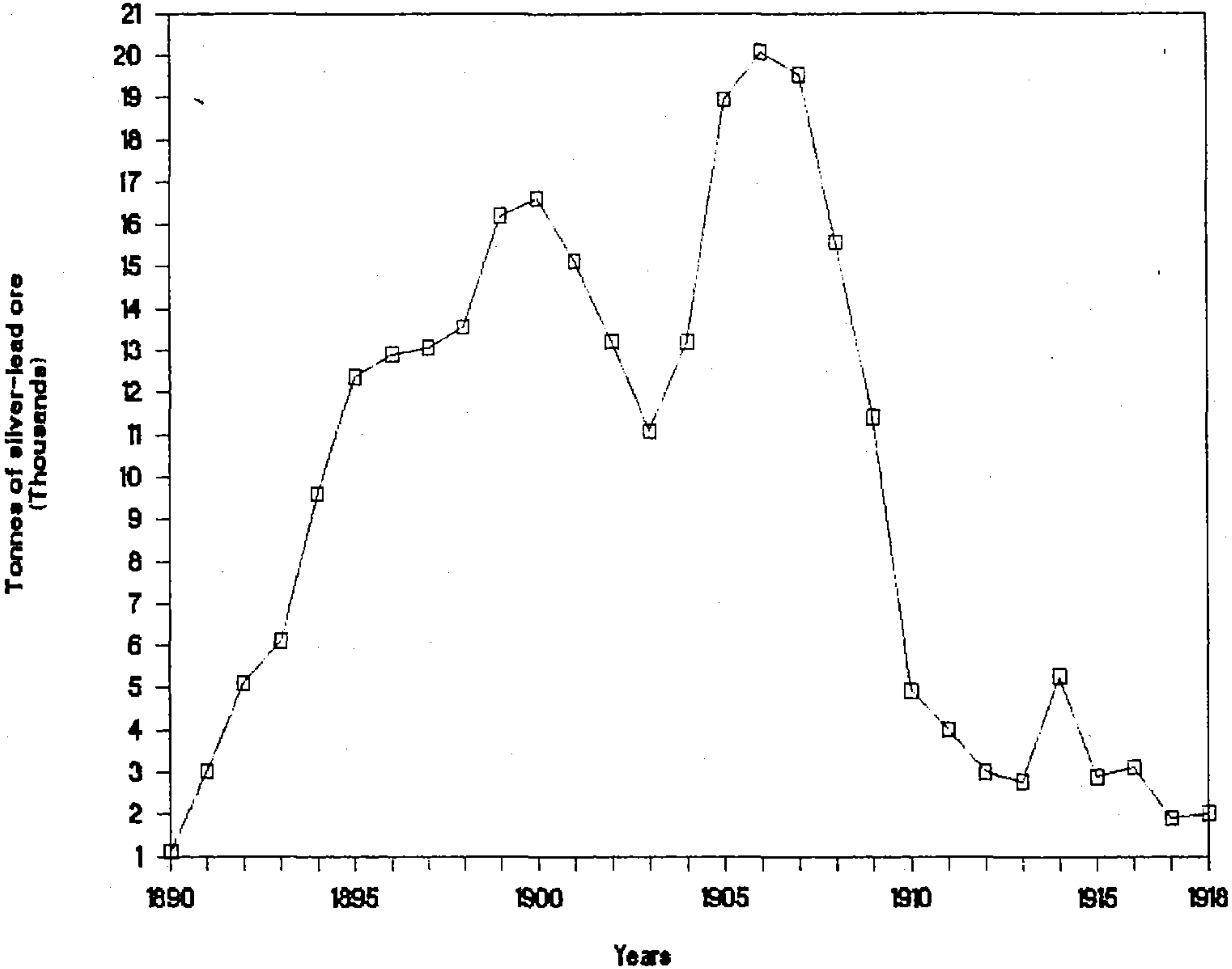

FIGURE 3

the operation. No. 1 shaft was sunk to a depth of 18 metres and No. 2 shaft to a depth of 12 metres.

Ore production was 530 tonnes between 1901 and 1905. Contained in this galena ore was 255 tonnes of lead and 50,000 ounces of silver. From 1906 the mine was taken over by Austral Valley Mining Co. and production from these mines may have been included in the figures for the total company.

2.3 Austral Valley Mine

This area was first explored in 1887. Some adits were driven but minimal production resulted due to the heavy inflows of water. The lodes were small and scattered and discouraged deep mining. Within the valley a flux quarry was mined to supply the smelter. Between 1907 and 1913, 875 tonnes of galena ore, 4290 tonnes of flux, 100 tonnes of sphalerite and 9 tonnes of pyrite were produced. The metal content is estimated at 810 tonnes of lead, 50 tonnes of zinc, and 33,000 ounces of silver.

In 1946, Zeehan Exploration drilled 5 holes near the flux quarry but results were generally disappointing.

2.4 Oceana Mine

In 1887, the first leases were taken up and later in 1892 Oceana Silver Mining Company was formed to mine the orebody below the gossan (Blissett, 1962). Production was at a rate of 41 tonnes of oxidised ore per day which was railed to the Argenton Smelting Co 2.4 kilometres away. This production ceased in 1893 when the smelter ceased production.

During the period, three prospecting shafts were connected by 215 metres of drives along the line of the lode. The old main shaft was started and later completed to 44 metres with levels being driven at 10 metre intervals and later at 24 metre intervals (Jack, 1960). The total production by 1893 was 1015 tonnes of ore assaying 39% Pb and 14 1/2 ozs Ag.

The workings were all to the north of the main shaft which later collapsed and all mining ceased.

There are two open cuts, Hall and Fox, to the south of the main shaft but there are no records available of production (Jack, 1960). The open cuts which were in the oxidised ore, were also mined by Zeehan Mines Pty. Ltd.

In 1946, North Broken Hill and Broken Hill South recommenced exploration in the area. After two years of exploration only the Oceana deposit appeared to be prospective. In 1950 Zeehan Mines Pty Ltd was formed jointly between North Broken Hill and Broken Hill South to reopen Oceana Mine. Considerable development was done between 1952 and 1953 and in 1954 the mine commenced production. The production continued until 1960 when it was closed due to low metal prices and the high inflows of water of 11,365m^3 per day. The production from the mine over its life is detailed in the table following.

<u>Oceana Mine Production</u> (Blissett 1962)

Year	Ore (tonnes)	Concen-trates (tonnes)	Silver (Kg)	Lead (tonnes)	Zinc (tonnes)
To 1893		1016	411	396	
1898		525	Est.283	Est.254	
1909		23	Est. 9	Est. 9	
1925		Est. 21	7	8	13
		1585	710	667	13
1954	8105	1241	1169	907	
1955	15932	2838	2307	2064	
1956	15690	2394	2022	1760	
1957	18204	2106	2178	1989	
1958	23026	3457	2712	2473	
1959	30823	4351	3799	3192	
1960	18456	2929	2538	2088	
	130,236	19,316	16,725	14,473	
	130,236	20,901	17,435	15,140	13

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2.4.1 Oceana Workings (1954-1960)

The following description of the workings is drawn extensively from Jack, 1960, description.

The development and mining, carried out by Zeehan Mines Pty. Ltd, was quite extensive within the ore zones south of the Mine fault and to the north of South Oceana fault. A circular 3.7 metre diameter shaft with two winding compartments was sunk to 61 metres (Jack, 1960). It was noted by Jack that the first 30 metres were within weak, partly decomposed limestone. From 61 metres to the final depth of 198 metres the shaft was converted to a four compartment, rectangular shaft. This was to facilitate extra space for rising mains.

The levels mined were No. 1 (46m/150ft), No.2 (92m/300ft), No. 3 (128m/420ft), No. 4 (137m/450ft), No. 5 (165m/540ft), No. 6 (195m/640ft). No. 4 level was mined solely for a sump.

Crosscuts were mined west from the shaft to intersect the orebody. The shaft is 37 metres from the ore bodies on No. 1 level. This distance drops to 24 metres on No. 6 level. The ore body dips in an easterly direction towards the shaft and becomes narrower in width at the bottom (see figures following).

The mining was done by flat-back cut and fill stoping. Deslimed tailings were used as fill in the stopes.
On No. 2 and No. 3 levels an attempt was made to mine north through the Mine fault but development met large inflows of water and mud. One attempt to mine through on No. 2 level was abandoned after 50 metres north of the shaft; while on No. 3 level two attempts were made and abandoned after 52 metres and 88metres.

It is understood that the cut-off grade was approximately 11% Pb and the mill recovery grade of 96%. The Ore Reserve remaining at 11% Pb cut-off, 1.2m minimum mining width, and

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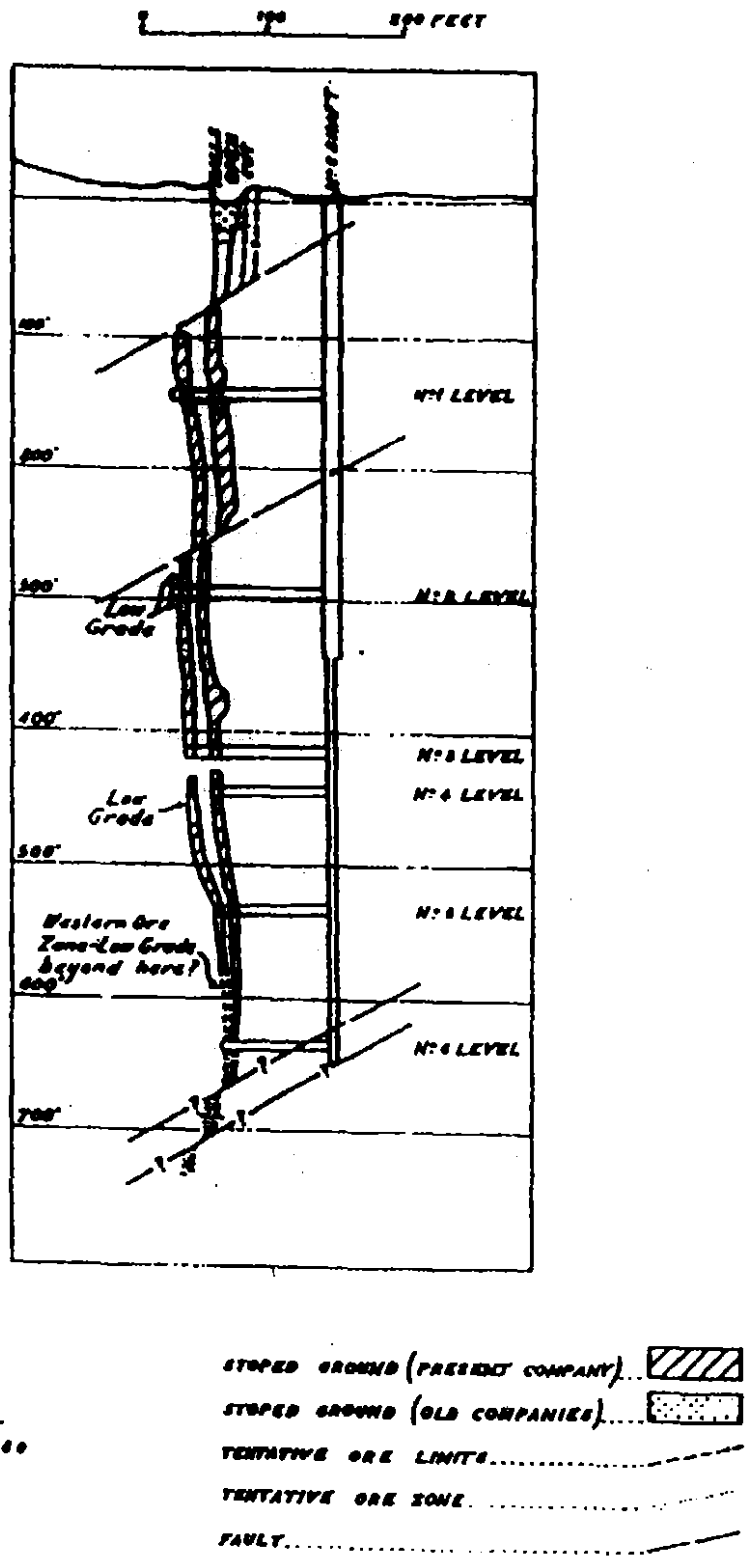
OCEANA MINE
SHAFT CROSS SECTION
(LOOKING NORTH)



R. JACK
GEOLOGIST
AUGUST [illegible]

FIGURE 4

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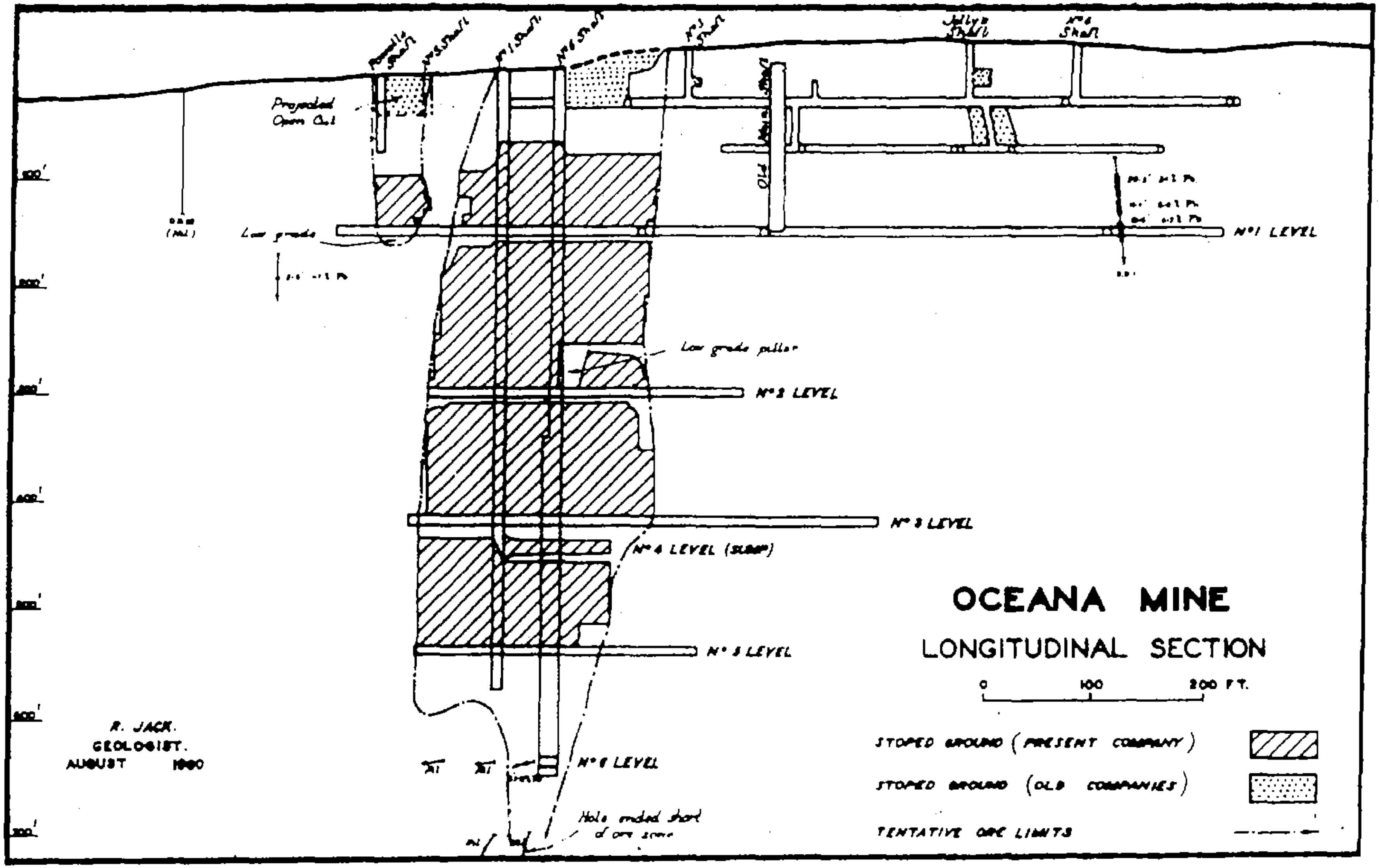

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FIGURE 5

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3.6 t/m^3 s.g. were calculated by Zeehan Mining to be 9,400 tonnes probable ore and 3,050 tonnes of possible ore.

The 480 m^3/hr was pumped from the mine by three four-stage turbine pumps and three Pomona pumps.

3.0 Ore Reserve

The Geological Resource of Oceana has been calculated from the drill holes drilled between 1979 and 1983 and the costeans. The Resource is divided up between ore types, epigenetic and stratiform, and within the epigenetic zone by depth. The Resource classification according to the recent Discussion Draft of October 1987 of the Reporting of Identified Mineral Resources and Ore Reserves, Report of the Joint Committee of the Australasian Institute of Mining and Metallurgy and the Australian Mining Industry Council is an Inferred Resource. It is noted that the recent changes to the ore classification system does not allow for this classification of resource to be determined as a mining reserve as the possible ore classification has been withdrawn.

The deposit is divided up into three areas.

Area 1. The epigenetic ore between the Oceana Fault and the Mine Fault to a depth of 100 metres. (see Enclosures 1, 2, 3, 4 and 5)

Area 2. The epigenetic ore between Oceana Fault and the Mine Fault from a depth of 100 metres to the bottom of the ore zone. (see enclosures 2, 3, 4 and 5)

Area 3. The stratiform ore south of the Mine Fault at a depth of between 200 metres and 300 metres. (see enclosure 2 and 6)

The Geological Inferred Resource of the three areas is:

1. Area 1 (Epigenetic zone)

0-50 metre	578,000 tonnes @ 4.3%Pb, 1.9%Zn, 31g/t Ag
50-100 metre	515,000 tonnes @ 6.7%Pb, 4.1%Zn, 59g/t Ag

2. Area 2 (Epigenetic zone)
below 100 metres
622,000 tonnes @ 13.2%Pb, 5.7%Zn, 112g/t Ag

3. Area 3 (Stratiform zone)
750,000 tonnes @ 12.0%Pb, 4.0%Zn, 89g/t Ag

Total 2,465,000 tonnes @ 9.4%Pb, 4.0%Zn, 75g/t Ag

These reserves are based on a specific gravity of 4.5t/m^3. The basis of the specific gravity is discussed in the geological report.

