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GREAT PYRAMID TIN DEPOSIT
NORTHEAST TASMANIA
RESOURCE ESTIMATES

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GREAT PYRAMID TIN DEPOSIT

This compilation contains the following reports, correspondence, memos etc:

- Geological Resource Estimates, Hall D.B., Wright R., 1983-4.
- Mining Study, Kaleta S., 1983.
- Preliminary Metallurgical Investigation of Tin Ore Deposit, Mineral Deposits Ltd, 1984 (Report No. 16.020.1/1)
- Correspondence regarding Report No. 16.020.1/1, Mineral Deposits Ltd, 1984.
- A Further Study of Cassiterite Occurrence in Middling Products from Testwork Report No. 16.020.1/1, Mineral Deposits, 1984.
- Great Pyramid Project Review, Platt J, 1984. Reply, Carter D.N., 1984.
- Comparison of Analytical Data from Bulk and their Equivalent Channel Samples at the Great Pyramid Tin Prospect, Carter D.N., 1985.
- Comments on data review, Foo K, 1985.

The various studies completed at Great Pyramid have often produced apparently conflicting results. However, the reserves calculated by Hall and Wright (viz 3.1 Mt at 0.22% Sn using 0.1% Sn cut off), supported by the grades reported by Carter, are considered to be sufficiently accurate to enable a meaningfuly ecnomic screening of the deposit to be made.

Under present economic conditions, the deposit is not considered viable.

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GREAT PYRAMID

GEOLOGICAL RESOURCE ESTIMATES
1983-1984

Report No.: 08.2959
BXP/AHO : D B Hall
BXH/TAS : R Wright

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CONTENTS

SUMMARY

INTRODUCTION

DATA RELIABILITY

DATA COLLATION AND PRESENTATION

PROBABLE RESOURCE BLOCK DEFINITION

PROBABLE RESOURCE CALCULATION

POSSIBLE RESOURCE ESTIMATE

UNDERGROUND SAMPLING

GRADE COMPARISONS

HIGH GRADE RESOURCE ESTIMATE

APPENDICES

I : SUMMARY OF DRILLING AND SAMPLING PROGRAMMES AT GREAT PYRAMID

II : CHANNEL AND BULK SAMPLING - LEVEL DISTRIBUTION

III : ADIT SAMPLING - CORRELATION WITH INTERPRETED RESOURCE OUTLINES

TABLES

1. TOTAL INTERCEPTS USED IN PROBABLE RESOURCE CALCULATION - 0.1% Sn CUT-OFF

2. NORTH BLOCK PROBABLE RESOURCE - 0.1% Sn CUT-OFF

3. SOUTH BLOCK PROBABLE RESOURCE - 0.1% Sn CUT-OFF

4. BROCKS BLOCK PROBABLE RESOURCE - 0.1% Sn CUT-OFF

5. TOTAL GREAT PYRAMID PROBABLE RESOURCE - 0.1% Sn CUT-OFF

6. TOTAL INTERCEPTS USED IN PROBABLE RESOURCE CALCULATION - 0.2% Sn CUT-OFF

7. NORTH BLOCK PROBABLE RESOURCE - 0.2% Sn CUT-OFF

8. SOUTH BLOCK PROBABLE RESOURCE - 0.2% Sn CUT-OFF

9. BROCKS BLOCK PROBABLE RESOURCE - 0.2% Sn CUT-OFF

PLANS

Level Plans and Cross Sections are in separate folders.

VOL 2 PLANS 580-602

VOL 3 PLANS 565-579

005

SUMMARY

Utilising all previous drilling data, a geological resource estimate has been calculated for the Great Pyramid tin project in northeast Tasmania.

The tin mineralization can be separated into three separate areas, referred to in this report as "Blocks", and the tonnage and grade are given below, using a 0.1% Sn cut-off.

North Block:	897,300	tonnes at 0.19% Sn
South Block:	1,980,200	tonnes at 0.24% Sn
Brocks Block:	252,700	tonnes at 0.17% Sn
Total:	3,130,200	tonnes at 0.22% Sn

It is stressed that this estimate is preliminary only, and has been calculated utilising the "envelope" principle, using data that is probably inaccurate. The estimate has been prepared in order to verify previous estimates prepared by Aberfoyle and BHP and to provide the basis of a preliminary economic evaluation.

INTRODUCTION

The Great Pyramid tin project is within EL10/80, northwest of Scamander in northeast Tasmania. SCOA is carrying out exploration of the area, after entering a Joint Venture agreement with BHP In 1982.

Various reports have been prepared by BHP and Shell, containing the results of work completed to date. These reports should be referred to for all relevant geological, geochemical, geophysical and drilling details.

Prior to this resource estimate by SCOA, three other estimates had been made, and are summarised below:

Assessment by	In Situ Reserves (indicated)	(indicated + "inferred")	Method of Determination
Paringa-Aberfoyle (1969-74)	4mT at .3%Sn		Based on cross-sections
BHP (1981)	4.10mT at .22% Sn	8.29mT at .19%Sn	Triangulation, based on levels (to 90 level)
BHP (1981)	3.3mT at .26%Sn	-	Rectangular, based on 170 level only with correction applied to all levels.
BHP (1981)	*(442,101T at .20% Sn)		
	**(658, 527T at .19% Sn)	-	Use of sludge assays in diamond holes (BPD series only).

- tonnage for North Block in "possible" category (*) with correction for estimated increase in grade from sludge assays (**).

The BHP estimates were based on all available data as at September 1981, at the conclusion of the BHP drilling programme. Details are presented in BHP report on EL 10/80 for the six months ended 21 January 1982. Their calculations assumed equal weighting of grades determined from all sources.

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This report outlines the methodology of the SCOA resource estimates, and the results obtained.

A summary of previous drilling, channel and bulk sampling programmes is included as Appendix I.

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DATA RELIABILITY

The aim of the resource estimate is to provide a tonnage and grade that can be used in a preliminary economic evaluation, in order to indicate whether the deposit has the potential to be economically viable.

The data used in the estimate is unreliable, and it is difficult to assign a degree of confidence to the results. Every assay value is assigned an equal weight, since there is no way of assigning a factor to allow for the different drilling methods.

From investigations made by BHP on sludge assays from diamond drilling, it is apparent that diamond drilling undervalues the grade. It is also probable that percussion drilling undervalues the grade, but this cannot be quantified. With the tin mineralization being associated with quartz in steeply dipping joints and fractures, it is more than likely that the percussion drilling has not recovered all the cassiterite.

There is no way of allowing for any variation in sample collection, preparation and analysis.

DATA COLLATION AND PRESENTATION

All available drillhole survey and assay data was collected, essentially from BHP, and an ex-BHP geologist (D. Steele) was employed to transfer it to the SHLMET system data sheets. This work was instigated by A. Donaldson of SCOA, and supervised by R. Wright (BXH, Tasmania).

An initial plot of the drillhole profiles on cross-sections indicated that all the Aberfoyle series of percussion drillholes were at variance with topographic and other drillhole levels. The collar elevations of all these holes were manually corrected to a best fit contour map of the hill and the BHP drilling data. Aberfoyle had apparently not surveyed in the percussion drillhole collars. BHP were able to locate some of the Aberfoyle holes and did survey the collar positions.

For presentation purposes a series of 24 cross sections were plotted on a bearing of 320° magnetic, essentially parallel to the line of the Aberfoyle grid drilling, and perpendicular to the mineralized joint set. Section interval is approximately 15 metres, and drillholes not on the section line are projected to the nearest section. Each hole is annotated with the hole number, drill collar co-ordinates and level, and the tin assay value in ppm for each sample intercept.

A series of level plans at 10 metre intervals were also produced, from RL 1070 to RL 1210. On each level plan, each drillhole intercept is annotated, and the weighted tin assay value for the 10 metre interval above the level is annotated.

PROBABLE RESOURCE BLOCK DEFINITION

In view of the uncertainties inherent in the assay data being used in this estimate, it was decided to use the envelope method to outline the resource.

A cut-off grade of 0.1% Sn was used to define the intercepts of each drillhole on a cross-section. The top of the block assigned to each hole was taken at the first intercept greater than 0.10% Sn, and the bottom of the block at the last intercept greater than 0.1% Sn, provided that the weighted average grade for the total intercept was also greater than 0.1% Sn. Varying numbers of individual intercepts of less than 0.1% Sn are thus included in the total intercept.

The boundaries of the blocks assigned to drillholes are placed at the mid-point between mineralized and barren holes, and where there is no adjacent hole on the section, the boundary has been placed 15 metres from the mineralized intercept.

Many of the Aberfoyle percussion drillholes bottomed in mineralization, either by design or due to loss of air pressure. In these cases the lower boundary has been extrapolated from adjacent holes, but usually kept close to the base of the holes.

The resource blocks are outlined on the E-W cross-sections, together with the grade of the total intercept.

The boundaries were then transferred to the level plans, and the boundaries smoothed to present a more "geological" outline.

The resource outlined has been assigned to the "probable" category, with a reasonable degree of confidence on the tonnage estimate, but a very low confidence level on the grade.

PROBABLE RESOURCE CALCULATION

The tonnages have been calculated for each 10 metre level, using the "prismoidal" method, with an assigned specific gravity of 2.65. The volume of material in a level is calculated from the following formula:

$$\text{volume} = h \frac{(a_1 + a_2 + \sqrt{a_1 \times a_2})}{3}$$

where: h = height between level (i.e. 10 metres)
a_1 = area of base of level
a_2 = area of top of level

The areas were calculated for each level using a planimeter.

The average grade for each level was calculated using an arithmetic average of all intercepts on the level, enclosed in the resource boundaries. It was decided that the unreliability of the data did not warrant calculating individual drillhole areas of influence and calculating a weighted average assay.

The "probable" in-situ geological resource is estimated to be 3.13 million tonnes at 0.22% Sn. All the data is summarised in Tables 1 to 5.

POTENTIAL RESOURCE ESTIMATE

There are significant gaps in the drilling pattern, adjacent to mineralized drill holes (i.e. >0.1% Sn). The cross-cutting diorite dyke appears to influence the grade of mineralization, particularly in South Block, with significantly higher grade zones close to the dyke. On many of the cross-sections the host rocks have not been drill tested close to the dyke.

All those areas between mineralized holes and the dyke have been included as a "potential" resource.

With many of the Aberfoyle percussion drill holes bottoming in mineralization, there is scope in many places for increasing the tonnage available.

Boundaries have been placed around "potential" areas on the cross-sections, with a great deal of subjectivity. The outlines were transferred to the level plans, and the boundaries "smoothed" to present a more geologically acceptable outline. From the level plans a tonnage has been calculated, but no grade can be assigned.

The "potential" tonnage is summarised below:

North Block	:	1,200,000 tonnes
South Block	:	1,100,000 tonnes
Brocks Block	:	200,000 tonnes
Total Potential		2,500,000 tonnes

UNDERGROUND SAMPLING

A series of 14 Adits were excavated in the early 1900's, presumably by the original company that attempted to mine the deposit - Great Pyramid Tin Company.

There are no records available, to Shell's knowledge, of the tin grade of the material extracted, if it was assayed or processed at all.

Channel sampling in many of the adits was done by the Great Pyramid Tin Company and by the Troy Tin Syndicate in 1909 to 1911, and in 1914, respectively.

The Tasmanian Mines Department carried out some minor bulk sampling in areas of richer old workings in 1957 and 1963. The tin values from these samples was consistently higher than previously indicated.

BHP in 1964-1965 mapped all the adits, and in 1980-81 carried out some limited channel and bulk sampling to provide new information regarding grade distribution and comparisons with previous sampling.

The absolute values of the channel sampling by the Great Pyramid Tin Company and the Troy Tin Syndicate are not reported in any of the data obtained by Shell, but have been represented on plans as 0.1% Sn intervals, e.g. 0.2 to 0.3%. The variability of the grades reported supports recent evidence of the patchiness of the cassiterite distribution.

Continuous channel samples were collected by Shell in three adits (North, C and 1SLL) to enable comparison between actual assay results and calculated block grades in areas included in the ore reserve estimate. Samples were collected on a lithological basis. Sample sizes varied depending on sample width. Most samples collected were in excess of 5 kgs in weight. Geological information and assay results are documented on Plan D/LH 01/502, 503, 531-534. Channel sample results are indicated below:

Channel Sample Assay Results

Adit	No. of Metres	Bulk Grade
North (west wall)	70.45m	0.119% Sn
North (east wall)	12.2m	0.072% Sn
C	62m	0.23% Sn
1SLL	40.1m	0.23% Sn

(an average overall grade of 0.175% Sn)

GRADE COMPARISONS

An attempt has been made to assess the effect of adit channel sampling and bulk sampling on the overall drill-estimated grade. The paucity of reliable data prevents any meaningful quantifiable comparison of grades.

Recent bulk sampling by SCOA indicated a head grade significantly higher than a calculated grade from channel sampling, 0.39% as against 0.22%. Unfortunately the bulk sample was a composite of samples taken from different adits, so no direct comparison can be made with individual areas.

BHP collected, processed and analysed four bulk samples, from North, C, 2SLL and 2NLL Adits. The tin grade tended to be equal to or slightly higher than grade from channel sampling.

A substantial channel sampling programme was carried out by Great Pyramid Tin Mines N.L. in 1909, and the Troy Tin Syndicate in 1914. Unfortunately the actual results are not available, but have been recorded as ranges of values at 0.1% Sn intervals, e.g. 0.1-0.19%.

Appendix III presents in "note format" a summary of the adit sampling results, compared to drill indicated results.

The actual results obtained from the various underground sampling programmes have not been incorporated into the reserve calculation. However, the outlines of the reserve have been adjusted to allow for indicated mineralization in areas of no drill information.

Of the two major drilling programmes completed to date, the Aberfoyle percussion drilling has indicated a higher grade than the BHP diamond/percussion drilling programme (SCOA Report 08/1905). The mean value of all percussion drill samples is 0.14% Sn, and for all diamond drill samples is 0.10% Sn. However, many of the percussion holes did not penetrate the mineralized horizon, whereas the diamond drilling was testing non-mineralized zones, both laterally and vertically. It is significant that BHP drilled two percussion holes adjacent to two diamond drillholes (1m. separation), and the mean tin value was 40% higher from the percussion samples.

A separate study by BHP of the sludge assays from diamond drilling, indicate an increase of up to 30% in the mean tin value in the sludge samples.

HIGH GRADE RESOURCE ESTIMATE

In order to assess the potential for higher grade zones within the "envelope" of plus 0.1% Sn, blocks were outlined using a 0.2% Sn cut-off.

A higher grade block was outlined using the following parameters:

- individual intercepts in drillhole ≧ 0.2% Sn;
- minimum of 2 adjacent intercepts in each drillhole;
- a single intercept of less than 0.2% Sn was included if it could be "carried" by an intercept of ≧ 0.2% Sn either side of it;
- a block requires a minimum of 2 adjacent drillhole intercepts either on the same or adjacent section.

The same prismoidal method of calculation was used as for the 0.1% cut-off.

As expected, the 0.2% cut-off produces several seperate blocks of reserves, with the only major one that has reasonable continuity occurring in South Block, between levels 140 and 190, close to the dyke. An estimated tonnage of 306,000 tonnes is indicated, at a grade of 0.46% Sn, in this area.

Intercepts used, and tonnage and grade estimates, are report in Tables 6,7,8 and 9.

APPENDIX I

Summary of Drilling and Sampling Programmes at Great Pyramid

DRILLING PROGRAMMES

Aberfoyle: Percussion (1970):	H1-H137
Aberfoyle: Diamond (1970):	GPY1-GPY6
BHP: Percussion (1965):	PDUS1-PDUS21 PDUS24-PDUS26
BHP: Diamond (1980):	BPD1,2A,2B,3,4,5,7,9A,9B,10,11,12 & 8 (sludge only)
BHP: Diamond (1965):	DDS-1
TMD: Diamond (1976):	MD1-MD4

CHANNEL SAMPLING

1. Great Pyramid Tin Co. (1909):

'C' Adit:	W. Wall some high grade $\frac{>0.5\%}{15m}$; $\frac{>0.5\%}{12m}$
'B' Adit:	N. Wall, 3 x 3m sample <0.2%
2 NLL Adit:	W. Wall. Low grade, minor >0.2%
3 NLL Adit:	S & W. Wall. Min or low grade.
3 SLL Adit:	E. Wall: 0.1-0.2%.
North Adit:	60% of E Wall: High grade patch $\frac{>0.2\%}{22m}$

2. Troy Tin Syndicate (1914):

2 SLL Adit:	40% of both walls, patchy low grade. Both walls of drive : med-high grade.
'C' Adit:	30% of both walls, alternate; patchy low and high grade.
2 NLL Adit:	"X-Cut": 60% of N & S walls, low grade.
3 NLL Adit:	Both walls. Traces only.
'A' Adit:	20% sampled : trace → 0.2%.
North Adit:	40% both walls : trace → >0.5%.

3. BHP (1980):

2 NLL (Drive):	100% of N. wall 50% of S. wall	$\frac{0.27\%}{46m}$

4. Shell (1983):

North Adit:	85% of W. wall. ~ 20% of E. wall.	Variable lithologies & grades.

'C' Adit:	70% of W. wall.	High and low grades average about 0.23%
1 SLL:	60% of W. wall.	Variable grade, average is 0.226%.

BULK SAMPLES

1. Tasmanian Mines Department (1957):

2 SLL Adit:	98m W. wall.
Drive:	10m N. wall; 26m N. wall.
'C' Adit:	16m W. wall; 42m E. wall.
2 NLL Drive:	16m N wall; 43m N. wall
1 SLL Adit:	W. wall @ end.

2. BHP (1980):

North Adit:	E. Wall 16-18m	PBS-1	0.50% Sn
'C' Adit:	W. Wall 34-36m	PBS-2	0.30% Sn
2 SLL Drive:	N. Wall 1-3m	PBS-3	0.65% Sn
2 NLL Adit:	E. Wall 19-21m	PBS-4	0.35% Sn

3. Shell (1983):

3 separate samples bulked together. Overall grade 0.37% Sn

North Adit:	21.7-24.4m W. wall	4 drums
'C' Adit:	11-13m W. wall	4 drums
1 SLL Adit:	48.65-51.15 W. wall	3 drums

1 sample taken, not assayed.

2 NLL Adit

APPENDIX II

Channel and Bulk Sampling -

Level Distribution

CHANNEL & BULK SAMPLING - LEVEL DISTRIBUTION

190L:

North Adit: (194m RL)	*Channel Samples:*	Great Pyramid Tin Co. Troy Tin Syndicate Shell
	Bulk Samples:	BHP Shell

180L:

'C' Adit: (183m RL)	*Channel Samples:*	Great Pyramid Tin Co. Troy Tin Syndicate Shell
	Bulk Samples:	TMD BHP Shell
'F' Adit (181m RL)	*Channel Samples:*	Troy Tin Sydnicate
'E' Adit: (182m RL)	*Channel Samples:*	Great Pyramid Tin Co.
'A'Adit: (181m RL)	*Channel Samples:*	Troy Tin Syndicate
'B' Adit: (188m RL)	*No Records*	

160L:

2 NLL Adit: (166m RL)	*Channel Samples:*	Great Pyramid Tin Co. Troy Tin Syndicate BHP
	Bulk Samples:	TMD BHP
1 NLL Adit: (163m RL)	*No Sampling.*	

140L:

3 NLL Adit: (142m RL)	*Channel Samples:*	Great Pyramid Tin Co. Troy Tin Syndicate
3 SLL Adit: (142m RL)	*Channel Samples:*	Great Pyramid Tin Co.
1 SLL Adit: (143m RL)	*Channel Samples:*	Shell Troy Tin Syndicate
	Bulk Samples:	TMD Shell

130L:

2 SLL Adit: (137m RL)	*Channel Samples:*	Troy Tin Syndicate
	Bulk Samples:	TMD BHP
Brocks Adit: (135m RL)	*Channel Sampling:*	TMD (1963)

170L:

'D' Adit: (174m RL)	*No Sampling.*

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APPENDIX III

Adit Sampling - Correlation with Interpreted Resource Outlines

190L

North Adit:

- Approximate level: 194m.
- Affects the 190L blocks in the North Block.
- Channel sampling indicates variability of Sn content with lithology.
- Average grade of Shell channel sampling = 0.12% Sn.
  Average grade of Shell Level Reserve = 0.15%.
- High grade of channel sampling, and BHP bulk sample ~ 14m from portal @ junction. Just within boundary of Shell interpreted zone of >0.2% Sn.

17WE: ~ 0.1% Sn.

18WE: Low grade x <0.1%.

19WE: Area of 0.18% Sn/4.25m.

20WE: Close to section of 0.28% Sn/4.9m in Adit adjacent to H17 - x = 0.14% above 190L.

21WE: Not sampled by Shell. Original sampling indicates 0.3-0.5% Sn.
Projection of drill collar to X section gives upper limit of mineralization at floor of adit. Only Shallow zone.

Conclusions:

- Overall grade of Shell sampling 0.12%, as against 0.14% for total level grade.
- High grade area is within higher grade area outlined on 190L reserve plan.

- Conflicting lithologies as mapped by BHP and Shell. Poor recognition of rock types.

- High grade bulk sample by BHP correlates with high grade channel sampling ~ 0.5% Sn.

180L

'C' Adit:

- Approximate level 183 RL.
- Affects 180L blocks in South Block.
- More distinct association of Sn with the sandstone/quartzite units, and sandstone/shale units.
- Shell channel sampled W. wall over 62m, average grade 0.23% Sn.
- Located in an area of variable drill indicated grade, but estimated at ≤ 0.2% Sn.
- Located sub-parallel to Aberfoyle drill sections, and between lines.
- Higher grade channel sampling by Shell, e.g. 27.5m @ 0.35% Sn and 6.4m @ 0.45% Sn.
- Tasmanian Mines Department bulk sampling grades do not correlate well with channel sampling grades, one about same grade: ~ 0.4-0.5%, and one indicated low, 0.1-0.2% in area of 0.35% in channel.

7WE: Portal at this section. Starts in high grade, < 0.5% Sn.

8WE: Area of <0.2% Sn from drillhole samples, high grade channel and bulk samples (>0.5%).

9WE: Interpreted <0.2% Sn area. 3m from BHP bulk sample @ 0.3% Sn; variable grade channel sampling by Shell; high grade channel sampling in 1909.

10WE: Located between low and medium grade drillholes, PDUS5 and H24. Edge of high grade of ore in channel sampling, 6.4m @ 0.45% Sn. Close to TMD bulk sample of 0.4-0.5% Sn.

11WE: Low grade area in drilling, <<0.1% Sn. No Shell channel sampling, close to <0.01%.

12WE: No Shell channel sampling. Low grade, <0.1% Sn; values from 1909 channel sampling.

Conclusions:

- Good correlation of grade with lithology.
- High grade mineralization in channel and bulk sampling of quartzites.
- Definitely barren in north part of adit from just north of 10WE section.

'E' Adit:

- Approximate level 182m RL.
- Situated between North and South Block on west side of hill.
- No drill information.
- Some average mineralization reported in channel sampling in 1909, 20m @ 0.1-0.3%, & 12m @ 0.1-0.3%.

'A' Adit:

- Approximate level 181 RL.
- Portal SE of North Block, north trending. End of adit just in interpreted limit of mineralization.
- Sampling in 1914 indicated low grade at end face and walls, <0.1%, <0.2%.
- Geology shows some unsampled quartzite and sandstone.

'F' Adit:

- Approximate level 181 RL.
- Affects east side of South Block.
- Drilling indicates medium to high grade.
- Channel sampling in 1914 (Troy Tin Syndicate) gave variable results, but all above 0.1% Sn, and up to 0.5% Sn.

160L

2 NLL Adit:

- Approximate level 166m RL.
- Affects 170L blocks in North Block.
- Channel sampling results indicate correlation of cassiterite with favourable lithologies - quartzites and sandstones. Much of the obvious shale units were not sampled in the early days, presumably the operators recognized that the shales were relatively barren.
- Sampling indicates variable grade, higher grade possibly related to more strongly fractured zones.
- TMD bulk sampling of high grade channel sampled area (i.e. >0.5% Sn); results were 0.83%, 0.37% and 0.42%.
- BHP bulk sample (PBS-4) from quartzite unit in area of 0.2%-0.4% Sn, returned a value of 0.35% Sn.
- BHP channel sampling of drive indicates average grade of 0.27% Sn over 46m.
- Average grade of probable, drill-indicated reserve at 160L, estimated at 0.26% Sn.

20WE: End of adit, indicated by channel sampling at 0.1-0.2% Sn, drilling (H4) indicates 0.22%.

21WE: No channel sampling at this point; lithology shown as sandstone and shale. Drilling (H6) indicates <0.1% Sn.

West drive is medium to high grade in drill indicated low grade area.

22WE: Area of low grade channel sampling in adit. Drill assays <0.1% Sn (H10).
Drive channel sampling up to 0.34%.
Indicates higher grade than drilling.

23WE: Low grade at adit level. Higher grade above and below.
Channel sampling very variable, but weak.

24WE: Low grade. Shale lithology. Outside reserve.

1 NLL Adit:

- No recorded sampling.
- Dangerous, with foul air.

140L

1 SLL Adit: (143m RL): within South Block

- Shell channel sampled the southern half, and the Troy Tin Syndicate channel sampled the northern half in 1914.
- Level block grade estimated from drilling at 0.27% SN.
- Overall grade of SCOA sampling is 0.23%.
- Mainly quartzite, sandstone and some shale.
- North end sampling indicated low grade, with some >0.2% in the walls at the north end of adit.

3WE: Through dolerite dyke, no tin values.

4WE: Start of higher grade area, channel sampling indicates 22.55m at 0.29% Sn.

Close to holes H41 and GPY-5, both with high grade Sn, 0.52% and 0.49% respectively; not reflected in channel sampling.

5WE: 7m from PDUS-20 which contain high grade values.
Channel sampling indicates 0.1-0.2% Sn.
Close to edge of probable reserve outline.

6WE: Beneath percussion drill outlined high grade zone, H54 at 0.72% Sn.
8m laterally from GPY-6 with >0.2% Sn.
Good potential.
Channel sampling indicates very low grade.

7WE: No drilling data. Below base of percussion drilling, all finished in good grade minieralization.
Channel sampling gave very low grades.

8WE: Interpreted average grade area, but above actual level. Good potential.
Channel sampling give 0.1-0.3%.

3 SLL Adit: (142m RL)

- East wall channel sampled by Great Pyramid Tin Co. (1909). All values indicated 0.1-0.2% Sn in the quartzites and sandstones.
- Outside area of drill delineated reserves.

3 NLL Adit: (142m RL)

- Channel sampled by Troy Tin Syndicate (1914) and partly by Great Pyramid Tin Co. (1909).
- Patchy values in shale, including a single sample >0.5% Sn, others 0.1-0.4% Sn.
- Some potential for reserves in this area; may require some testing.

130L

Brocks Adit: (135m RL)

- Through lower levels of Brocks Block.
- Evidence of old workings, some stoping and winzing.
- Quartzite, and mixed quartzite, sandstone and shale.
- Only recorded sampling by TDM in 1963; essentially barren, but some values <0.5% near old workings.

2 SLL Adit: (137m RL)

- Through lower levels of South Block.
- Channel sampled by Troy Tin Syndicate (1914), particularly in quartzite, sandstone, and mixed sandstone/shale lithologies.
- Very variable grade, <0.1 to >0.5% Sn.
- Overall high grade in drive off adit, particularly in the quartzite.
- Only 1 drillhole at this level is close to the adit, H38.
- Bulk sampling by TMD only in higher grade areas, confirms grade.
- Extends area of probable reserve, and would effectively upgrade this level. Drill estimated grade 0.25% Sn.

2WE: Just outside area of low grade reserve.

3WE: Edge of low grade reserve area, but close to high grade channel sampling.

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4WE: Adjacent to base of H38, a low grade hole, average 0.1% Sn.
Variable grade in adit 0.1-0.5%.
Medium to high grade in drive.

5WE: Area of no drill information.
Adit channel sampling is medium to high grade, close to limit of favourable lithology.
Drive sampling low to high grade, 0.1% to >0.5% Sn.
Good potential for extra tonnes and grade.

6WE: Outside drill indicated reserves. No channel sampling, probably because of shale lithology.

7WE: Some medium grade values associated with quartzite and sandstone/shale.
No drill data.
Some potential for tonnage/grade.

8WE: Some low grade channel samples, 0.1-0.2%.
Beneath interpreted limit of drill indicated reserve.
Some potential for tonnage/grade.