The mining methods proposed to mine the deposit at Oceana are detailed in the next section. The resultant mining reserve allowing for those areas not mineable and the expected recoveries, is:-

Area 1	
0-50 metres	550,000 tonnes @ 4.3%Pb, 1.9%Zn, 31g/t Ag
50-70 metres	200,000 tonnes @ 6.7%Pb, 4.1%Zn, 59g/t Ag
Area 2	
	440,000 tonnes @ 13.2%Pb, 5.7%Zn, 112 g/t Ag
Area 3	680,000 tonnes @ 12.0%Pb, 4.0%Zn, 89 g/t Ag
Total	1,870,000 tonnes @ 9.5%Pb, 3.8%Zn, 74g/t Ag

4.0 Mining

4.1 Mine Design

The geology and mineralisation of the mining area is described in the geological report. The Mining Areas 1 and 2 lie between the Oceana Fault and Mine Fault in the epigenetic zone and Area 3 lies south of the Mine Fault in the stratiform zone. Area 3 is south of the previous workings in the stratiform zone. The host rock is the Gordon Limestone.

4.1.1 Area 1

Due to the bad ground conditions in the Epigenetic zone near surface, only open pit mining can be contemplated. A Preliminary pit design of the reserve went to a depth of 70 metres. The design assumed a bench height of 10 metres at 60° face slopes and 5 metre berms giving an overall slope angle of 43°. This relatively low angle reflects the instability of the Gordon Limestone when drained.

Due to the poor grade of the zone it was assumed that there was no dilution and a recovery of ore of 95%. This is an optimistic assumption as there may be considerable dilution.

The mining will be made difficult with the high volumes of ground water expected as well as the high rainfall. A considerable amount of the ore will be rippable but the mining costs reflect the adverse ground conditions.

It is expected that a water course will have to be put in to direct the water from Mount Zeehan around the western edge of the open pit.

The stripping ratio of the open pit is 6.9:1 waste:ore. The mining method will be conventional open cut mining with excavators and trucks with dozers to rip the ore. It is expected that the northern wall of the pit will have to be drilled and blasted from the outset as it is in Moina Sandstone.

The ore will be trucked from the open pit to the stockpile prior to processing in the Plant on the north-eastern side of the pit (see Figure 6).

4.1.2 Area 2

The ore in this area , below 100 metres, is assumed to be accessed by decline and mined by mechanised cut and fill. The ground conditions are expected to be very poor and so with intense alteration and weathering open stoping cannot be considered.

With such poor ground conditions it was assumed that only 70% of the deposit in this area could be extracted. A mining method with greater ground support such as square set and fill stoping may be required but for this report the more optimistic assumption was made.

The decline access was assumed to be mined to a size of 4.5 m x 5.5 m that would allow for truck haulage of the ore from the mine. The entrance is planned to be from the open pit and then decline in the footwall of the orebody. The gradient of the decline being 1 in 8 or 12.5%.

The mining plan allows for the underground mine to commence after the open cut.

The mechanised cut and fill will entail drilling the next lift in section using a drilling jumbo and rock bolts and meshing installed after. The ore will be mucked by LHD loader to ore passes in the footwall, if ground conditions permit. The orepasses will be emptied through chutes onto the trucks for transport to surface.

The width of the orebody will be between 5 and 10 metres.

4.1.3 Area 3

The decline will be extended south from Area 2 to Area 3 which is approximately 200 metres south. Zeehan Mines Pty Ltd were unable to mine north through the Mine Fault from their working south of the fault on their No. 2 and No. 3 levels (92 metres and 128 metres below ground respectively).

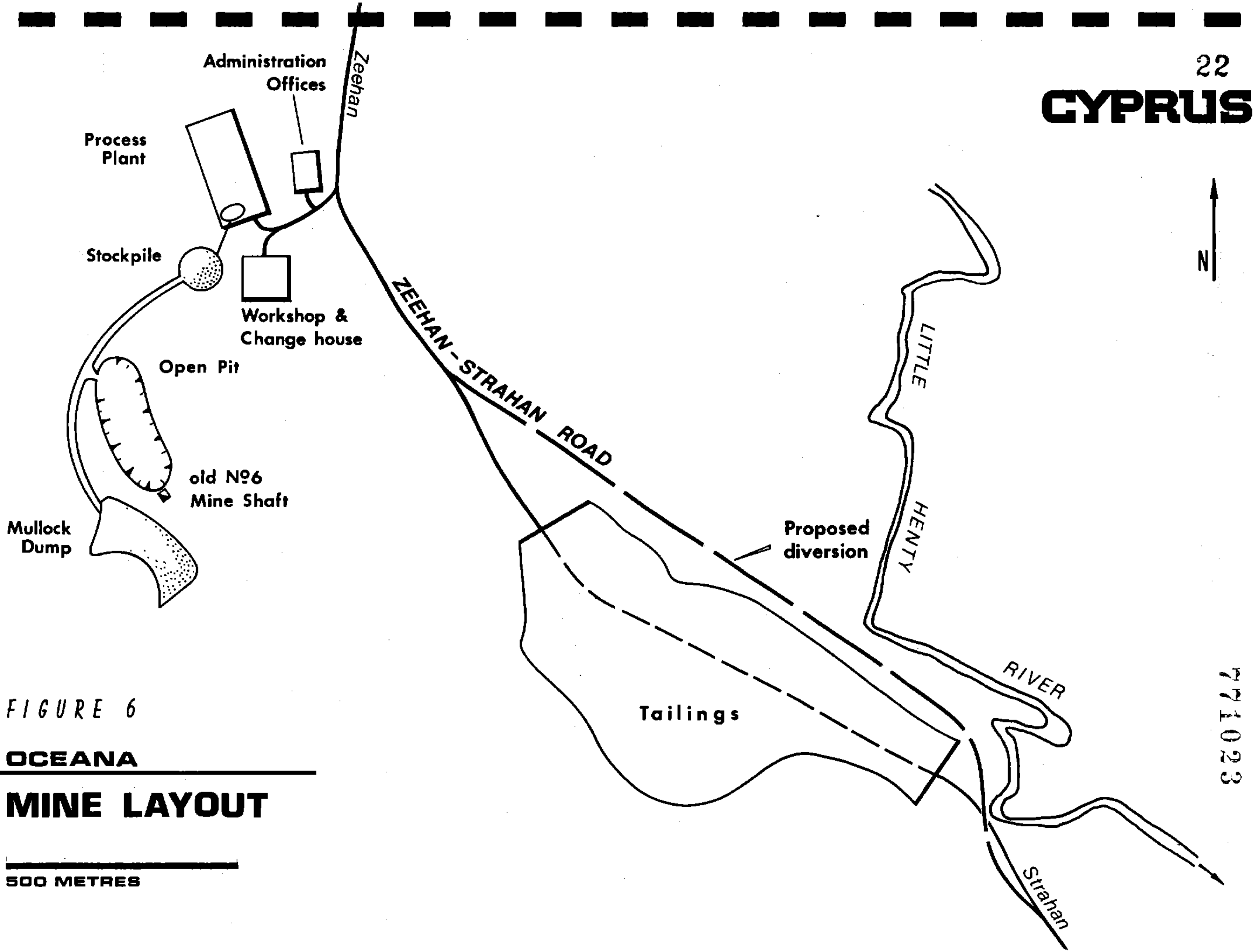

FIGURE 6

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MINE LAYOUT

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It is planned to mine the decline through the Fault at approximately 200 metres below surface as it is expected that the fault will be less disturbed and ground conditions will be better. If this is not the case then the decline will have to be mined west and round the fault (see surface geology map showing the Mine Fault terminating at the South Oceana Fault in the Geological Report).

The mining method for Area 3 will be mechanised cut and fill. The ground conditions should be suitable for this method. Zeehan Mines used flat-back cut and fill when stoping out the one around No. 6 shaft and it expected that similar ground conditions prevail through the stratiform ore zone. It was assumed that 90% of the deposit in this area could be extracted. The width of the orebody is approximately 6 metres.

Within Area 3 there are several intersections of mineralised zone to the footwall side of the orebody. This zone varied in thickness between 1 and 2 metres. The zone was not included within the geological resource or mining reserve as the mining cost of this very thin zone and the depth below surface of the zone would be considerably higher than the other resources and so be more uneconomic than the stated resources.

4.2 Underground Drainage

The volume of water in the underground workings is expected to be high. Zeehan Mines were pumping approximately $8m^3$/minute prior to closure in 1960 and at a greater depth this volume is likely to rise as the cone of depression of the water table increases in surface area.

The large volume of water expected will necessitate a substantial pump station to be constructed initially at the base of Area 2. Water will be collected in a settler prior to discharging into a sump. Multistage centrifugal pumps will pump the water to a surface sump where it will be channelled across the orebody beyond the depression cone in the water table.

When mining Area 3, a second pump station will be established and water will be pumped up to the Area 2 pump station.

In addition to these two main pump stations, local drainlines will be necessary within the workings, or even channels along the side of the drives.

5.0 Metallurgy

5.1 Metallurgical Testing

The testwork was carried out on the samples from drill core at Roseberry where the ore was put through the standard flotation test for the Roseberry circuit. It was concluded by Electrolytic Zinc (E.Z.) that the two samples of ore from Ares 1 and 2 were incompatible with the Roseberry circuit because of the low recoveries. The sample of ore from Area 3 was compatible but the low zinc values suggested that ore was not economically attractive (see Appendix 1). As the sample gave poor results, Electrolytic Zinc concluded that a separate flotation circuit would have to be put in at the Roseberry Plant to handle Oceana's ore. For this reason this report assumes a treatment plant at site.

5.2 Process Plant Description

The process flowsheet is based on typical industry practise for a relatively coarse grained lead/silver/zinc ore with simple mineralogy. The following flowsheets detail overall layout, and comminution, lead flotation, zinc flotation and process water sections of the plant.

5.2.1 Crushing and Crushed Ore Storage

Ore is received from the mine into a run-of-mine ore bin, from which it is fed via a variable speed apron feeder to a

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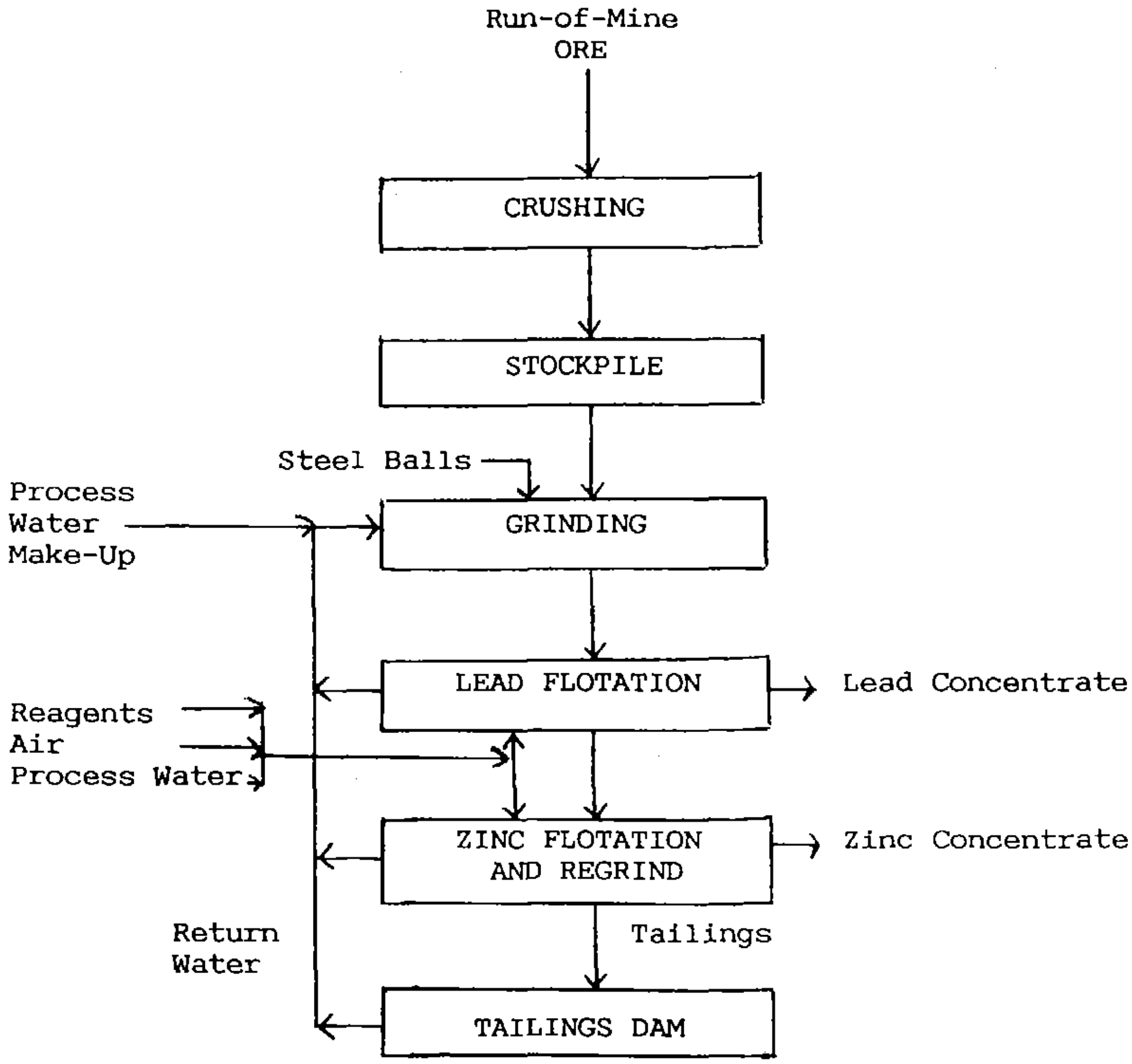

OCEANA PROSPECT

OVERALL

CONCEPTUAL FLOWSHEET

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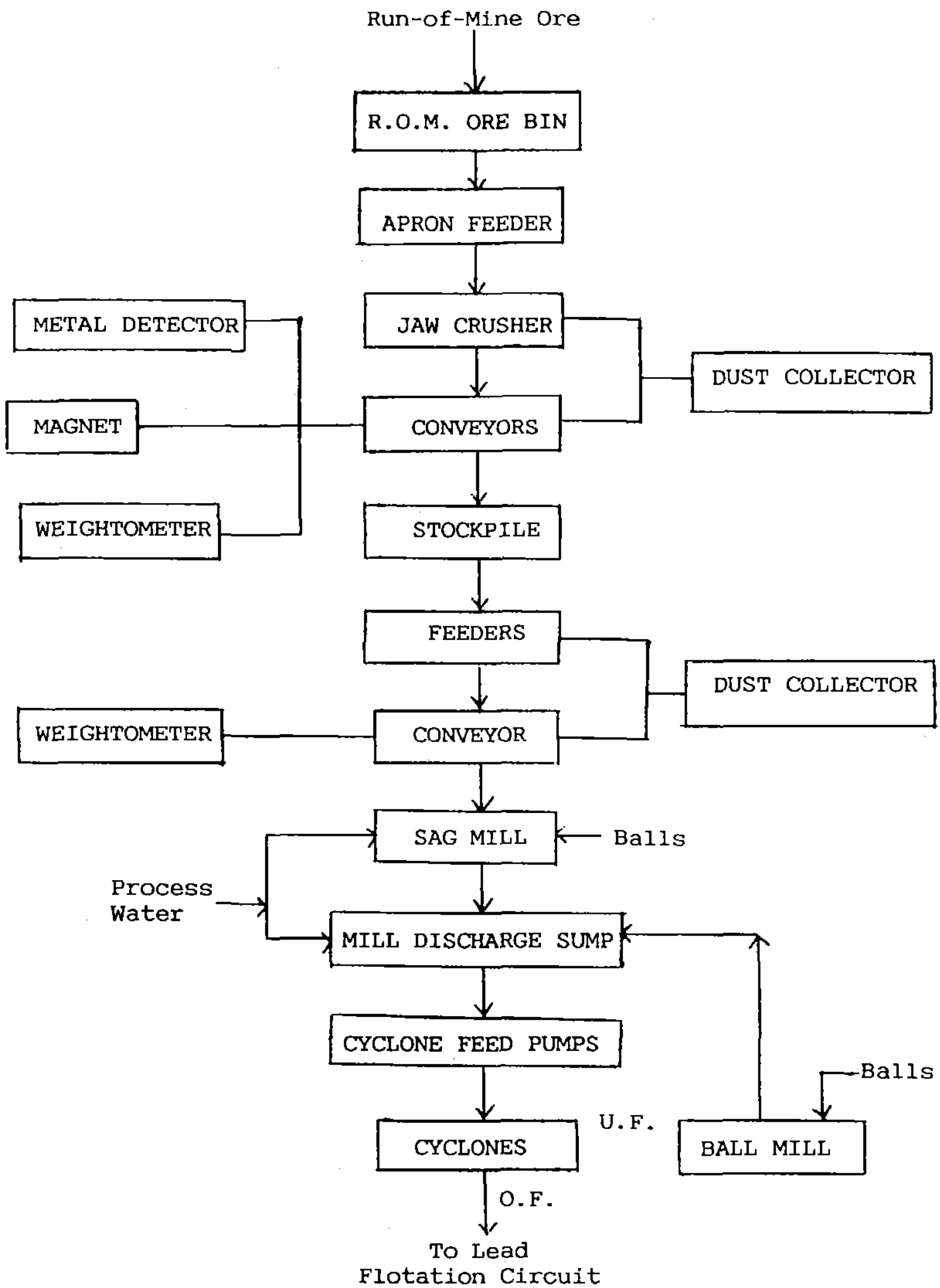

<u>Oceana Prospect</u>

<u>Conceptual Flowsheet</u>

<u>Crushing & Grinding</u>

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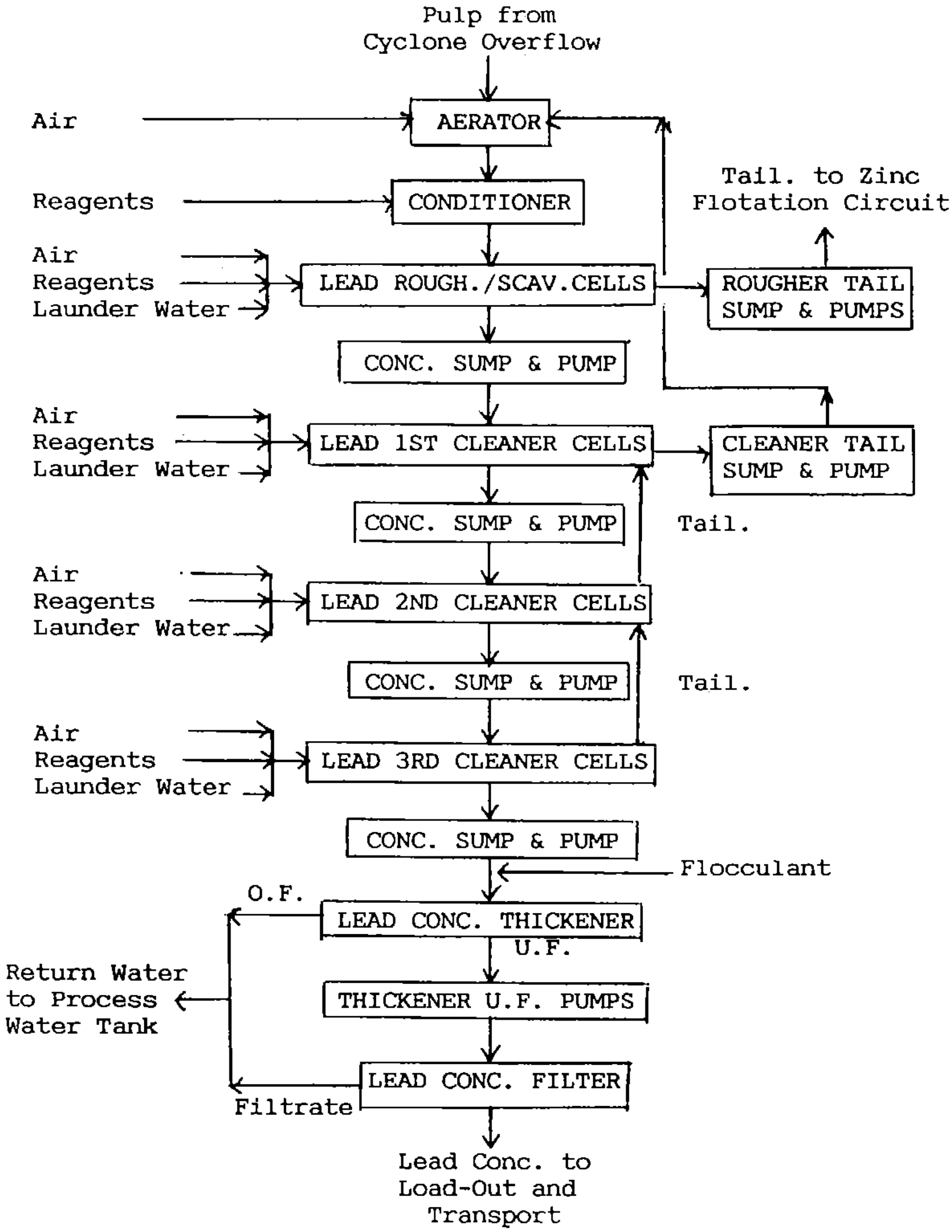

OCEANA PROSPECT

CONCEPTUAL FLOWSHEET

LEAD FLOTATION CIRCUIT

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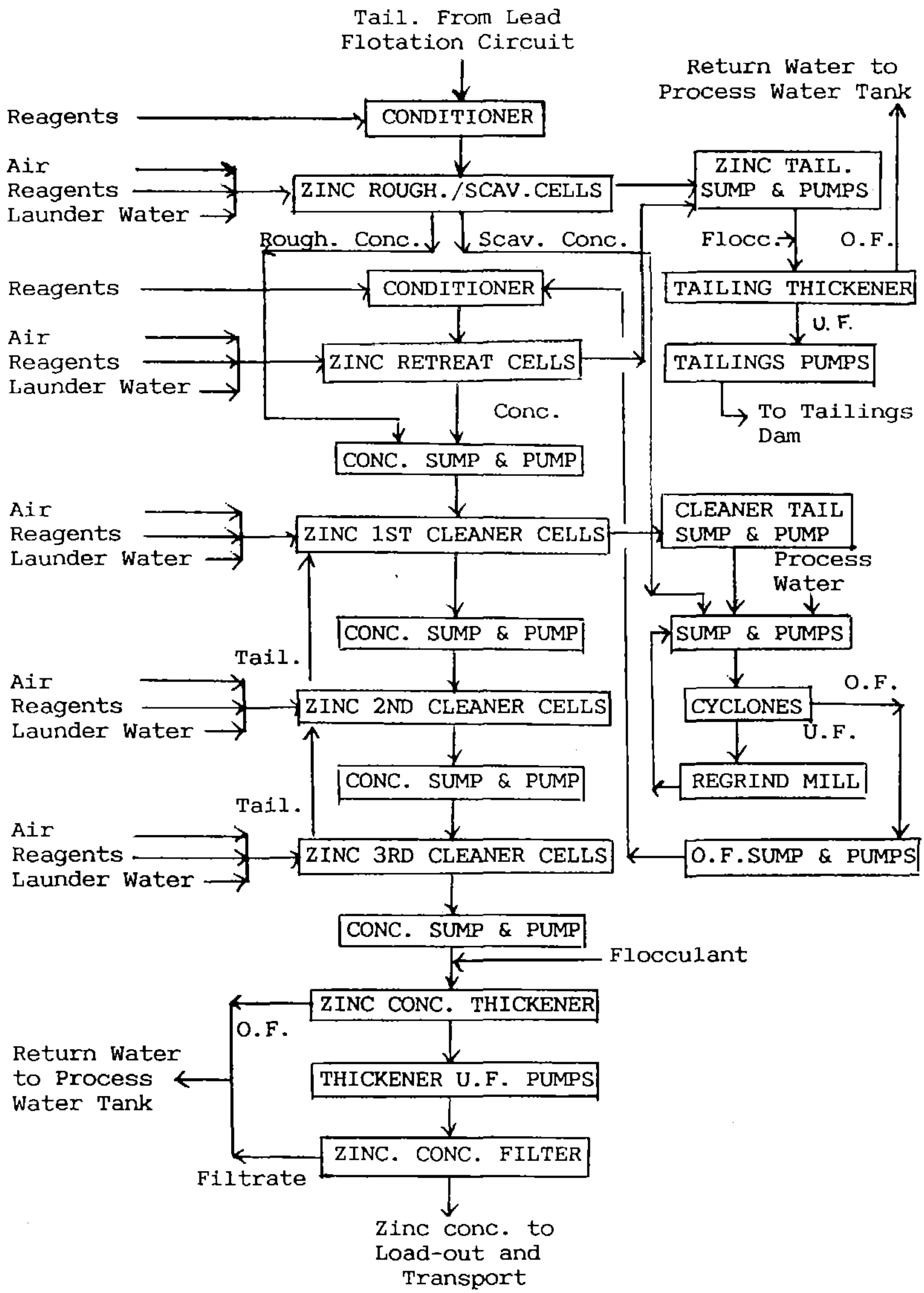

OCEANA PROSPECT

CONCEPTUAL FLOWSHEET

ZINC FLOTATION CIRCUIT

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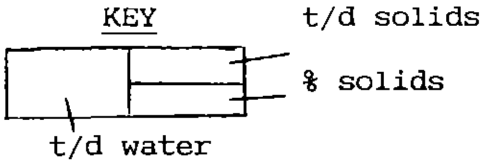

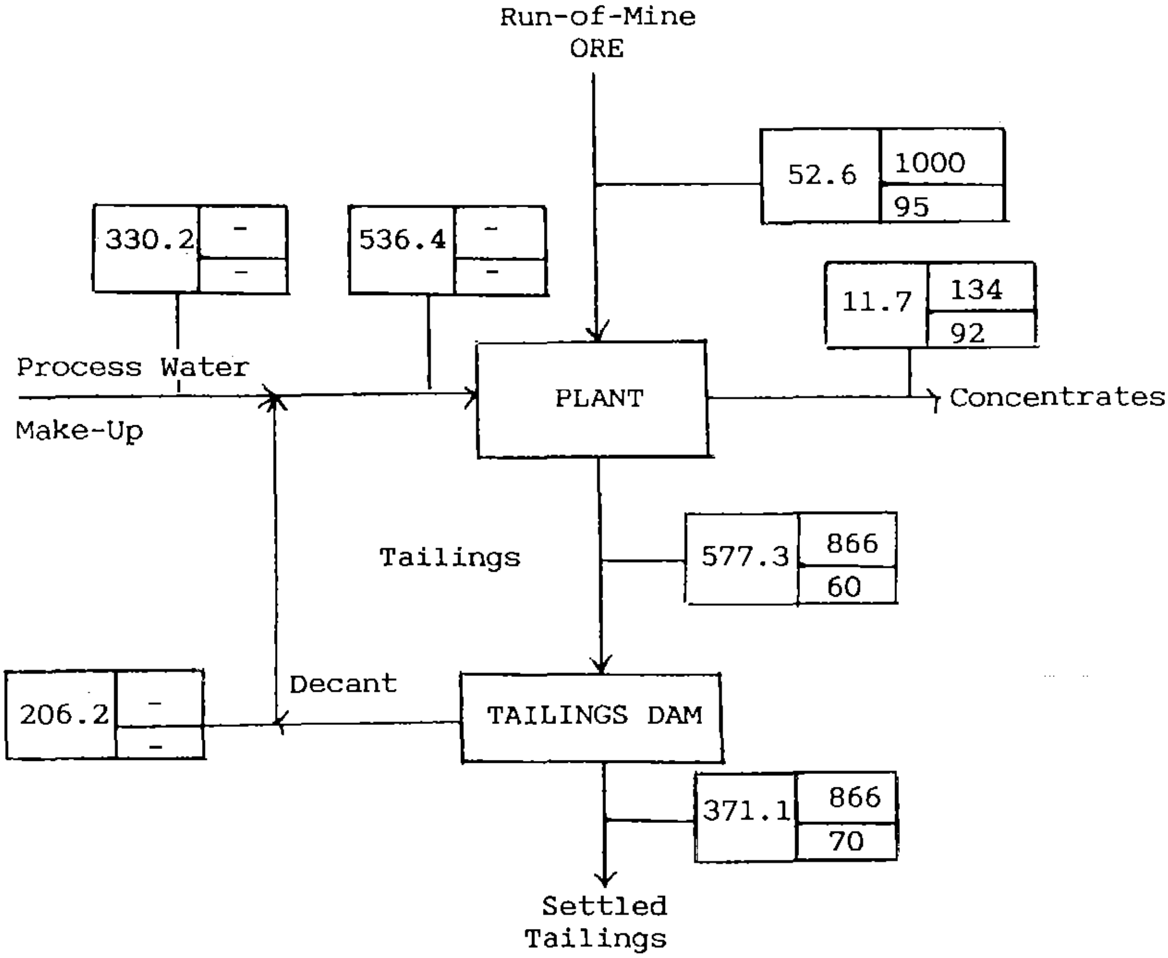

Notes

1. Balance is for average grades

2. No alloances for evaporation or rainwater catchment

3. Potable water excluded

OCEANA PROSPECT

OVERALL PROCESS

WATER SYSTEM

double toggle jaw crusher. Crushed ore is conveyed to an open stockpile.

The crusher discharge is equipped with a dry dust collector. The discharge conveyer incorporates an electromagnet and metal collector. A weightometer measures the quantity of crushed ore deposited on to the stockpile.

5.2.2 Ore Reclaim, Grinding, and Cyclone Classification

Crushed ore is withdrawn from the stockpile by two variable speed slot feeders which discharge onto a conveyer feeding a SAG mill. A weightometer is provided to measure the feed rate.

Water is added to the SAG mill feed and the resulting pulp discharges via a trommel into a sump. The trommel is equipped with a reverse screw to return oversize to the mill.

Ball mill discharge is also received by the sump, and the combined pulp is pumped to a group of cyclones. Water is added to the sump to control the cyclone feed pulp density as required.

Cyclone overflow passes on as feed to the lead flotation circuit. Underflow gravitates to the ball mill feed for further grinding.

The ball mill discharges through a trommel to remove scats which are deposited into a boxed concrete area.

The mills and associated equipment are located over a bunded slab containing a sump and sump pump. Floor slopes are graded to facilitate clean-up.

5.2.3 Lead Flotation

Cyclone overflow is passed through an aeration tank followed by a conditioner, then is fed to lead rougher/scavenger flotation cells. MIBC frother, lime, sodium cyanide, and xanthate, are added to the conditioner. Further additions of xanthate and frother are made to the intermediate junction boxes as required. Rougher/scavenger concentrate is collected in a sump and pumped to the first cleaner cells. Tailing is pumped to the zinc rougher/scavenger circuit.

The concentrate passes through three stages of cleaning cells with reagent additions as appropriate. The final concentrate is thickened, filtered, then stored ready for load-out and transport. Tailing from each stage flows to the previous stage. First cleaner tailing is recycled to the aerator. Thickener overflow water and filtrate are collected in process water tank for reuse.

5.2.4 Zinc Flotation

Tailing from the lead scavenger cells is received by a conditioner, then fed to zinc rougher/scavenger cells. MIBC frother, copper sulphate, lime, and xanthate and frother are made to the intermediate junction boxes as required. Rougher concentrate is collected in a sump together with concentrate from the retreat circuit and pumped to the first cleaner cells. It then passes through three stages of cleaning with reagent additions as appropriate. Final concentrate is thickened, filtered, then stored ready for load-out and transport. Tailing from each stage flows to the previous stage. First cleaner tailing is transferred to the regrind circuit. Thickener overflow water and filtrate are collected in the process water tank for reuse.

In the regrind circuit, first cleaner tailing and scavenger concentrate are treated in a ball mill/cyclone system to achieve a grind of 80% passing 38 microns. The cyclone overflow product is pumped to a conditioner which feeds retreat cells. Reagents are added as appropriate. Retreat concentrate reports to the rougher concentrate sump. Tailing joins the scavenger circuit tailing and the combined pulp is pumped to a tailing thickener. Thickened pulp is pumped to the tailings retention dam. Thickener overflow is sent to the process water tank together with decant solution from the tailings dam.

Facilities are provided to receive, store, prepare, and distribute the various reagents to the plant addition points.

Floor sumps and sump pumps are provided where appropriate.

5.2.5 Sandfill Station

After the open pit has been completed, there will be a requirement for deslimed tailings to be piped underground for fill. The sandfill station will require a cyclone bank to separate the slimes out. The deslimed tailings will be pumped underground when required and the slimes will be pumped to the tailings dam.

5.3 Plant Design Criteria

In the absence of meaningful metallurgical test data, the following parameters are based on industry figures. The figures are on the optimistic side to allow the maximum potential of the project to be evaluated. The size of the crusher and mills is given in table following as is the concentrate production of the various ore types.

5.3 PLANT DESIGN CRITERIA

1.0	**General**		
1.1	Annual Throughput:		
	Ore processed	t/y (dry)	365,000
1.2	Grinding/Flotation Schedule:		
	Days/year		365
	Hours/day		24
	Shifts/day		3
1.3	Grinding/Flotation Capacity:		
	Availability/uilisation	%	90
	Design daily throughput	t/d	1000
1.4	Ore Grade:		
	Lead	%	9.5
	Zinc	%	3.8
	Silver	g/t	74
1.5	Ore Characteristics:		
	Specific gravity		4.5
	Moisture content	%	5
	Angle of repose	degrees	37
	Abrasion index		0.21
	Bulk density		3.0
2.0	**Crushing & Stockpiling**		
2.1	Operating Schedule:		
	Weeks/y		52
	Days/week		6
	Shifts/day		1
	Hours/shift		8
2.2	Capacity:		
	Daily throughput	t/d	1167
	Availability/utilization	%	80
	Design rate	t/h	182

2.3 Performance:

Type	Single stage jaw, double toggle	
R.O.M. feed size, 80% passing	mm	850
Product size, 80% passing	mm	150
Crushing work index	Kwh/t	20

2.4 Stockpile:

Type		Open
Live Capacity	t(dry)	2000
Equiv. plant operating time	hr.	48

3.0 <u>Grinding</u>

3.1 Capacity:

Daily throughput	t/d	1000
Design rate	t/h	41.7

3.2 SAG Milling:

Feed size, 80% passing	mm	150
Product size, 80% passing	micr.	600
SAG work index	Kwh/t	18
Operating pulp density	% solids	65
Ball Charge, v/v	% v/v	0-12
% Critical speed	%	72
Liners		steel
Trommel	Reverse screw type	

3.3 Ball Milling:

Product size, 80% passing	micr.	75
Bond work index	Kwh/t	12
Operating pulp density	% solids	70
Ball Charge, v/v	%	40
Liners		rubber

3.4 Classification:

Overflow pulp density	% solids	35
Underflow pulp density	% solids	70
Circulating load	%	250

3.5 Regrinding (zinc.):

Product size, 80% passing	micr.	38
Bond work index	Kwh/t	18
Operating pulp density	% solids	70
Ball charge v/v	%	40
Liners		rubber

3.6 Regrind Classification (zinc.):

Overflow pulp density	% solids	27
Underflow pulp denisty	% solids	70
Circulating load	%	50

4.0 <u>Flotation</u>

4.1 Circuit Design:

	Time (min.)	% Solids
Lead aeration	5	35
Lead conditioning	5	35
Lead rougher/scavenger	25	35
Lead 1st cleaner	20	30
Lead 2nd cleaner	15	25
Lead 3rd cleaner	15	20
Zinc conditioning	5	33
Zinc rougher/scavenger	30	33
Zinc retreat conditioning	5	28
Zinc retreat	10	20
Zinc 1st cleaner	20	25
Zinc 2nd cleaner	15	25
Zinc 3rd cleaner	15	20

4.2 Concentrates:

		Lead Conc.	Zinc Conc.
Production:			
t/y		40,077	19,053
t/d		122	58
Grade:			
Lead	%	70.0	8.3
Zinc	%	3.2	50.0
Silver	g/t	412	33

Recoveries:			
Lead	%	92	4.1
Zinc	%	11.7	73.0
Silver	%	85.0	2.4

5.0 Thickening (Lead & Zinc Conc.)

Unit area	m²/t/d	1.0
Underflow pulp density	% solids	60
Flocculant	g/t	50

6.0 Filtration (Lead & Zinc. Conc.)

Cake moisture	%	8
Rate, Zinc	Kg/m²/hr	100
Lead	Kg/m²/hr	200

7.0 Reagents & Consumables

	Kg/t Ore
Balls	1.4
Lime	0.5
Xanthate	0.26
Sodium Cyanide	0.14
MIBC	0.03
Copper Sulphate	1.0
Flocculant	0.01

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MAJOR PLANT EQUIPMENT SIZES

The following crushing and grinding equipment have been selected based on typical industry data.