TABLE 1

TOTAL INTERCEPTS USED IN PROBABLE RESOURCE CALCULATION

0.1% Sn Cut-Off

1. North Block

SECTION	HOLE NO.	RL FROM	RL TO	TOTAL DEPTH	% Sn	
17WE	H88	196	215	19.0	0.17	
18WE	BPD2A	188	207	19.0	0.11	
	PDUS26	175.5	217	41.5	0.14	
	H87	186	203	17.0	0.14	
	H1	183.5	200.5	17.0	0.32	*
19WE	BPD1	169.5	212.0	42.5	0.14	
	H133	179.0	212.0	33.0	0.20	*
	H28	180.5	211.0	30.5	0.19	*
	H186	182.5	211.0	28.5	0.13	
	H2	184.0	206.5	22.5	0.10	
20WE	BPD5	166	180	14.0	0.17	
	BPD4	169	190	21.0	0.11	
	PDUS15	176	206	30.0	0.14	*
	H17	168	205	37.0	0.27	*
	H4	159	201.5	42.5	0.22	*
	H135	157.5	168.5	11.0	0.14	
21WE	H29	171.0	197.5	26.5	0.12	*
	H8	151	197	46.0	0.26	*
	H7	150	194	44.0	0.16	*
	H6	151.5	188.5	37.0	0.18	
	H5	149	180	31.0	0.45	

* Drill Hole Bottomed in Mineralization (>0.1% Sn)

1. North Block (Cont'd)

SECTION	HOLE NO.	RL FROM	RL TO	TOTAL DEPTH	% Sn	
22WE	BPD3	147.5	170	22.5	0.14	
	PDUS14	143.5	190	46.5	0.14	
	H16	159	186.5	27.5	No Assays	
	H10	144.5	187.5	43.0	0.16	
	H11	175	179.5	4.5	0.18	
23WE	H14	144	176	32.0	0.25	*
	H15	144	180.5	36.5	0.20	
24WE	H13	170.5	174	3.5	0.13	
	PDUS13	164.5	177.5	13.0	0.22	

2. South Block

SECTION	HOLE NO.	RL FROM	RL TO	TOTAL DEPTH	% Sn	
2WE	H124	140.5	148	7.5	0.17	
	H124	114.5	125	10.5	0.19	*
	H123B	109.5	149	39.5	0.27	*
	GPY2	91.5	110	18.5	0.12	
3WE	BPD10	134	158.5	24.5	0.17	*
	H122	127	155	28.0	0.26	*
4WE	H38	138	162.5	24.5	0.10	*
	H37	134	169	35.0	0.14	*
	BPD10	110	134.5	24.5	0.20	
	H41	134.5	165	30.5	0.36	*
	GPY5	109	165	56.0	0.30	
	H85	124.5	129.5	5.0	0.24	*
	H82	94	119.5	25.5	0.34	
	H126	88.5	102	13.5	0.27	*
5WE	PDUS20	134	167	33.0	0.36	
	H42	131	167.5	36.5	0.22	*
	BPD11	105	131.5	26.5	0.10	
	H81	102.5	136	33.5	0.15	*
6WE	H57	156	179	23.0	0.12	*
	GPY6	148.5	183.5	35.0	0.30	
	H54	153.5	178	24.5	0.72	*
	H43	143.5	172	28.5	0.40	
	H80	108.5	137	28.5	0.18	

2. South Block (Cont'd)

SECTION	HOLE NO.	RL FROM	RL TO	TOTAL DEPTH	% Sn	
7WE	H105	154.5	163.5	9.0	0.35	
	H55	162.5	188.5	26.0	0.37	*
	BPD9A	178.5	193	14.5	0.15	*
	H44	148.5	173	24.5	0.39	*
	H119	111.5	128.5	17.0	0.15	*
	H130	102	134	32.0	0.15	*
	H128	118	127.5	9.5	0.24	*
	H53	167.5	188.5	21.0	0.26	*
8WE	H96	161	171.5	10.5	0.22	
	MD4	161.5	172	10.5	0.21	
	PDUS25	158.5	171	12.5	0.15	
	GPY3	109	185	74.0	0.32	
	H56	160.5	185	24.5	0.30	*
	H79	165	191.5	26.5	0.11	*
	H52	170.5	195	24.5	0.33	*
	BPD9A	140.5	183	42.5	0.17	
	H45	150	176	26.0	0.13	
	H131	119	140.5	21.5	0.30	
9WE	H92	162.5	184	21.5	0.15	*
	H103	172.5	190.5	18.0	0.64	
	H26	167.5	200	32.5	0.23	*
	MD3	164	189.5	25.5	0.17	*
	H25	165.5	199	33.5	0.12	*
	PDUS7	170	200	30.0	0.29	
	H51	167	195	28.0	0.50	*
	H46	160.5	177.5	17.0	0.26	*
	H117	157	161.5	4.5	0.17	*

2. South Block (Cont'd)

SECTION	HOLE NO.	RL FROM	RL TO	TOTAL DEPTH	% Sn	
10WE	H111	140	167	27.0	0.42	
	H109	176	186.5	10.5	0.20	
	H91	178.5	184.5	6.0	0.21	
	PDUS5	172.5	194	21.5	0.13	
	H24	174	207.5	33.5	0.22	
	H22	174.5	201.5	27.0	0.18	*
	H47	178	184	6.0	0.34	
11WE	H108	149	187.5	38.5	0.15	
	H32	165.5	193	27.5	0.17	*
	H21	169.5	206	36.5	0.15	*
	PDUS8	169	201.5	32.5	0.08	
	H49	151	189.5	38.5	0.13	
12WE	H113	164	177	13.0	0.16	
	H34	164.5	175	10.5	0.23	
	H20	173.5	201	27.5	0.24	*
	H50	170	185	15.0	0.17	
13WE	H114	147	166.5	19.5	0.20	*
	H30	183	194	11.0	0.15	
	PDUS6	183	197	14.0	0.11	
	H19	185.5	206.5	21.0	0.29	*
14WE	H36	183	194	11.0	0.20	

3. Brocks Block

SECTION	HOLE NO.	RL FROM	RL TO	TOTAL DEPTH	% Sn	
2WE	H65	136.5	170.5	34.0	0.17	*
	H64	151.5	161	9.5	0.34	*
3WE	H66	141.5	172	30.5	0.20	*
	PDUS2	137.5	172	34.5	0.12	
	H63	137	167.5	20.5	0.17	
4WE	H72	150	156	6.0	0.29	
	H67	161	172	11.0	0.25	
	H62	130.5	162	31.5	0.18	
5WE	H73	133	142	9.0	0.13	
	H68	148	160	12.0	0.23	
	H61	165	177.5	12.5	0.18	
6WE	H69	135.5	141.5	6.0	0.27	
	H60	142.5	148.5	6.0	0.32	
	H95	147	156	9.0	0.17	

TABLE 2

GREAT PYRAMID

BLOCK: NORTH CATEGORY: PROBABLE DATE: SEPT. 84

LIFT	A1 SQ. M.	A2 SQ. M.	$\sqrt{A_1 x A_2}$	VOL. $\frac{h}{3}(A1+A2+\sqrt{A1xA2})$	ORE TONNES (+) @ S.G.=2.65	GRADE % Sn	TONNES Sn
200-210	4980	1910	3084	33245	88100	0.21	185
190-200	7025	4980	5915	59735	158300	0.15	235
180-190	6930	7025	6975	69765	184900	0.22	405
170-180	7310	6930	7115	71190	188700	0.21	395
160-170	5210	7310	6170	62305	165100	0.21	345
150-160	3330	5210	4165	42350	112200	0.14	160
140-150		3330					
TOTAL					897300	0.19	1725

TABLE 3

GREAT PYRAMID

BLOCK: SOUTH CATEGORY: PROBABLE DATE: SEPT. 84

LIFT	A1 SQ. M.	A2 SQ. M.	$\sqrt{A_1 \times A_2}$	VOL. $\frac{h}{3}(A1+A2+\sqrt{A1 \times A2})$	ORE TONNES (+) @ S.G.=2.65	GRADE % Sn	TONNES Sn
200-210	2315					0.20	
190-200	8075	2315	4324	49045	130000	0.18	235
180-190	11725	8075	9730	98435	260800	0.24	625
170-180	10455	11725	11072	110840	293700	0.23	675
160-170	11065	10455	10755	107585	285100	0.25	715
150-160	7315	11065	8997	91255	241800	0.36	870
140-150	5745	7315	6483	65145	172650	0.26	450
130-140	7605	5745	6610	66535	176300	0.23	405
120-130	6295	7605	6920	69400	183900	0.18	330
110-120	3995	6295	5015	51015	135200	0.21	285
100-110	1620	3995	2544	27200	72100	0.18	130
90-100	625	1620	1006	10835	28700	0.16	45
80-90		625					
TOTAL					1980200	0.24	4765

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TABLE 4

GREAT PYRAMID

BLOCK: BROCKS CATEGORY: PROBABLE DATE: SEPT. 84

LIFT	A1 SQ. M.	A2 SQ. M.	$\sqrt{A_1 \times A_2}$	VOL. $\frac{h}{3}(A1+A2+\sqrt{A1 \times A2})$	ORE TONNES (+) @ S.G.=2.65	GRADE % Sn	TONNES Sn
170-180	1890					0.24	
160-170	3060	1890	2405	24515	65000	0.17	110
150-160	3030	3060	3045	30450	80700	0.19	155
140-150	2630	3030	2825	28285	74900	0.15	110
130-140	230	2630	778	12125	32100	0.19	60
120-130		230					
TOTAL					252700	0.17	435

TABLE 5

GREAT PYRAMID

BLOCK: TOTAL CATEGORY: PROBABLE DATE: SEPT. 84

LIFT	A1 SQ. M.	A2 SQ. M.	$\sqrt{A_1 \times A_2}$	VOL. $\frac{h}{3}(A1+A2+\sqrt{A1 \times A2})$	ORE TONNES (+) @ S.G.=2.65	GRADE % Sn	TONNES Sn
NORTH					897300	0.19	1725
SOUTH					1980200	0.24	4765
BROCKS					252700	0.17	435
TOTAL					3130200	0.22	6925

TABLE 6

TOTAL INTERCEPTS USED IN PROBABLE RESOURCE CALCULATION

0.2% Sn Cut-Off

1. North Block

SECTION	HOLE NO.	RL FROM	RL TO	TOTAL DEPTH	% Sn
19WE	H133	200	206	6.0	0.41
	BPD1	188.5	193	4.5	0.27
	H28	180.5	188	7.5	0.24
	"	202	208	6.0	0.37
20WE	H17	174.5	185	10.5	0.35
	"	199	203.5	4.5	0.67
	H4	159	179	20.0	0.27
	"	195.5	201	5.5	0.26
21WE	H8	154	164.5	10.5	0.27
	"	180	193.5	13.5	0.46
	H7	157.3	161.8	4.5	0.25
	"	188	191	3.0	0.41
	H6	172.8	181.3	8.5	0.37
	H5	156.5	179.5	23.0	0.56
22WE	H10	172	186	14.0	0.34
23WE	H14	144	150	6.0	0.33
	"	162	172.5	10.5	0.38
	H15	148.5	153	4.5	0.31
	"	160.5	166.5	6.0	0.38
	"	174	180	6.0	0.31
24WE	PDUS13	164.5	173.5	9.0	0.27

2. South Block

SECTION	HOLE NO.	RL FROM	RL TO	TOTAL DEPTH	% Sn	
2WE	H124	116	120.5	9.0	0.21	
	"	140.5	143.5	3.0	0.29	
	H123B	114	123	9.0	0.27	
	"	135.5	149	13.5	0.35	
3WE	BPD10	139.5	142.5	3.0	0.25	
	H122	130.7	143	12.3	0.35	
4WE	H37	134	140	6.0	0.27	
	BPD10	129.5	134.5	5.0	0.53	
	GPY5	133	165	32.0	0.41	
	H41	134.5	165	30.5	0.36	
	H82	100	120	20.0	0.40	
	H126	88.5	91.5	3.0	0.90	
5WE	PDUS20	141.5	163.5	22.0	0.52	
	H42	141.5	167.5	26.0	0.29	
	H81	116	123.5	7.5	0.26	
6WE	GPY6	144	176	32	0.46	
	H54	153.5	178	24.5	0.72	
	H43	143.3	151	7.7	0.30	
	H43	161.6	172.2	10.6	0.71	
	H80	127	133	6.0	0.43	
7WE	H55	162.8	176.5	13.7	0.42	*
	"	181.2	188.7	7.5	0.40	
	H53	167.2	171.8	4.6	0.58	*
	"	178	185.5	7.5	0.29	
	H44	153	159.2	6.2	0.64	
	"	168.5	173	4.5	0.70	

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2. South Block (cont'd)

SECTION	HOLE NO.	RL FROM	RL TO	TOTAL DEPTH	% Sn
7WE	H119	116	120.5	4.5	0.24
	H130	117.5	120.5	3.0	0.25
	H128	118.5	121.5	3.0	0.43
8WE	H90	156	159	3.0	0.52
	GPY3	148.5	173	24.5	0.49
	H52	172.5	175.5	3.0	0.33
	"	186.5	194.0	7.5	0.57
	BPD9A	169	174.5	5.5	0.38
	H45	171.5	174.5	3.0	0.27
	H131	131.5	137.5	6.0	0.66
9WE	H103	183.2	191	7.8	1.33
	H26	168	174	6.0	0.56
	H25	165.5	170	4.5	0.33
	PDUS7	169.7	182.5	12.8	0.43
	"	186	193.5	7.5	0.30
	H51	171.5	190	18.5	0.66
	H46	166.7	177.5	10.8	0.33
10WE	H24	185	195.5	10.5	0.35
	H22	182	186.5	4.5	0.29
	H47	177.5	182	4.5	0.41
12WE	H20	178	181	3.0	1.17
	H50	171.5	174.5	3.0	0.40
13WE	H19	185.5	196	10.5	0.34
	"	199.5	205.5	6.0	0.34

3. Brocks Block

SECTION	HOLE NO.	RL FROM	RL TO	TOTAL DEPTH	% Sn
2WE	H65	137	146	9.0	0.42
	H64	151.8	161	9.2	0.34
3WE	H66	155.5	158.8	3.3	0.51
	"	165	172.5	7.5	0.31
	PDUS2	159.5	163.2	3.7	0.37
	"	168.7	172.2	3.5	0.33
	H63	161.5	167.5	6.0	0.45
4WE	H72	153	156	3.0	0.86
	H67	161.2	172.4	11.2	0.29
	H62	153.2	162.5	9.3	0.28
5WE	H68	148	158.5	10.5	0.25
	H61	166.5	169.5	3.0	0.23
	"	172.5	175.5	3.0	0.22
6WE	H69	135.5	141.5	6.0	0.27
	H60	142.5	147	4.5	0.39

TABLE 7

GREAT PYRAMID

BLOCK: NORTH CATEGORY: PROBABLE - H1 GRADE DATE: OCT. 84

LIFT	A1 SQ. M.	A2 SQ. M.	$\sqrt{A_1 \times A_2}$	VOL. $\frac{h}{3}(A1+A2+\sqrt{A1 \times A2})$	ORE TONNES (+) @ S.G.=2.65	GRADE % Sn	TONNES Sn
200-210	950					0.57	
190-200	1230	950	1080	10850	28700	0.42	
180-190	2290	1230	1680	17300	45800	0.36	
170-180	2050	2290	2170	21700	57500	0.35	
160-170	1570	2050	1800	18100	47900	0.34	
150-160		1570					
TOTAL					180000	0.36	650

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TABLE 8

GREAT PYRAMID

BLOCK: SOUTH CATEGORY: PROBABLE - H1 GRADE DATE: OCT. 84

LIFT	A1 SQ. M.	A2 SQ. M.	$\sqrt{A_1 \times A_2}$	VOL. $\frac{h}{3}(A1+A2+\sqrt{A1 \times A2})$	ORE TONNES (+) @ S.G.=2.65	GRADE % Sn	TONNES Sn
200-210	400	-				0.34	
190-200	1890	400	870	10550	27900	0.39	
180-190	2195	1890	2040	20400	54100	0.55	
170-180	3875	2195	2920	29950	79400	0.42	
160-170	2770	3875	3280	33100	87700	0.50	
150-160	2790	2770	2780	27800	73700	0.54	
140-150	2100	2790	2420	24350	64600	0.35	
130-140	290	2100	780	10550	28000	0.40	
120-130	690	290	450	4750	12600	0.24	
TOTAL					428000	0.43	1840

NB: Does not include material East of Dyke.

TABLE 9

GREAT PYRAMID

BLOCK: BROCKS **CATEGORY:** PROBABLE - H1 GRADE **DATE:** OCT. 84

LIFT	A1 SQ. M.	A2 SQ. M.	$\sqrt{A_1 \times A_2}$	VOL. $\frac{h}{3}(A1+A2+\sqrt{A1 \times A2})$	ORE TONNES (+) @ S.G.=2.65	GRADE % Sn	TONNES Sn
170-180	540	-				0.30	
160-170	1270	540	830	8800	23300	0.31	
150-160	435	1270	740	8160	21600	0.31	
140-150	645	435	530	5370	14200	0.37	
130-140		645					
TOTAL					59100	0.32	190

MEMORANDUM

TO: BEM (BCD)

FROM: AHO BEM/2:SK:YP

DATE: February 9, 1983

SUBJECT: GREAT PYRAMID PROSPECT

Introduction

The object of this memorandum is to provide sufficient mining information for "in-house" evaluation of the Great Pyramid Prospect.

The ore reserves calculation used for the mining study comes from "Great Pyramid Tin Prospect - Statistical Study of Drillhole Data" prepared by BEC/2. The mine equipment selection and costing has been carried out independently by BEM/2 using current costs from the Australian Mining Industry. The mining part of the evaluation is presented in 2 separate cases:

Case 1: Mining done by SCOA's own group
Case 2: Mining carried out on a contract basis

General Description

The project area is located near Scamander in NE Tasmania. SCOA entered the prospect by Joint Venture with the licence holder BHP later in 1982. The area has low infrastructure requirements as sufficient power and water supply, ample labour market and good roads surround the prospect.

Geological Basis for Mining Study

The Great Pyramid prospect was first located in 1909. Since that time numerous investigations by adit sampling, percussion drilling and diamond drilling have been carried out. The geology of the deposit has been described in BHP Exploration Report "Specification for Exploration of the Great Pyramid Mine" (Hine R. and Goode A., 1980).

An interbedded and gradational sequence of sandstones, siltstones and shales of the Siluro - Devonian Mathinna Beds form the host to the tin mineralisation.

The quartzite (more correctly, a silicified sandstone) which forms the Great Pyramid ridge is generally a hard, grey massive rock. Bedding is

usually difficult to recognise although it is parallel to a well developed jointing direction. In the main outcrops, beds are commonly several metres in thickness, separated by very thin (1-2 cm) shale beds.

The mudstones, siltstones and shales range in thickness from a few cm up to 2m. They are sheared, incompetent rocks, which pinch and swell along strike.

Observations both underground and on surface indicate most bedding strikes approximately north west and dips about 70 degrees south west, although many variations of strike and dip do occur. Asymmetrical folds have been observed plunging 20 to 30 degrees south east.

A major north west trending fault possessing a steep northerly dip transects the Great Pyramid prospect. Due to the absence of marker horizons the extent of displacement across the fault is unknown, although a large displacement is implied by the existence of two structurally distant zones on either side.

The bulk of the tin mineralisation comprises an area of cassiterite bearing close spaced joints. These joints strike about 070 degrees (at, or near, right angles to the strike of the bedding) and dip 60 to 70 degrees to the north west. The mineralised joints are generally 1-5mm in width and are locally lined with quartz and limonite.

The cassiterite occurs as grains generally coarser than 0.1mm (100u) with one or more of the following minerals: muscovite, fluorite, siderite, sulphides (arsenopyrite, pyrite, sphalerite, galena), tourmaline and wolframite.

Jointing and veining are best developed in the thicker, more massive sandstone and quartzite beds and, while present in the interbedded sandstones and shales, mineralisation is rare.

Two zones of mineralisation have been defined to date (a 'north' zone and a 'south' zone) which have been interpreted as being truncated by the fault described earlier. Some high grade tin intersections have been reported from the fault zone which probably has a width of 3-5 metres.

A narrow, non-mineralised dyke of intermediate composition (a 'quartz dolerite' in the petrological descriptions) crosses the deposit on an ENE-WSW trend.

The major controls of 'veinlet' style mineralization at Great Pyramid, therefore appear to be as follows:

1. Rock type: there appears to have been a strong preference for mineralised veinlets to develop in competent sandstones and quartzites.

2. Dense, close-spaced jointing, oriented normal to the local fold axis.

3. Proximity to a throughgoing fault. However, the importance of the fault as the principal conduit for ore fluids is questionable.

Drilling and Sampling Programmes

The following observations have been made regarding the drilling/sampling to date:

1. The BHP percussion holes ('PDUS' Series) were drilled on too widely spaced centres (50-60 metres) to adequately test the mineralised zones. Only 3 holes intersected significant mineralisation.

2. The Aberfoyle - Paringa percussion holes ('H' Series) were drilled generally on a grid pattern to test soil anomalies, at a spacing of 50' (15m) on lines 100' (30m) apart. This is the most extensive borehole coverage although it does not include all areas of indicated and potential mineralisation. Several boreholes were terminated in mineralised zones due to loss of air pressure.

3. The Mines Department diamond holes ('MD' Series) were drilled parallel to the main joint set and hence do not give reliable grade estimates.

4. The Aberfoyle - Paringa vertical diamond holes ('GPY' Series) have recorded some of the best intersections to date e.g. GPY 5 (33m at 0.36% Sn from 0-33m), GPY 6 (41m at 0.40% from 0-41m) and GPY 3 (33m at 0.38% from 0-33m and 20m at 0.38% from 56-57m).

5. The BHP diamond drillholes ('BPD' Series) were drilled on bearings normal to the mineralised joint set. During this campaign it was noticed that a significant proportion of the cassiterite was reporting to the drilling sludges. Sludges were collected and assayed over 3 metre intervals where sufficient water return was achieved. Two diamond holes, BPD2 and BPD9, were abandoned and percussion holes drilled alongside the discontinued diamond holes.

Statistics

The major findings of the statistical study were that:

1. The tin assay grades derived from the Aberfoyle - Paringa percussion drilling program ('H' Series) tend toward a lognormal distribution with similar estimates of the mean tin grade.

2. There is a considerable variability of grades over short distances. Corresponding downhole intervals from diamond holes and the adjacent percussion holes show no correlation between tin grades. This suggests that either the 'nugget effect' is very significant and/or that sampling and assaying problems have been considerable.

3. There is no apparent statistical correlation between tin grade and fracture density in either quartzite/sandstone, or in the quartzite and

interbedded shale lithologies.

4. It would not be possible to use kriging for ore reserve estimation given the drillhole data available at the present time. This is because the semi-variogram range (or zone of influence) is less than the drillhole and line spacing for the 'H' Series percussion holes.

Ore Reserve Estimates

The 'H' Series holes were used to derive reserve estimates at different cut-off grades using inverse distance squared weighting technique. The estimated reserves at a 0.1% tin cut-off grade are presented below:

Level	Tonnes	Mean Sn Grade (%)
2	103,723	0.260
3	267,649	0.204
4	389,518	0.246
5	442,683	0.205
6	578,170	0.263
7	342,938	0.232
8	274,057	0.201
9	156,650	0.188
10	133,400	0.175
11	48,403	0.218
12	33,216	0.235
13	25,838	0.116
Total	2.8 mln tonnes	0.225% Sn

Mining

The following mine design parameters and production criteria were derived or assumed for the mine equipment selection and the mine facilities construction:

Mill Feed	300,000 tonnes/year
Mine Production	
- Ore	1,225 tonnes/day
- Waste	940 tonnes/day
Total Movement	2,165 tonnes/day
Mining Ratio	0.8
Mine Life	8.5 years
Operating Time	245 days/year, 1 shift,

8 hours/shift

Ore Density 2.65 tonnes/m3

MINING COSTS

Clearing and Mine Development

All clearing of the hill from the forest and forest litter removal as well as the first 65,000 tonnes of overburden removal (entire level 1) will be done by a contractor specialising in this type of work. The contractor will also develop 1.3 km of hauling road connecting South and North part of the pit with the crusher area. The overall supervision of the clearing and mine development work will be SCOA's responsibility.

Cost Per Year

Type of Work	Area (ha)	Cost
Litter Removal	12	25,000
Bench & Road Development (1km road - $140,000)		180,000
Pre-production Overburden Removal		115,000
Drainage		10,000
Total Mine Development		$ 330,000

Production Mining

Drilling

Gardner Denver RDC-16B rotary drills are going to be used to drill holes of 6 metres depth and nominal 100mm diameter. Drilled holes will be sampled to provide more accurate information for grade control.

Drilling Data

Tonnes to be drilled	530,000/year
Drill availability	90%
Utilisation	60%
Drill fleet	2

Operating Cost

Operating supplies	10,000
Power	26,000
Maintenance	40,000
Total Operating	$ 76,000
Unit Cost	0.14 $/tonne mined

Capital Cost

2 x Gardner Denver RDC-16B drill	$510,000
+ Contingency (10%)	51,000
Total Capital	$561,000

Blasting

The blast design and optimisation will be carried out by the Mine Planner/Grade Controller. The blasting practices will be carried out accordingly to the local rules and regulations.

Consumable Cost		92,000
Labour		46,000
Equipment Cost	Operating $	Capital $
Mixing Unit	12,000	50,000
1 Hydraulic Rock Breaker	4,000	40,000
1 Bobcat	1,000	14,000
1 Magazine (Explosives)	1,000	10,000
1 AN Magazine	1,000	80,000
	19,000	
Total Operating	$157,000	
Total Capital		$194,000
+ Contingency 10%		20,000
		$214,000
Unit Cost - Blasting	0.29 $/tonne mined	

Loading & Hauling

A conventional open pit system using loader and truck was chosen for ore and overburden removal from the pit. Cat 966 (3.1/m~) rubber tyred front-end loaders and 20 tonne dump trucks were selected. The mining fleet (for 2km hauling cycle) will consist of 2 rubber tyred front-end loaders and seven 20 tonne trucks, together with ancilliary equipment such as grader, 1 dozer and 1 water cart.

Loader Selection:

Daily Production	2,165 tonnes
Availability	75%
Utilisation	75%
Fleet Units Required	2 Cat 966 (3.1 m~)

Truck Selection:

Dipper Size	3.1 m~
Tonnes Per Dipper	4.6
No. of Passes	(4-5)
Truck Capacity 20 Tonnes	(Komatsu HD-200)
No of Trucks Required	5

Labour Cost (1 Loader Operators + 4 Truck Operators) x 23,000 =	115,000
Maintenance Cost	
- Loaders $18/Loader/oper hr	35,000
- Trucks $17/Truck/oper hr	133,000
Power	
- Loaders	26,000
- Trucks	74,000
Total Operating Loading & Hauling	$ 383,000
Unit Cost - Loading & Hauling	0.72 $/tonne mined

Capital Costs

2 x Cat 966	434,000
5 x Komatsu HD200	875,000
	$1,309,000
+ 10% Cost	130,000
	1,439,000

060

Mine Facilities

Labour (1 first aid)	23,000
Supplies (25% of labour cost)	6,000
	29,000

Unit Cost Mine Facilities 0.05 $/tonne mined

Auxilliary Equipment

Auxilliary equipment is going to be required for minor production tasks.

Equipment	Number	Oper Cost	Cap Cost
Cat D7	1	18,000	241,000
Grader	1	8,000	156,000
Washdown Unit	1	1,000	50,000
Water Cart	2	4,000	30,000
		39,000	485,000
		+ 10% Cont	50,000
			535,000
Manpower		23,000	
Maintenance		80,000	

Total Auxilliary Equipment Operating Cost 142,000

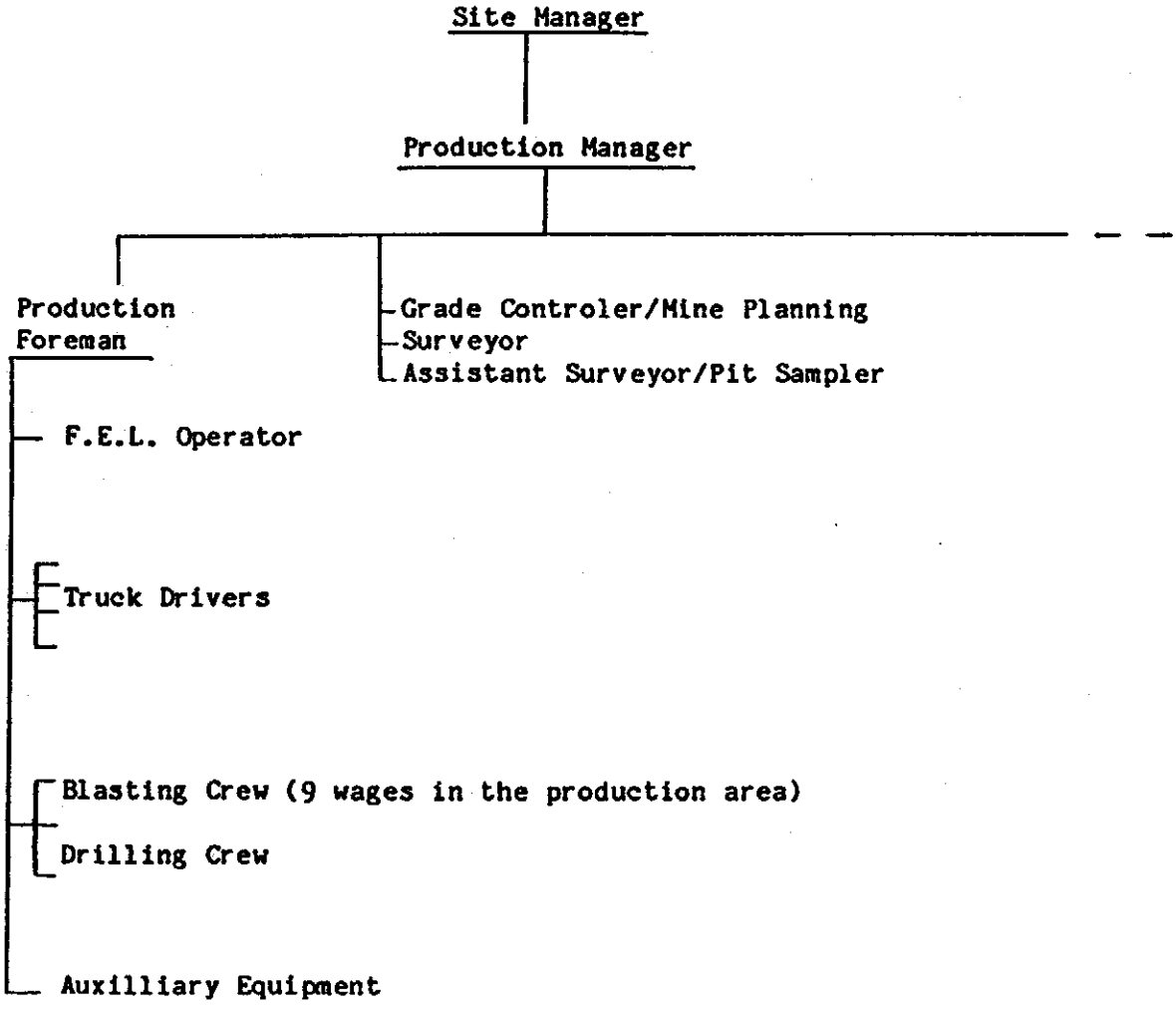
Unit Cost 0.27 $/tonne mined

Maintenance

There will not be a pit workshop. All repairs will be done in the main workshop. The main workshop will be equipped to repair the mining equipment as well as the plant and the concentrator.

General Mine Expense

The following organizational chart is proposed to facilitate administration cost estimation.



4 men x $26,000 = 104,000
Operating Administration Cost is $104,000
Unit Cost 0.19 $/tonne mined

Summary

The project area is located near Scamander in NE Tasmania. The bulk tin mineralisation comprises of an area of cassiterite bearing close spaced mineralised joints lined with quartz and limonite. Two zones of mineralisation have been defined: North Zone and South Zone. A narrow, non mineralised dyke of intermediate composition (quartz-dolerite) crosses the deposit and separates the zones.

Potential tonnage and grades of tin so far discovered were calculated from assay results of the Aberfoyle percussion holes ('H' Series) drilled at 15 metres spacing on lines 30 metres apart using inverse

distance squared weighting technique. The estimated reserves at 0.1% tin cut-off grade are 2.8 mln tonnes 0.225% Sn. For this evaluation, the mining reserves of 2.615 mln tonnes 0.22% Sn (after 5% dilution factor) were selected.

Production rate of 300,000 tonnes of ore/annum was assumed with operations consisting overburden removal, drilling, blasting, loading and trucking. A workforce of some 13 people will be employed at the mine site. No provision has been made for housing at this stage of the project evaluation.

The calculated unit cost for mining operations maintained with the SCOA's group (Case 1) is 1.66$/tonne mined. The capital cost of the mine development and mine equipment is $445,000 and 2,750,000 respectively.

The cost unit for the same mining operation quoted by Roche Bros. Pty. Ltd. (Case 2) is 2.36 $/tonne mined. Both cases were compared (see Attachment 1) and it was shown that Case 1 is more favourable as the NPV of the outlays is approximately 7.06 mln compared with 7.229 mln in the second case.

Conclusions

It is concluded that the difference between 2 outlays of the costs while mining using SCOA's mining group or a contractor is relatively insignificant and practical factors like gaining practical experience for the Group, prefered method of financing project should decide the selection.

Stan Kaleta.