Description	Size	Motor Kw
Double toggle jaw crusher	1067 x 1219 mm (42 x 48 inch)	110
SAG mill	4.88 x 1.52 m (16 x 5 ft)	315
Ball mill	2.90 dia. x 3.05 m (9½ dia. x 10 ft)	315
Regrind ball mill	1.83 dia. x 2.13 m (6 dia. x 7 ft)	90

ORE TYPE	ZINC CONCENTRATE QUANTITY T/100T ORE	% RECOV. Zn	GRADE % Zn	LEAD/SILVER CONCENTRATE QUANTITY T/100T ORE	% RECOV. Pb	GRADE Pb	% RECOV. Ag	GRADE g/t Ag
Average	5.8	73	50	12.2	92	70	85	412
0 - 50 M	2.2	58	50	5.6	84.6	65	74.5	412
50 - 100 M	6.4	77.6	50	8.7	90.6	70	84.6	574
BELOW 100 M	9.3	82	50	17.0	96.5	75	86.5	570
STRATIFORM	6.2	77.5	50	15.4	96	75	86	407

EQUIVALENT ORE HEAD GRADES

ORE TYPE	% ZINC	% LEAD	g/t SILVER
Average	3.8	9.5	74
0 - 50 M	1.9	4.3	31
50 - 100 M	4.1	6.7	59
BELOW 100 M	5.7	13.2	112
STRATIFORM	4.0	12.0	89

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5.4 Plant Feed Schedule

The schedule of mill feed for the six years production is shown on the following table.

Mill Feed Schedule (dry tonne basis)

	tonnes	%Pb	%Zn	g/t Ag
Year 1	365,000	4.3	1.9	31
Year 2	365,000	5.5	3.0	45
Year 3	365,000	12.8	5.6	109
Year 4	365,000	12.3	4.4	95
Year 5	365,000	12.0	4.0	89
Year 6	45,000	12.0	4.0	89
Total	1,870,000	9.5	3.8	74

6.0 Site Location, Infrastructure, Tailings Dam

6.1 Site Location

The layout of the site is shown in figure 6. The location of the plant is on the ridge between the Oceana Valley and the Austral Valley. As the Gordon Limestone becomes unpredictable when the water table is lowered, it is prudent to locate the plan away from the Limestone. The Austral Valley may provide further plant feed from the prospective areas and so the plant will be located well if drilling proves these areas.

6.2 Infrastructure

The road access to the site, as mentioned in other sections, is good. The electrical power grid is connected to Zeehan, 2.4 kilometres north of the site. It will not be too difficult for an extension to be brought down to the mine site.

Water is readily available from the pumping of the mine workings. Sewerage will be required to be treated at the site.

6.3 Tailings Dam

The proposed facility has a maximum storage area of 33 hectares within an overall catchment of 57 hectares. The tailings would be confined in a cross valley type facility consisting of a main southern embankment and a smaller dam to the north. The dam is located to the east of the Oceana valley and will be south of the treatment plant.

At present the Zeehan-Strahan road passes through the valley and it will be necessary to direct the road around the proposed facility.

Due to the high rainfall it was necessary to locate the dam where the catchment was low. Water may have to be discharged from the dam into the Little Henty River. An allowance for treatment of this water has been made within the cost estimates to ensure no pollutants are discharged into the river.

7.0 Environment

7.1 The local Environs

The proposed Retention Licence area is crown land, with the proposed tailings dam being within a timber reserve.

The Oceana deposit lies in the Oceana Valley (see Plate 1) at the base of Mount Zeehan which is a major feature overlooking the area and is 702 metres high (see plate 2). Mount Zeehan overlooks the Little Henty River Valley which flows south to the east of Mount Zeehan before flowing west to the coast at the foot of the hill.

There is a walking track to the top of Mount Zeehan which commences from the Oceana valley. The start of this track

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PLATE 1: View of Oceana looking south with old mine workings in right foreground

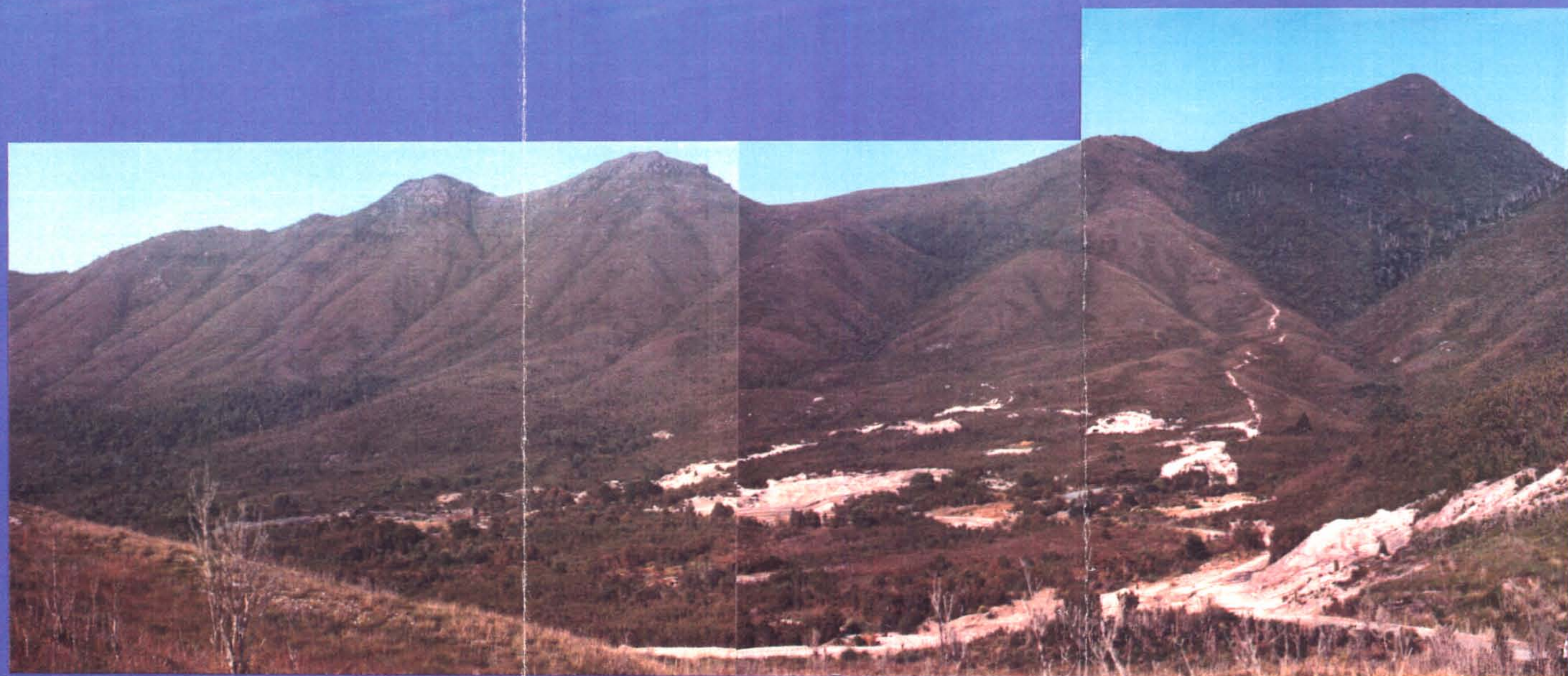

PLATE 2: View of Mount Zeehan looking west with Oceana Valley to left

will need to be moved to the outside of the main mining area. In the construction of the access road to the mine a car park for hikers will be allowed for and necessary connections to the original walking track up the hill.

Zeehan Town, which is just north of the proposed area, has a population of 1610 and is connected by sealed road to the port of Burnie, located 140 kilometres to the north. The town is the service city for the Renison Tin Mine which is 17 kilometres by road to the north. Zeehan is also connected to Strahan by sealed road which passes through the proposed Retention Licence area.

Reopening of the mine would have a very positive affect on the town of Zeehan, where the population has been contracting.

7.2 Meteorological Data

The climate is temperate with high rain falls especially in the winter months. The average annual rainfall and temperatures which are given in the following tables, indicate the climate.

	Zeehan Rainfall	(1890 - 1968) Mean No. Rainy Days.
January	139 mm	17
February	114	14
March	151	18
April	216	20
May	240	21
June	252	20
July	265	23
August	263	24
September	230	22
October	221	22
November	190	20
December	165	18
Annual	2446 mm	Annual 239

Zeehan Temperature (1908 - 1968)

	Max(°C)		Min(°C)
January	19.5		8.8
February	20.2		9.5
March	18.8		8.4
April	15.4		6.9
May	13.2		5.5
June	11.2		3.8
July	10.8		3.4
August	11.5		3.8
September	13.2		4.8
October	14.9		5.7
November	16.3		6.9
December	18.2		8.2
Av. Annual	15.3	Av. Annual	6.3

Coldest recorded -7°C

Warmest recorded 37.3°C

7.3 Vegetation

Button-grass covers much of the area which is typical of swampy flats with acid, peaty soils where the climate is wet and mild. There is a considerable thickness of alluvial gravels in the Oceana valley which underlays the soils. In the gullies, there are ti-trees and young eucalypts which are quite dense in places.

7.4 Monitoring

Water quality monitoring will be required from the discharge of the mine pumps. It is expected that the water from the limestone rock will be pure but it will be necessary to contain the water prior to discharge in the natural drainage system to ensure water quality.

Monitoring will also be required around the base of the two tailings dam walls. If water levels rise within the dam then controlled release of water will be necessary through water purifiers to ensure the quality of water being released.

8.0 Cost Estimate

8.1 Capital Cost Estimate

8.1.1 Open Cut Mining

The Open cut mining will be carried out by contractors. This reduces the capital cost of the development of the open cut to the prestripping cost, the mobilisation fee of the contractor and the establishment of the dewatering bores, pumps and redirecting of water courses.

1.	Pumps, dewatering holes	$1,000,000
2.	Prestripping and site Preparation	$ 500,000
3.	Contractor Mobilisation	$ 120,000
	Total	$1,620,000

8.1.2 Underground Mining

The underground mine will be mined by Cyprus employees apart from the decline which will be done by a contractor.

The mine equipment will be purchased new. The life of the underground mine is approximately three and a half years so some salvage value can be realised on the equipment after the reserves are exhausted.

The mine will require two main pump stations underground as mentioned in section 3. The underground equipment will be maintained in surface workshops which will be located close to the trucking ramp of the Open pit. Changing rooms, first aid station etc, will also be required for the underground work force.

Recruitment of the employees will be a capitalised expense. The total underground mine capital cost is as follows:-

1.	Decline to Area 2	$4,000,000
2.	Decline from Area 2 to Area 3	$ 500,000
3.	Mine Equipment	$5,300,000
4.	Area 2 Pump Station and piping	$2,000,000
5.	Area 3 " " " "	$1,000,000
6.	Workshops	$ 500,000
7.	Changing rooms, etc	$ 150,000
8.	Recruitment	$ 200,000
		13,650,000

8.1.3 Process Plant

The estimate of the plant is based on data from other studies and projects. Included within the capital cost of the plant is the process plant, associated buildings such as admin office, laboratories, control room etc, in-plant services, tailings dam, earthworks, plus engineering, procurement, construction and project management charges.

After the mining of the open pit, a sandfill station will be required for the mechanised cut and fill mining. The station will deslime the tailings so that they are suitable for fill underground.

1.	Crushing and stockpiling	$1,600,000
2.	Grinding and classification	$3,600,000
3.	Flot., reagents & conc. dewatering	$5,700,000
4.	Services, buildings & tails dam	$1,700,000
5.	Earthworks	$ 500,000
6.	E.P.C/Proj.Man/Temp Works (15%)	$2,000,000
7.	Sandfill Station	$1,000,000
	Total	$16,100,000

8.1.4 Infrastructure and Services

The tailings dam location will necessitate the diversion of the Zeehan-Strahan Road.

The power and water reticulation of the process plant and mine is costed in this section. The power will be brought from the electrical grid which is presently connected to Zeehan. The drainage water from the mine will be used in the Process Plant. The Environmental Impact Study would be required to fulfil the Mining Lease application.

An administration office and storage area will be located close to the Process Plant.
Recruitment of the employees for the process plant and administration as well as the senior mine management is included in this section.

1.	Road Diversion	$ 50,000
2.	Power Connection and Reticulation	$ 800,000
3.	Water Supply	$ 100,000
4.	Environmental Review	$ 300,000
5.	Office and store	$ 200,000
6.	Recruitment	$ 250,000
	Total	$1,700,000

8.1.5 Working Capital

The working capital is based on 2 months of the annual operating costs. This capital is reclaimable in the final year of the operation; but for the purposes of this study this amount is assumed to pay for rehabilitation of the mine site.

8.1.6 Salvage Value

A salvage value of the major process plant items (e.g. crusher, mills, etc) and U/G equipment was estimated at 10% of purchase price. This value is equivalent to 5% of original value of the plant and underground capital.

8.2 Operating Cost Estimate

8.2.1 Open pit mining (Area 1)

The open pit mining will be done by contractors. The cost of mining is assumed to be $4/BCM for waste rock and $5/BCM for ore. With a mining depth of 70 metres, this equates to a cost of $7.25/tonne of ore at the stripping ratio of 6.9:1.

The operating cost of the mine office and associated work during the open pit mining is estimated at $800,000 per year. The total cost of pumping the mine is estimated at $1,500,000 per year.

With the production rate at 365,000 tonnes per year the cost per tonne of ore is $14.35 and with a contingency of 5% gives a total of $15.10/tonne.

8.2.2 Underground Mining (Area 2 and 3)

The underground mining was assumed to be carried out by Cyprus and not by contractors.

The following costs were estimated for mechanised cut and fill mining.

			$/t
1.	Supervision and Engineering		2.80
2.	Development		12.00
3.	Stoping and trucking		10.00
4.	Ground Support		2.00
5.	Pumping and Drainage		5.50
6.	Ventilation		1.20
7.	Supply		0.30
8.	Fill storage and delivery		3.80
9.	Miscellaneous		1.50
		Sub-Total	39.10
		Contingency	2.00
		Total	$41.10/t

8.2.3 Process Plant

The process plant costs are based on typical reagent consumption rates in the industry. It is expected that sodium sulphide will be necessary for the ores from Areas 1 and 2 as it is expected to be partially oxidised. The following costs were estimated:

		$/t of ore feed
1.	Labour, staff and wages	5.00
2.	Power	3.10
3.	Consumables	3.50
4.	Maintenance supplies	1.50
5.	Miscellaneous	2.00
		15.10
6.	Add-on for sodium sulphide	0.80
		15.90

These costs were based on the unit costs of consumables of:-

1.	Power	8c/Kwh
2.	Lime	$ 200/t
3.	Cyanide	$1950/t
4.	Copper sulphate	$ 965/t
5.	Xanthate	$1750/t
6.	MIBC	$2400/t
7.	Flocculant	$4000/t
8.	Steel balls	$ 880/t
9.	Sodium Sulphide	$ 700/t

8.2.4 Administration

The costs included in this section are the administration section of the operation, including accounts, stores requisitions etc, environmental matters and miscellaneous including lease payments, security etc. The following annual costs are assumed:

1.	Administration	$400,000
2.	Environmental monitoring	$300,000
3.	Miscellaneous	$200,000
		$900,000

This equates to $2.50/t of ore.

9.0 Marketing

9.1 Lead

9.1.1 Uses

Lead is used as metal in the pure form or as an alloy or as a chemical. The major use is in lead batteries which are used in back-up power systems, automobiles and motive power such as forklifts, aircraft tugs, etc.

The use of lead as a petrol additive for automobiles is reducing as countries introduce legislation to restrict its usage. Lead is also used in bearings because of its wearing properties and being a natural lubricant.

Lead is used in hospitals etc for protection against x-ray and is used for containers of toxic wastes.

Lead-oxide paint is one of the major areas of lead in the chemical form. 75% of the use of lead in the United States in the glass and ceramics industry is in the manufacture of colour TV tubes (USBM, 1985).

Lead cable sheathing is used in underwater and, to a reducing extent, in underground cables.

Other minor uses of lead metal are lead shot, small calibre bullets, ballast, counter-weights and sinkers for fishermen. Other minor uses of oxides of lead are dyes, matches, rubber substitutes, adhesives and in oil refining.

The end uses of lead in the two major user countries, Japan and the United States and in Australia are given in the table following. The drop in use of lead in petrol is dramatically shown in the US from 89,000 tonnes in 1983 to 29,000 tonnes in 1986.

9.1.2 Consumption

The following table details the recent consumption of lead by country.