Case 1

Unit Cost - Mining Operations

$ / tonne mined

Drilling	0.14
Blasting	0.29
Loading & Hauling	0.72
Auxilliary Mine Equipment	0.27
General Mine Expense	0.19
Mine Facilities	0.05
	1.66

Attachment 1

Comparison of Mining Alternatives

Case 1 -
Case 2 -

Year	0	1	2	3	4	5	6	7	8
Case 1	2.75	.88	.88	.88	.88	.88	.88	.88	.88
Case 2		1.25	1.25	1.25	1.25	1.25	1.25	1.25	1.25

0.88 x 6.2469 = 5.497

where 6.2469 = Present value of $1 received annually for 9 years at 8%

As the payment was not made by the end of year 1, it is necessary to bring once more the cash from year 1 to 0.

5.497 x 0.9259 = 5.089

where 0.9259 the present value of $1 factor for 1 year at 8%.

The same logics follow case 2 calculation

1.25 x 6.2469 = 7.808

7.808 x 0.9259 = 7.229

Annual depreciation = - 344

Tax gain = - 158 (46%)

NPV of the tax gains = - 778

Case 1	Case 2
2.75	
+ 5.089	
7.839	
- .778	
7.061	7.229

GREAT PYRAMID - CAPITAL COSTS A$'000 (MID 1983)

		-2	-1	0	1	2-7	8	WHOLE
FEASIBILITY:	100% SCOA	727						
	SHARED	100						
	PRE-PRODUCTION		410	35				
MINE CAPEX MINE EQUIPMENT				2,749				
PROCESSING CAPEX			4,000	5,000	250	-		
GENERAL CAPEX			1,000	750				
TOTAL								
WORKING CAPITAL			-----	ALLOW 3 MONTHS OP COSTS			-----	

APPENDIX 3: GENERAL

The General section covers those aspects of the project not covered in appendicies 2.1 and 2.3.

a) Operational Costs

(i) Capital Costs

	$
Site Preparation	100,000
Compensation	100,000
Laboratory	200,000
Maintenance Facilities	200,000
Offices	100,000
General Site Services (telephone/telex/ power etc.)	150,000
Off Site Costs (Senior Staff Housing) (1 x $100,000) (2 x $ 80,000) (5 x $ 65,000)	585,000
Sub Total	1,435,000
Contingency	315,000
Grand Total	1,750,000

(ii) Operating Costs

Senior Metallurgist	0.75	50,000	40%	52,500
Administration Supt.	1.0	30,000	40%	42,000
Administration Dept.	7.0	20,000	25%	175,000
Total Personnel	8.75			269,500
Office Costs (rentals, supplies, insurances etc.)				210,000
Total Administration Op Costs				479,000

Transport of Concentrate $200/tonne

Feasibility Study

Object No 1:	To demonstrate relationship of drilling to ore	
Procedure:	Exploratory Driving : Re sampling of adits etc.	
Allow	$500/metre and 250 metres (1.8m x 1.8m) (2100 tonnes)	$125,000
	Allow 1000 Samples at $30	$ 30,000
	Allow Supervision 1.5 x 3 x 8,000	$ 36,000
	Allow Surveys	$ 15,000
	Allow Sampling/metallurgical testwork etc.	$100,000
	Allow Reporting	$ 24,000
	Allow unbudgetted items (20%)	$ 66,000
	Allow contingency (20%)	$ 79,000
		$475,000
Object No 2:	To demonstrate project costs/feasibility	
Allow	Feasiblity Study SCOA/Consultant Say 20 man months x $10,000	$200,000
Allow	Management (Corporate/Government/ Public Contact)	$ 60,000
Subtotal		$735,000
	Allow + 12½% Company Overheads	$827,000

069

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GREAT PYRAMID

THE SHELL COMPANY OF AUSTRALIA LIMITED
METALS DIVISION

PRELIMINARY METALLURGICAL INVESTIGATION

OF TIN ORE DEPOSIT

FROM TASMANIA

GREAT PYRAMID

Report No. 16.020.1/1

MINERAL DEPOSITS LIMITED
MINERAL TECHNOLOGY DIVISION
BUNDALL - QUEENSLAND
AUSTRALIA

INDEX

1.	INTRODUCTION		
	1.1	Test Material	1
	1.2	Objective	1
	1.3	Terms of Reference	1
	1.4	Conclusions	2
2.	TEST EQUIPMENT AND PROCEDURES		
	2.1	Sample Preparation	4
	2.2	Grinding and Screening	4
	2.3	Hydrosizing	5
	2.4	Spiral Tests	5
	2.5	Table Tests	6
	2.6	Mineralogical Examination	6
	2.7	Tin Assays	6
	2.8	Sizing Analysis	6
3.	RESULTS AND DISCUSSION		
	3.1	Sample Processing Flowsheet	7
	3.2	Head Assay	7
	3.3	Ore Crushing and Milling	7
	3.4	Hydrosizing	9
	3.5	Spiral Tests	10
	3.6	Wet Table Testwork	11
	3.7	Magnetic Separation	12
	3.8	Recommendations	12
4.	APPENDICES		14

1. INTRODUCTION

1.1 Test Material

Eleven drums of tin ore were submitted by the Shell Company of Australia, Metals Division, from a deposit in Tasmania. The sample weighed 3.17 tonnes dry, and was nominally -100mm. The tin content was expected to be 0.2-0.25% Sn.

1.2 Objective

The objectives of the testwork were to:

1.2.1 Derive an estimate of the mass and metallurgical balances for the elements in the programme.

1.2.2 Produce a concentrate by gravity methods assaying greater than 30% Sn and a rejectable gravity tailing.

1.2.3 Upgrade the tin gravity concentrate to a saleable product.

1.3 Terms of Reference

1.3.1 Correspondence from Mr. Bruce Spiers of Shell Metals Division (SMD) outlining the test programme requirements, dated March 16, 1983.

1.3.2 Initial verbal communications between Mr. Bruce Spiers of SMD and Dr. Roger Richards of Mineral Deposits Limited (MDL) on February 21, 1984.

1.3.3 Letter of confirmation from Mr. Bruce Spiers, SMD, to Dr. Roger Richards, MDL, dated February 22, 1984.

1.3.4 A quotation, dated March 1 1984, from Dr. Roger Richards, MDL, for testwork as outlined.

1.3.5 Correspondence from Mr. Stan Kaleta, SMD, accepting the test proposal and submitting SMD's Order No. 48214.

1.4 Conclusions

The data produced from the testwork suggests the following:

1.4.1 A gravity concentrate of greater than ~~30%~~ 65% Sn content is achievable, using classification and wet gravity techniques.

1.4.2 A significant proportion of the feed, possibly as high as 30% or better, could be rejected as a final gravity tailing.

Not proven → (see p.8).

1.4.3 Liberation of over 20% of the total tin content of the feed in a size suitable for gravity concentration is achievable by grinding to less than 400 microns.

So what? →

1.4.4 Improvements in recovery of tin by gravity concentration in a production plant over the results achieved by the testwork carried out are possible by regrinding select middling products, recirculation of various product streams within the flowsheets, and better operating techniques achievable on a continuous operating basis.

Obviously.

073

1.4.5 Introducing different size feeds to tne separate gravity circuits may improve separation efficiency although this was not confirmed in this test programme.

1.4.6 Upgrading the gravity concentrates to saleable product by magnetic separation techniques appears possible.

1.4.7 No other valuable mineral was upgraded to any significant degree.

M. V. ATFIELD,
CHIEF METALLURGIST.

P. A. BUTLER,
SNR. PROJECT TECHNICIAN.

G. F. BALDERSON,
MANAGER - TECHNICAL SERVICES.

2. TEST EQUIPMENT AND PROCEDURES

2.1 Sample Preparation

The total sample was initially weighed and then thoroughly mixed. Approximately two tonnes were then split out by cone and quarter techniques for feed for the pilot test circuit. A further representative sample was obtained by coning and quartering and later ground and riffled to obtain a head assay.

2t feed
Contains 40-50kg Sn
= 60-70kg SnO_2

4" feed ??

2.2 Grinding and Screening

The sample was initially crushed in a jaw crusher and then screened on a vibrating screen fitted with a 13mm aperture punch plate screen cloth. The screen oversize was recirculated to the jaw crusher for further reduction. The screen undersize constituted feed to the rod mill for grinding.

No pretesting with heavy liquids therefore arbitrary choice of grind size, therefore the value of this work greatly reduced

The rod mill product was screened on a 1m diameter Kason vibrating screen fitted with a 400 microns aperture screen cloth. The oversize was returned to the rod mill (for further grinding) and the undersize flowed through a series of four hoppers to allow dewatering. Overflow from the fourth hopper which carried some fine slime was not retained.

The total rod mill ground product was then placed on a concrete pad to drain the excess water. The ground material was then thoroughly mixed and a sample was obtained by a coning and quartering procedure for moisture, head assay and screen analysis.

2.3 Hydrosizing

The Hydrosizer (of similar design to a Concenco three-cell model constricted plate classifier) was fed at approximately 60% solids w/w at 2tph. The aim was to produce three products: i.e. coarse, medium and slimes fractions. The three classifier underflow fractions were collected in 200 litre drums and the classifier overflow fines fraction was collected in a large tank. The classifier underflows were then sampled and weighed, and moisture determinations and size assays were performed on representative samples. The data indicated that the products from cells 1 and 2 could be combined to give three fractions; i.e. coarse, medium and fine. The fine fraction was grab-sampled during classifier operation. The grab sample was size analysed and assayed for tin.

2.4 Spiral Tests

The coarse and medium fractions were passed separately over a Reichert type LG7 single-start spiral separator in open circuit. The spiral was fed at approximately 2.5 tph and 40% solids. Three spiral products (concentrate, middling and tailing) were produced. The spiral splitters were set by eye to obtain good tin recoveries to concentrate. All products were weigned, sampled and assayed. The concentrate was also size analysed and the size fractions assayed for tin.

2.5 Table Tests

Tne spiral concentrates were individually passed over a laboratory-scale wet shaking table, adjustments being made by eye to produce acceptable performance. Splitters were set below the cassiterite line for tne concentrate take and just into the silica line for middling/tailing take. All products (concentrate, middling, tailing) were weighed, sampled and assayed.

Details? Reproducibility? Were tests examined to determine efficiency of test? Cleaning tests?

Magnetic Separation!

2.6 Mineralogical Examination

The table concentrates were individually passed over a laboratory-scale Carpco induced-roll magnet to further upgrade the tin concentrate. Three fractions were produced: highly susceptible magnetics, 0.35 amp magnetics, and a non-magnetics fraction. The fractions from the magnetic separation were weighed, assayed for tin, and subjected to microscopic examination to determine the degree of liberation of tin and also identify other minerals present.

Were these the only mineralogical tests? Middlings?

At the end of the test! →

What were being removed!

2.7 Tin Assays

Samples of less than 10% Sn were assayed for tin by volatilising the samples in ammonium iodide and subsequent reading against standards using an Atomic Absorption Spectrophotometer. The test products greater than 10% Sn were titrated (following a peroxide fusion and nickel reduction) against iodine with starch as tne indicator.

Reliability?

2.8 Sizing Analysis

Sizing analysis was performed using British Standard sieves. Samples were wet screened at 38 microns prior

077

3. RESULTS AND DISCUSSION

3.1 Sample Processing Flowsheet

Table 1 shows the weight distribution and tin assay and distribution to the various stage final products. The sample processing flowsheets (Appendices 4.1.1, 4.1.2, 4.1.3) show each stage of the processing as well as the weight splits and tin grade and distribution

3.2 Head Assay

That's what you get by coming + quartering at 4" for a head assay!

3.2.1 The head assayed 0.385% Sn, which was significantly higher than the expected 0.2-0.25% Sn. This figure is confirmed by the other assays in the flowsheet.

?

3.2.2 The head tin assays for most stages calculated back from the stage products gave reasonable agreement - considering the number of tin assays involved.

3.2.3 The final ground product assayed 0.414% Sn,. which was a little higher than the head assay; the increase being due mainly to the upgrading effect of the lower tin content slime rejected in the storage hopper overflow.

7% of the Sn in feed was lost in this assay!

3.3 Ore Crushing and Milling

Why weren't fines removed before milling?
Why no effort to explain the high slimes losses?

The finisned ground product size and tin analysis data is tabulated in Appendix 4.2. The fines content of this data is lower than true mill output, as it does not include the fines lost as overflow from the final settling bin in the series of four bins. The size

078

TABLE 1: Overall Flowsheet Performance

	Wt. %		% Sn	Sn Dist
FINAL TIN CONCENTRATES				
Coarse size magnetic non-mags	<0.1%		70.85	14.33
Medium size magnetic non-mags	<0.1%		66.63	5.94
Combined	<0.2%	(Calc)	69.10	20.27
FINES FRACTIONS				
Fine slime ex ground product	22.8%		0.286	16.96 ?
Hydrosizer overflow	28.8%		0.220	19.00
Combined	51.6%	(Calc)	0.250	35.96 !
REJECTABLE FRACTIONS				
Coarse size spiral tailing	23.7%		0.110	6.96
Medium size spiral tailing	12.6%		0.070	1.66
Combined	36.3%	(Calc)	0.096	8.62
HIGH GRADE TIN GRAVITY PRODUCT				
Coarse size wet shaking table concentrate	0.1%		47.92	15.65
Medium size wet shaking table concentrate	<0.1%		42.36	6.51
Combined	<0.2%	(Calc)	45.70	22.16
MIDDLING FRACTIONS				
Coarse size spiral middling	4.6%		0.34	3.97
Medium size spiral middling	3.4%		0.28	1.78
Coarse size shaking table mid	0.4%		17.50	19.44 Locked !
Medium size shaking table mid	0.2%		20.80	6.28 Locked !
Coarse size shaking table tail	1.7%		0.28	1.28
Medium size shaking table tail	1.6%		0.18	0.51
Combined	11.9%		--	33.26

?? How do we know that these are rejectable?

* where are the mag sep rejects?

3.4 Hydrosizing

The Hydrosizer produced four products, of which the two coarser fractions were combined to give an approximate split of 35:30:35 by weight.

Then why not repeat the experiment !!!!

Hydrosizer performance was retarded due to problems with the initial operation and the presence of some undispersed slime particles in the feed, which tended to report to cell No. 1 underflow; thus increasing the fine slimes content to underflow.

The results recorded in Appendix 4.8 show, however, that even under the previously described operating conditions, upgrading of the tin content of cell Nos. 1 and 2 combined product did occur with 56.95% of the tin in the feed to the classifier reporting in 39.38% of the original feed weight, resulting in the combined 1 and 2 cell product assaying 0.51% Sn; whereas the original feed assayed 0.41% Sn. The medium size fraction, or No. 3 cell product, assayed 0.31% Sn; somewhat below headfeed grade, due to depletion of recoverable tin by cell Nos. 1 and 2. This fraction did, however, contain significant liberated tin as shown by later testwork.

i.e. Suck-it-and-see testwork.

The slime or overflow fraction was downgraded to 0.22% Sn, with a -38 micron rejection to overflow of approximately 80% despite the initial operational problems.

What is surprising about this!

It is interesting to note that the finer fractions recovered in the underflow products were much higher in tin content than similar size fractions in the overflow.

Appendix 4.3 is a graph of size vs. cumulative weight for the products and head.

O.K In Appendix 4.4 the Hydrosizer separation efficiency is plotted.

4.6 missing Appendices 4.5 - 4.7 are the size analysis and tin assays of the size fractions for the three final products which includes also dry weight and tin distribution calculated on a feed basis.

3.5 Spiral Tests

3.5.1 The results of both the coarse and medium size spiral tests are summarised in Appendix 4.9. The spiral concentrate size analysis and tin assays for the size fractions are tabulated in Appendix 4.10.

3.5.2 The test data revealed that reasonable results were achieved, particularly as there was only sufficient material for one test run. Further adjustment should improve performance.

The same sample could have been re-run repeatedly. →

3.5.3 The coarse size test data showed that 85.69% of the tin in the feed could be recovered in 22.16% of the original feed weight. The size analysis data on the concentrate and the feed showed significant tin upgrading was achieved on all size fractions.

Coarse Spiral

3.5.4 The medium size test data showed that 90.11% of the tin in the feed could be recovered in 29.43% of the original feed weight. The size analysis data on the concentrate and the feed indicated significant tin upgrading on each size fraction. Recovery of tin in the -38 micron fraction was also being achieved.

Medium Spiral

3.5.5 The combined spiral test data indicated that 36.3% of the original feed weight could be discarded for a loss of 8.62% of the Sn in the original feed.

Promising (if we knew why +50% wt had already been rejected as slimes)

3.6 Wet Table Testwork

3.6.1 The wet table performance data for both the coarse and medium fractions are summarised in Appendix 4.11.

3.6.2 The coarse size test data showed that a concentrate grade of 47.92% Sn could be achieved. The recovery to the concentrate was 42.78% of the total tin in the feed, in 5.43% of the original feed weight.

3.6.3 The data from the medium size testwork showed similar results to the coarse size testwork. Slightly lower concentrate grade was produced (42.36% Sn), but a slightly higher recovery was achieved. Overall tin loss to tailing was low.

3.6.4 In an operating situation, table middling would circulate back to the head feed or be reprocessed in a separate circuit. Wet table performance would, therefore, be improved.

✓, but by how much? The test method chosen has yielded minimal data

3.6.5 Chemical analysis of both wet table concentrates (Appendix 4.13) indicates only minor quantities of most contaminants present, with the exception of iron-bearing minerals which are readily rejected by magnetic separation.

3.7 Magnetic Separation

Why selected?

3.7.1 The results of the magnetic separation of both table concentrates are summarised in Appendix 4.12.

3.7.2 The data from the coarse size testwork indicated that high grade concentrates (70.85% Sn) could be produced at high recoveries (91.23%) of the tin in the feed using magnetic separation.

3.7.3 The medium-size test data showed that a high grade concentrate (66.63% Sn) could be produced at high recoveries (91.23%) of tin.

3.7.4 The magnetic separation testwork on both table products showed that a significant proportion of the tin recovered from the gravity flowsheet was in a liberated form. The data also showed that the concentrates contained high proportions of other heavy minerals which were mainly magnetic in nature.

Not necessarily

3.7.5 A dry induced roll type magnet was used for the test programme. In an operating circuit, a wet high intensity magnet would most likely be selected for the duty.

3.8 Recommendations

No!!!
A high-quality bench-scale amenability study is required

Sufficiently encouraging results have been produced from the testwork to justify a recommendation for a further larger scale test programme which could be approached in two ways:

(1) Produce either one or a number of bulk samples of 20 - 30 tonnes, the number depending on the variability of the metallurgical characteristics of the ore body, and processing the samples on a more detailed basis at our Bundall laboratory.

It is irresponsible to propose construction of a pilot plant on the basis of testwork of the degree of accuracy & definition of the report. We haven't even looked for the liberation size !!!

(2) Design and construct a pilot plant which could operate on a continuous basis on-site, processing large bulk samples from various areas of the ore body.

The most satisfactory approach on a metallurgical basis is the pilot plant proposal, which would produce much more accurate data for full-scale flowsheet development. However, some factors which may influence a decision on the scope of additional work are:

(a) Availability of large bulk samples;

(b) Magnitude of the full-scale operation envisaged;

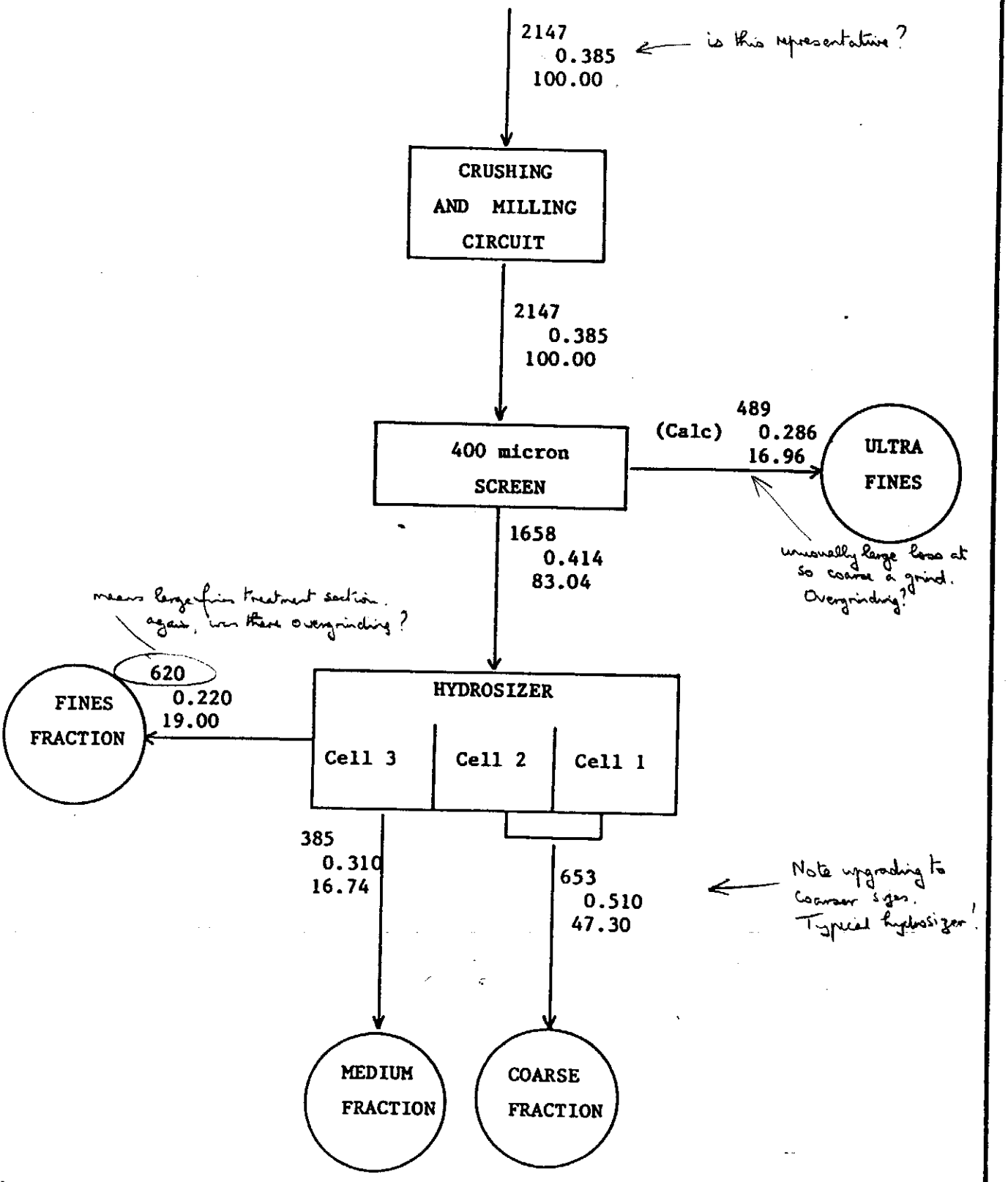
(c) Timing of the project;

(d) Variability of the metallurgical characteristics of the ore body.

4. APPENDICES

APPENDIX 4.1

Flowsheets



Key

Weight (kg)
% Sn
Mineral dist. to head feed (%)

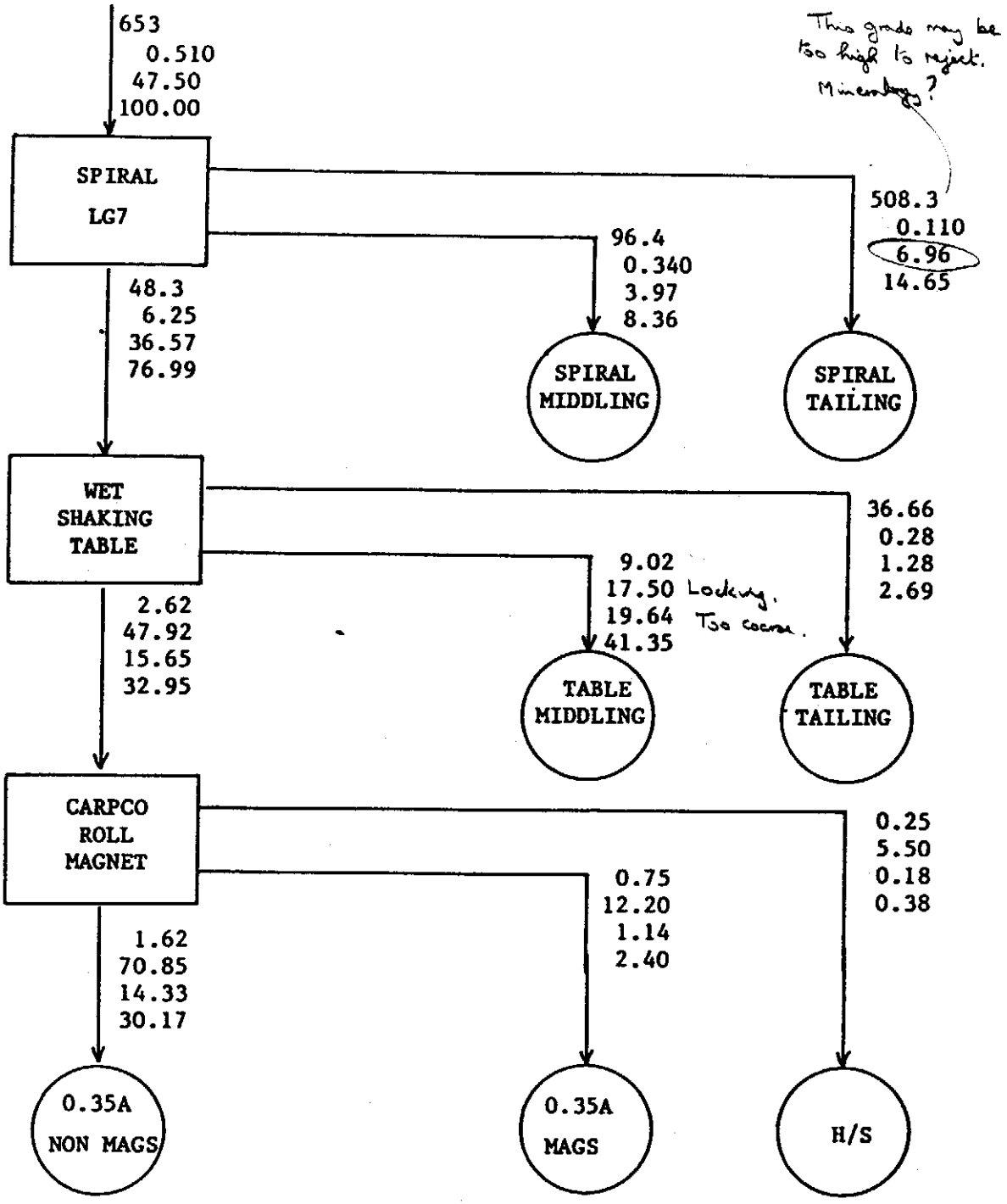
RESEARCH AND ENGINEERING DEPARTMENT
MINERAL DEPOSITS LTD.—SOUTHPORT, QUEENSLAND

APPENDIX 4.1.1

SHELL METALS - TIN ORE FLOWSHEET
COMMINUTION STAGE

DRAWN PAB:PJY

DATE 28.6.1984



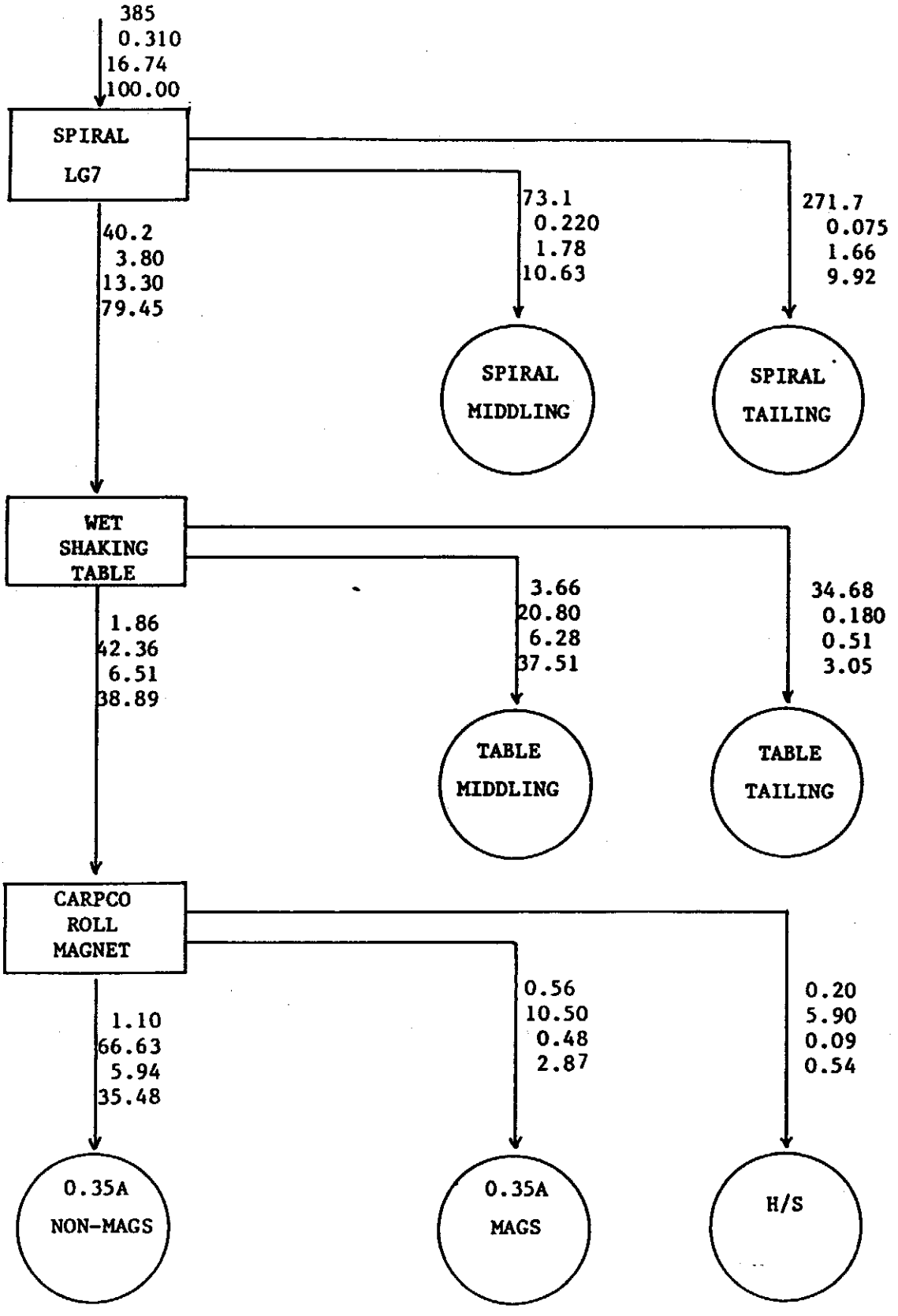
This grade may be too high to reject. Mineralogy?

Looking. Too coarse.

KEY

Weight (kg)
% Sn
Mineral dist. to head feed (%)
Mineral dist. to stage feed (%)

RESEARCH AND ENGINEERING DEPARTMENT **MINERAL DEPOSITS LTD.—SOUTHPORT, QUEENSLAND**	APPENDIX 4.1.2
SHELL METALS - TIN ORE FLOWSHEET CONCENTRATION STAGE, COARSE FRACTION	DRAWN PAB:PJY DATE 28.6.1984



KEY

weight (kg)
% Sn
Mineral dist. to head feed (%)
Mineral dist. to stage feed (%)

RESEARCH AND ENGINEERING DEPARTMENT **MINERAL DEPOSITS LTD.—SOUTHPORT, QUEENSLAND**	APPENDIX 4.1.3
SHELL METALS - TIN ORE FLOWSHEET CONCENTRATION STAGE, MEDIUM FRACTION	DRAWN PAB:PJY DATE 28.6.1984

APPENDIX 4.2: FINISHED GROUND PRODUCT

SHELL MINERALS TIN ORE.

SIZE (um)	WT. %	CUM WT%	ASSAY %Sn	CUM ASSAY	DISTN %	CUM DISTN
425	0.81	0.81	0.515	0.515	1.03	1.03
300	17.21	18.02	0.486	0.487	20.69	21.72
212	14.80	32.82	0.425	0.459	15.56	37.28
125	17.45	50.27	0.418	0.445	18.04	55.32
90	10.07	60.34	0.400	0.437	9.96	65.28
63	9.42	69.76	0.362	0.427	8.43	73.72
38	4.48	74.24	0.388	0.425	4.30	78.02
-38	25.76	100.00	0.345	0.404	21.98	100.00
TOTAL	100.00	0.00	0.404	0.000	100.00	0.00

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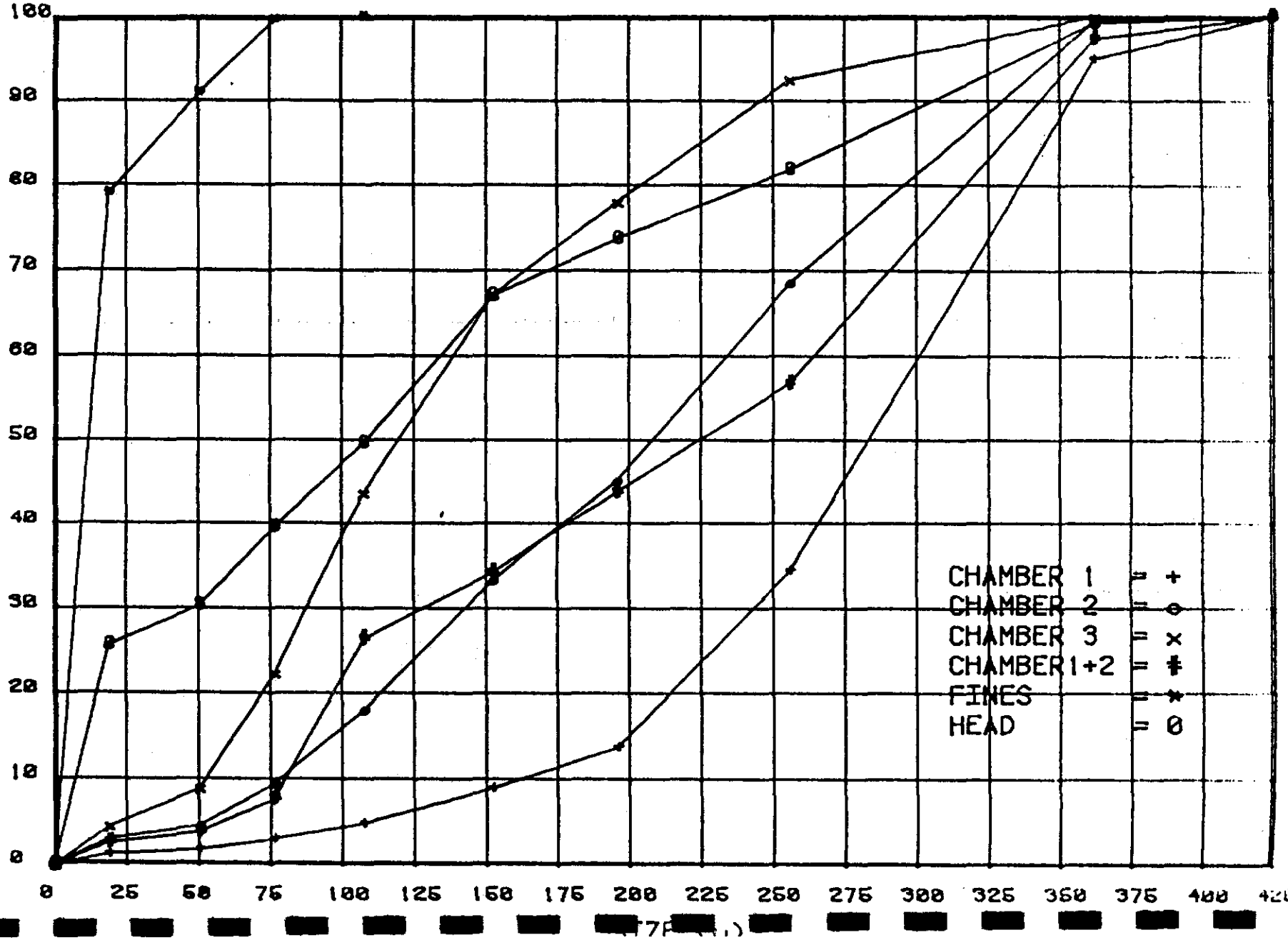
HYDROSIZER PERFORMANCE FOR SHELL MINERALS TIN ORE.



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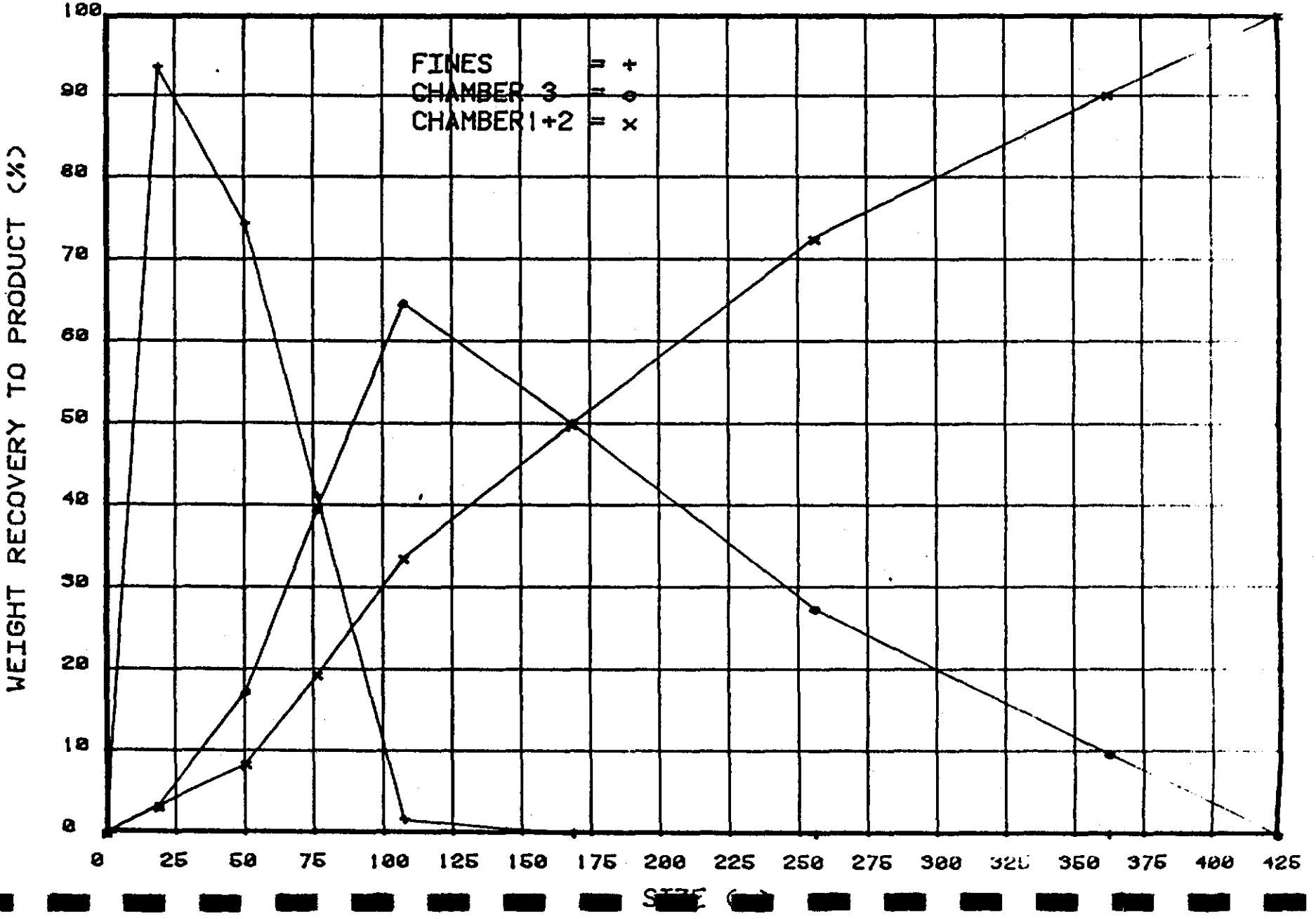
HYDROSIZER PERFORMANCE FOR SHELL MINERALS TIN ORE.



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APPENDIX 4.5: HYDROSIZER CELLS NOS. 1 & 2 UNDERFLOW

SHELL METALS TIN ORE. C 1+2

SIZE (um)	WT. %	CUM WT%	ASSAY %Sn	CUM ASSAY	DISTN %	CUM DISTN
425	2.64	2.64	0.32	0.32	1.65	1.65
300	40.46	43.10	0.35	0.35	27.95	29.60
212	22.50	65.60	0.42	0.38	18.62	48.22
125	20.13	85.73	0.65	0.44	25.54	73.76
90	6.56	92.29	0.87	0.47	11.11	84.87
63	3.86	96.15	1.02	0.49	7.68	92.56
38	1.27	97.42	1.32	0.50	3.27	95.83
-38	2.58	100.00	0.83	0.51	4.17	100.00
TOTAL	100.00	0.00	0.51	0.00	100.00	0.00

SHELL METALS TIN ORE. C 1+2 FEED BASIS

SIZE (um)	WT. %	CUM WT%	ASSAY %Sn	CUM ASSAY	DISTN %	CUM DISTN
425	1.04	1.04	0.32	0.32	0.94	0.94
300	15.93	16.97	0.35	0.35	15.92	16.86
212	8.85	25.84	0.42	0.38	10.61	27.47
125	7.93	33.76	0.65	0.44	14.55	42.01
90	2.58	36.35	0.87	0.47	6.33	48.34
63	1.52	37.87	1.02	0.49	4.38	52.72
38	0.50	38.37	1.32	0.50	1.86	54.59
-38	1.02	39.38	0.83	0.51	2.37	56.96
TOTAL	39.38	0.00	0.51	0.00	56.96	0.00

APPENDIX 4.7: HYDROSIZER SLIME OVERFLOW

SHELL METALS TIN ORE. FINES

SIZE (um)	WT. %	CUM WT%	ASSAY %Sn	CUM ASSAY	DISTN %	CUM DISTN
425	0.00	0.00	0.00	0.00	0.00	0.00
300	0.00	0.00	0.00	0.00	0.00	0.00
212	0.00	0.00	0.00	0.00	0.00	0.00
125	0.00	0.00	0.00	0.00	0.00	0.00
90	0.35	0.35	0.10	0.10	0.16	0.16
63	8.65	9.00	0.06	0.06	2.47	2.64
38	11.91	20.91	0.09	0.08	4.67	7.31
-38	79.09	100.00	0.25	0.22	92.69	100.00
TOTAL	100.00	0.00	0.22	0.00	100.00	0.00

SHELL METALS TIN ORE. FINES FEED BASIS

SIZE (um)	WT. %	CUM WT%	ASSAY %Sn	CUM ASSAY	DISTN %	CUM DISTN
425	0.00	0.00	0.00	0.00	0.00	0.00
300	0.00	0.00	0.00	0.00	0.00	0.00
212	0.00	0.00	0.00	0.00	0.00	0.00
125	0.00	0.00	0.00	0.00	0.00	0.00
90	0.13	0.13	0.10	0.10	0.04	0.04
63	3.23	3.37	0.06	0.06	0.57	0.60
38	4.45	7.82	0.09	0.08	1.07	1.67
-38	29.58	37.39	0.25	0.22	21.21	22.88
TOTAL	37.39	0.00	0.22	0.00	22.88	0.00

APPENDIX 4.8: HYDROSIZER PERFORMANCE DATA

Product	Wt. %	% Sn	Sn Distn.
No. 1 & 2 Cell U/F (Coarse Size)	39.4	0.510	56.6
No. 3 Cell U/F (Medium Size)	23.2	0.310	20.2
No. 3 Cell O/F (Slime Fines)	37.4	0.220	23.2

095

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APPENDIX 4.9

SHELL METALS TIN ORE, COARSE FRACTION SPIRAL TEST.

PRODUCT	TPH	%WT.	%SOL	% Sn	UGR	DIST
CON	0.00	7.40	100.00	5.25	10.446	77.30
MID	0.00	14.76	100.00	0.34	0.568	8.39
TAIL	0.00	77.84	100.00	0.11	0.184	14.31
Feed	0.00	100.00	100.00	0.60	1.000	100.00
Prd.1-2	0.00	22.16	100.00	2.31	3.867	85.69

Ratio of Conc <3.

Too high.

SHELL METALS TIN ORE, MED. FRACTION SPIRAL TEST.

PRODUCT	TPH	%WT.	%SOL	% Sn	UGR	DIST
CON	0.00	10.44	100.00	3.80	7.611	79.45
MID	0.00	18.99	100.00	0.28	0.561	10.65
TAIL	0.00	70.57	100.00	0.07	0.140	9.89
Feed	0.00	100.00	100.00	0.50	1.000	100.00
Prd.1-2	0.00	29.43	100.00	1.53	3.062	90.11

Ratio of Conc
8:1

Better
(Wilfley Riban?)

APPENDIX 4.10

SHELL METALS TIN ORE, COARSE FRACTION SPIRAL CON.

SIZE (um)	WT. %	CUM WT%	ASSAY %Sn	CUM ASSAY	DISTN %	CUM DISTN
425	0.98	0.98	4.97	4.97	0.78	0.78
300	15.08	16.06	9.43	9.16	22.76	23.54
212	13.36	29.42	10.16	9.61	21.73	45.27
180	7.42	36.84	8.02	9.29	9.52	54.79
125	24.48	61.32	4.71	7.46	18.45	73.25
90	22.00	83.32	4.41	6.66	15.53	88.77
63	11.78	95.10	3.89	6.31	7.33	96.11
38	3.52	98.62	4.24	6.24	2.39	98.50
-38	1.38	100.00	6.80	6.25	1.50	100.00
TOTAL	100.00	0.00	6.25	0.00	100.00	0.00

Sn

↑ Very even grade distrib.

SHELL METALS TIN ORE, MED.FRACTION SPIRAL CON.

SIZE (um)	WT. %	CUM WT%	ASSAY %Sn	CUM ASSAY	DISTN %	CUM DISTN
300	8.45	8.45	0.40	0.40	2.45	2.45
212	8.64	17.09	0.58	0.49	3.63	6.09
180	5.18	22.27	0.62	0.52	2.33	8.42
125	15.91	38.18	0.75	0.62	8.66	17.07
90	23.28	61.46	0.77	0.67	13.00	30.07
63	24.22	85.68	1.36	0.87	23.89	53.97
38	9.59	95.27	2.82	1.06	19.62	73.58
-38	4.73	100.00	7.70	1.38	26.42	100.00
TOTAL	100.00	0.00	1.38	0.00	100.00	0.00

APPENDIX 4.11

SHWELL METALS TIN ORE, COARSE FRACTION TABLE TEST.

PRODUCT	TPH	%WT.	%SOL	% Sn	UGR	DIST
CN	0.00	5.43	100.00	47.92	7.879	42.78
MID	0.00	18.67	100.00	17.50	2.877	53.72
TAIL	0.00	75.90	100.00	0.28	0.046	3.49
Feed	0.00	100.00	100.00	5.08	1.000	100.00
Prd.1-2	0.00	24.10	100.00	24.35	4.004	96.51

SHELL METALS TIN ORE, MED.FRACTION TABLE TEST.

PRODUCT	TPH	%WT.	%SOL	% Sn	UGR	DIST
CON	0.00	4.53	100.00	42.36	10.560	48.89
MID	0.00	9.11	100.00	20.80	5.185	47.24
TAIL	0.00	86.26	100.00	0.18	0.045	3.87
Feed	0.00	100.00	100.00	4.01	1.000	100.00
Prd.1-2	0.00	13.74	100.00	28.07	6.996	96.13

APPENDIX 4.12

SHELL METALS TIN ORE, COARSE FRACTION MAG SEP.

PRODUCT	TPH	%WT.	%SOL	% SN	UGR	DIST
MAGS	0.00	61.69	100.00	70.85	1.484	91.57
MID	0.00	28.63	100.00	12.20	0.256	7.32
NON-MAGS	0.00	9.68	100.00	5.50	0.115	1.12
Feed	0.00	100.00	100.00	47.73	1.000	100.00
Prd.1-2	0.00	90.32	100.00	52.26	1.095	98.88

SHELL METALS TIN ORE, MED.FRACTION MAG SEP.

PRODUCT	TPH	%WT.	%SOL	% Sn	UGR	DIST
MAGS	0.00	59.22	100.00	66.63	1.541	91.23
MID	0.00	30.17	100.00	10.50	0.243	7.32
NON-MAGS	0.00	10.61	100.00	5.90	0.136	1.45
Feed	0.00	100.00	100.00	43.25	1.000	100.00
Prd.1-2	0.00	89.39	100.00	47.69	1.103	98.55

APPENDIX 4.13: QUANTITATIVE XRF SCAN OF TABLE CONCENTRATES

A. COARSE SIZE FRACTION TABLE CONCENTRATES

Assay Range	Elements Detected
>5	Sn
0.05% - 0.5%	Ti, Cr, Mn, Zr
50 - 500 $g.t^{-1}$	Ni, Cu, Zn, Y, Nb
50 $g.t^{-1}$	As, Rb

b. MEDIUM SIZE FRACTION TABLE CONCENTRATES

Assay Range	Elements Detected
>5	Sn
0.05% - 0.5%	Ti, Cr, Mn, Zr
50 - 500 $g.t^{-1}$	Ni, Cu, Zn, Y, Nb
50 $g.t^{-1}$	As, Rb

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MINERAL DEPOSITS LIMITED

(INCORPORATED IN NEW SOUTH WALES)

81 ASHMORE ROAD, BUNDALL, QUEENSLAND AUSTRALIA 4217

MVA:PJY

July 6, 1984.

The Manager,
The Shell Company of
Australia Limited,
Metals Division,
Box 872K G.P.O.,
MELBOURNE. VIC. 3001.

Attention Mr. John Collins

Dear Sir,

We refer to our conversation of July 3, and your subsequent telex re our report No. 16.020.1/1 on testwork on a tin ore sample submitted by your company. The items in your telex are noted, and commented on as follows:

Item 1: The mineralogical examination of wet table products as recorded in our scope of work February 22, 1984, refers to table products only. We feel the report adequately covered the table concentrate with observations and the magnetic separation backed up by chemical analysis. However, we will carry out further work on the following: (a) table concentrates; (b) table middling; (c) coarse spiral middling, as these fractions contain the bulk of the tin in the middling fractions. A further report will be issued.

Item 2: The general directives of the correspondence between MDL and SMD (see enclosed correspondence) did not state as a requirement application of the testwork to a flowsheet design; in fact, the implication of the correspondence from SMD was that the flowsheet had been fixed and that the testwork was part of a confirmation of the overall flowsheet concept.

We will, however, comment on the results re a plant flowsheet design as per your request as follows:

A general flowsheet concept would consist of the following basic stages:

POSTAL ADDRESS: P.O. BOX 5044 GOLD COAST MAIL CENTRE, BUNDALL 4217 QUEENSLAND, AUSTRALIA
TELEX: MINDEP AA40438 CABLES: MINDEPOSIT BUNDALL QUEENSLAND TELEPHONE: (075) 39-9055

The Shell Company of
Australia Limited

July 6, 1984.

1. Primary crushing.
2. Secondary and tertiary crushing.
3. Autogenous or semi-autogenous grinding with (2) deleted.
4. Primary grinding with (3) deleted.
5. Gravity concentration - magnetic separation.
6. Regrind - gravity concentration - magnetic separation.
7. Flotation.

1-4: Crushing-Grinding

No testwork was conducted on the crushing or grinding characteristics of the ore. However, by observation of the crushing and grinding carried out to generate samples for the gravity concentration testwork, the ore appears to be relatively hard with resultant high energy requirement for reduction. No comments can be made on the ore susceptibility to autogenous and semi-autogenous grinding, although there could be distinct advantages with regard to operation and capital costs and simplicity of flowsheet if the ore could be ground by either of these methods.

5. Gravity Concentration

If the ore is as hard as suspected, there would be distinct advantages in energy savings in the grinding circuit by being able to reject coarse tailings. The gravity concentration testwork indicated that at least 30% of the original feed could be rejected as a final tailing at relatively coarse sizes and still maintain acceptable tin losses.

The proportion of the original feed that could be rejected to final tailing by gravity concentration in practice cannot be defined without having a detailed study carried out on the liberation characteristics of the ore and the gravity concentration process.

In other words, it may be possible to initially grind coarser than -0.400mm, which should result in less "slime" tin being generated. Gravity concentratiosn followed by a regrind of select middling products and furtner gravity concentration may improve both the recovery of tin to final product and also the proportion of the original feed rejected by the gravity

concentration stage, thus reducing the energy requirement for preparation of a flotation feed.

Initial magnetic separation testwork on final gravity concentrate showed significant upgrading could be achieved by rejection of the magnetic fraction of the heavy mineral gravity concentrate while maintaining acceptable tin losses. Final non-magnetic concentrate grades of between 66% and 70% Sn were achieved. Magnetic separation after gravity concentration in a flowsheet can therefore be justifiably included at this stage. Although an induced roll dry magnetic separator was used in the test programme, in practice the testwork indicated that acceptable results could be achieved on a Wet High Intensity Magnetic Separator (WHIMS). Further investigation on the performance of a WHIMS would be required.

6. Flotation

No flotation testwork has been conducted so comment is difficult. However, until the pretreatment stages are defined the actual feed to a flotation circuit cannot be generated; so meaningful testwork at this stage would be difficult to achieve.

Item 3: A further recommended programme of testwork would consist of three phases:

Phase 1:

(a) A fundamental liberation test programme to define the optimum size to which the ore should be ground initially to achieve a maximum recovery of tin by gravity concentration while maintaining maximum rejection of the solids to tailings and a minimum generation of tin "slimes".

(b) Further liberation testwork on select middling fractions followed by gravity concentration to increase tin recovery and increase tailing rejection at the gravity concentration stage.

(c) Further investigations on upgrading the gravity concentrate.

The work would be conducted on a bench scale with small samples - say ten to 20 kg - from the remainder of the sample supplied. Grinding would be conducted in bench scale wet grinding mills and the ground products concentrated on laboratory scale wet shaking tables.

The basic data produced from the bench scale test programme will allow the better definition of a full scale test programme.

Phase 2: The results from the bench scale testwork plus the additional work carried out on table and spiral products from the current testwork will allow a decision to be made on whether additional testwork is justified on products such as spiral middling and wet shaking table tailings which have been generated from the current testwork. This testwork, if completed, would take the form of further gravity concentration/regrind stages to assess whether further improvements to the current test circuit, such as increased rejection of tailings and additional recovery of tin by gravity methods, was achievable.

Phase 3: The scope of work of Phase 3 would be dependent on the findings of Phases 1 and 2, and also the finding of the current report. If a large scale laboratory testing programme is recommended, the form would probably be:

1. Generate large bulk samples of 20-35 tonnes.

2. Define crushing-grinding characteristics of the ore which is needed to select crushing-grinding equipment.

3. Carry out crushing-grinding test programme.

4. Full scale gravity concentration programme.

5. Flotation testwork programme.

The Shell Company of
Australia Limited

July 6, 1984.

We hope that we have clarified the points that you have raised, and will despatch to you the additional test data as soon as it comes to hand.

Yours faithfully,

M. V. ATFIELD,
CHIEF METALLURGIST.

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THE SHELL COMPANY OF AUSTRALIA LIMITED

METALS DIVISION

A FURTHER STUDY ON CASSITERITE OCCURRENCE IN MIDDLING PRODUCTS FROM TESTWORK REPORT NO.16.020.1/1

REPORT NO. 16.020.1/2

MINERAL DEPOSITS LIMITED
MINERAL TECHNOLOGY DIVISION
BUNDALL QUEENSLAND
AUSTRALIA

INDEX

1. INTRODUCTION 1

2. TESTWORK 1

2.1 Material 1
2.2 Procedure 1

3. RESULTS OF TESTWORK 2

4. CONCLUSIONS 6

C. 109

1. INTRODUCTION

This report covers the further examination of the spiral and table middlings products that were produced from testwork reported by Mineral Deposits Limited (MDL) (Report No. 16.020.1/1). The examination followed verbal discussion between Mr. John Collins of Shell Metals Division (SMD) and Mr. Max Atfield of MDL.

The object was to determine the degree of liberation of the tin in the middling products.

2. TESTWORK

2.1 Material

The samples examined were:

2.1.1 Coarse fraction spiral middling;
2.1.2 Medium fraction spiral middling;
2.1.3 Coarse fraction table middling;
2.1.4 Medium fraction table middling.

2.2 Procedure

The samples were individually processed in the following manner:

2.2.1 Sink-float separation using Bromoform (s.g. 2.90).
2.2.2 Magnetic separation of 2.90 s.g. sink material to produce three clean products:
2.2.2.1 Highly susceptible Ilmenite/Magnetite;
2.2.2.2 0.35 amp Magnetics (Ilmenite);
2.2.2.3 0.35 amp Non-magnetics;
using a laboratory-scale Carpco Induced Roll Magnet.

2.2.3 Further heavy liquid sink-float separation using Clerici's Solution (s.g. 4.05) on the 0.35 amp Non-magnetic fraction.

2.2.4 All samples were subjected to the standard HCl-Zinc block test procedure to allow ease of identification of the Cassiterite particles by microscopic examination.

2.2.5 The samples were then screened at +180 microns, +106 microns and -106 microns so as to observe the size distribution of the Cassiterite grains.

3. RESULTS OF TESTWORK

3.1 Table 1 below shows the weight distributions of the heavy liquid and magnetic separations:

TABLE 1

Spiral Middlings	Coarse Fraction	Medium Fraction
S.G. 2.90 Float	93.63	94.45
H/S - Magnetite	0.49	0.29
0.35 amp Mags	0.34	0.10
0.35 amp N/mags s.g. 4.05 float	5.24	4.66
s.g. 4.05 sink	0.30	0.50
Table Middlings	**Coarse Fraction**	**Medium Fraction**
S.G. 2.90 float	22.55	34.15
H/S - Magnetite	4.77	2.27
0.35 amp Mags	25.78	18.80
0.35 amp N/mags s.g. 4.05 float	24.16	35.48

3.1.1 Comment

Microscopic examination of the 2.9 s.g. floats fractions and the highly susceptible magnetic fractions from the four products revealed that no significant free or composite Cassiterite grains occurred in these fractions.
Microscopic examination of the 0.35 amp Magnetic fractions indicated that less than 0.5% of the grains present were fine Cassiterite grains which were present in sizes below 106 microns. These fine Cassiterite grains probably reported to the Magnetic fraction as a result of mechanical entrapment.

3.2 Tables 2 - 5 summarise the results of the size analysis and microscopic examination of the size fractions of the 0.35 amp Non-magnetics from each product where most of the tin occurs.

check meaning

The degree to which the tin minerals are locked was estimated on a volumetric basis by observation and must be accepted on that basis.

TABLE 2: Coarse Fraction Spiral Middlings

	DEGREE OF LIBERATION				
FRACTION	LIBERATED Sn MINERALS	HIGH Sn MIN. CONT	MED/HIGH Sn MIN CONTENT	MED/LOW Sn MIN CONTENT	LOW/TRACE Sn MIN CONTENT
N/M -4.05 +180	—	—	—	8	92
+160	—	—	—	7	93
-106	—	—	—	2	98
N/M +4.05 +180	18	31	31	6	15
+106	21	16	15	8	40
-106	28	7	5	—	60

Comment

The results show that a significant proportion of the Sn minerals are liberated in each size fraction.

A high proportion of the tin minerals occur in the fractions which have a liberation potential. Sn minerals do occur to some degree throughout most of the range.

TABLE 3: Medium Fraction Spiral Middlings

FRACTION	DEGREE OF LIBERATION				
	LIBERATED Sn MINERALS	HIGH Sn MIN.CONT	MED/HIGH Sn MIN CONTENT	MED/LOW Sn MIN CONTENT	LOW/TRACE Sn MIN CONTENT
N/M -4.05 +180	—	—	—	1.5	98.5
+106	—	—	—	1.0	99.0
-106	—	—	—	Trace	100
N/M +4.05 +180	5	4	1	—	90
+106	5	2	1	—	92
-106	40	—	—	—	60

Comment

The results show that the bulk of the tin minerals occur as liberated grains. The liberated grains are much finer than the coarse spiral middling.

TABLE 4: Coarse Fraction Spiral Middlings

FRACTION	DEGREE OF LIBERATION				
	LIBERATED Sn MINERALS	HIGH Sn MIN.CONT	MED/HIGH Sn MIN CONTENT	MED/LOW Sn MIN CONTENT	LOW/TRACE Sn MIN CONTENT
N/M -4.05 +180	—	—	—	8.5	91.5
+106	—	—	—	3.0	97
-106	—	—	—	1.0	99
N/M +4.05 +180	35	29	27	6	3
+106	21	23	15	5	36
-106	41	14	6	5	54

Comment

The results indicate that a significant proportion of the Sn minerals are liberated, with a significant proportion occurring below 106 microns.

A high proportion of the Sn minerals occur in the fractions which have a liberation potential.

Sn minerals occur to some degree throughout most of the range.

TABLE 5: Medium Fraction Table Middlings

FRACTION	DEGREE OF LIBERATION				
	LIBERATED Sn MINERALS	HIGH Sn MIN.CONT	MED/HIGH Sn MIN CONTENT	MED/LOW Sn MIN CONTENT	LOW/TRACE Sn MIN CONTENT
N/M -4.05 +180	—	--	---	9.5	90.5
+106	—	—	—	4	96
-106	—	—	—	1	99
N/M +4.05 +180	11.5	6	9.5	6.5	66.5
+106	13.5	3.5	9.0	3.5	70.5
-106	25.0	8.0	3.5	—	63.5

Comment

The data show that a significant proportion of the Sn minerals is liberated and is relatively fine.

A high proportion of the Sn minerals occurs in the fractions which have a liberation potential.

Sn minerals occur to some degree throughout most of the range.