Lead: Principal Uses
Thousand Metric Tons (From the International Lead and Zinc Study Group)

	Annual Totals: 1983	1984	1985	1986	Year to Date: 1986	1987
Australia (Refined lead only)					**Jan**	**- Mar**
Batteries	24	24	27	27	-	6
Cable Sheathing	3	1	1	2	-	0
Pipe, Sheet, Foil, Shot		9	10	10	-	1
Chemicals	7	7	6	7	-	2
Alloys	3	2	2	2	-	0
Miscellaneous	2	4	4	4	-	1
Total	**45**	**47**	**50**	**52**	**-**	**10**
Japan (Refined lead only)					**Jan**	**- Oct**
Batteries	207	239	253	262	215	210
Cable Sheathing	23	20	19	13	12	10
Pipe and Sheet	17	16	15	14	11	10
Chemicals	69	75	73	62	51	50
Alloys	16	18	18	17	14	15
Miscellaneous	27	23	19	21	19	18
Total	**359**	**391**	**397**	**389**	**322**	**308**
United States(1) (Lead in all forms including scrap)					**Jan**	**- Oct**
Batteries	807	866	841	854	676	768
Cable Sheathing	10	12	15	17	14	16
Pipe and Sheet	39	41	35	32	28	31
Tetraethyl	89	79	46	29	-	-
Chemicals	69	77	73	70	84(2)	80(2)
Ammunition	44	48	50	44	41	43
Alloys	69	60	61	52	48	38
Miscellaneous	21	24	27	27	19	22
Total	**1148**	**1207**	**1148**	**1125**	**910**	**998**

(1) Monthly and year to date figures for current year include estimates of consumption by companies reporting annually.
(2) Including tetraethyl.

The consumption of lead in the United States, the major user, has been dropping over the last three years but this has been countered by a substantial rise in Asian consumption and to a lesser extent Europe, which has seen lead consumption rise marginally over the period.

Refined Lead Consumption by Country (Thousand Metric tons)
(From the International Lead and Zinc Study Group)

	Annual Totals:				Year to Date:	
	1983	1984	1985	1986	1986 JAN -	1987 NOV
Europe	**1558**	**1617**	**1609**	**1645**	**1513**	**1491**
France	196	209	208	205	190	185
Germany, F.R.	318	357	346	359	331	320
Italy	229	233	230	232	215	221
Spain	102	107	116	112	103	115
United Kingdom	293	295	274	282	259	261
Yugoslavia	116	95	116	145	133	110
Other	304	321	319	310	282	279
Africa	**106**	**101**	**121**	**117**	**107**	**108**
Algeria	10	11	15	13	12	11
Egypt	28	28	30	30	27	28
South Africa	44	42	48	49	45	50
Others	24	20	28	25	23	19
America	**1448**	**1525**	**1508**	**1501**	**1381**	**1388**
Brazil	49	64	73	92	84	85
Canada	97	122	100	105	98	79
Mexico	89	110	125	84	75	82
United States	1134	1143	1121	1119	1031	1049
Others	79	86	89	101	93	93
Asia	**639**	**676**	**733**	**760**	**696**	**704**
India	60	60	72	77	70	74
Japan	359	391	397	390	356	347
Other	220	225	264	293	270	283
Oceania	**71**	**71**	**69**	**68**	**63**	**62**
Australia	59	59	59	60	56	53
New Zealand	12	12	10	8	7	9
Total	**3822**	**3990**	**4040**	**4091**	**3760**	**3753**
Monthly Average	319	333	337	341	342	341

9.1.3 Price Outlook

The lead price over the last ten years is shown in the following figure. The price has come down substantially since 1979 with significant lows in 1983 and 1986. There has been an improvement over the last year and this year.

The latest forecasts for the lead price over the next six years are given below in US dollars. An Australian dollar equivalent is given assuming an exchange rate of AUS$1.00 = US$0.80 (current). The forecast shows that the lead price is likely to drop over the short term and then picking up again in 1992 and 1993. Although the price does pick up, the increase in price from 1988 to 1993 is still well behind the current inflation rate of 6.9% and only reflects an annual increase of 3%.

	1988	1989	1990	1991	1992	1993
Lead Price (L.M.E. Price)						
USc/lb	28	26	23	27	34	33
Lead Price (Aus. Dollar Equ)						
Aus$/tonne	770	720	635	745	940	910

9.2 Zinc

9.2.1 Uses

Zinc is used extensively in galvanising steel structural components for roofing, reinforcement bars, air ducts etc. Rolled zinc is used in dry cell battery cases, weather stripping, photo engraving plates, pipelines and offshore drilling platforms.

Zinc dust is used in the sherardizing process to protectively coat nuts, bolts, etc.

The other use of zinc is in the compound form. Zinc chemicals are used in fungicides, wood preservatives, varnishes, dyes, inks, pharmaceuticals and linoleum. Zinc is

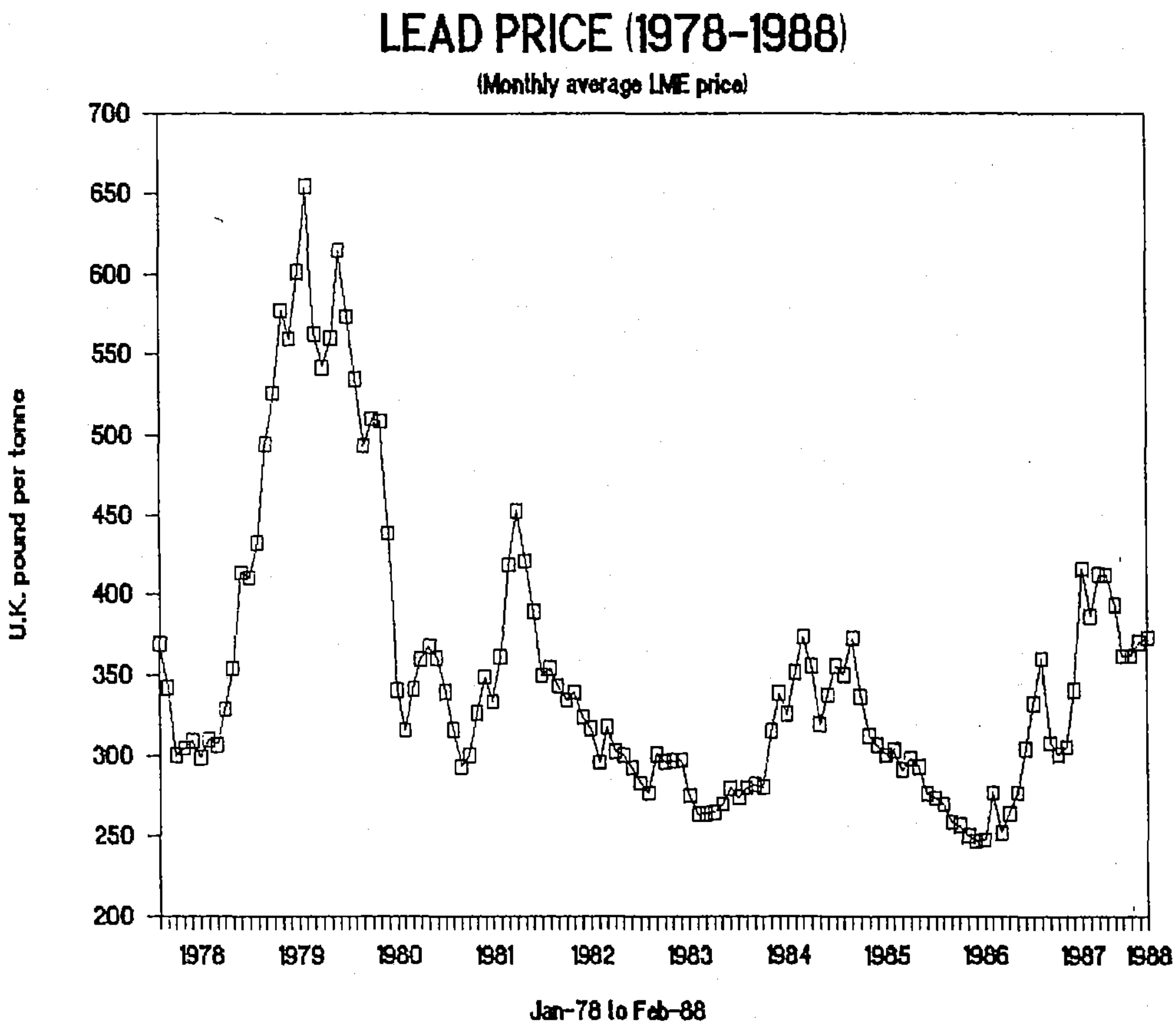

FIGURE 7

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used as an accelator and activator in vulcanising rubber. Zinc properties to inhibit corrosion are used in paint primers. Some welding and soldering fluxes contain zinc also.

The end uses of zinc in the two major user countries, Japan and the United States and in Australia are given in the table following.

9.2.2 Consumption

The following table details the recent consumption of zinc by country.

Zinc: Principal uses
Thousand metric tons (From the International Lead and Zinc Study Group)

	Annual Totals: 1983	1984	1985	1986	Year to Date: 1986	1987
Australia (Slab zinc only)					**Jan**	**- Mar**
Galvanizing	49	48	57	56	-	13
Brass	7	7	7	6	-	2
Diecasting Alloys	8	8	7	6	-	1
Rolled Zinc	1	2	2	1	-	0
Zinc Dust	2	2	2	2	-	0
Zinc Oxide	4	4	6	4	-	1
Miscellaneous	2	2	1	2	-	1
Total	**73**	**73**	**82**	**77**	**-**	**18**
Japan (Refined zinc incl. secondary distilled zinc)					**Jan**	**- Oct**
Galvanising	455	466	452	439	365	365
Diecasting Alloys	124	116	130	127	107	93
Brass	95	102	100	99	85	85
Rolled Zinc	31	29	28	26	21	20
Oxides, powders	41	39	43	39	34	23
Miscellaneous	25	23	27	23	20	15
Total	**771**	**775**	**780**	**753**	**632**	**601**
United States(1)(Slab zinc incl. secondary refined)					**Jan**	**- Oct**
Galvanising	373	375	362	366	296	328
Diecasting Alloys	213	232	218	175	168	134
Brass	108	126	78	74	62	72
Rolled Zinc	56	57	48	29	30	29
Zinc Oxide:						
from slab	36	37	44	40	34	39
from concentrate	38	48	42	22	23	12
Miscellaneous	20	21	19	23	16	19
Total	**844**	**896**	**811**	**729**	**629**	**633**

(1) Monthly and year to date figures for current year include estimates of consumption by companies reporting annually.

The consumption of zinc has been rising modestly in the three major continents Europe, Asia and America. The most significant increase in consumption is in Asia where there has been a 9% increase between 1984 and 1986.

9.2.3 Price Outlook

The price of zinc over the last ten years is given in the following figure. The price has been rising steadily over the period of 1978 to 1985. Since the peak in 1985, the zinc price has fallen but over the last year it has been improving again.

The latest forecasts for the zinc price over the next six years are given below in US dollars. An Australian equivalent is given assuming an exchange rate of AUS$1.00 = US$0.80 (current). The current high price of zinc is believed to be caused by a temporary tightening of supplies due to increased consumption by producers of galvanised steel. This is only expected to be temporary and the price is expected to fall back next year with a moderate increase per year till 1993. A recent forecast by a leading British Bank predicts a zinc price of USc51/lb in 1991 but even this more optimistic forecast represents only a 1.5% increase in price per year from USc48/lb in 1988.

	1988	1989	1990	1991	1992	1993
Zinc Price (European Producer Price) USc/lb	48	42	43	44	45	45
Zinc Price (Aust. Dollar equiv) AUS$/tonne	1320	1160	1190	1210	1240	1240

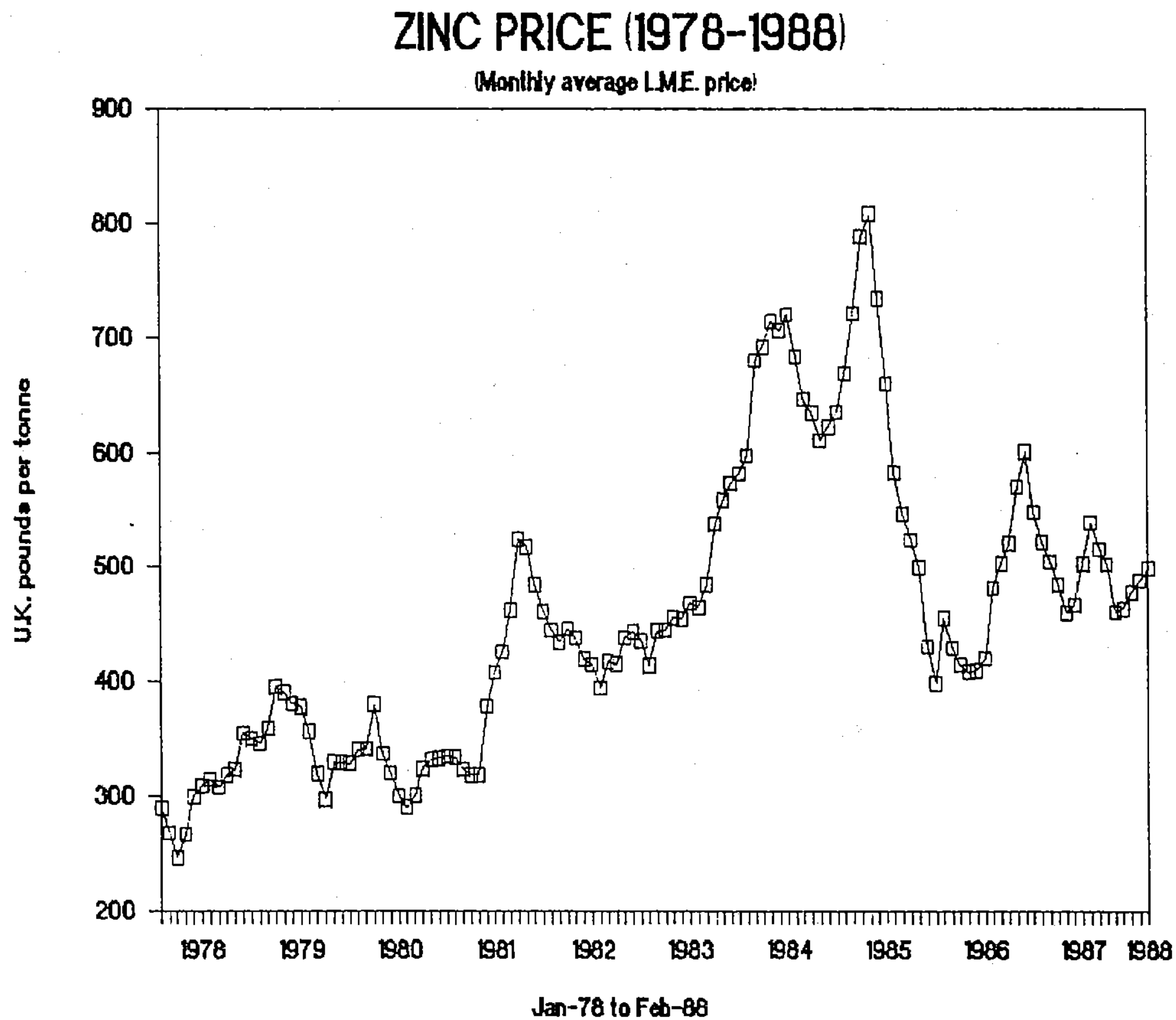

FIGURE 8

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Zinc Consumption by Country (Thousand metric tons)
(From the International Lead and Zinc Study Group)

	Annual Totals:				Year to date:	
	1983	1984	1985	1986	1986 JAN - NOV	1987 JAN - NOV
Europe	**1638**	**1683**	**1665**	**1706**	**1578**	**1578**
France	271	282	247	260	243	226
Germany, F.R.	405	425	410	434	405	417
Italy	208	210	218	232	212	228
Spain	107	101	103	100	93	105
United Kingdom	117	182	189	182	168	174
Yugoslavia	90	102	105	90	83	83
Others	440	381	393	408	374	345
Africa	**156**	**169**	**153**	**164**	**149**	**158**
Algeria	15	16	9	21	19	19
Egypt	17	20	18	18	17	16
South Africa	83	90	84	83	75	86
Others	41	43	42	42	38	37
America	**1360**	**1470**	**1491**	**1528**	**1404**	**1498**
Brazil	102	114	141	151	136	152
Canada	144	146	156	146	131	142
Mexico	88	101	101	92	85	100
United States	924	980	962	999	923	962
Others	102	129	131	140	129	142
Asia	**1307**	**1304**	**1341**	**1418**	**1301**	**1315**
India	125	130	130	134	123	119
Japan	771	774	780	753	691	666
Korea, Rep.	109	124	120	154	141	183
Others	302	276	311	377	346	347
Oceania	**102**	**97**	**108**	**101**	**95**	**93**
Australia	83	77	87	81	76	75
New Zealand	19	20	21	20	19	18
Total	**4563**	**4723**	**4758**	**4917**	**4527**	**4642**
Monthly Average	380	394	397	410	412	422

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9.3 Silver

9.3.1 Uses

In the undeveloped countries the main use of silver is in jewellery compared to the developed countries where it is used in more serviceable areas. (USBM, 1985)

The major use of silver is in the production of photographic materials. Following that, the other major use is electronic and electrical products where Silver properties of high electrical and thermal conductivity and its resistance to corrosion are used. Also pure silver is used in low to medium current switching devices but it is often alloyed with copper to improve resistance to arcing and wear characteristics.

Silver is used in making batteries as it has a high energy per unit size and weight, but due to their expense their use is limited to space and defence.

The other main area is for decorative purposes. This can be broken down into three areas jewellery, silverware (silver-copper alloy) and silver plating.

Mirrors, calalysts, dental analgams, coins, medallions and solder are all minor uses of silver.

9.3.2 Consumption

The following table details the recent consumption of silver by end use for the United States.

US INDUSTRIAL CONSUMPTION OF SILVER BY END USE (Mkg)

	1986	1985	1984
Photographic materials	1.81	1.80	1.72
Electrical and electronic products:			
Batteries	0.12	0.08	0.08
Contacts and conductors	0.95	0.86	0.80
Jewellery	0.24	0.18	0.18
Silver Ware	0.22	0.22	0.22
Brazing alloys and solders	0.20	0.17	0.18
Commemoratives	0.13	0.08	0.08
All other areas	0.30	0.30	0.30
Total industrial consumption	3.97	3.69	3.56

Source: Handy & Harman

9.3.3 Price Outlook

The silver price over the last ten years is given in the following figure. The price took a very spectacular rise in 1980 when the Hunt brothers attempted to corner the market, but after they failed the price fell until 1982, when there was a considerable improvement. The peak in 1983 has not been repeated and the price has been falling steadily since, with only a slight improvement in 1987.