4. CONCLUSIONS

4.1 The data show that all four products (which contain the bulk of the tin in the middling fractions) examined contained significant proportions of liberated tin which could be recovered by further gravity concentration.

4.2 The bulk of the tin minerals in the medium spiral middlings is liberated and therefore further grinding is not justified.

4.3 The coarse spiral middlings and both table middlings contain a high proportion of potentially recoverable tin minerals in the form of composites and investigation on regrinding and further gravity concentration is justified.

4.4 The data show that significant proportions of the other liberated or near-liberated heavy minerals are occurring in the middlings fractions and some may be rejected with careful control of the process.

4.5 A significant proportion of the Sn values is reporting to the -106 micron fraction in all middling products. The testwork originally reported indicated that Cassiterite particles finer than 106 microns could be recovered by gravity concentration. However, grinding initially to a coarser top size may reduce the tendency to overgrind valuable heavy minerals.

M. V. Atfield

M. V. ATFIELD,
CHIEF METALLURGIST.

P. A. Butler

P. A. BUTLER,
SENIOR PROJECT TECHNICIAN.

Bundall,
July 19, 1984.

NOTE

To : MTE/2
BXG - B.A., Melbourne
From : MTE/24
Cc. : MTE, MTE/23, CF
BXP, BXHE, BXHE/1, B.A. Melbourne
Date : 15th November, 1984

Subject : Great Pyramid Project Review

1. This note summarises observations and comments, discussed with BXG and BXP in Scamander, Tasmania, and Melbourne, Victoria , on 9 and 10 October 1984, which followed on my review of the Great Pyramid data package at Scamander on 8/9 October 1984. An appendix carries a number of additional observations from B.A. which both complement and qualify the comments made hereunder. A more detailed examination of selected aspects of the review will be carried out in due course.

2. The Great Pyramid prospect has a composite physical configuration of a well defined hill feature with prolific outcrop, sparingly vegetated. It has been extensively accessed by both Forestry and Exploration interests. Exploration work consists of 2 diamond drilling programmes (23 holes total); 3 surface costeans; a percussion drilling programme on a 15 m (along lines) by 30 m (line spacing) grid (127 holes); and 15 adits from various entry points around the circumference of the hill, basically at 2 distinctive levels of entry.

3. The adits, which total 1250 m in length (driven over the period 1909-1914) were designed to follow, rather than crosscut mineralization, but end up doing about an equal amount of both. The adit walls have been cumulatively channel sampled at various periods since the adits were driven - most recently by BA in three key locations. Selected bulk (large volume) samples have also been taken from the adits in recent years for both comparative grade analysis and metallurgical testwork.

4. The total output of this work effort adds up to a large and well knit database, fully adequate for a preliminary appraisal of mineral reserves. The project is bona fide advanced stage, but in that the data in hand was gathered by a number of operators over an extended number of years, the interrelationships and compatibility of the constituent data types need further rationalisation.

5. The geological model for the Great Pyramid deposit has been very positively defined both from the gross sense in its regional setting, and from the local model sense, with its structural and geological controls to mineralization.

6. The deposit is hosted in a sequence of interbedded Lower Devonian quartzites, sandstones and shales, which strike NW and maintain a general dip of 70° to the SW. Quartzite and sandstone predominate in the Great Pyramid hill.

7. A set of close spaced joints follows the regional NE-SW fracture trend (at right angles to bedding) and are best developed in the quartzites. Sn mineralization (as cassiterite) is contained by the joints, which dip at about 70° NW. The joints are up to 5 mm wide, but normally in the order of 1 mm, with or without quartz healing. Some are partially open. Joint intensity is in the order of 80 or so per m, but may be double this in the main mineralized areas. Hence mineralization relates lithologically to quartzite, and varies in grade proportionally with joint density.

8. Three distinctive "blocks" of preferentially enhanced Sn mineralization, known as North, South and Brocks blocks, have been defined within the Great Pyramid geological sequence.

9. The Great Pyramid mineral reserves have been assessed by BA and their predecessors on the project, on the basis of the percussion drilling sample results, which form the most general and consistent data set in the database. The percussion holes were drilled vertically at about 150 mm diameter (unconfirmed) and are up to 60 m deep. A number of the holes bottom is significant Sn mineralization.

10. The results of diamond drilling have not been used in mineral reserves estimation, apart from one instance of a limited area assessment by BHP using their own 19 hole programme. The BHP holes were oriented parallel to bedding and normal to joint direction - those holes in the second diamond drilling programme (Tasmanian Mines Dept.), were drilled normal to bedding strike. The latter was out of step with the (now) recognised geological controls to Sn mineralization (see also appendix, item 1).

11. Comparative grade studies (BHP) have shown that diamond drilling produces consistently lower grade results than does percussion drilling. There is convincing evidence in the drill core that significant Sn values are being lost to sludge during drilling. Records report numerous drill operational problems in the intensely jointed ground. The question of diamond drill grade validity calls for further investigation.

12. The results of adit wall channel sampling are illustrative of the lateral variability of Sn grades, but are at the same time supportive overall of reserve block grades estimated from the percussion drilling data. They lend a measure of confidence to the reserve estimation. Extension of channel sampling to provide comprehensive and BA validated coverage of all adits is worth consideration in a future work programme.

13. In that the geological continuity of the deposit is assured, the grade consistency of the database is the definitive criterion in reserve estimation. Using the percussion drilling, with channel sample support, a number of reserve estimates have been made, most notably by BA and their project predecessors BHP and Aberfoyle.

14. The estimates range from 3 million tonnes grading 0.20% Sn up to 4 million tonnes grading 0.30% Sn for the three identified mineralized blocks. The figures are a measure of the limits of the drilled deposit. There remains potential for extension at depth which can add to the estimated tonnage.

15. The methods used for mineral reserves estimation were basically polygonal/area of influence type, incorporating preferred variations depending on who did the estimation. At this stage of the project it is not possible to envisage that the data could have been more correctly treated in an alternative manner. A mineral reserve made by BA via the medium of SHLMET using an inverse distance weighting technique gave an outcome within the lower range of the polygonal estimates (see also appendix, item 2).

16. A statistical analysis of Sn grade data (SHLMET) demonstrated unimodality and highlighted discrepancies between diamond drill and percussion drilling grades. Acceptably structured and stable downhole semivariograms were calculated on diamond drill grades, and indicated a range of influence of only 5 m. The impracticality of designing a drilling programme to cope with this limitation is obvious.

17. The mineral reserve tonnage is assured in view of the clear cut definition of the geological framework to the mineral reserve. It can be confidently anticipated therefore that a deposit tonnage of at least 3 million tonnes, and as much as 4 million tonnes is available. Additional work will serve to confirm rather than amend the figures, and might be aimed at giving a better reading on the quality of the potential at depth, since this direction offers the latitude for tonnage increase.

18. In consideration of mineral reserve grade, it must be acknowledged that the percussion drill holes are unable to adequately define the short distance grade variability of the Great Pyramid deposit. However, they do demonstrate the consistency of occurrence of Sn mineralization within the 3 reserve blocks.

19. Mineral reserves within the indicated grade range are acceptable, and what needs to be established is whether or not (a) they are at the top or bottom of the range, or (b) they could exceed the upper limit, and if so by how much.

20. There are two lines of investigation which could be followed to improve the projects current perception of grade:

20.1.1 The results of a diamond drilling programme, in which the holes are correctly oriented with regard to geological controls, should, given reasonable core recovery, provide a more definitve reading on grade than would an equivalent percussion drilling programme. This situation appears reversed at Great Pyramid.

20.1.2 Confirmation of the probable mechanism of Sn losses during coring has been noted above.

20.1.3 A further possibility to consider could be that of preferred orientation to the distribution of Sn mineralization in the joints, which the diamond drilling may have missed. This is not very likely and it is recognised that it would be difficult to define.

20.1.4 For discussion purposes it may be worth assuming that the distribution of Sn mineralization on a single joint is fairly general, with some allowance for variability. Each joint plane and the mineralization that it contains, might then be seen as a unique "shoot" of large planar extent, but minimal thickness. It is necessary to maximise on the planar dimension to approach representative grade and obtain the optimum sample. This builds up to the critical size for sample definition, beyond which it does not matter. If this assertion has any validity it is then clear that the percussion drill holes, on both diameter and inclination attitude to joint planes, take a substantially larger sample of a mass of "shoots" than do the diamond drill holes.

20.1.5 The diamond drill holes tend to lose Sn from core in open fractures - but there can also be gravitationally inspired or open fracture losses of Sn in percussion holes. The volumetric aspect of sample definition, and the Sn recovery problem might be advantageously studied further.

20.2 Bulk Sampling

20.2.1 Sample volume will reach maximum potential in bulk samples. Most parties who have examined the Great Pyramid data have suggested that a rationalisation of grade accuracy could be made via bulk sampling. A number of such samples have been taken from adits.

20.2.2 The samples might best be described as "large" rather than "bulk" since the latter term carries a very specific and purposeful correlation. There seems little doubt that their size at Great Pyramid offers the opportunity to define the grade factor, and hence to strengthen this aspect of the mineral reserve estimation.

20.2.3 The most noteworthy bulk samples were taken by BHP and BA, who based their sample sites on the results of their channel sampling in the North, C and ISLL adits. The BHP bulk samples came from the North, C, 2SLL and 2NLL adits, and the first two named give grade comparison with BA channel samples. All bulk samples were taken from sites where additional support can be derived from the results of channel sampling by the Great Pyramid Tin Co. (1909). Although GPTC grades are available only rounded off to the nearest 0.1% Sn, they compare reasonably well with the modern bulks.

20.2.4 The credentials of the bulk samples are queried on the following grounds:

- Their selectivity. The samples seem to have been taken from locations selected to show better than average grades.
- Their precision. The samples were taken with regard to supplying a gross volume of rock, rather than a precise volume of rock cut to clearly defined limits. The method of sample excavation has been necessarily somewhat crude. For these reasons comparison of bulk and channel sample grades may not be justified.

- Their definitive quality. The BA bulk samples were composited prior to grade analysis, and the composite grade cannot be related to its constituent grades. It may be noted that BA's bulk sample was intended for metallurgical requirements, and at the time of its selection average deposit grade, and the upgrading option, was not an issue (see also appendix, item 3).

20.2.5 Comparative grade data, as far as can be assessed for the BA and BHP bulks and channels are as follows:

Source	Adit Sampled	% Sn Channel Sample	% Sn Bulk Sample	Volume
BA	1SLL	0.28		3 drums
	C	0.32		4 "
	North	0.17		4 "
Mean		0.25	0.39	11 "
BHP	C	0.51	0.30	
	North	0.36	0.50	
	2SLL		0.45	
	2NLL		0.35	

20.2.6 The figures suggest that bulk sample grades offer a better and perhaps more definitive result than does the conventional percussion drilling/channel sampling, but they by no means confirm it.

20.2.7 Any error in the precision of bulk sample location, particularly in view of the evident lateral variability of the Sn mineralization, can effect the outcome. The bulk samples give higher than mean mineral reserve estimate grades, due at least in part to selectivity. There remains however a possibility that some form of overall deposit upgrading could be feasible. This presents a promising area for investigation, provided that accurate sampling controls can be applied to any further bulk samples taken.

124

21. The key factor for Great Pyramid is grade, not tonnage. It is suggested that an attempt to demonstrate the upgrading potential of the deposit might be made based on a single mineral reserve block, both to limit logistical problems and to utilise the channel sampling already completed (see also appendix, item 4).

22. South Block might be the preferred area. In it, both C and 1SLL adits have been extensively channel sampled by BA, and the balance of channel sampling could be completed fairly easily. This would be followed up by a channel sampling programme on 2SLL adit.

23. The 3 adits in South Block would require some form of rehabilitation prior to the channel sampling.

24. From the results of the full channel sampling programme, bulk sample sites might be selected to reflect various anticipated grade levels, the most critical of which to the project is that around the mineral reserve grade. Upgrading from the 0.25 - 0.30% Sn range will provide the optimum benefit to the project. Upgrading will keep lower grade low, and make higher grades even higher. Neither will impact significantly on the bulk of the reserve which relates to mean grade.

25. The physical limits of bulk samples in-adit should be clearly set with respect to those of the channels.

26. A measure of the comparative accuracy of percussion drilling and diamond drilling might be made at the same time, with reference to bulk and channel sample grades as may be relevant.

27. The adits will also make ideal sites for underground confirmatory or exploratory diamond drilling.

Attachment

J.W. Platt
MTE/24

id 1579

APPENDIX

M E M O R A N D U M

DATE : 5 November, 1984

FROM : BXHE/1 - BA, MELBOURNE

TO : MTE/2 - BIM

cc : BXP, BXHE

GREAT PYRAMID PROJECT REVIEW

After examining MTE24's well presented Great Pyramid Project Review of 17/10/84, a number of additional points have been raised or need clarification.

1) Diamond drill hole results (section 10 of memo).

Results of all diamond drill holes have been incorporated in the SHLMET print out. Those portions of the holes which satisfy the conditions necessary for them to be included in the mineral reserve calculations, have been treated as the percussion holes and included in the mineral reserve calculations. The holes drilled along the strike of the veins, i.e. at right angles to the sections, appear on the sections as small intercepts covering a zone projected from where the holes enter and leave the area of influence of each section. Where applicable, these intersections have been incorporated in the mineral reserve calculations.

2) Method of Mineral Reserve Estimation (section 15).

The "polygonal/area of influence" method may not be the best description for the method used to calculate the reserves. The BA estimate was based on what could be referred to as the "envelope" principle, at 0.1% Sn cut-off. Mineralised blocks were outlined on the cross sections and transferred to the level plans. The volume was calculated from the level plans by multiplying the area of the 0.1% Sn zone on that level, by the constant 10 m vertical distance between levels.

The grade was calculated as an arithmetical mean of the average grade of each drill hole. Thus, no particular area of influence was given to the individual holes within the reserve block. A trial calculation of grade, using polygonal zones of influence around each hole produce similar results. However, considering the possible nugget effect resulting from the small sample represented by the drill holes, the method used is considered to give a more realistic grade estimation.

3) Bulk Sampling (section 20.2.4)

While it is quite true that the BHP bulk samples were selected from sites with better than average grades, the BA sample was not. The intention of BA was to select a sample which represented the average grade of the deposit. The calculated estimated grade of 0.25% Sn of the bulk, is in line with the average grade of 0.22% Sn for the deposit.

The precision of the sampling is considered to be quite high in two dimensions. Along the adit wall, extreme care was taken to collect the bulk samples over specifically measured distances. The samples were

taken over measured, and marked, previously sampled channels for which the grade was known. The irregularities of the back face indicate that a perfect parallelepiped was not taken. However, the size of the samples (minimum 3 drums or about 1 tonne), should have been sufficient to eliminate any nugget effect caused by these irregularities.

4) Grade (sections 21-25)

BA is in complete agreement that grade is the key factor for Great Pyramid. The programme of bulk sampling, as presented in September, and about to commence will result in 14 samples, of +0.5 tonnes each, being collected from North Block (6 samples), South Block (6 samples and surrounding areas (2 samples). The samples have been selected from the results of previous sampling and cover 0.1-0.65% Sn grades. A variety of rock types have also been selected (Tables 1, 2).

It is felt that the upgrading of results in the 0.1-0.35% Sn range is extremely important and that to concentrate on the 0.25-0.3% Sn range as suggested, may be too restrictive.

D N CARTER

TABLE 1 BULK SAMPLING GREAT PYRAMID (BSS1-16)

No.	Location/Adit	Adit Wall	Distance from Portal (m)	Level (ASL)	Grade % Sn	Lithology	To be Channel Sampled
	Adit Samples						
BSS1	N*	E	14	195	0.5	Qtzite	Yes
BSS2	N	W	54.0-55.8 m	195	0.22	Sst/stt	No
BSS3	N	W	70.0-71.5 m	195	0.145	Sh	No
BSS4	E	N	40.0-42.0 m	185	0.1(?)	Qtzite	Yes
BSS5	F	S	9.0-10.0 m	185	0.15(?)	Sst/sh	Yes
BSS6	C	W	11.0-13.5 m	185	0.32	Sst/sh	No
BSS7	2SLL*	N	40 m	140	0.65	Qtzite	Yes
BSS8	2NLL**	N	54 m	165	0.33	Qtzite	No
BSS9	2NLL*	E	20 m	165	0.35	Qtzite	Yes
BSS10	2NLL	N	82 m BHP PCS17	165	0.23	Sst/sh	No
BSS11	1SLL**	W	48.6-51.2 m	145	0.28	Sst/sh	No
	Surface Samples						
BSS12	400E 175N			175	>0.3	Sst/Qtzite	Yes
BSS13	Nr BPD 10		(360E 155N)	165	<0.1	Sst/Qtzite	Yes
BSS14	Surface (end of MD1)		(315E 285N)	195	>0.1	Qtzite	Yes

* BHP Bulk Sample site.
** Billiton Sample collected not analysed.

Qtzite = quartzite; sst = sandstone; stt = siltstone; sh = shale.

TABLE 2

Grade (%Sn)	No. of Samples
< 0.1%	1
0.1–0.2	3
0.2–0.3	3
0.3–0.4	3
0.4–0.6	1
0.6–0.8	1
Uncertain	2
	14

Lithology	No. of Samples
Qtzite	6
Sst/Qtzite	2
Sst/stt	1
Sst/sh	4
sh	1
	14

130

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BILLITON AUSTRALIA, THE METALS DIVISION OF
THE SHELL COMPANY OF AUSTRALIA LIMITED

SCAMANDER, EL 12/78

COMPARISON OF ANALYTICAL DATA FROM BULK AND THEIR EQUIVALENT CHANNEL SAMPLES AT THE GREAT PYRAMID TIN PROSPECT

Author: D.N. Carter
Report No.: 08.2958
Date: August, 1985
Copy No.:

Distribution: 1. BHP, Melbourne
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CONTENTS

SUMMARY

1. INTRODUCTION

2. SAMPLE LOCATIONS

3. SAMPLING PROCEDURES

4. SAMPLE PREPARATION

5. RESULTS

6. CONCLUSIONS

Reference

FIGURES

No.	Title	Scale
1.	Location Plan	1:100,000
2.	Great Pyramid, Bulk Sample Locations	1:1,000

TABLES

1. Historical Metallurgical Tests
2. Bulk Sampling Details
3. Analytical Results (only +8mm portion through roll mill)
4. Analytical Results (entire sample through roll mill)

APPENDIX

I Report on Preparation and Analysis of Samples from Tasmania

SUMMARY

Preliminary work on bulk samples taken at Great Pyramid suggested that the analytical data obtained from channel samples may have resulted in an underestimation of the grade of the deposit.

To investigate this posssibility, fourteen bulk samples (each +500 kgms) were collected, prepared and analysed. Results from three laboratories were compared and contrasted to those obtained from channel samples which were taken at the same sites as the bulks. No significant differences were noted between the two data sets.

1. INTRODUCTION

Since the late 1960s, Aberfoyle, BHP and Shell have jointly or individually been assessing the Great Pyramid tin deposit. Each of these companies recognised that the grade of the deposit was one of the most critical factors in evaluating the economic potential of the resource.

Early efforts by the Mines Department (Table 1) to determine the viability of the project by establishing recovery rates, were done on material with tin grades well above the average (0.22%) calculated by Shell (Hall D.B., 1984).

Aberfoyle, in 1971, and BHP/Shell in 1983, completed basic recovery tests on more representative material. In Aberfoyle's tests, a 72 kgm composite sample of percussion drill chips, assaying 0.296% Sn, achieved a 63.5% recovery in a concentrate containing 47.6% Sn. BHP/Shell obtained a 54.5% recovery of a 53% Sn concentrate from a 546.7 kgm sample assaying 0.22% Sn. However, each of these companies expressed doubt as to what was a truly representative sample from the deposit. Discrepencies were continually noted when comparing analytical data between adjacent percussion and diamond drill holes as well as between a few bulk and their corresponding channel samples. This was highlighted early in 1984 when Shell conducted a series of recovery tests on a bulk sample which, according to earlier geochemical results from channel samples, should have contained 0.25% Sn. However, after analysing all the products of this test, the grade of the initial sample was calculated to be 0.414% Sn. Because the initial split on this sample was carried out on 10 cm material, a grain size well above the liberation size of the cassiterite, some doubts were expressed as to the reliability of the result. Nevertheless, it was considered possible that the smaller $\pm$5 kgm samples, taken as channels or from drill holes, may not have been sufficiently large to indicate the tin grade at any particular location. It was therefore proposed to collect a number of bulk samples (about 500 kgm each) at various sites, and compare the results from those obtained from channel samples taken at the same location.

2. SAMPLE LOCATIONS

Sample sites were selected so that data would be available from areas with a variety of tin grades and rock types (Fig. 2). However, the majority of

samples were taken from areas of predominantly sandstone/quartzite, with tin grades thought to be in the 0.15-0.3% range (Table 2).

3. SAMPLING PROCEDURES

Due to the nature of the sampling (jack hammers underground), Mines Department approval was necessary before any sampling was possible. Most of the adit entrances required some attention, while poor ground conditions in two of the tunnels forced some of the originally selected sites to be abandoned.

Most of the bulk sample sites corresponded to well marked channel sample sites. Where comparative channels were not available, but grades known from drill hole data, the channel and bulk samples were taken consecutively from the same location.

Each bulk sample was collected on a ground sheet and loaded into 200 litre drums for shipment to the laboratory.

4. SAMPLE PREPARATION

In an attempt to avoid any possible nugget effects, the following sample preparation route was requested from the laboratory:

1. Crush jaw crusher.
2. Jaw crusher discharge crush in roll crusher
3. Split roll crusher discharge on 10 way rotary splitter
4. Combine 2x10 way splits (100 kgs)
5. Re-split 10 ways
6. Take 1x10 kg split
7. Crush to -2 mm
8. Re-split 10 ways
9. Assay for Sn in triplicate : 1 Mineral Deposits (AAS)
   1 Comlabs (XRF)
   1 Amdel (XRF)

In addition, a second 100 kgm split (step 4) was collected for three of the samples (numbers 2b, 3b, 6b) and prepared for analysis using the same procedure.

A report on this work (Appendix I) indicates that the agreed sample flowsheet was not initially followed. According to the report, after initial crushing, the -8 mm fraction passed directly to the rotary divider prior to further size reduction by the roll crusher. As a result, the laboratory was required to process all the samples again ensuring that all material passed through the roll crusher prior to splitting and subsequent analysis.

5. RESULTS

Analytical results from the two sets of data are, to all intents and purposes, identical. Table 3 lists the results from the samples not passed through to roll crusher while Table 4 lists the data from those which did pass through the crusher.

While there are some discrepencies, in general the analytical results from the bulks are similar to those of the corresponding channel samples. Samples 1 and 7 are clearly anomalous and the sharp variation from the rest of the data suggests there may have been an original error in sample preparation or in sample location.

The majority of the data indicates that there is no significant difference between results gained from the larger and smaller samples. No indication of any significant assay variation due to nugget effect is indicated, with the three final subsets within each sample giving almost identical analyses within each of the three laboratories. In addition, the 100 kgm subsamples for samples 2, 3 and 6 gave results almost identical to those of the original.

The accuracy of the analyses is considered to be high, since the data from the three laboratories for any given sample are virtually the same.

6. CONCLUSIONS

The data indicates that there is no significant difference in analytical results between the bulk samples (each about 500 kgms) and their equivalent channel samples (each about 5 kgm). As a consequence of this, it can be concluded that a large scale programme of bulk sampling over the deposit would not give significantly different analytical results to those already obtained from the channel and rock sampling.

It is therefore suggested that results from an additional bulk sampling programme are unlikely to significantly alter the average grade figures used in the present ore reserve estimate.

Reference

Hall, D.B., 1984, Great Pyramid, Preliminary Resource Estimates 1983-4 (Shell Rep. 08.2959)

TABLE 1 : HISTORICAL METALLURGICAL TESTS

DATE	COMPANY	WORK COMPLETED	RESULTS	RECOVERIES (%)	CONCENTRATE (% Sn)	ORIGINAL GRADE (% Sn)	SAMPLE WEIGHT
1945	Mines Dept.	Size analysis	73% of -3 mm crushing measured -1.7 mm 48% of -3 mm crushing measured -0.5 mm Free SnO_2 in all -0.7 mm fractions.			0.64	
		Gravity concentration	-420 µm :	78.6	22.0		?
			-250 µm :	80.6	27.0		Composite of
			-180 µm :	80.2	27.0		2 samples
1945	Mines Dept.	Size analysis				0.64	as above
		Floatation	2 lbs Cetynal/ton+pine oil	72.8-81.8	6.4-11.1		
			3 lbs Cetynal/ton+pine oil	81.1-85.8	8(?)		
1963	Mines Dept.	Gravity concentration	a) Initial grind	79.26	61.6	2.17	1216 lbs
			Initial grid & regrinding +250 µm tailings	84.55	59.7		
			above and +150 µm tailings	87.05	59.0		
			b) Initial grind	73.81	50.5	0.76	1316 lbs
			above & +250 µm tailings	77.25	48.9		
			above & +150 µm tailings	79.76	46.9		
			c) Initial grind	60.59	46.9	0.47	676 lbs
			above & +250 µm tailings	66.61	43.8		
			above & +150 µm tailings	71.37	41.3		

Table 1 (continued)

DATE	COMPANY	WORK COMPLETED	RESULTS	RECOVERIES (%)	CONCENTRATE (% Sn)	ORIGINAL GRADE (% Sn)	SAMPLE WEIGHT
1971	Aberfoyle	Size analysis	32% Sn in -850+500 μm, 33% in -500+300 μm.			0.29	140 lbs (percussion drill chips)
		Gravity concentration	Various results from work on size fractions.	63.5	47.6		
		Magnetic separation	Removed impurities - "necessary phase of the treatment".				
1983	Shell/BHP	Size analysis	47% Sn in -300+75 μm range Liberation commences in +3.18 mm fraction.			0.22	546.7 kgs
		Heavy liquid tests :	2.95 t/m^3 : 1mm particle size necessary for effective pre-concentration	sink 73.0		1.30	(composite)
			2.7 t/m^3 : as above				
		Cassiterite grain analysis	90% cassiterite suitable for gravity separation.				
		Magnetic separation	High density iron minerals - necessary phase of operation.				
		Gravity concentration	Results from several size fractions.	54.5	52.9	0.22	
1984	Shell	Size Analysis	54% Sn in -300 to +125 μm +20% total Sn liberated at -0.4 mm			0.414	2 tonnes
		Hydrosizing				(calculated)	
		Spiral Tests	Coarse:86% Sn from 22% of original feed Medium:90% Sn from 29% of original feed				
		Gravity concentration	Concentrate, middling, tail results given	43	48		
		Magnetic separation	Results in increased concentrate grade				

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TABLE 2 BULK SAMPLING DETAILS

No.	Location/Adit	Distance from Portal (m)	Level (ASL)	Grade % Sn	Lithology
Adit Samples					
BSS 1	N*	14	195	0.5%	Qtz
BSS 2	N#	54.0-55.8	195	0.22%	Sst/stt
BSS 3	N#	70.0-71.5	195	0.145%	Shale
BSS 4	E	40.0-42.0	185	0.1%	Qtz
BSS 5	F	9.0-10.0	185	0.15%	Sst/shale
BSS 6	C#	11.0-13.5	185	0.32%	Sst/shale
BSS 7	2SLL*	40	140	0.65%	Qtz
BSS 8	2NLL	54	165	0.33%	Sst + Qtz
BSS 9	2NLL	22	165	0.3%	Qtz/shale
BSS 10	2NLL	82	165	0.23%	Qtz + shale
BSS 11	1SLL**	48.6-51.2	145	0.28%	Sst/shale
Surface Samples					
BSS 12		390E 185N	175	0.3%	Sst/Qtz
BSS 13	Near BPD 10	360E 155N	165	0.1%	Sst/Qtz
BSS 14		315E 285N	180	0.1%	Qtz

* BHP Bulk sample site.
** Billiton sample from earlier sampling programme.
\# Prepared in duplicate by Mineral Deposits Ltd.

Qtz = quartzite, Sst = sandstone, stt = siltstone

TABLE 3 : ANALYTICAL RESULTS (only +8 mm portion through roll mill)

SAMPLE NUMBER	BHP BULK (%)Sn	COMLABS (%Sn)**				AMDEL (%Sn)**				MINDEPS (%Sn)*				CHANNEL (%Sn) **
		1	2	3	Average	1	2	3	Average	1	2	3	Average	
1	0.5	-	-	-	-	-	-	-	-	-	-	-	-	0.29
2a	-	0.100	0.105	0.110	0.105	0.114	0.108	0.112	0.111	0.107	0.092	0.096	0.098	0.22
2b	-	0.105	0.105	0.105	0.105	0.114	0.112	0.104	0.110	0.091	0.084	0.083	0.086	-
3a	-	0.018	0.019	0.019	0.019	0.018	0.021	0.018	0.019	0.025	0.015	0.015	0.018	0.145
3b	-	0.018	0.017	0.017	0.018	0.018	0.021	0.018	0.019	0.025	0.015	0.015	0.018	-
4	-	0.185	0.205	0.200	0.197	0.198	0.220	0.204	0.207	0.212	0.182	0.192	0.197	0.13
5	-	0.260	0.260	0.270	0.263	0.298	0.288	0.268	0.285	0.258	0.228	0.236	0.241	0.25
6a	-	0.240	0.240	0.250	0.243	0.254	0.248	0.256	0.253	0.231	0.214	0.225	0.223	0.32
6b	-	0.245	0.240	0.230	0.238	0.260	0.244	0.236	0.247	0.235	0.218	0.219	0.224	-
7	0.65	-	-	-	-	-	-	-	-	-	-	-	-	0.37
8	-	0.315	0.315	0.300	0.310	0.346	0.382	0.346	0.358	0.290	0.251	0.262	0.267	0.33
9	-	0.240	0.250	0.240	0.243	0.242	0.262	0.258	0.254	0.257	0.207	0.224	0.229	0.30
10	-	0.155	0.160	0.160	0.158	0.172	0.180	0.186	0.179	0.178	0.151	0.157	0.162	0.23
11	-	0.360	0.360	0.410	0.377	0.388	0.360	0.354	0.367	0.315	0.298	0.306	0.306	0.28
12	-	0.240	0.225	0.220	0.228	0.280	0.280	0.285	0.282	0.279	0.250	0.250	0.260	0.232
13	-	0.260	0.275	0.260	0.265	0.266	0.268	0.288	0.274	0.243	0.224	0.227	0.231	0.335
14	-	0.009	0.009	0.009	0.009	0.011	0.010	0.010	0.010	0.016	0.016	0.015	0.016	0.0115

* analysis by atomic absorption spectrophotometry

** analysis by x-ray fluorescence

0471371

TABLE 4 : ANALYTICAL RESULTS (entire sample through roll mill)

ASSAYS %SN:

	MDL			AMDEL			COMLABS			MDL	AMDEL	COMLABS
										AVGE	AVGE	AVGE
SAMPLE	1	2	3	1	2	3	1	2	3			
BSS 2A	.101	.097	.090	.108	.112	.110	.110	.100	.105	.096	.110	.105
BSS 2B	.091	.094	.090	.114	.112	.112	.110	.110	.100	.092	.113	.107
BSS 3A	.015	.010	.015	.018	.019	.020	.017	.015	.016	.013	.019	.016
BSS 3B	.010	.010	.010	.018	.020	.020	.017	.017	.015	.010	.019	.016
BSS 4	.194	.169	.188	.220	.208	.222	.205	.205	.195	.190	.217	.202
BSS 5	.295	.278	.275	.284	.308	.282	.260	.280	.270	.283	.291	.270
BSS 6A	.242	.228	.230	.258	.268	.246	.230	.225	.235	.233	.257	.230
BSS 6B	.232	.210	.211	.250	.246	.246	.220	.215	.220	.218	.247	.218
BSS 8	.265	.263	.245	.310	.302	.288	.305	.305	.305	.258	.300	.305
BSS 9	.243	.223	.224	.238	.230	.258	.225	.235	.235	.230	.242	.232
BSS 10	.176	.169	.174	.174	.144	.178	.175	.165	.165	.173	.165	.168
BSS 11	.370	.336	.333	.332	.332	.358	.315	.335	.330	.346	.341	.327
BSS 12	.237	.225	.224	.232	.264	.258	.230	.240	.230	.229	.251	.233
BSS 13	.267	.247	.249	.266	.254	.260	.265	.235	.235	.254	.260	.245
BSS 14	.012	.010	.010	.010	.010	.010	.010	.010	.011	.011	.010	.010

047139

144

APPENDIX I

Report on Preparation and Analysis of Samples from Tasmania

by Mineral Deposits Limited

047140

145

THE SHELL COMPANY OF AUSTRALIA LIMITED

REPORT NO. 16.020.1/2

REPORT ON PREPARATION AND ANALYSIS OF SAMPLES FROM TASMANIA

MINERAL DEPOSITS LIMITED

MINERAL TECHNOLOGY DIVISION

BUNDALL - QUEENSLAND

AUSTRALIA

047141

CONTENTS

1.0 INTRODUCTION

1.1	Material	1
1.2	Objectives	1
1.3	Terms of Reference	1
1.4	Summary	2
1.5	Conclusions	2

2.0 TEST EQUIPMENT

2.1	Machines	4
2.2	Test Rigs	4
2.3	Analytical Systems	4

3.0 TREATMENT AND PROCEDURES

3.1	Procedures	6
3.2	Results	6
3.3	Discussion	6

147

1.0 INTRODUCTION

1.1 MATERIAL

The samples as received consisted of damp, lump ore in 200 litre drums. Twelve samples, each weighing about 500 kg, were received, labelled BSS 2, BSS 3, BSS 4, BSS 5, BSS 8, BSS 9, BSS 10, BSS 12, BSS 13, BSS 14, C.ADIT and 1 SLL. Samples C.ADIT and 1 SLL were subsequently renumbered BSS 6 and BSS 11 respectively.