The price of silver is projected to lie between a low value of US$5.00/ounce and a high of US$7.00/ounce with a medium value of US$6.00/ounce. This range is projected over short and long terms. These values equate to AUS$6.25, 7.50 and 8.75 per ounce.

9.4 Concentrate Sales

As detailed in the previous section there will be four types of ore from the three Areas of the deposit. The estimated recoveries of lead, zinc and silver and the lead and zinc concentrates grades are given in section 5.

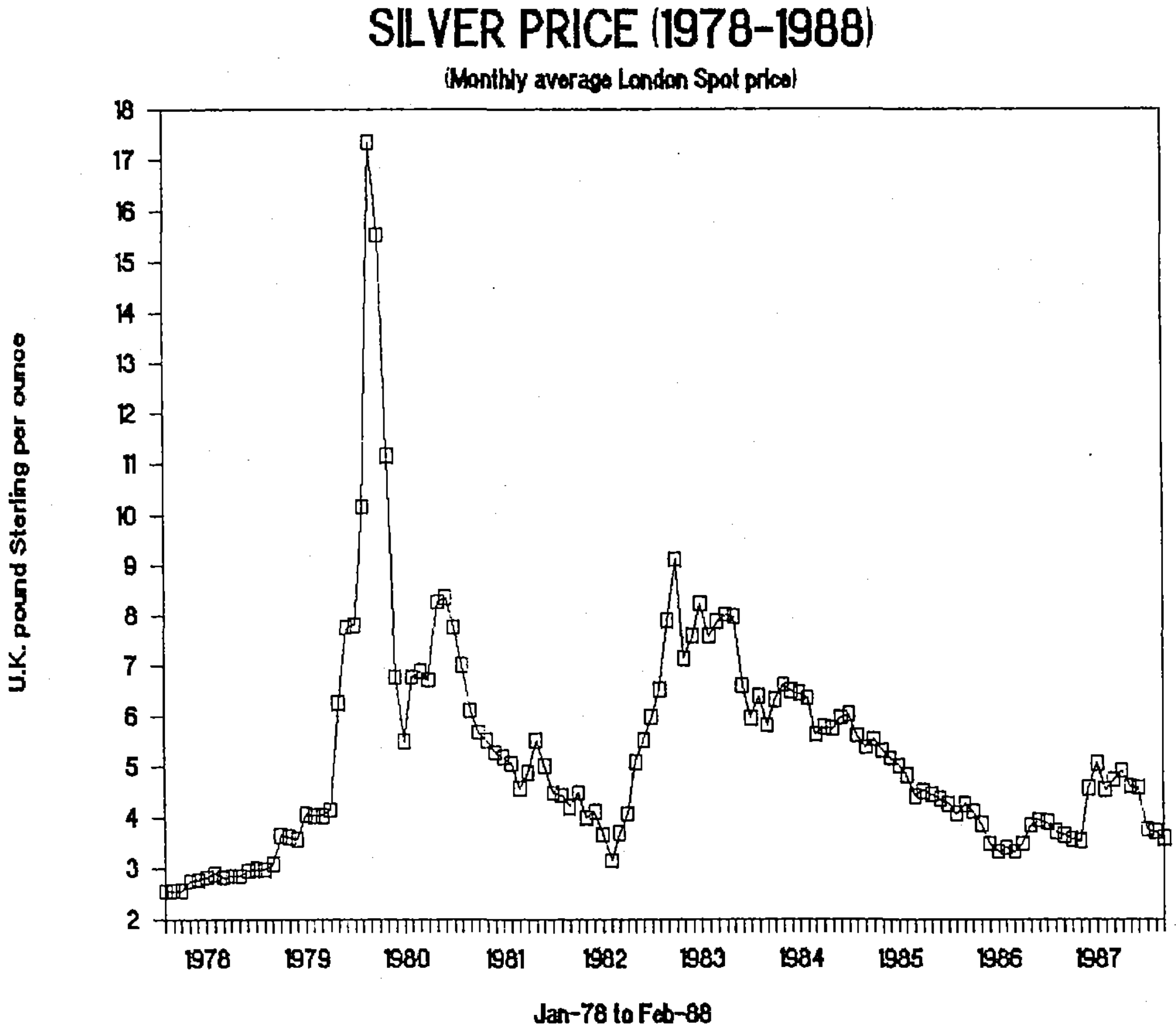

FIGURE 9

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It is assumed that the concentrates will be trucked to the rail head at Rosebery where they will be railed to the port of Burnie. From there they will be shipped to a smelter. For zinc concentrate, it may be Risdon, Tasmania, main-land Australia or Japan. For lead concentrate it may be main-land Australia or Japan.

The product value given below assumes freight overseas and smelter and refining charges in Japan. It was assumed that the concentrates had no impurities and so did not incur any penalties from the smelters.

	Concentrate tonnes/annum	Value AUS$/tonne	Value (Ore Basis) AUS$/tonne
Zinc	19,053	262	15.34
Lead	40,077	467	57.56
			72.90

These values are based on 1000 tonnes per day throughput and metal prices of:

Lead	AUS$	950/tonne
Zinc	AUS$	1350/tonne
Silver	AUS$	8/oz

For the individual ore types the product values are:

	$/t of ore
Area 1	
0-50 metres	28.20
50-70 metres	56.40
Area 2	109.20
Area 3	90.30
Average	72.90

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10.0 Economic Evaluation

The capital and operating parameters and product value used in the base case analysis were from the previous sections. The project will be subject to the Tasmanian Governments royalty and Australian Federal Income Tax and these costs are included. All mining reserves were included for the base case.

The result of the base case analysis was that the project can repay its capital investment only (i.e. the discounted cashflow rate of return (DCFROR) is zero (Table 10.1). As has been mentioned in previous sections, this result is taken after optimistic recovery factors are used. For the project to meet a 15% rate of return, which is considered necessary for the project to become attractive for investment, the prices received for all three metals must increase by 30% (table 10.2). The second analysis is without the reserves in Area 2 as it is doubtful whether these reserves can be mined due to the deep weathering found near the fault (see details in geological report accompanying). The result of the second analysis is a DCFROR of -20% (see table 10.3). For this case to reach necessary returns there needs to be a 90% increase in the prices of the three metals (see table 10.4).

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CYPRUS GOLD AUSTRALIA CORPORATION
OCEANA PROJECT

TABLE 10.1

(All figures in Thousands Australian Dollars 1st Quarter 1988) 08-Jun-88

YEAR		0	1	2	3	4	5	6	TOTAL
PRODUCTION (tonnes)									
Area 1 - Open pit									
0-50m			365000	185000					550000
50-70m				180000	20000				200000
Area 2 - U/G					345000	95000			440000
Area 3 - U/G						270000	365000	45000	680000
TOTAL PRODUCTION		0	365000	365000	365000	365000	365000	45000	1870000
CAPITAL COSTS									
Opencut		1620							1620
Underground				12150	1500				13650
Plant		16100							16100
Admin & Services		1700							1700
Working Capital		1700							1700
TOTAL CAPITAL		21120	0	12150	1500	0	0	0	34770
OPERATING COSTS	$/t								
Mine Open pit 0-50m	10.50	0	3833	1943	0	0	0	0	5775
Mine Open pit 50-70m	27.50	0	0	4950	550	0	0	0	5500
Mine U/G - Area 2	41.10	0	0	0	14180	3905	0	0	18084
Mine U/G - Area 3	41.10	0	0	0	0	11097	15002	1850	27948
Plant (Area 1&2)	15.90	0	5804	5804	5804	1511	0	0	18921
Plant (Area 3)	15.10	0	0	0	0	4077	5512	680	10268
Admin & Services	2.50	0	913	913	913	913	913	113	4675
TOTAL OPERATING		0	10549	13609	21446	21502	21426	2642	91171
REVENUE	$/t								
Area 1 - Open pit									
0-50m	28.20	0	10293	5217	0	0	0	0	15510
50-70m	56.40	0	0	10152	1128	0	0	0	11280
Area 2 - U/G	109.20	0	0	0	37674	10374	0	0	48048
Area 3 - U/G	90.30	0	0	0	0	24381	32960	4064	61404
REVENUE FROM PRODUCT		0	10293	15369	38802	34755	32960	4064	136242
Other Revenue (Salvage Value)			0	0	0	0	0	1488	1488
ROYALTY		0	257	384	1164	869	824	139	3637
INCOME TAX		0	0	0	3853	2251	1597	16	5024
CASHFLOW		-21120	-513	-10774	10840	10134	9113	2755	435

NPV @15% -9625
Rate Of Return 0%

Metal Prices
Lead 950 Aus$/t
Zinc 1350 Aus$/t
Silver 8.00 Aus$/oz

REVENUE INCREASE 0%

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CYPRUS GOLD AUSTRALIA CORPORATION
OCEANA PROJECT

TABLE 10.1

ROYALTY & TAX CALCULATION

08-Jun-88

	0	1	2	3	4	5	6	TOTAL
Revenue from product	0	10293	15369	38802	34755	32960	4064	136242
Other Revenue (Salvage)	0	0	0	0	0	0	1488	1488
Depreciation (20%)	0	3884	3884	6314	6614	6614	2730	27310
Operating Costs	0	10549	13609	21446	21502	21426	2642	91171
Annual Rylty Profit	0	-4140	-2124	11043	6640	4920	180	16519
Royalty Rate	5.0%	5.0%	5.0%	6.0%	5.0%	5.0%	5.0%	
Royalty - Profit based (A)	0	-207	-106	663	332	246	9	936
Proceeds	0	10293	15369	38802	34755	32960	5551	137730
Transport Costs	0	1170	1710	3995	3463	3240	399	13978
Net Proceeds	0	9123	13659	34807	31292	29719	5152	123751
Royalty Rate	2.5%	2.5%	2.5%	3.0%	2.5%	2.5%	2.5%	
Royalty - Net Proc. based (B)	0	257	384	1164	869	824	139	3637
Royalty (greater of A or B)	0	257	384	1164	869	824	139	3637
Income Bf. Tax - Royalty	0	-4397	-2508	9878	5771	4096	41	12881
Tax Rate	39.0%	39.0%	39.0%	39.0%	39.0%	39.0%	39.0%	39.0%
Tax Payable	0	0	0	3853	2251	1597	16	5024
Income after Tax	0	-4397	-2508	6026	3520	2499	25	7858

CYPRUS GOLD AUSTRALIA CORPORATION
OCEANA PROJECT

TABLE 10.2

(All figures in Thousands Australian Dollars 1st Quarter 1988)

08-Jun-88

YEAR		0	1	2	3	4	5	6	TOTAL
PRODUCTION (tonnes)									
Area 1 - Open pit									
0-50m			365000	185000					550000
50-70m				180000	20000				200000
Area 2 - U/G					345000	95000			440000
Area 3 - U/G						270000	365000	45000	680000
TOTAL PRODUCTION		0	365000	365000	365000	365000	365000	45000	1870000
CAPITAL COSTS									
Opencut		1620							1620
Underground				12150	1500				13650
Plant		16100							16100
Admin & Services		1700							1700
Working Capital		1700							1700
TOTAL CAPITAL		21120	0	12150	1500	0	0	0	34770
OPERATING COSTS	$/t								
Mine Open pit 0-50m	10.50	0	3833	1943	0	0	0	0	5775
Mine Open pit 50-70m	27.50	0	0	4950	550	0	0	0	5500
Mine U/G - Area 2	41.10	0	0	0	14180	3905	0	0	18084
Mine U/G - Area 3	41.10	0	0	0	0	11097	15002	1850	27948
Plant (Area 1&2)	15.90	0	5804	5804	5804	1511	0	0	18921
Plant (Area 3)	15.10	0	0	0	0	4077	5512	680	10268
Admin & Services	2.50	0	913	913	913	913	913	113	4675
TOTAL OPERATING		0	10549	13609	21446	21502	21426	2642	91171
REVENUE	$/t								
Area 1 - Open pit									
0-50m	28.20	0	10293	5217	0	0	0	0	15510
50-70m	56.40	0	0	10152	1128	0	0	0	11280
Area 2 - U/G	141.96	0	0	0	48976	13486	0	0	62462
Area 3 - U/G	117.39	0	0	0	0	31695	42847	5283	79825
REVENUE FROM PRODUCT		0	10293	15369	50104	45182	42847	5283	169078
Other Revenue (Salvage Value)			0	0	0	0	0	1488	1488
ROYALTY		0	257	384	1754	1581	1285	169	5431
INCOME TAX		0	0	0	8031	6039	5274	479	17130
CASHFLOW		-21120	-513	-10774	17375	16060	14863	3480	19370

NPV @15%	-185
Rate Of Return	15%

Metal Prices	
Lead	1235 Aus$/t
Zinc	1755 Aus$/t
Silver	10.40 Aus$/oz

REVENUE INCREASE 30%

CYPRUS GOLD AUSTRALIA CORPORATION
OCEANA PROJECT

TABLE 10.2

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ROYALTY & TAX CALCULATION

30-May-88

	0	1	2	3	4	5	6	TOTAL
Revenue from product	0	10293	15369	50104	45182	42847	5283	169078
Other Revenue (Salvage)	0	0	0	0	0	0	1488	1488
Depreciation (20%)	0	3884	3884	6314	6614	6614	2730	27310
Operating Costs	0	10549	13609	21446	21502	21426	2642	91171
Annual Rylty Profit	0	-4140	-2124	22345	17066	14808	1399	49354
Royalty Rate	5.0%	5.0%	5.0%	7.0%	7.0%	6.0%	5.0%	
Royalty - Profit based (A)	0	-207	-106	1564	1195	888	70	3404
Proceeds	0	10293	15369	50104	45182	42847	6770	170565
Transport Costs	0	1170	1710	3995	3463	3240	399	13978
Net Proceeds	0	9123	13659	46109	41719	39607	6371	156587
Royalty Rate	2.5%	2.5%	2.5%	3.5%	3.5%	3.0%	2.5%	
Royalty - Net Proc. based (B)	0	257	384	1754	1581	1285	169	5431
Royalty (greater of A or B)	0	257	384	1754	1581	1285	169	5431
Income Bf. Tax - Royalty	0	-4397	-2508	20591	15485	13522	1229	43923
Tax Rate	39.0%	39.0%	39.0%	39.0%	39.0%	39.0%	39.0%	39.0%
Tax Payable	0	0	0	8031	6039	5274	479	17130
Income after Tax	0	-4397	-2508	12561	9446	8249	750	26793

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CYPRUS GOLD AUSTRALIA CORPORATION
OCEANA PROJECT

TABLE 10.3

(All figures in Thousands Australian Dollars 1st Quarter 1988)

08-Jun-88

YEAR		0	1	2	3	4	5	6	TOTAL
PRODUCTION (tonnes)									
Area 1 - Open pit									
0-50m			365000	185000					550000
50-70m				180000	20000				200000
Area 2 - U/G									0
Area 3 - U/G					345000	335000			680000
TOTAL PRODUCTION		0	365000	365000	365000	335000	0	0	1430000
CAPITAL COSTS									
Opencut		1620							1620
Underground				13650					13650
Plant		16100							16100
Admin & Services		1700							1700
Working Capital		1700							1700
TOTAL CAPITAL		21120	0	13650	0	0	0	0	34770
OPERATING COSTS	$/t								
Mine Open pit 0-50m	10.50	0	3833	1943	0	0	0	0	5775
Mine Open pit 50-70m	26.50	0	0	4770	530	0	0	0	5300
Mine U/G - Area 2	41.10	0	0	0	0	0	0	0	0
Mine U/G - Area 3	41.10	0	0	0	14180	13769	0	0	27948
Plant (Area 1&2)	15.90	0	5804	5804	318	0	0	0	11925
Plant (Area 3)	15.10	0	0	0	5210	5059	0	0	10268
Admin & Services	2.50	0	913	913	913	838	0	0	3575
TOTAL OPERATING		0	10549	13429	21150	19665	0	0	64791
REVENUE	$/t								
Area 1 - Open pit									
0-50m	28.20	0	10293	5217	0	0	0	0	15510
50-70m	56.40	0	0	10152	1128	0	0	0	11280
Area 2 - U/G	109.20	0	0	0	0	0	0	0	0
Area 3 - U/G	90.30	0	0	0	31154	30251	0	0	61404
REVENUE FROM PRODUCT		0	10293	15369	32282	30251	0	0	88194
Other Revenue (Salvage Value)			0	0	0	1488	0	0	1488
ROYALTY		0	257	384	807	793	0	0	2242
INCOME TAX		0	0	0	1447	1820	0	0	644
CASHFLOW		-21120	-513	-12094	8878	9460	0	0	-15389

NPV @15%	-16926	Metal Prices		REVENUE INCREASE	0%
Rate Of Return	-20%	Lead	950 Aus$/t		
		Zinc	1350 Aus$/t		
		Silver	8.00 Aus$/oz		

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CYPRUS GOLD AUSTRALIA CORPORATION
OCEANA PROJECT

TABLE 10.3

ROYALTY & TAX CALCULATION

30-May-88

	0	1	2	3	4	5	6	TOTAL
Revenue from product	0	10293	15369	32282	30251	0	0	88194
Other Revenue (Salvage)	0	0	0	0	1488	0	0	1488
Depreciation (20%)	0	3884	3884	6614	6614	0	0	27610
Operating Costs	0	10549	13429	21150	19665	0	0	64791
Annual Rylty Profit	0	-4140	-1944	4518	5460	0	0	3895
Royalty Rate	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	
Royalty - Profit based (A)	0	-207	-97	226	273	0	0	195
Proceeds	0	10293	15369	32282	31738	0	0	89682
Transport Costs	0	1170	1710	3187	2974	0	0	9041
Net Proceeds	0	9123	13659	29095	28764	0	0	80640
Royalty Rate	2.5%	2.5%	2.5%	2.5%	2.5%	2.5%	2.5%	
Royalty - Net Proc. based (B)	0	257	384	807	793	0	0	2242
Royalty (greater of A or B)	0	257	384	807	793	0	0	2242
Income Bf. Tax - Royalty	0	-4397	-2328	3711	4666	0	0	1652
Tax Rate	39.0%	39.0%	39.0%	39.0%	39.0%	39.0%	39.0%	39.0%
Tax Payable	0	0	0	1447	1820	0	0	644
Income after Tax	0	-4397	-2328	2264	2846	0	0	1008

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CYPRUS GOLD AUSTRALIA CORPORATION
OCEANA PROJECT

TABLE 10.4

(All figures in Thousands Australian Dollars 1st Quarter 1988) 08-Jun-88

YEAR		0	1	2	3	4	5	6	TOTAL
PRODUCTION (tonnes)									
Area 1 - Open pit									
0-50m			365000	185000					550000
50-70m				180000	20000				200000
Area 2 - U/G									0
Area 3 - U/G					345000	335000			680000
TOTAL PRODUCTION		0	365000	365000	365000	335000	0	0	1430000
CAPITAL COSTS									
Opencut		1620							1620
Underground				13650					13650
Plant		16100							16100
Admin & Services		1700							1700
Working Capital		1700							1700
TOTAL CAPITAL		21120	0	13650	0	0	0	0	34770
OPERATING COSTS	$/t								
Mine Open pit 0-50m	10.50	0	3833	1943	0	0	0	0	5775
Mine Open pit 50-70m	26.50	0	0	4770	530	0	0	0	5300
Mine U/G - Area 2	41.10	0	0	0	0	0	0	0	0
Mine U/G - Area 3	41.10	0	0	0	14180	13769	0	0	27948
Plant (Area 1&2)	15.90	0	5804	5804	318	0	0	0	11925
Plant (Area 3)	15.10	0	0	0	5210	5059	0	0	10268
Admin & Services	2.50	0	913	913	913	838	0	0	3575
TOTAL OPERATING		0	10549	13429	21150	19665	0	0	64791
REVENUE	$/t								
Area 1 - Open pit									
0-50m	28.20	0	10293	5217	0	0	0	0	15510
50-70m	56.40	0	0	10152	1128	0	0	0	11280
Area 2 - U/G	207.48	0	0	0	0	0	0	0	0
Area 3 - U/G	171.57	0	0	0	59192	57476	0	0	116668
REVENUE FROM PRODUCT		0	10293	15369	60320	57476	0	0	143458
Other Revenue (Salvage Value)			0	0	0	1488	0	0	1488
ROYALTY		0	257	384	2279	2288	0	0	5208
INCOME TAX		0	0	0	11808	11855	0	0	21040
CASHFLOW		-21120	-513	-12094	25083	25156	0	0	16513

NPV @15% 144
Rate Of Return 15%

Metal Prices
Lead 1805 Aus$/t
Zinc 2565 Aus$/t
Silver 15.20 Aus$/oz

REVENUE INCREASE 90%

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CYPRUS GOLD AUSTRALIA CORPORATION
OCEANA PROJECT

TABLE 10.4

ROYALTY & TAX CALCULATION

30-May-88

	0	1	2	3	4	5	6	TOTAL
Revenue from product	0	10293	15369	60320	57476	0	0	143458
Other Revenue (Salvage)	0	0	0	0	1488	0	0	1488
Depreciation (20%)	0	3884	3884	6614	6614	0	0	27610
Operating Costs	0	10549	13429	21150	19665	0	0	64791
Annual Rylty Profit	0	-4140	-1944	32556	32685	0	0	59158
Royalty Rate	5.0%	5.0%	5.0%	7.0%	7.0%	5.0%	5.0%	
Royalty - Profit based (A)	0	-207	-97	2279	2288	0	0	4263
Proceeds	0	10293	15369	60320	58963	0	0	144945
Transport Costs	0	1170	1710	3187	2974	0	0	9041
Net Proceeds	0	9123	13659	57133	55989	0	0	135904
Royalty Rate	2.5%	2.5%	2.5%	3.5%	3.5%	2.5%	2.5%	
Royalty - Net Proc. based (B)	0	257	384	2111	2064	0	0	4816
Royalty (greater of A or B)	0	257	384	2279	2288	0	0	5208
Income Bf. Tax - Royalty	0	-4397	-2328	30277	30397	0	0	53950
Tax Rate	39.0%	39.0%	39.0%	39.0%	39.0%	39.0%	39.0%	39.0%
Tax Payable	0	0	0	11808	11855	0	0	21040
Income after Tax	0	-4397	-2328	18469	18542	0	0	32909

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References

King,D and Blisset, A.H. A statistical and Geological Review of the Zeehan Silver-Lead Mines, West Tasmania. Proc. Aust. Inst. Min. Met., No. 228, December 1968.

Blissett, A.H. Geological Survey Explanatory Report of Zeehan one mile Geological Map. Tasmanian Department of Mines, 1962.

Jack, R. Report on the Oceana Mine - Zeehan. Technical Report No. 5. Tasmania Department of Mines, 1960.

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APPENDIX 1

REVIEW OF E.Z. TESTWORK

Some very preliminary flotation tests were carried out by Electrolytic Zinc. The objective of the tests was restricted to checking the response of Oceana ore to the standard procedure used to test Rosebery and Que River ore.

Three samples were tested - high and low grade samples from the northern zone and a sample of lead ore from the southern zone.

The results are summarised in the table below.

E.Z. concluded that northern zone ore is probably incompatible with the present Rosebery feed, because of its poor recovery properties. However they acknowledged that the results obtained did not represent maximum recoveries or typical concentrate grades to be expected from the ore if it were treated separately using different conditions. They felt that southern zone ore is reasonably compatible, though its low zinc content makes it less attractive than the current feed which contains 12-14% Zn.

In reviewing the data it is noted that one of the northern zone samples is extremely high grade, while the other is very low grade. Thus neither of them are typical of the orebody. Another point is that E.Z. produced a copper concentrate according to their normal practice. The Oceana ore, however, only contains about one tenth or less copper, so that the production of a separate copper concentrate is inappropriate.

The data does not include anything on impurities.

Comments on the results for each sample are as follows:

Sample No. 21335, Hole 79-2, 111-112m (Northern Zone)

High grade lead and zinc concentrates were achieved, but losses to tailing were high, especially for zinc. E.Z. reported the material as being slightly oxidised and postulated that oxidation could be part of the reason for the high losses. If this is the case, sodium sulphide activation may be of benefit. Silver loss to tailing was similar to lead. No comparative silver data was given for Rosebery.

Sample No. 21349, Hole 79-2, 125-126m (Northern Zone)

Lead and zinc concentrate grades, in fact, matched Rosebery data, despite E.Z.'s statement to the contrary. However, losses of lead and zinc were very high. Silver content of tailing was low at 1 g/t.

Sample No. 21609, Hole 80-4, 304-305m (Southern Zone)

A good recovery of relatively high grade lead concentrate was achieved. Loss to tailing was acceptable. Silver recovery was also good. The zinc head grade is too low for the production of a separate zinc concentrate.

Future metallurgical testwork should be undertaken on typical average grade northern zone material. If it is determined that a significant part of the zone is weathered, then two composites should be tested, weathered and unweathered. Once the best treatment conditions have been established, variations in response should then be examined by testing samples of various grades.

A similar approach will be necessary for southern zone ore if this is to be included in the reserves.

The testwork would typically examine such factors as grind size, flotation times, various reagent combinations including sodium sulphide activation, and the effect of concentrate regrind. Parallel mineralogical work will be essential.

RESULTS OF E.Z. FLOTATION TESTS

SAMPLE	PRODUCT	WT %	ASSAY				RECOVERY			
			Pb	Zn	Cu	Ag	Pb	Zn	Cu	Ag
No. 21335 Hole 79-2 111-112 m High Grade North Zone	Cu Conc.	1.37	45.50	25.50	0.23	985	2.45	0.85	3.96	6.74
	Pb Conc.	25.44	67.70	12.20	0.07	451	67.70	7.55	22.40	57.31
	Zn Conc.	24.84	11.10	52.30	0.08	99	10.84	31.61	24.99	12.28
	Final Tail.	48.35	10.00	51.00	0.08	98	19.01	59.99	48.65	23.67
	Head Assay		25.44	41.0	0.08	200.2				
No. 21349 Hole 79-2 125-126 m Low Grade North Zone	Cu Conc.	1.20	11.00	4.70	0.08	99	7.48	2.45	4.11	21.16
	Pb Conc.	2.31	24.20	21.50	0.03	115	31.57	21.52	2.95	47.31
	Zn Conc.	2.78	5.10	26.00	0.11	30	8.01	31.33	13.03	14.85
	Final Tail.	93.71	1.00	1.10	0.02	1	52.94	44.69	79.91	16.68
	Head Assay		1.77	2.31	0.02	5.6				
No. 21609 Hole 80-4 304-305 m Lead ore South Zone	Cu Conc.	1.20	53.30	2.10	0.25	838	7.02	5.03	11.00	17.31
	Pb Conc.	7.63	70.00	1.60	0.03	396	58.84	24.45	8.43	52.02
	Zn Conc.	5.22	38.00	5.10	0.09	226	21.84	53.31	17.29	20.31
	Final Tail.	85.95	1.30	0.10	0.02	7	12.31	17.21	63.28	10.36
	Head Assay		9.08	0.50	0.03	58.1				
Rosebery Standard Test Rosebery Ore 77 % Que River Ore 23%	Cu Conc.	6.91	18.10	24.57	7.17		32.18	17.47	61.19	
	Pb Conc.	7.77	19.27	21.83	1.50		38.52	17.45	14.40	
	Zn Conc.	29.56	2.72	19.46	0.48		20.69	59.18	17.52	
	Final Tail.	55.76	0.60	1.03	0.10		8.61	5.91	6.89	
	Head Assay		3.89	9.72	0.81					

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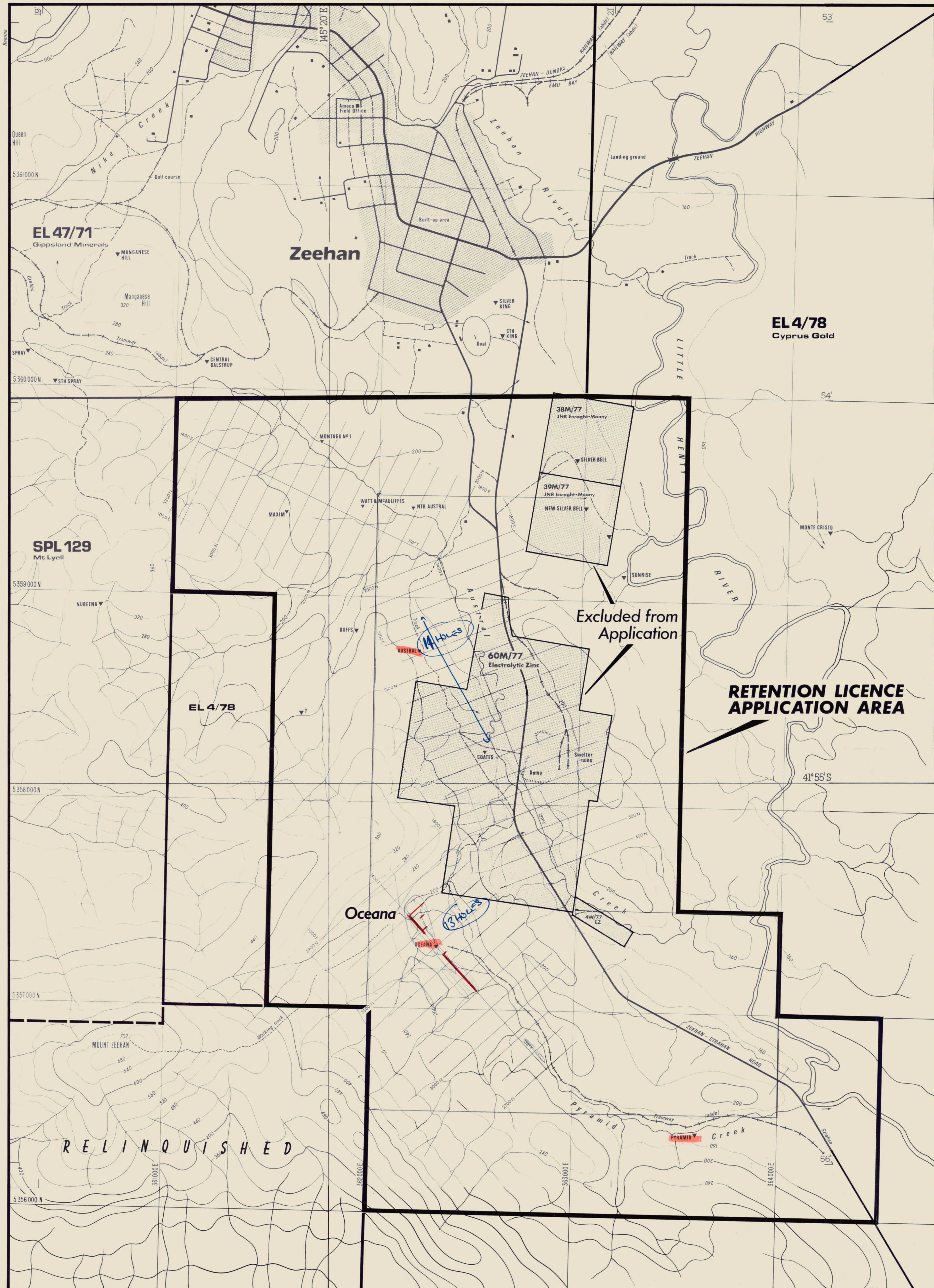

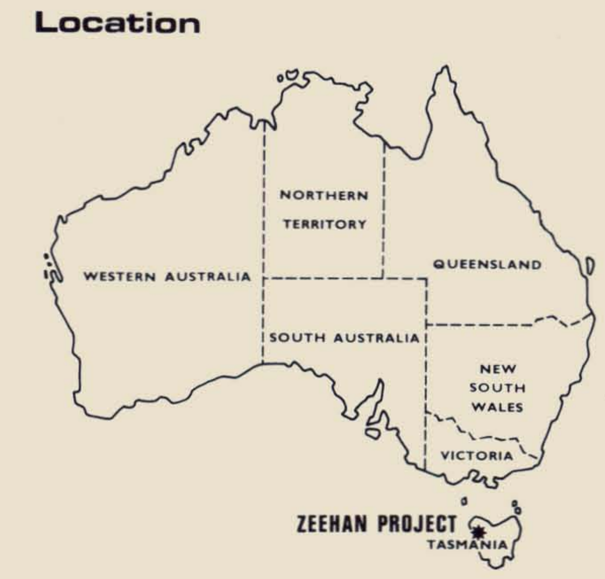

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Project	ZEEHAN	Nº A-78-60B
Project Partner		

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RETENTION LICENCE
APPLICATION AREA

Map Ref. ANG	K-55-5	Latitude	41°55'S	Longitude	145°20'E
Surveyed	P. Jones	Date	1980	Scale	1:10000
Drawn	S. Fowler	Date	May 1983	Drawing Nº	M83-2055

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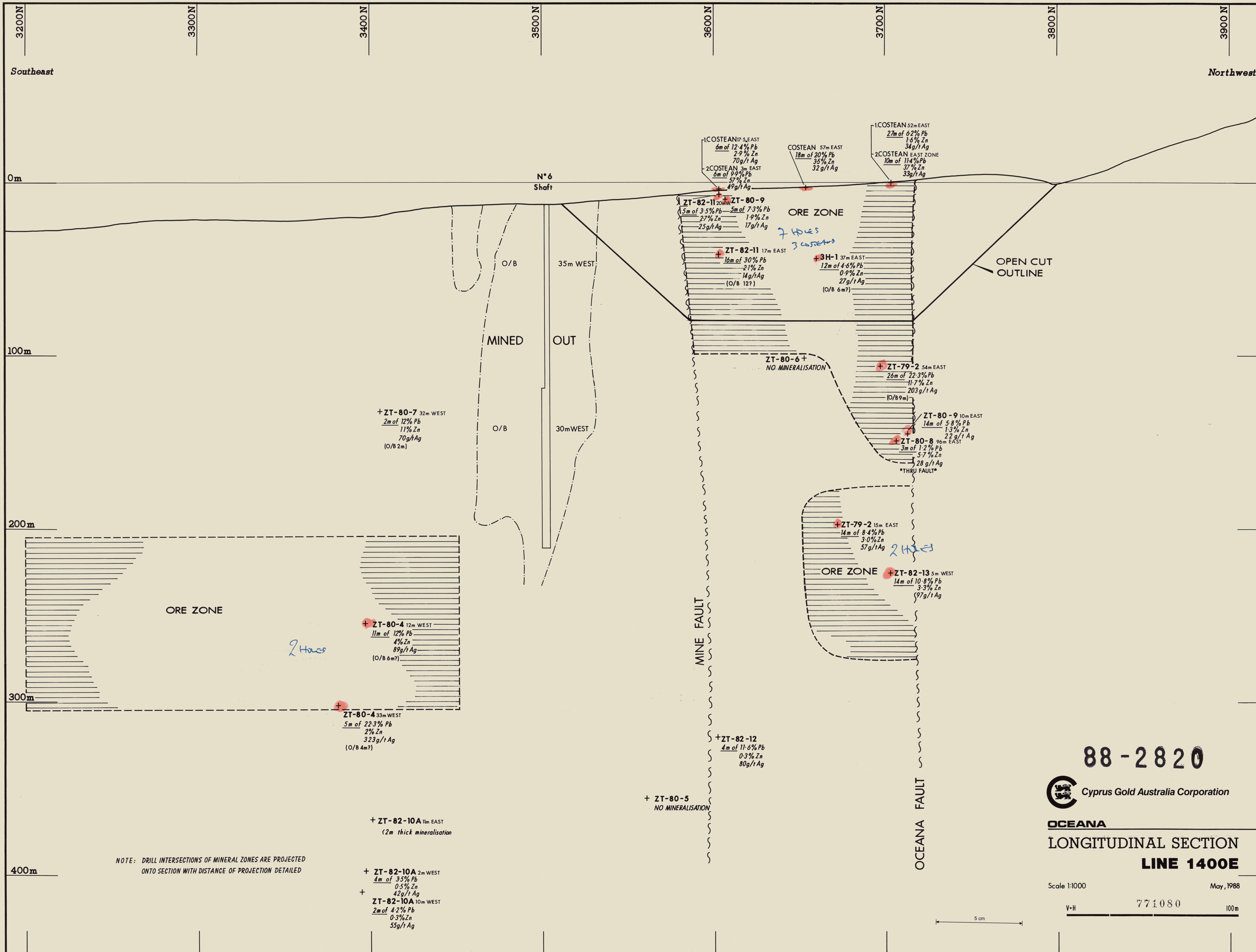
Enclosure A