1.2 OBJECTIVES

The samples were to be processed to give representative sub-samples for tin assay. Sub-samples were to be analysed by two independent laboratories by X-ray fluorescence (XRF) techniques, and by Mineral Deposits Limited (MDL) by atomic absorption (AA) spectrophotometry.

One hundred kilogram sub-samples of each sample were to be retained for further treatment if required. The full sample treatment flowsheet is given in Section 3.1.

1.3 TERMS OF REFERENCE

1.3.1 Telex from Mr. M. V. Atfield, MDL, to Mr. Derek Carter, Shell - STX 1274/84 of August 28, 1984.

1.3.2 Telex from Mr. Derek Carter, Shell, to Mr. M. V. Atfield, MDL - November 29, 1984.

1.3.3 Telex from Mr. M. V. Atfield, MDL, to Mr. Derek Carter, Shell - STX 3523/84 of December 18, 1984.

1.3.4 Telex from Mr. Derek Carter, Shell, to Mr. Peter Mills, MDL - January 21, 1985.

148

1.4 SUMMARY

Table 1 shows average results from the three laboratories:

TABLE 1: Average % Sn in Sub-Sample

SAMPLE	MDL	AMDEL	COMLABS
BSS 2	0.098	0.111	0.105
BSS 2	0.086	0.110	0.105
BSS 3	0.018	0.019	0.019
BSS 3	0.018	0.019	0.017
BSS 4	0.197	0.207	0.197
BSS 5	0.241	0.285	0.263
BSS 6	0.223	0.253	0.243
BSS 6	0.224	0.247	0.238
BSS 8	0.267	0.358	0.310
BSS 9	0.229	0.254	0.243
BSS 10	0.162	0.179	0.158
BSS 11	0.306	0.367	0.377
BSS 12	0.260	0.282	0.228
BSS 13	0.231	0.274	0.265
BSS 14	0.016	0.010	0.009

1.5 CONCLUSIONS

1.5.1 MDL assays are generally lower than the other two, except for very low concentrations of tin.

1.5.2 In most cases, the difference between MDL and Comlabs assays is comparable with the difference between Comlabs and Amdel assays.

1.5.3 Amdel assays are highest in all but one case.

1.5.4 MDL assays are lowest in all but four cases.

149

1.5.5 Considering the accuracy and reliability of the XRF assays, MDL results can be used to give a conservative estimate of the tin content of the samples.

A. P. MILLS,
SENIOR METALLURGIST,
MINERAL TECHNOLOGY DIVISION,
MINERAL DEPOSITS LIMITED.

M. V. ATFIELD,
MANAGER - TECHNICAL SERVICES,
MINERAL TECHNOLOGY DIVISION,
MINERAL DEPOSITS LIMITED.

Bundall,
April 12, 1985.

150

2.0 TEST EQUIPMENT

2.1 MACHINES

For primary crushing, a jaw crusher was used followed by an 8mm square aperture woven wire screen with screen oversize crushed in a roll crusher.

A special 10-way rotary divider was used to give 50 kg splits of each sample, and 100 kg sub-samples were obtained by combining two 50 kg splits.

A small jaw crusher was used to crush to -2mm, all material being hand-sieved to ensure it passed this size. Oversize was recrushed.

2.2 TEST RIGS

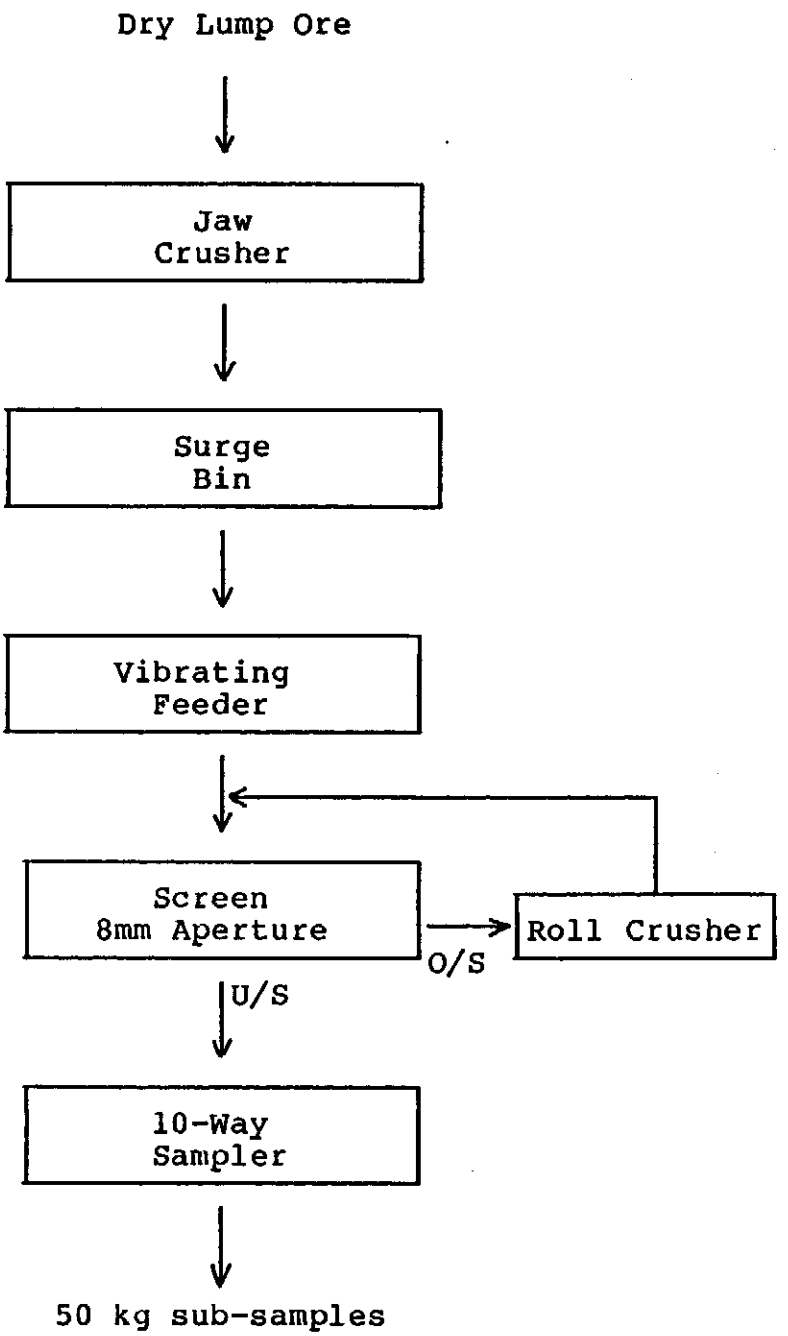
The jaw crusher, screen, roll crusher and 10-way sampler were set up in the circuit shown in Figure 1, and each 500 kg sample processed in turn. The remainder of the sample treatment was handled as a batch operation.

2.3 ANALYTICAL SYSTEMS

A final 1.25 kg sub-sample from each sample was assayed by standard AA spectrophotometry techniques at MDL's laboratory. The method consisted of pulverisation of the sample, then ammonium iodide sublimation followed by dissolution in hydrochloric acid. The spectrophotometer used was a Techtron AA-5.

The two independent laboratories were Australian Mineral Development Laboratories (Amdel) and Comlabs, both of Adelaide, S.A. XRF analyses were performed by the pressed powder method, whereby the sample is pulverised and pressed into a pellet in a carrying medium, for presentation to the X-ray beam.

FIGURE 1: Sample Crushing Flowsheet



3.0 TREATMENT AND PROCEDURES

3.1 PROCEDURES

Figure 2 (following) shows the sample treatment procedure in flowsheet form. At the stage of producing 100 kg splits from the preliminary crushing, samples BSS 2, BSS 3 and BSS 6 were treated in duplicate; i.e. two separate 100 kg splits were taken of each for further treatment.

The final subdivision of the ground sample was carried out to eight ways in a riffle, rather than ten ways as had been done earlier in the sample treatment. The selection of three sub-samples for analysis was done at random. Each sub-sample was analysed in triplicate by the methods mentioned in Section 2.3.

3.2 RESULTS

Table 2 (following) gives the complete set of assay results.

3.3 DISCUSSION

The AA technique used by MDL has an accuracy of $\pm$10% relative. The XRF methods employed by Amdel and Comlabs have an accuracy of $\pm$5% relative. This suggests that the results are reasonably consistent between laboratories, and the results obtained by MDL give a conservative estimate of the grades of tin present in the samples.

153 FIGURE 2: Sample Treatment and Preparation Flowsheet

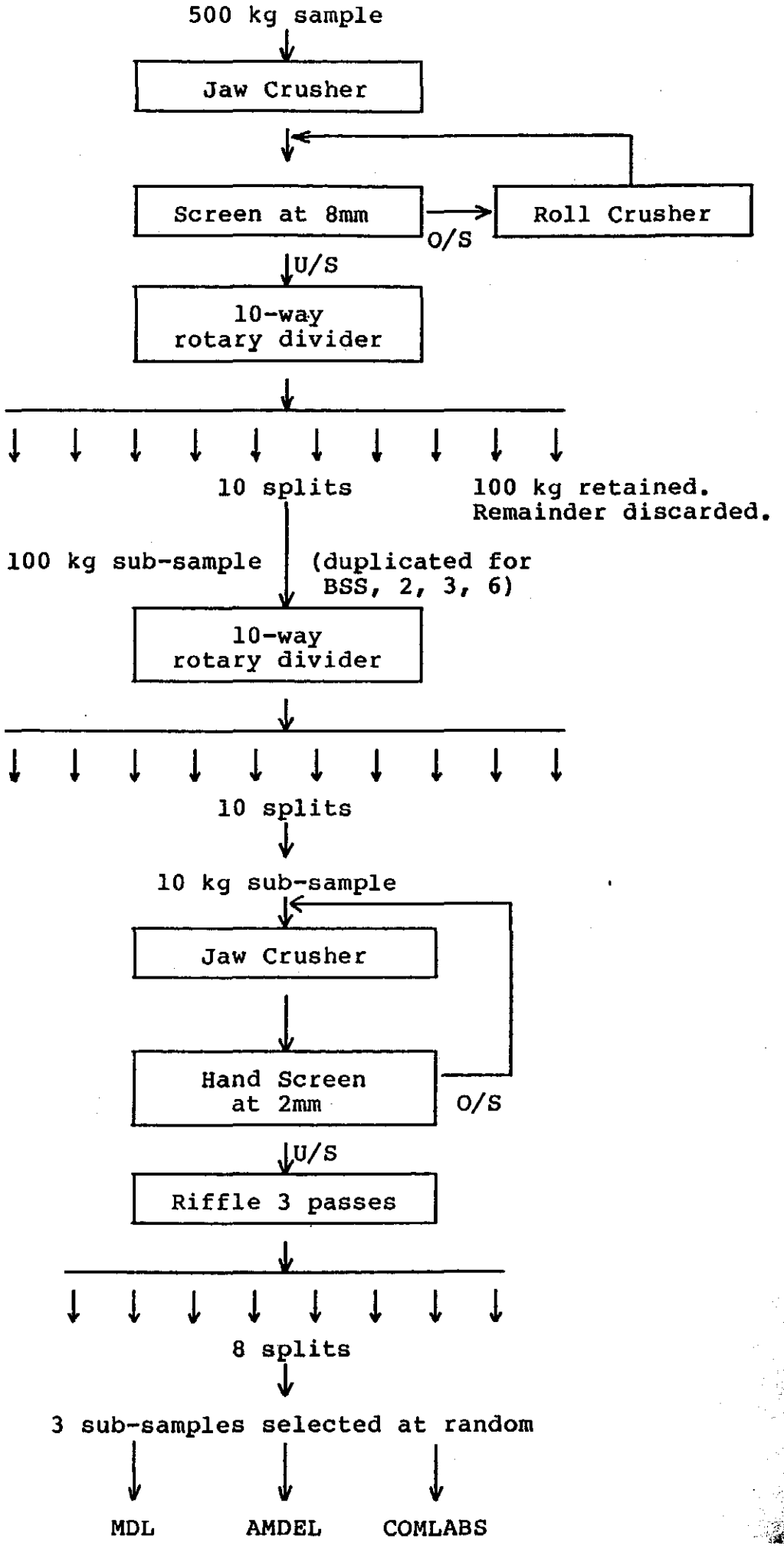


TABLE 2: Combined Results from All Laboratories

ASSAY %SN	MDL - AA				AMDEL - XRF				COMLABS - XRF			
SAMPLE	1	2	3	Avrge	1	2	3	Avrge	1	2	3	Avrge
BSS 2	0.107	0.092	0.096	0.098	0.114	0.108	0.112	0.111	0.100	0.105	0.110	0.105
BSS 2	0.091	0.084	0.083	0.086	0.114	0.112	0.104	0.110	0.105	0.105	0.105	0.105
BSS 3	0.025	0.015	0.015	0.018	0.018	0.021	0.018	0.019	0.018	0.019	0.019	0.019
BSS 3	0.025	0.015	0.015	0.018	0.018	0.021	0.018	0.019	0.018	0.017	0.017	0.017
BSS 4	0.212	0.182	0.196	0.197	0.198	0.220	0.204	0.207	0.185	0.205	0.200	0.197
BSS 5	0.258	0.228	0.236	0.241	0.298	0.288	0.268	0.285	0.260	0.260	0.270	0.263
BSS 6	0.231	0.214	0.225	0.223	0.254	0.248	0.256	0.253	0.240	0.240	0.250	0.243
BSS 6	0.235	0.218	0.219	0.224	0.260	0.244	0.236	0.247	0.245	0.240	0.230	0.238
BSS 8	0.290	0.251	0.262	0.267	0.346	0.382	0.346	0.358	0.315	0.315	0.300	0.310
BSS 9	0.257	0.207	0.224	0.229	0.242	0.262	0.258	0.254	0.240	0.250	0.240	0.243
BSS 10	0.178	0.151	0.157	0.162	0.172	0.180	0.186	0.179	0.155	0.160	0.160	0.158
BSS 11	0.315	0.298	0.306	0.306	0.388	0.360	0.354	0.367	0.360	0.360	0.410	0.377
BSS 12	0.279	0.250	0.250	0.260	0.280	0.280	0.285	0.282	0.240	0.225	0.220	0.228
BSS 13	0.243	0.224	0.227	0.231	0.266	0.268	0.288	0.274	0.260	0.275	0.260	0.265
BSS 14	0.016	0.016	0.015	0.016	0.011	0.010	0.010	0.010	0.009	0.009	0.009	0.009

047149

156

047150

Aberfoyle Limited

INCORPORATED IN VICTORIA
25th Floor, 367 Collins Street, Melbourne Victoria 3000 Australia
Telephone: (03) 61 3661 Telex: AA31020

12 November 1985

Mr J Collins
Shell Company of Australia
Metals Division
P O Box 872K
MELBOURNE VIC 3000

Dear John,

Gt Pyramid Tin Deposit

I have read the reports you lent me and attach a copy of my notes. Additionally, I searched and found some work that we did in 1979 on a small sample. Because of other priorities my time on this job has been inadequate but I attach my thoughts to date.

Metallurgy

The testwork completed did so much but said so little. Neither the Department of Mines nor the Mineral Deposits work is very conclusive. For the weights of sample treated and the obvious expense, Shell has got poor value for money.

Nevertheless, there is confirmation that ore treatment should be regarded as a simple gravity separation exercise.

The Aberfoyle HLS work, in part confirmed by the TDM indicates a liberation from -600um and excellent potential for heavy media cyclone separation. Conventional crushing, grinding and gravity + tin flotation would also work well but would be more expensive. The technical key will be to make the heavy media separation work as well as the laboratory tests indicate. Heavy media cyclones (Renison, Cleveland) work generally well down to about 500um but I feel if "pushed" they could work on this ore down to 250um. Some South African work claims 100-75um separation (P Van Der Walt, Norten Tivedale).

The use of pre-concentration and direct fuming would also be possible if a high grade sink (say >10% Sn) is achieved at high (>85%) recovery but unless normal gravity processes do not work or exceptional circumstances exist, it is probably cheaper on this ore to use tables.

Flowsheets

I have drawn up some rough conceptual flowsheets for consideration but proper development of these will need more thought. At 0.22% Sn any flowsheet will have to be very simple to be economic. I favour the HMS/Cones/Table circuit, Number 1.

Economics

Your file note of 17/10/85 is not too far out but I would drop recovery from 90% to 80-85%, say 83% overall to be safe.

Milling costs of a simple HMS-gravity mill at 500,000 t/y should be much less than $12/tonne (without flotation) and may end up about $8/tonne.

		$/t
i.e.	Crushing	1.00
	Grinding	1.50
	HMS	1.50
	Gravity	1.00
	Other	2.00
	Mill Admin	1.50
		8.50

Ardlethan for instance has milling costs at about $10/t for 250,000 t/y rate which includes a large tin flotation section. Its head grade is currently about 0.25% Sn and we are making a small amount of profit.

The recent trauma in the tin market has left everything uncertain but at prices before the problems, a 0.22% Sn ore was valued at say:

$33/t @ A$15,000/t tin price.

At say A$10,000/t tin price, the value drops to $22/t.

— Tin Prices are net of smelting + Transport

Using this price and 83% recovery value you have $18/t revenue to cover your costs and capital.

Assuming your mining cost of $3/t and other overheads, etc of $3/t, overall costs excluding depreciation would be $15/t - a lean show!

Conclusions

- The tin price that emerges from ITC/LME movements in the next few weeks will very quickly tell you whether a project at Gt Pyramid has a hope. At say $10,000/t tin price the project would unlikely to be viable.

- The metallurgical work done to date has not properly pursued the major "hope" for this deposit, that is, innovative use of HMS. Indications from Aberfoyle and Tasmanian Department of Mines work are that a well run, simple HMS-Gravity plant could achieve 80-85% tin recovery at low milling cost, about $8-9/t. More testwork and economic studies may improve this but the tin price will be the major issue.

- The heavy liquid response of the Gt Pyramid ore is somewhat unique in my experience and confirmatory tests (Aberfoyle Metallurgy at Burnie could do this if you wish) on sized fractions from 3mm down to 45um should be done on representative sample. If the response is as good as previously indicated and if the tin price can be held above say A$10,000/t you have a tantalising prospect that I feel should not be dropped.

- Economics of this prospect could be considerably improved by scale (cf Newmonts Grampians Prospect) and if more resources were found to enable doubling or trebling of current size, greater confidence in the viability would result.

Overall, my "gut feeling" says that the prospect is marginal but not hopeless and if full advantage can be made of the ores unique HMS character, a viable project could result.

Yours sincerely,

Kevin A. Foo

Kevin A Foo
Manager - Metallurgical Development

Attachment

NOTES - GT PYRAMID TESTWORK

Tasmanian Department of Mines (1983)

- Good standard work but their usual difficult to read report.
- Sample seemed good size and representative of grade at 0.22% Sn.
- Sizing and natural grainsize indicate coarse cassiterite (compared with other Tasmanian ores) and ∿90% SnO_2 >30um, 66% +75um. Should be quite amenable to gravity separation.
- Heavy liquid test to +600um showed liberation appreciable at that size. Tail at 0.08% Sn and 98% weight was excellent result. Should have continued work down to 45um.
- Jig tests at lab scale unreliable.
- Larger scale tests - looks like sample was overground and lab separations inefficient.
- 54% recovery was a very poor result - perhaps the "clustering" was a real problem but I doubt it.
- Lack of flowsheet with results is annoying.
- Grind of 400-600um appears suitable if pre-concentration not used but pre-concentration with HM cyclone a distinct possibility.

Mineral Deposits (1984)

- 400um grind size about right.
- Spiral and table testwork indicated good gravity response but lack of good overall flowsheet/balance was annoying (again!)
- Weak conclusions - not quantified.
- Report - hard to follow in parts.
- Balances on flowsheets do not add up, not indicated which are calculated and which are measured weights and assays. AAS Sn assays unreliable.
- Spiral recovery 77-90% Sn to 6.25% Sn grade. From 0.51% feed, a good indication. Tail of 0.11% Sn a bit high for coarse fraction.
- Medium fraction 79% recovery to 3.8% Sn and 0.075% Sn tail good enough.
- Upgrading tests indicated no major problems.

160

Aberfoyle (1979) Technical Note (attached)

- Two samples only 750g and 850g provided.
- Heavy liquid tests and superpanned sinks.
- Veinlets 600-1000um SnO_2.
- Native copper?
- Projected 63≈80% recovery from feeds 0.2-0.6% Sn.
- Small scale tests.
- Table 1 2.96 SG (-600 +45um)

230501	Sink	1.9% Wt	28.6% Sn	86% Distribution
	Float	98.1% Wt	0.09% Sn	14% Distribution
	Feed		0.6% Sn	
230502	Float	97.7% Wt	0.06% Sn	31.3% Distribution
	Sink	2.3% Wt	5.6% Sn	68.7% Distribution
			0.19% Sn	

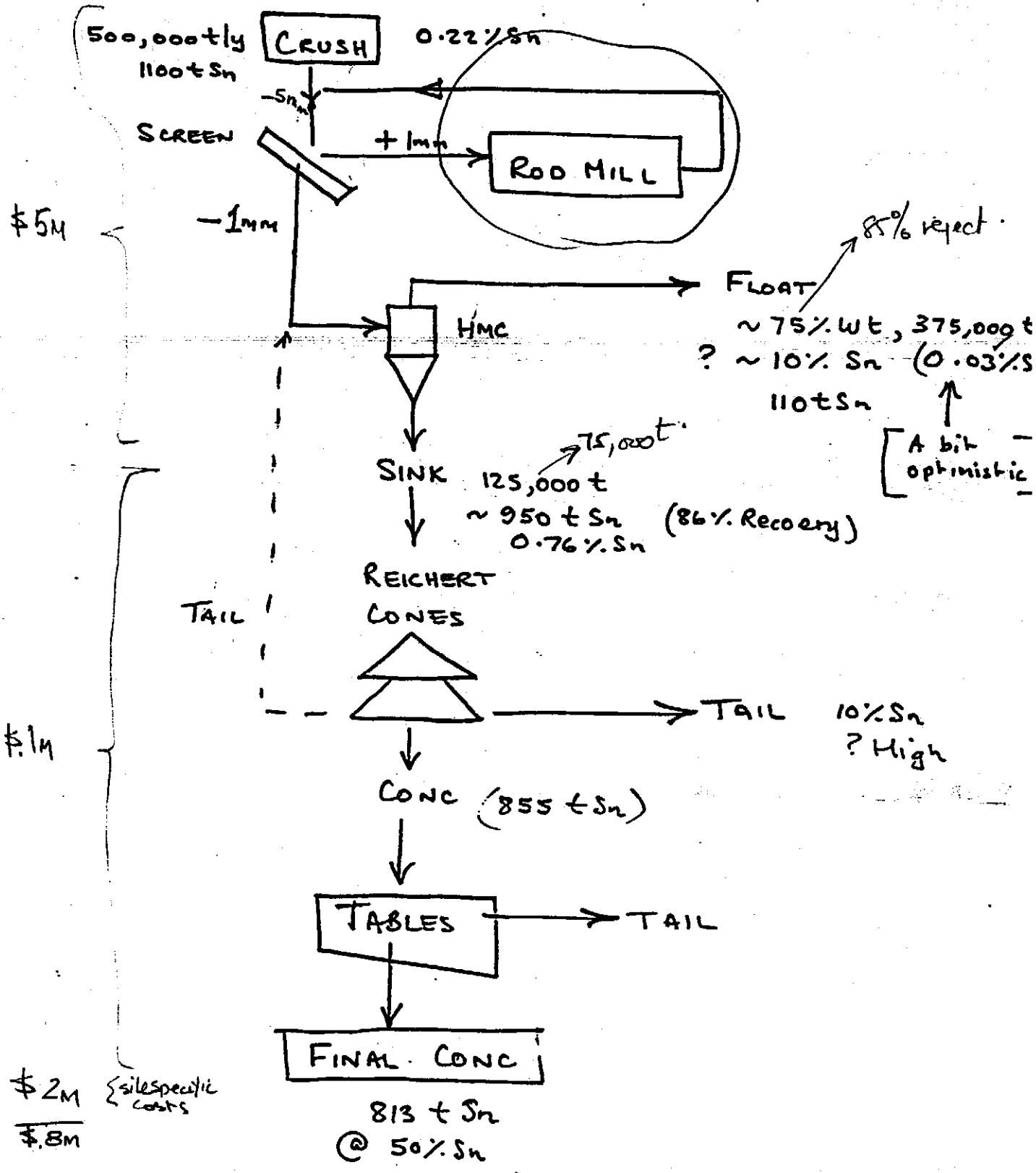
* HMS looks promising ± gravity precon.

Kevin. A. Foo

Kevin A Foo
12 November 1985

161

GT. PYRAMID – FLOWSHEET 1



74% O/all – should be more?