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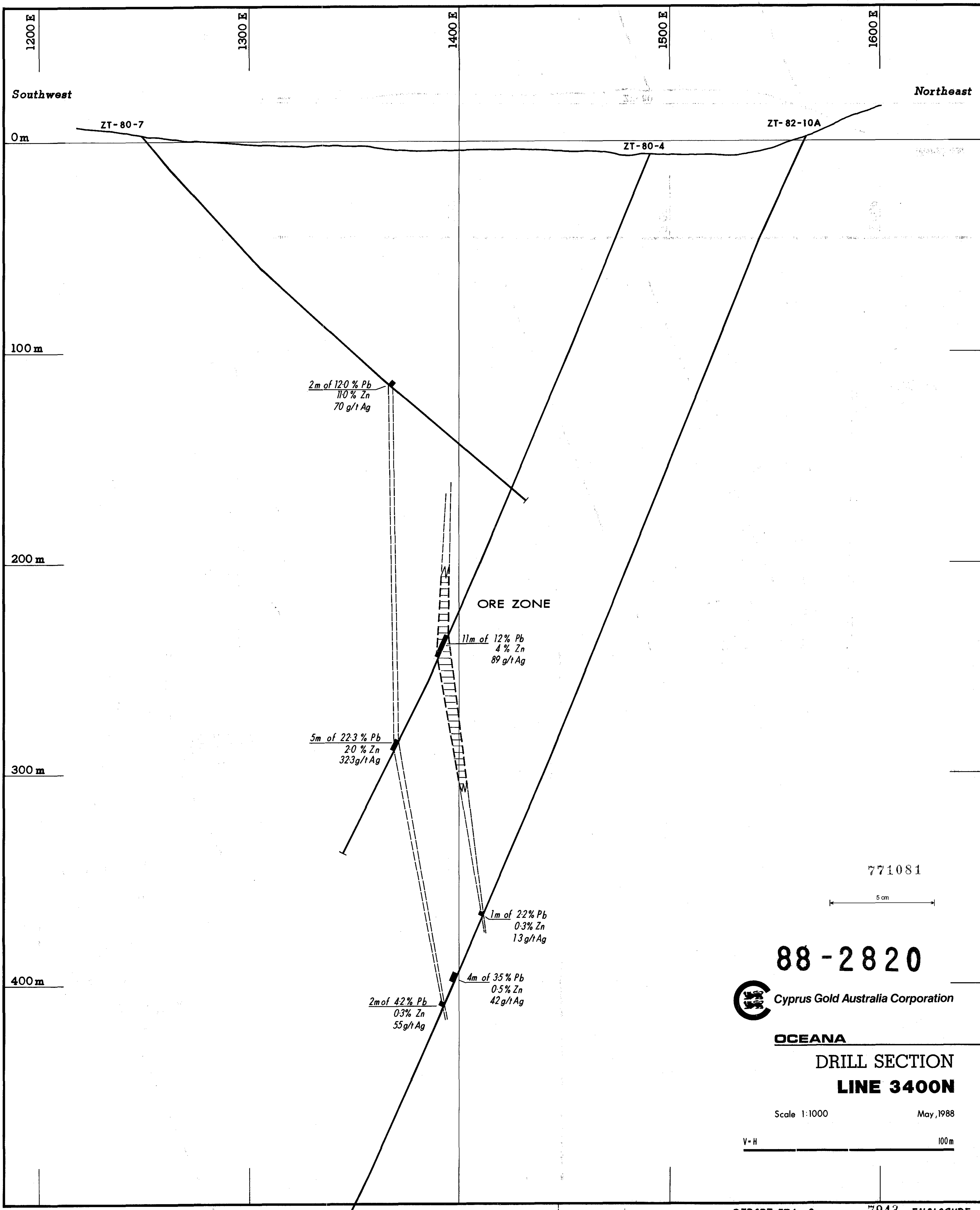
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1300 E
1400 E
1500 E
1600 E
Southwest
Northeast
ZT-80-7
ZT-82-10A
ZT-80-4
0 m
100 m
200 m
300 m
400 m
2m of 12·0 % Pb
11·0 % Zn
70 g/t Ag
ORE ZONE
11m of 12 % Pb
4 % Zn
89 g/t Ag
5m of 22·3 % Pb
2·0 % Zn
32·3 g/t Ag
1m of 2·2% Pb
0·3% Zn
13 g/t Ag
4m of 3·5 % Pb
0·5 % Zn
42 g/t Ag
2m of 4·2 % Pb
0·3% Zn
55 g/t Ag
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5 cm
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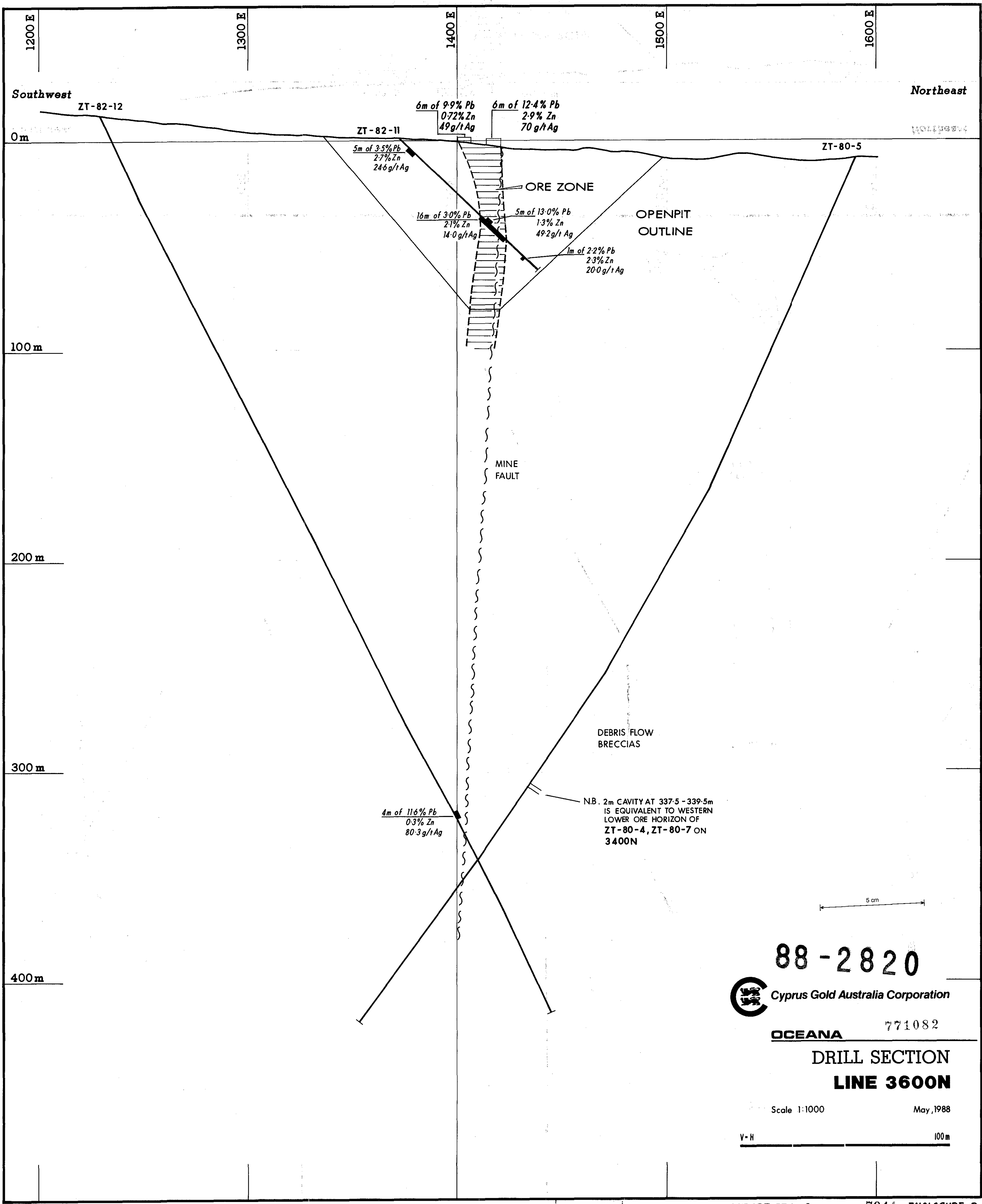
OCEANA
DRILL SECTION
LINE 3400N
Scale 1:1000
May, 1988
V=H
100m


1200 E
1300 E
1400 E
1500 E
1600 E
Southwest
Northeast
ZT-82-12
ZT-82-11
ZT-80-5
0 m
100 m
200 m
300 m
400 m
6m of 9·9% Pb
0·72% Zn
49 g/t Ag
6m of 12·4% Pb
2·9% Zn
70 g/t Ag
5m of 3·5% Pb
2·7% Zn
24·6 g/t Ag
ORE ZONE
16m of 3·0% Pb
2·1% Zn
14·0 g/t Ag
5m of 13·0% Pb
1·3% Zn
49·2 g/t Ag
OPENPIT
OUTLINE
1m of 2·2% Pb
2·3% Zn
20·0 g/t Ag
MINE
FAULT
DEBRIS FLOW
BRECCIAS
N.B. 2m CAVITY AT 337·5 - 339·5m
IS EQUIVALENT TO WESTERN
LOWER ORE HORIZON OF
ZT-80-4, ZT-80-7 ON
3400N
4m of 11·6% Pb
0·3% Zn
80·3 g/t Ag
5 cm
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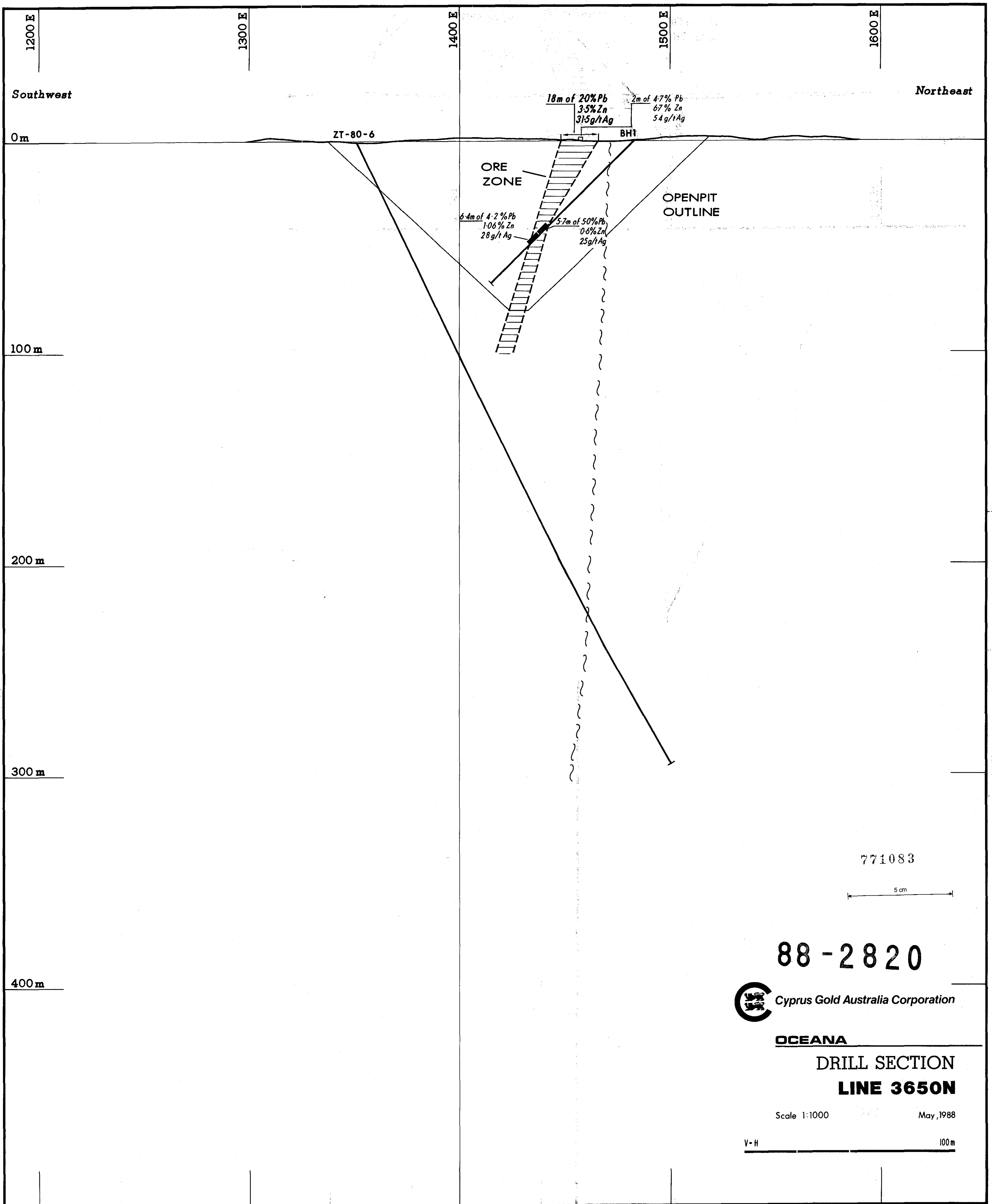
OCEANA
DRILL SECTION
LINE 3600N
Scale 1:1000
May, 1988
V=H
100m



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OCEANA

DRILL SECTION

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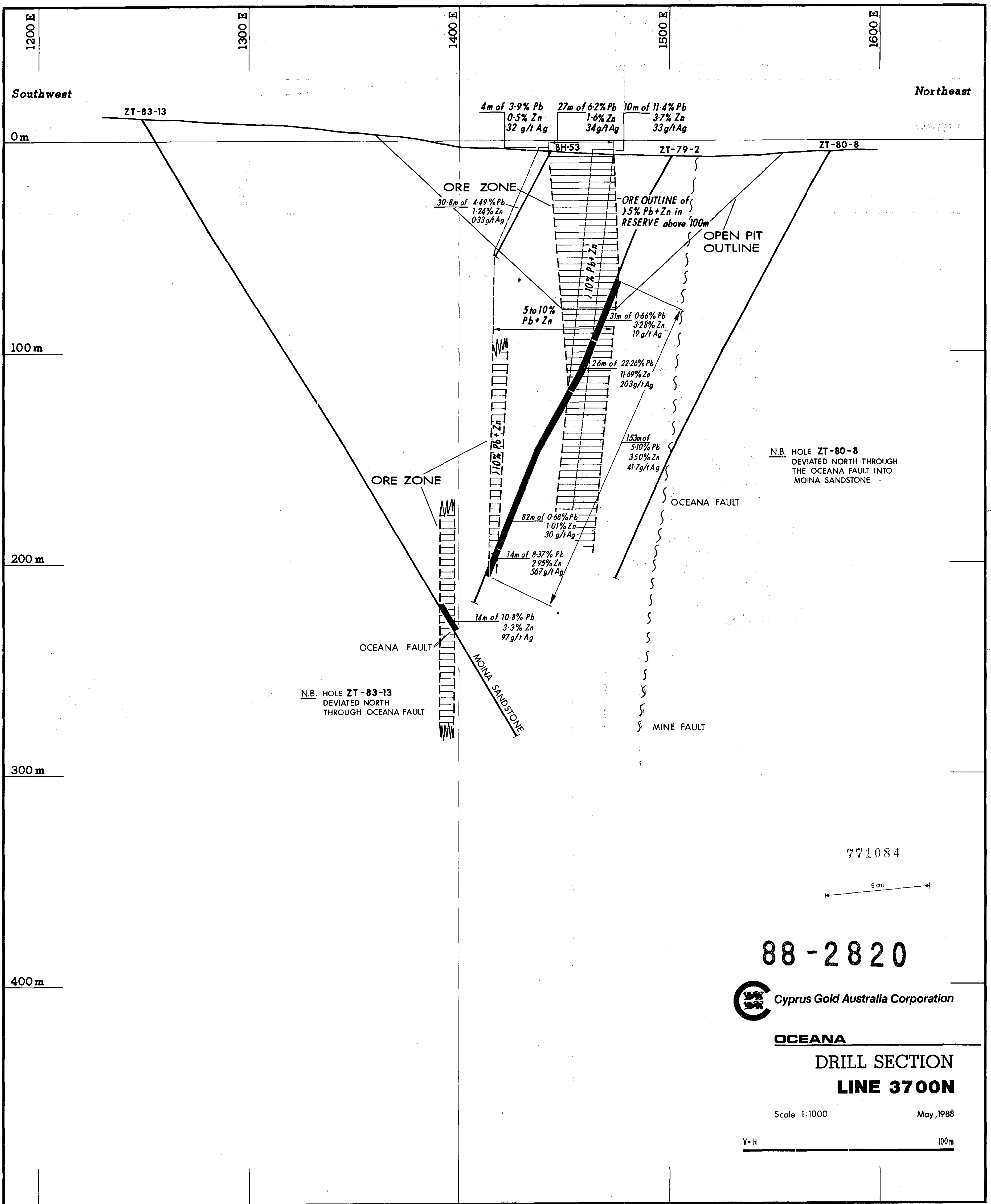
Scale 1:1000 May, 1988

V=H 100 m

REPORT 574 ·2· 7945 ENCLOSURE E



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OCEANA

DRILL SECTION

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V=H 100m

REPORT 574 ·2· 7946 ENCLOSURE F