[Should get > 80% in well run plant]

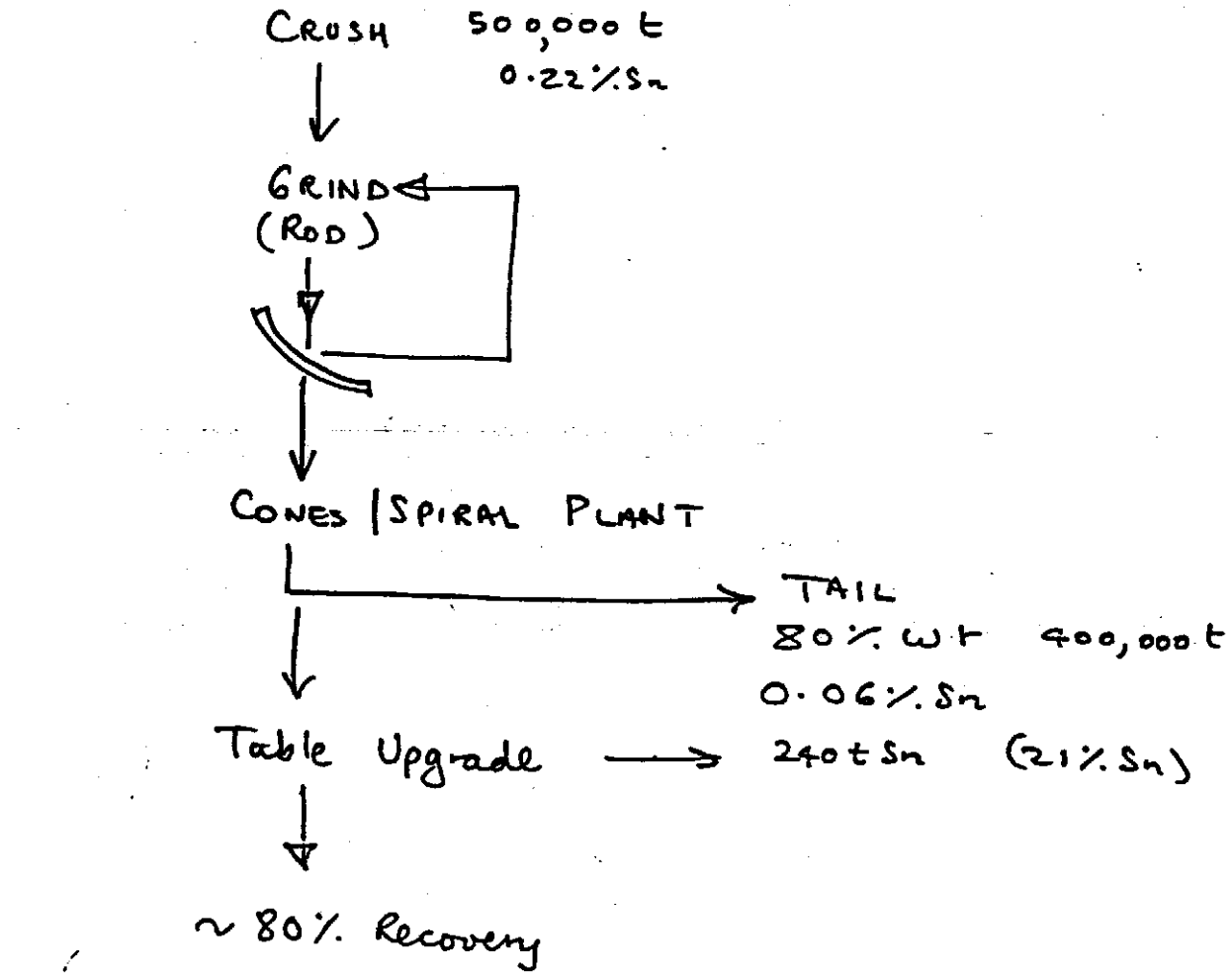
Op. Cost ~$8/t

Mine $4M

Infra $ 3M

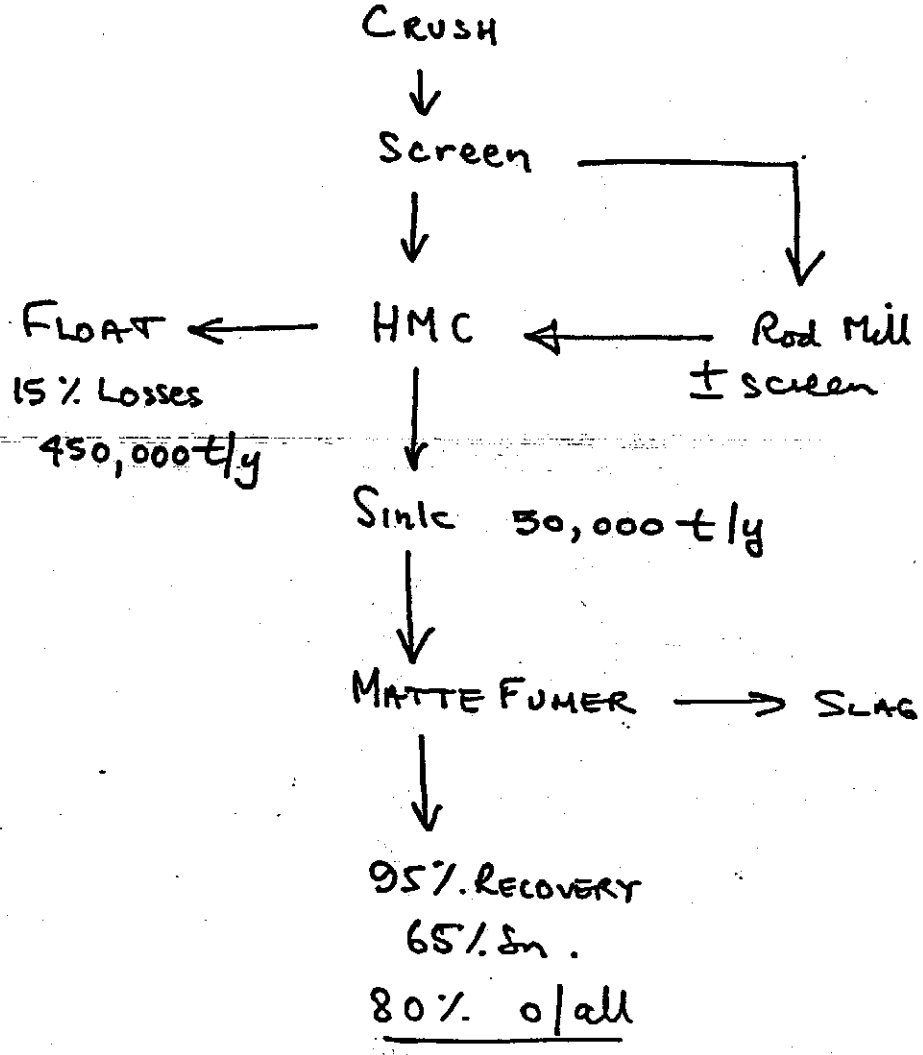
KAF Nov 85

GT. PYRAMID – FLOWSHEET 2



Operating Cost ~ $10/t

Ct. Pyramid – Flowsheet 3



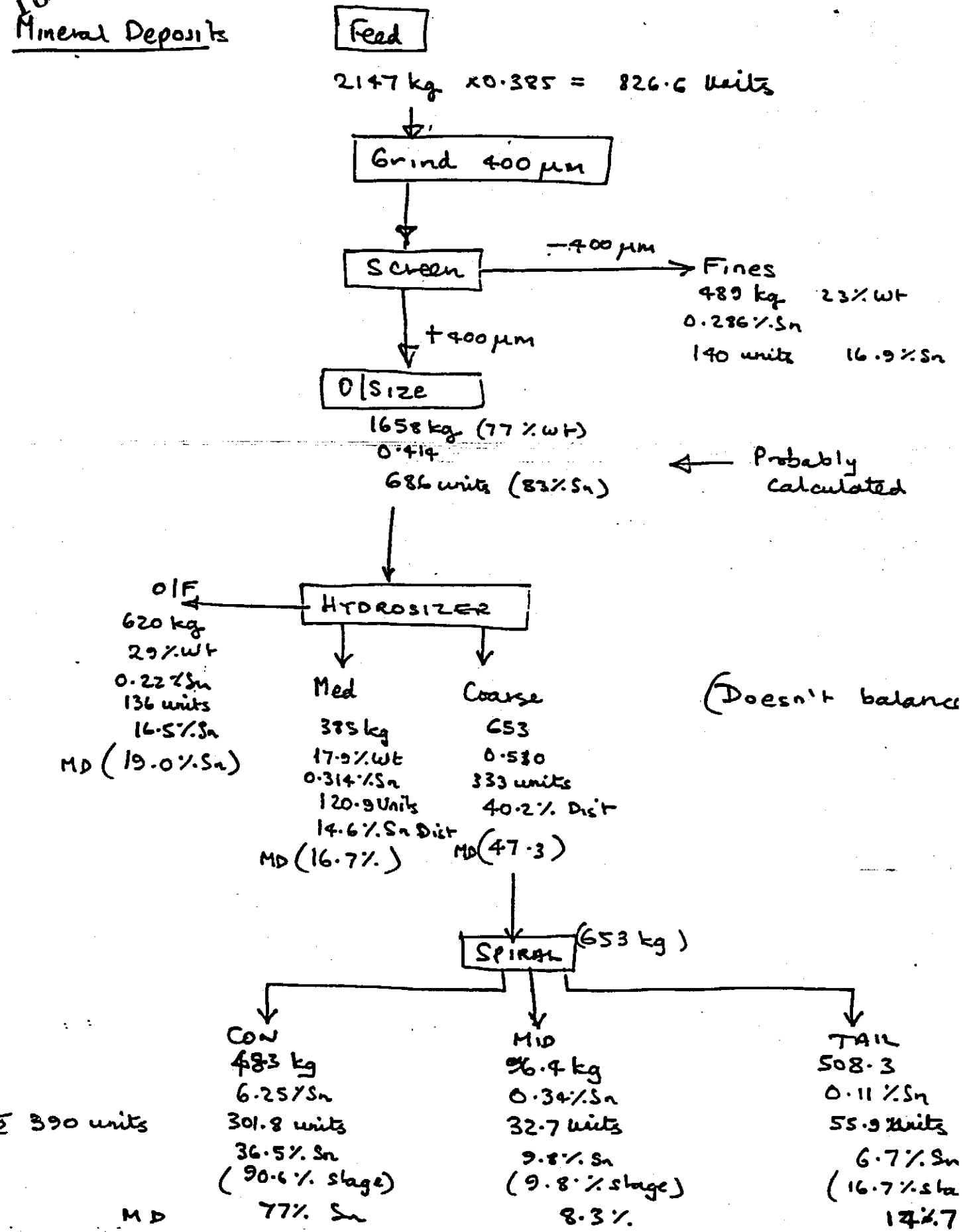
Assume: Excellent HMS separation
Smallish Fumer

Operating Cost ~ $8-10/t

KAF Nov '85

047158

Mineral Deposits



KAF Nov(

165

CLEVELAND TIN LIMITED

NEM/KC

Luina, Tasmania 7321, Austra
Telephone: 39 1180 Telex: 585

TECHNICAL NOTE

TO: R.J. ROBINSON

FROM: N.E. MOONY

DATE: 4/10/79

COPIES: C. Young, K. Palmer, S.M. Richards

SUBJECT: CHARACTERISATION OF "GREAT PYRAMID" CORE SAMPLE

SUMMARY

Due to the samples being small a full metallurgical investigation was not possible, but a brief ore characterisation showed the following:-

1) 90% of the cassiterite was finer than 300 micron.

2) There was less than 5% -5 micron cassiterite.

3) The host rock lends itself readily to pre concentration methods such as cones or trays.

4) The characteristics of this type of ore allow for ease of liberation due to the high concentration of cassiterite in veinlets.

5) Good recoveries were achieved with easily retreatable middlings.

6) Due to the small amount of concentrate, 230502 was difficul to upgrade to more than ½0.0% Sn. The relatively high tailing in the +45 micron fraction could be reduced by regrinding.

Samples have been submitted for mineralogical examination.

1. INTRODUCTION

Two small samples of drill core were received on 18/9/79 from Mr. C. Young at Wynyard. The samples were numbered:-

230501 total weight 750 gm

230502 " " 850 gm

A brief metallurgical analysis was requested with emphasis on pre concentration.

2. TEST PROCEDURE

The core was scanned mineralogically to judge a possible top liberation point, 600 micron was decided upon to allow maximum recovery.

The samples were reduced to 100% passing 600 micron, by passing the samples through a roll crusher until all the sample was reduced. The crushed product was then screened at 45 micron.

The +45 micron fraction was subjected to heavy liquid tests at 2.96 S.G and 3.3 S.G using Tetrabromo Ethane ($CHBr_2.CHBr$ and Di Iodomethane (CH_2I_2) respectively. The di iodomethan sink was superpanned to concentrate the free cassiterite and superpan tail plus tetrabromo ethane sink were combined to give a +45 micron middling product.

The -45 micron fraction was cyclosized and the +15 micron -45 micron fraction was superpanned. The +5 micron -15 micron fraction was reported as fine middlings. This fraction would be readily treated by oxide flotation. (Refer flowsheet attached)

3. DISCUSSION OF RESULTS

3.1 The Heavy Liquid

From a visual and cursory mineralogical examination the samples appeared very oxidised and weathered. Mineralisati occured as thin veinlets in a sandstone host rock. The veinlets were 600 micron to 1000 micron in width, and in some instances contained native copper with sulphides.

As a result of the above it was thought that a liberation point of 600 micron would give an upper limit for free cassiterite to allow maximum recovery. Both samples were reduced to 100% - 600 micron.

167

230501

Summary of Results

	Wt	Sn%	Sn Dist	Project Mill Re at 50%
Combined Concentrates	5.30	(38.13)	63.7	61.0
Coarse Middlings	4.12	16.18	21.0	16.8
-15 micron +5 micron Middlings	24.00	0.13	1.0	0.5
Combined Tailings	450.00	(0.08)	12.2	0.0
Slimes	50.56	0.13	2.1	0.0
Combined Feed	534.00	0.60		78.3

230502

Summary of Results

Combined Concentrates	2.12	(7.9)	13.7	11.6
Combined Middlings	11.90	5.2	50.7	40.6
-15 micron +5 micron Middling	24.00	0.13	2.7	1.4
Combined Tailings	506.00	0.06	24.9	9.4
Slimes	75.00	0.13	8.0	
Combined Feed	620.00	0.20	100.0	63.0

The results in table 1 show excellent separation and indicate that the sample would readily lend itself to pre concentration by cones on spirals. It would be likely that heavy media separation would work on this type of ore but the samples were too small to allow any judgement to be made.

The di iodomethane results have been incorporated in the superpanning and cannot be isolated.

3.2 Gravity Separation

The gravity separation was hindered by the other heavy minerals such as abundant native copper and sulphide, present in both samples. Sulphide removal was not possibl due to the small volume of sample.

230501

The gravity results show that 60% of the cassiterite is liberated above 45 micron with a 600 micron screening. From mineralogical examination, 22% of the cassiterite reporting as middlings at 16.2% Sn would realise at least 40% of the components at +45 micron, and an80% average recovery could be expected from these middlings. The tailings grade of 0.09% could be reduced with a finer grin

The -45 micron concentrate grade was lower than expected, but the recovery was very good at about 80%. The reason for the low concentrate is not clear, but sulphides are present.

The coarse middlings seem to contain abundant native copper.

230502

A final concentrate was not attempted due to the small amo of sample. The most significant point was the large amoun of middlings present which indicates that this material may not liberate as well as 230501 using 600 micron as the liberation point. Sulphides also caused difficulty in concentrating the coarser material and insufficient sample prevented the taking of a -45 micron concentrate.

4. CONCLUSIONS

The samples are too small to draw any definite conclusions but if they are in any way representative, then it could be concluded that the material warrants further investigat

The small amount of middlings present (i.e high grade heav liquid sinks) would enhance any gravity treatment process.

169

230501

+45 micron Gravity Treatment Results

	Wt(gm)	Wt%	Sn%	Sn Di
Concentrate One	1.18	0.22	52.4	19.3
Concentrate Two	2.58	0.48	36.7	29.3
Coarse Middlings	4.12	0.78	16.19	22.0
Coarse Tailings	409.00	76.59	0.09	11.5
-45 micron				
Concentrate One	0.72	0.13	29.00	6.3
Concentrate Two	0.86	0.16	30.40	8.1
-15 micron +5 micron Middlings	44.0	4.49	0.13	1.0
Fine Tailings	41.0	7.68	0.05	0.6
Slimes	50.54	9.47	0.13	2.0
Head Calc	534.00	100.00	0.60	100.0
Assay			0.63	
Summary				
Combined Concentrate	5.30	0.99	38.13	63.1
Combined Middlings	28.12	5.27	2.59	22.8
Combined Tailings	500.58	93.74	0.09	14.1
	534.00	100.00	0.60	100.0

230502

Concentrate One	0.82	0.13	11.4	7.
Concentrate Two	1.30	0.21	5.7	6.(
Coarse Middlings	9.51	1.92	5.2	50.(
-15 micron +5 micron Middlings	25.00	4.03	0.13	20.
Coarse Tailings	472.00	76.13	0.06	22.
Fine Tailings	34.00	5.48	0.11	3.
Slimes	77.37	12.10	0.13	7.
Summary				
Combined Concentrate	2.12	0.34	7.90	13.
Combined Middlings	36.90	5.95	1.76	53.
Combined Tailings	581.00	93.71	0.07	33.
	620.02	100.00	0.20	100.

170

230501

Heavy Liquid Separation T B E Float

-600 micron +45 micron fraction

S.G	Wt%	Sn	Sn Dist
2.96 Sink	1.89	28.60	86.0
Float	98.11	0.09	14.0
Head Calc	100.00	0.63	
Head Assay		0.63	X45

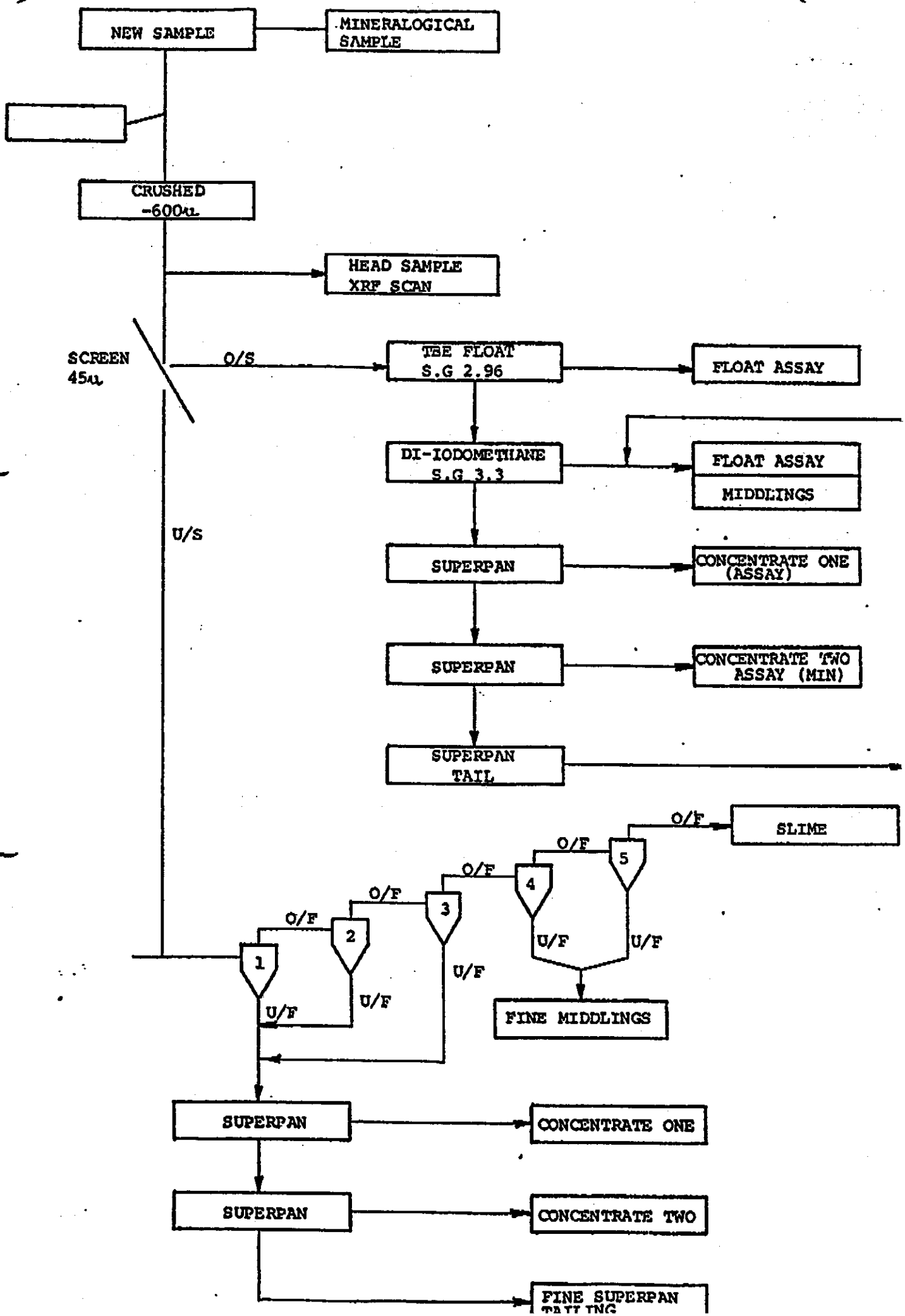
230502

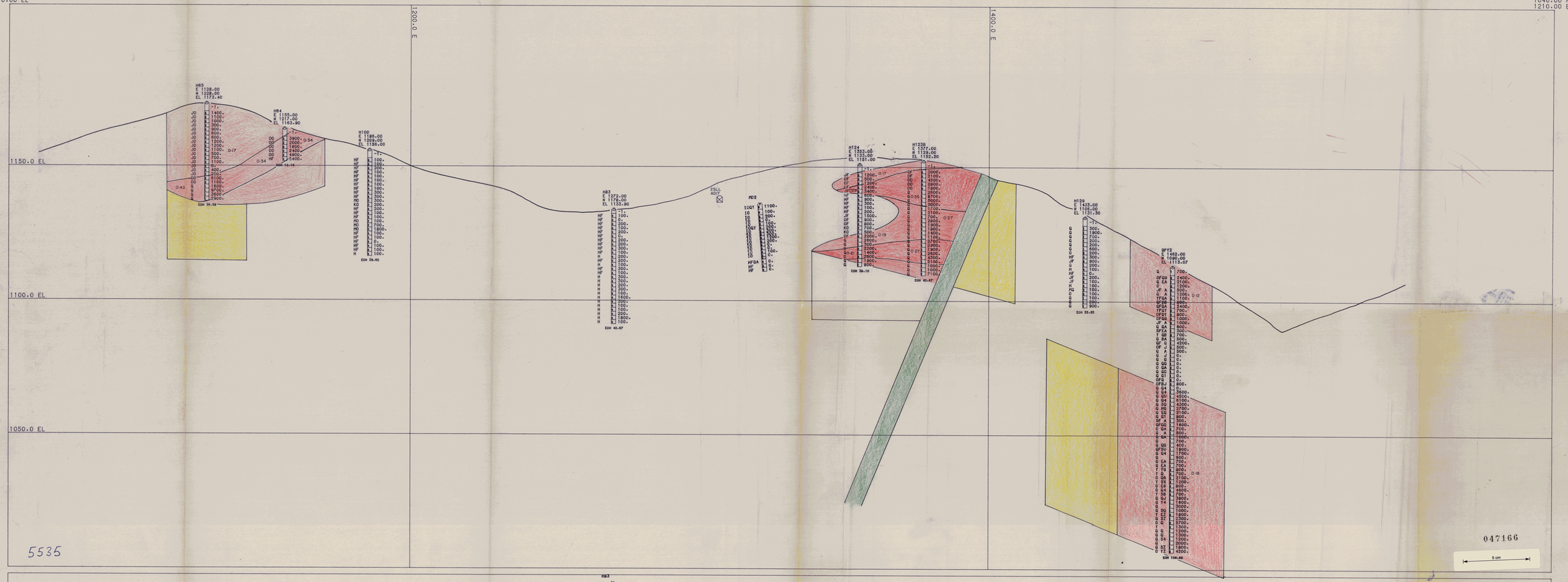
Heavy Liquid Separation T B E Float

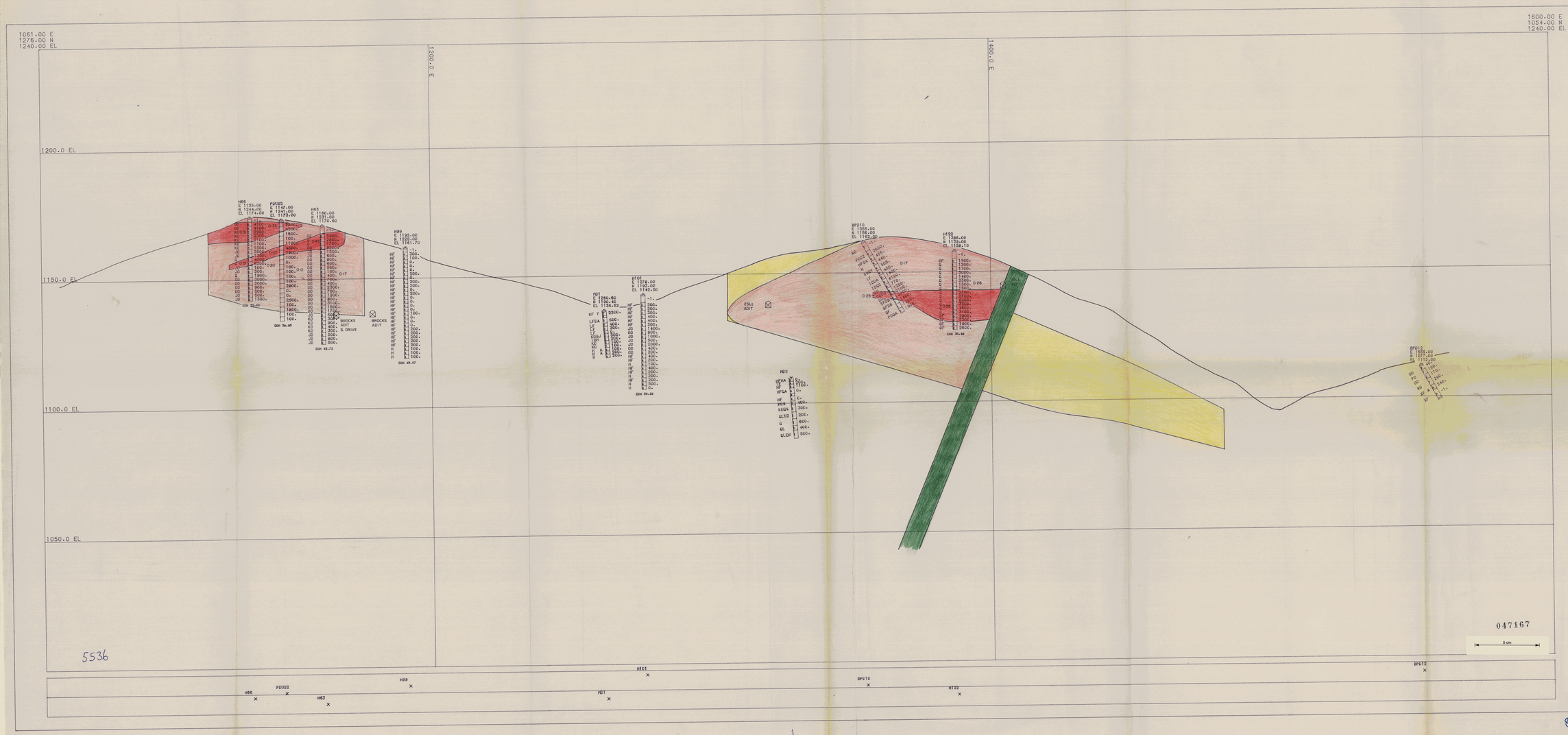
-600 micron +45 micron fraction

2.96 Sink	2.30	5.6	68.7
Float	97.70	0.06	31.3
Head Calc	100.00	0.19	100.0

if can be shown to be 5-10% into Sn + 80% recovery } - forcibly a proper


NEW SAMPLE
MINERALOGICAL SAMPLE
CRUSHED -600u
HEAD SAMPLE XRF SCAN
SCREEN 45u
O/S
U/S
TBE FLOAT S.G 2.96
FLOAT ASSAY
DI-IODOMETHANE S.G 3.3
FLOAT ASSAY
MIDDLINGS
SUPERPAN
CONCENTRATE ONE (ASSAY)
SUPERPAN
CONCENTRATE TWO ASSAY (MIN)
SUPERPAN TAIL
O/F
SLIME
5
4
3
2
1
U/F
FINE MIDDLINGS
SUPERPAN
CONCENTRATE ONE
SUPERPAN
CONCENTRATE TWO
FINE SUPERPAN


1061.00 E
1260.00 N
1210.00 EL
1595.00 E
1040.00 N
1210.00 EL
1200.0 E
1400.0 E
1150.0 EL
1100.0 EL
1050.0 EL
HB5
E 1128.00
N 1228.00
EL 1173.40
HB4
E 1155.00
N 1217.00
EL 1163.90
H100
E 1186.00
N 1258.00
EL 1196.00
HB3
E 1272.00
N 1178.00
EL 1133.90
2SLL
ADIT
MD2
H124
E 1353.00
N 1133.00
EL 1191.00
H123B
E 1377.00
N 1129.00
EL 1152.20
H129
E 1423.00
N 1105.00
EL 1131.30
GPT2
E 1462.00
N 1090.00
EL 1113.07
5535
047166
5 cm
SHLMET ORE EVALUATION SYSTEM
DRILLHOLE SECTION PLOT
NUMERIC LEGEND
/ SN /
LITHOLOGICAL DESCRIPTION CODES PLOTTED
LEGEND
PROBABLE RESOURCE ≥ 0·1% Sn
PROBABLE RESOURCE ≥ 0·2% Sn
POTENTIAL RESOURCE
VERTICAL SCALE = 1 : 500
HORIZONTAL SCALE = 1 : 500
SHELL COMPANY OF AUSTRALIA
METALS DIVISION
GREAT PYRAMID
SECTION 2 W-E
PRELIMINARY GEOLOGICAL
RESOURCE OUTLINES
FIG.NO:
REPT.NO:
ENCL.NO:
DRG.NO: D/LHDT/580
DATE:
AUTHOR:
DRAWN:
OFFICE:
5535
84-2532



SHLMET ORE EVALUATION SYSTEM
DRILLHOLE SECTION PLOT

NUMERIC LEGEND

/ SN /

LITHOLOGICAL DESCRIPTION CODES PLOTTED

LEGEND

PROBABLE RESOURCE ≥ 0·1% Sn

PROBABLE RESOURCE ≥ 0·2% Sn

POTENTIAL RESOURCE

VERTICAL SCALE = 1 : 500
HORIZONTAL SCALE = 1 : 500

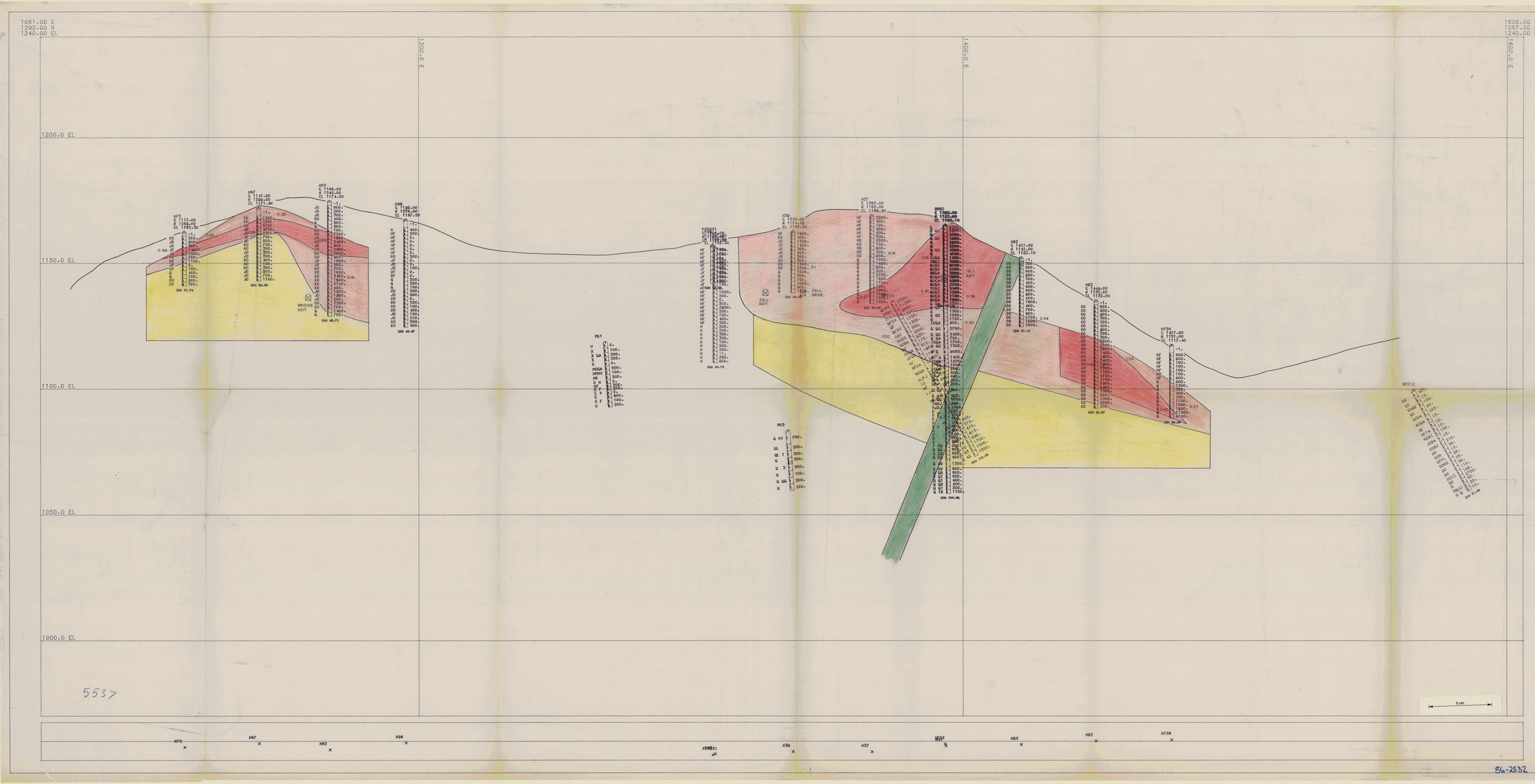
SHELL COMPANY OF AUSTRALIA
METALS DIVISION

GREAT PYRAMID
SECTION 3 W-E
PRELIMINARY GEOLOGICAL
RESOURCE OUTLINES

FIG.NO:	REPT.NO:
ENCL.NO:	DRG.NO: D/LH01/581
DATE:	AUTHOR:
DRAWN:	OFFICE:

5536

86-2532



047168

SHLMET ORE EVALUATION SYSTEM
DRILLHOLE SECTION PLOT

NUMERIC LEGEND

/ SN /

LITHOLOGICAL DESCRIPTION CODES PLOTTED

LEGEND

- PROBABLE RESOURCE ≥ 0·1% Sn
- PROBABLE RESOURCE ≥ 0·2% Sn
- POTENTIAL RESOURCE

VERTICAL SCALE = 1 : 500
HORIZONTAL SCALE = 1 : 500

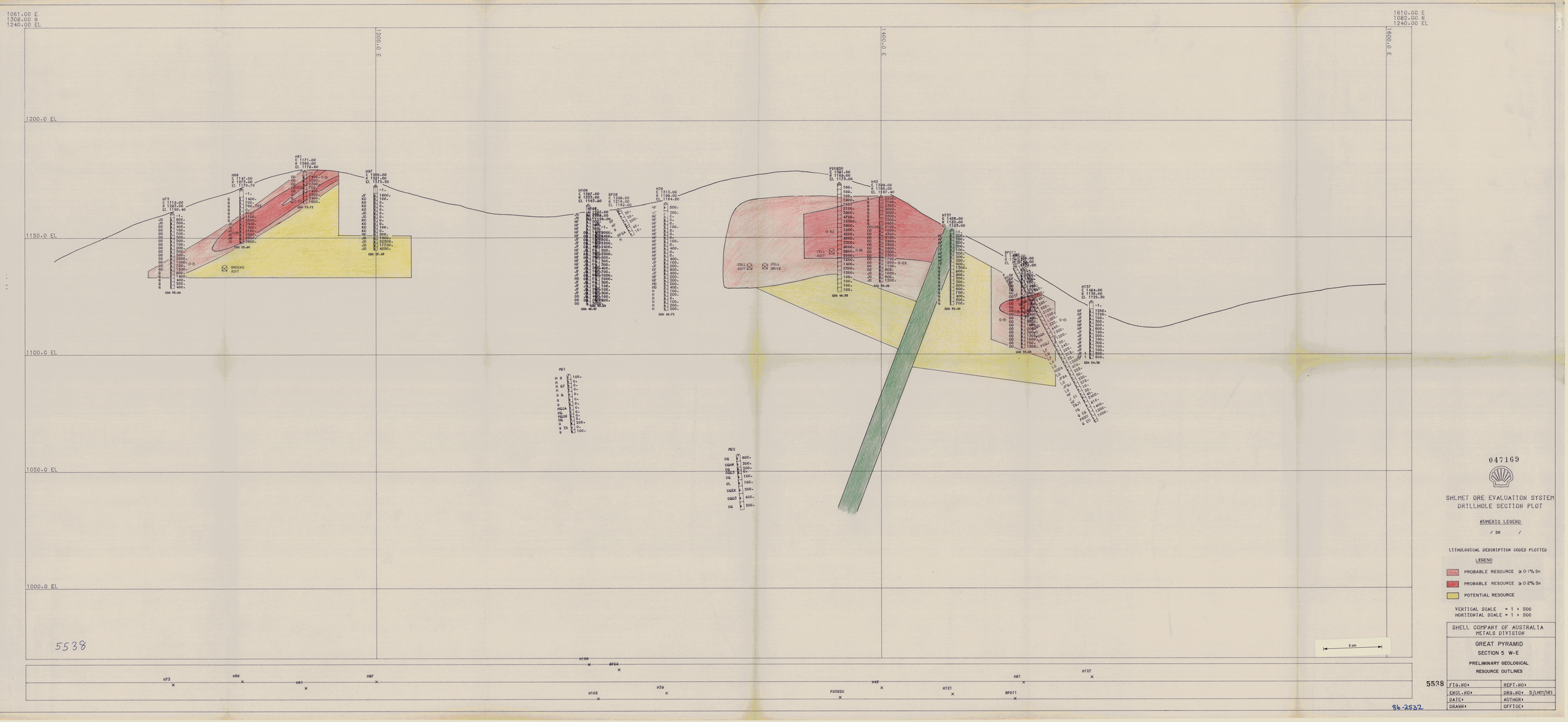
SHELL COMPANY OF AUSTRALIA
METALS DIVISION

GREAT PYRAMID
SECTION 4 W-E
PRELIMINARY GEOLOGICAL
RESOURCE OUTLINES

FIG.NO:	REPT.NO:
ENCL.NO:	DRG.NO: D/LH01/582
DATE:	AUTHOR:
DRAWN:	OFFICE:

5537

86-2532



5538

047169

SHELL ORE EVALUATION SYSTEM
DRILLHOLE SECTION PLOT

NUMERIC LEGEND

/ SW /

LITHOLOGICAL DESCRIPTION CODES PLOTTED

LEGEND

- PROBABLE RESOURCE ≥ 0·1% Sn
- PROBABLE RESOURCE ≥ 0·2% Sn
- POTENTIAL RESOURCE

VERTICAL SCALE = 1 : 500
HORIZONTAL SCALE = 1 : 500

SHELL COMPANY OF AUSTRALIA
METALS DIVISION

GREAT PYRAMID
SECTION 5 W-E
PRELIMINARY GEOLOGICAL
RESOURCE OUTLINES

5538 FIG.NO:	REPT.NO:
ENGL.NO:	DRG.NO: D/LHO1/583
DATE:	AUTHOR:
DRAWN:	OFFICE:

86-2522

1061.00 E
1325.00 N
1240.00 EL

1615.00 E
1097.00 N
1240.00 EL

1200.0 E

1400.0 E

1600.0 E

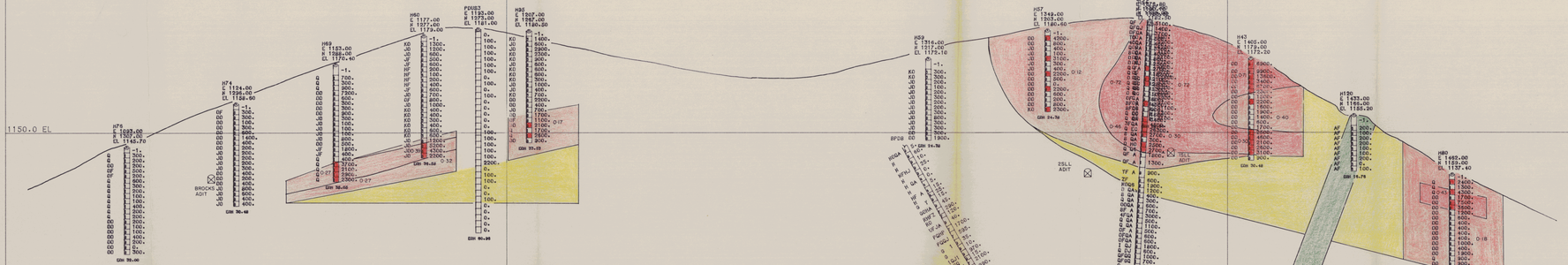
1200.0 EL

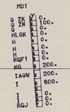
1150.0 EL

1100.0 EL

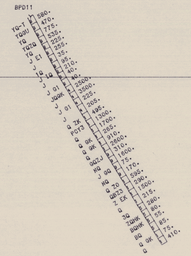
1050.0 EL

1000.0 EL



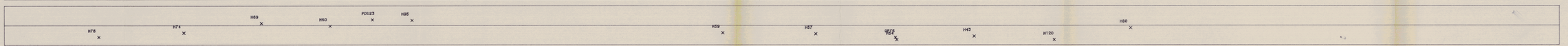






5539

5 cm



96-2532

047170

SHELL

SHLMET ORE EVALUATION SYSTEM
DRILLHOLE SECTION PLOT

NUMERIC LEGEND

/ SN /

LITHOLOGICAL DESCRIPTION CODES PLOTTED

LEGEND

- PROBABLE RESOURCE ≥ 0·1% Sn
- PROBABLE RESOURCE ≥ 0·2% Sn
- POTENTIAL RESOURCE

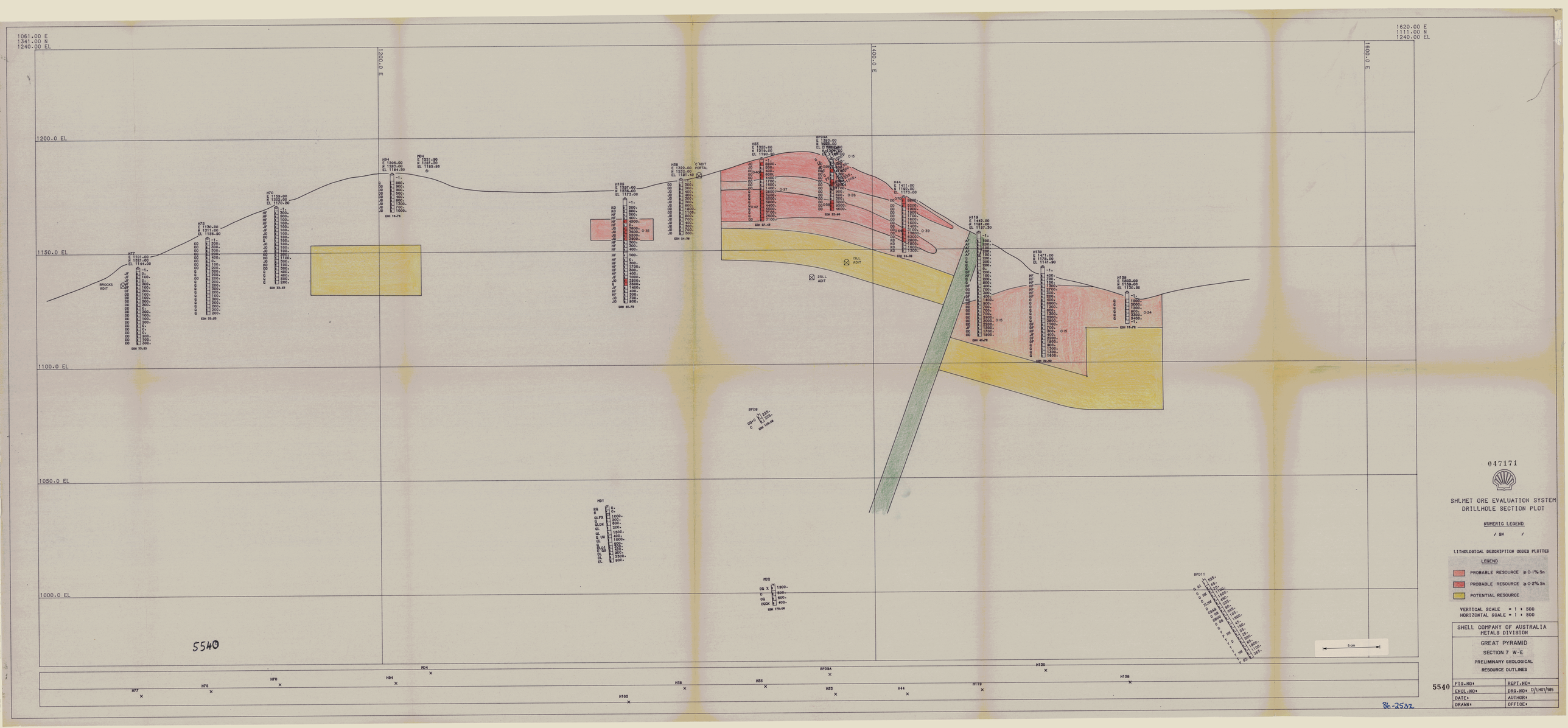
VERTICAL SCALE = 1 : 500
HORIZONTAL SCALE = 1 : 500

SHELL COMPANY OF AUSTRALIA
METALS DIVISION

GREAT PYRAMID
SECTION 6 W-E
PRELIMINARY GEOLOGICAL
RESOURCE OUTLINES

5539

FIG.NO:	REPT.NO:
ENGL.NO:	DRG.NO: D/LH01/584
DATE:	AUTHOR:
DRAWN:	OFFICE:



047171

SHELL

SHELMET ORE EVALUATION SYSTEM
DRILLHOLE SECTION PLOT

NUMERIC LEGEND

/ SH /

LITHOLOGICAL DESCRIPTION CODES PLOTTED

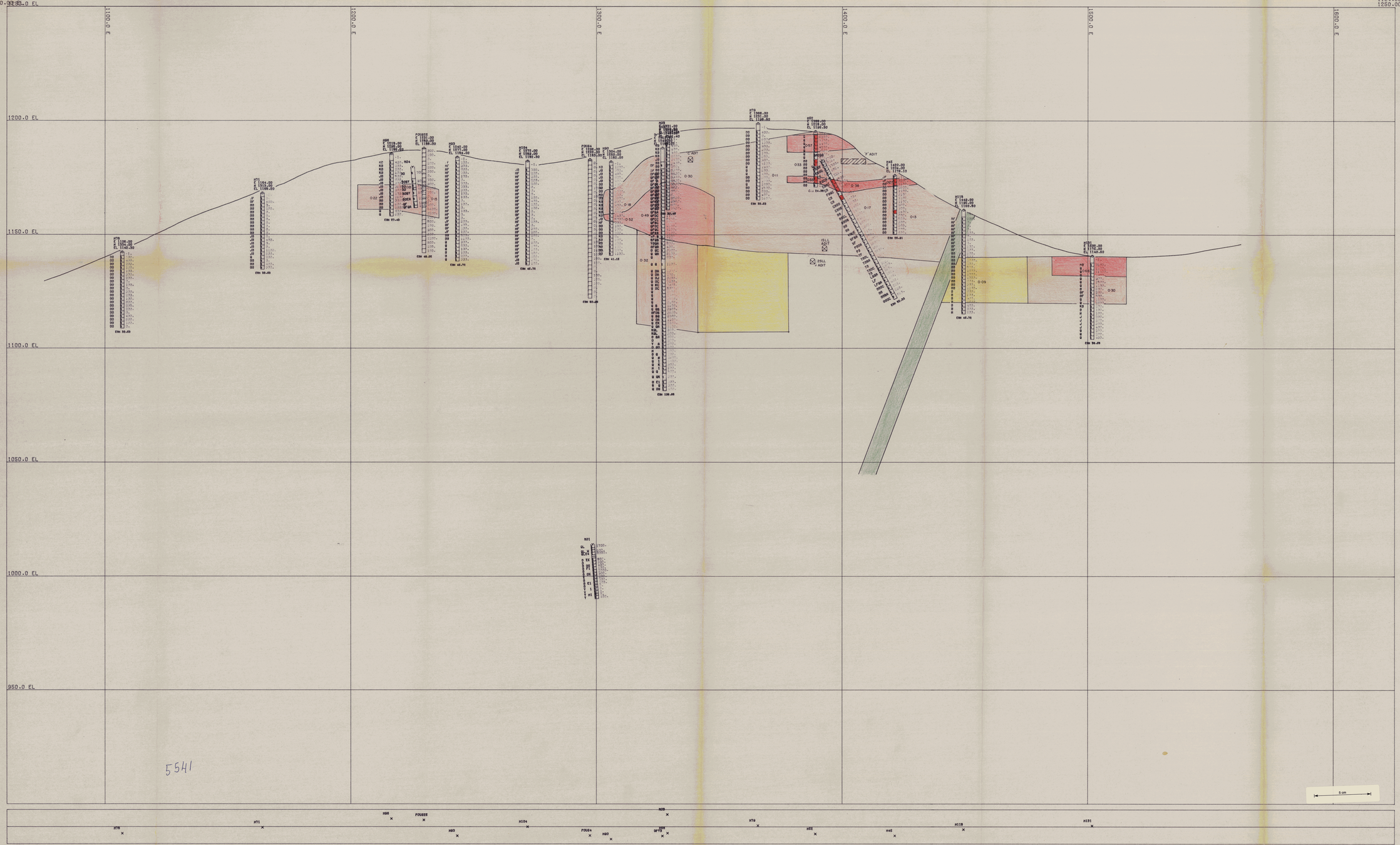
LEGEND

- PROBABLE RESOURCE ≥ 0·1% Sn
- PROBABLE RESOURCE ≥ 0·2% Sn
- POTENTIAL RESOURCE

VERTICAL SCALE = 1 : 500
HORIZONTAL SCALE = 1 : 500

SHELL COMPANY OF AUSTRALIA METALS DIVISION	
GREAT PYRAMID	
SECTION 7 W-E	
PRELIMINARY GEOLOGICAL RESOURCE OUTLINES	
5540	
FIG.NO:	REPT.NO:
ENGL.NO:	DRG.NO: D/LHOT/585
DATE:	AUTHOR:
DRAWN:	OFFICE:

86-2532



5541

047172

SHELL ORE EVALUATION SYSTEM
DRILLHOLE SECTION PLOT

NUMERIC LEGEND

/ DH /

LITHOLOGICAL DESCRIPTION CODES PLOTTED

LEGEND

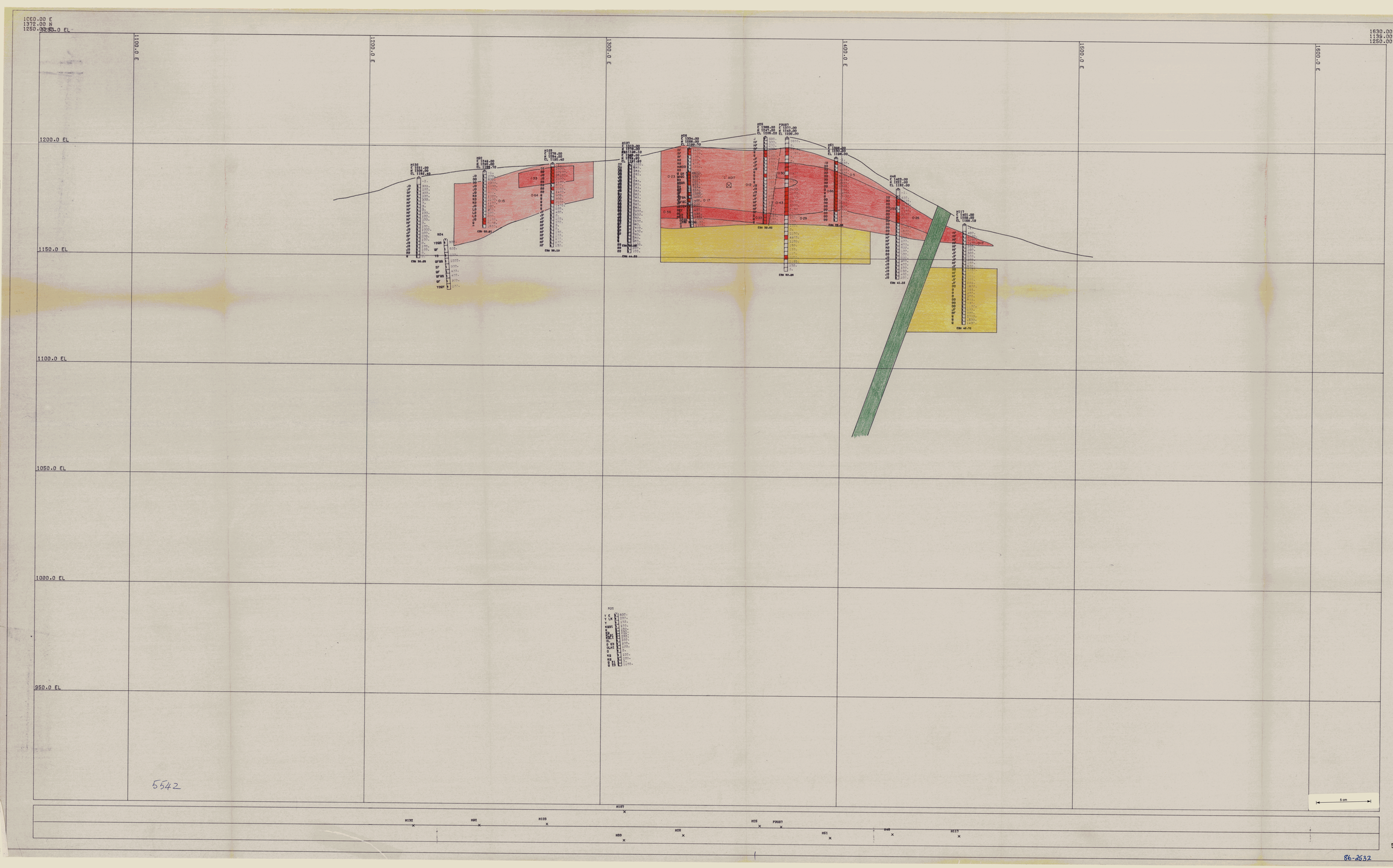
- PROBABLE RESOURCE ≥ 0·1% Sn
- PROBABLE RESOURCE ≥ 0·2% Sn
- POTENTIAL RESOURCE

VERTICAL SCALE = 1 : 500
HORIZONTAL SCALE = 1 : 500

SHELL COMPANY OF AUSTRALIA METALS DIVISION	
GREAT PYRAMID SECTION 8 W-E PRELIMINARY GEOLOGICAL RESOURCE OUTLINES	
FIG.NO:	REPT.NO:
ENCL.NO:	DRG.NO: D/LH05/586
DATE:	AUTHOR:
DRAWN:	OFFICE:

5541

86-2532



SHLMET ORE EVALUATION SYSTEM
DRILLHOLE SECTION PLOT

NUMERIC LEGEND

/ SN /

047173

LITHOLOGICAL DESCRIPTION CODES PLOTTED

LEGEND

PROBABLE RESOURCE ≥ 0·1% Sn

PROBABLE RESOURCE ≥ 0·2% Sn

POTENTIAL RESOURCE

VERTICAL SCALE = 1 : 500
HORIZONTAL SCALE = 1 : 500

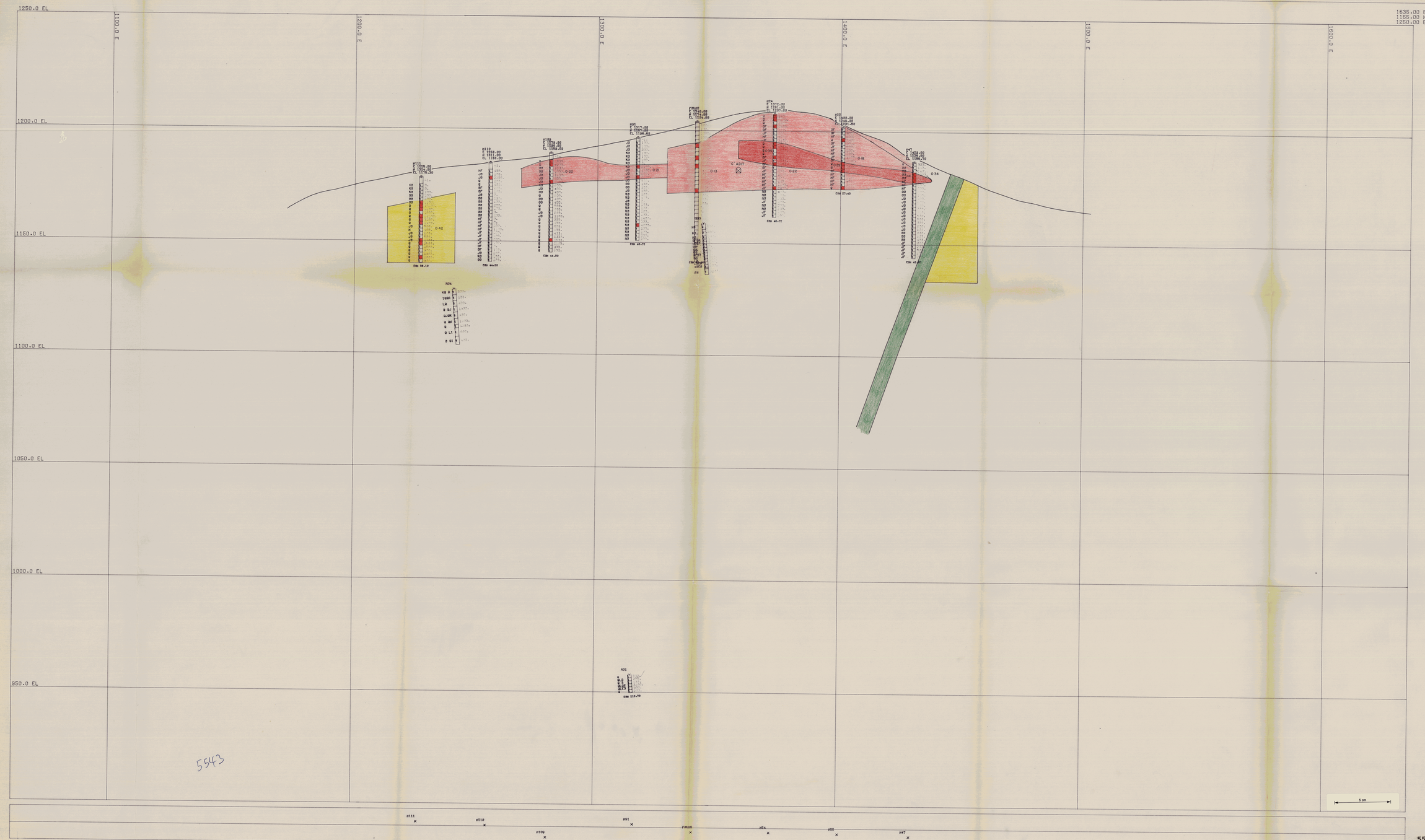
SHELL COMPANY OF AUSTRALIA
METALS DIVISION

GREAT PYRAMID
SECTION 9 W-E
PRELIMINARY GEOLOGICAL
RESOURCE OUTLINES

FIG.NO:	REPT.NO:
ENCL.NO:	DRG.NO: D/LH01/587
DATE:	AUTHOR:
DRAWN:	OFFICE:

5542

86-2532



5543

SHELL COMPANY OF AUSTRALIA
METALS DIVISION

SHELMET ORE EVALUATION SYSTEM
DRILLHOLE SECTION PLOT

NUMERIC LEGEND

047174
LITHOLOGICAL DESCRIPTION CODES PLOTTED

LEGEND

- PROBABLE RESOURCE ≥ 0·1% Sn
- PROBABLE RESOURCE ≥ 0·2% Sn
- POTENTIAL RESOURCE

VERTICAL SCALE = 1 : 500
HORIZONTAL SCALE = 1 : 500

SHELL COMPANY OF AUSTRALIA METALS DIVISION	
GREAT PYRAMID SECTION 10 W-E PRELIMINARY GEOLOGICAL RESOURCE OUTLINES	
FIG.NO:	REPT.NO:
ENGL.NO:	DRG.NO: D/LH01/588
DATE:	AUTHOR:
DRAWN:	OFFICE:

5543

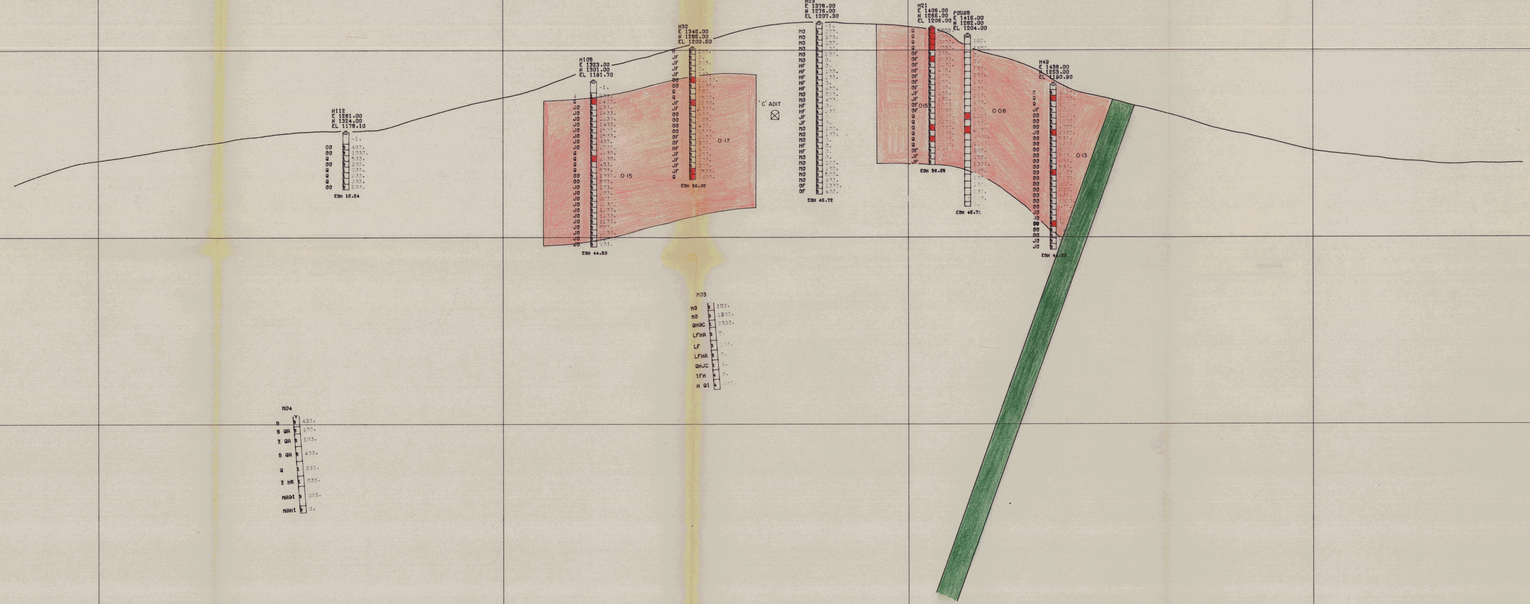
86-2532

1079.00 E
1400.00 N
1250.00 EL

1640.00 E
1168.00 N
1250.00 EL

1100.0 E | 1200.0 E | 1300.0 E | 1400.0 E | 1500.0 E | 1600.0 E

1200.0 EL
1150.0 EL
1100.0 EL
1050.0 EL
1000.0 EL
950.0 EL



5544

H11E H10E H02 H03 H01 FOUR H09

86-2532

SHLMET ORE EVALUATION SYSTEM
DRILLHOLE SECTION PLOT

NUMERIC LEGEND

047175
LITHOLOGICAL DESCRIPTION CODES PLOTTED

LEGEND

- PROBABLE RESOURCE ≥ 0·1% Sn
- PROBABLE RESOURCE ≥ 0·2% Sn
- POTENTIAL RESOURCE

VERTICAL SCALE = 1 : 500
HORIZONTAL SCALE = 1 : 500

5 cm

SHELL COMPANY OF AUSTRALIA
METALS DIVISION

GREAT PYRAMID
SECTION II W-E
PRELIMINARY GEOLOGICAL
RESOURCE OUTLINES

5544

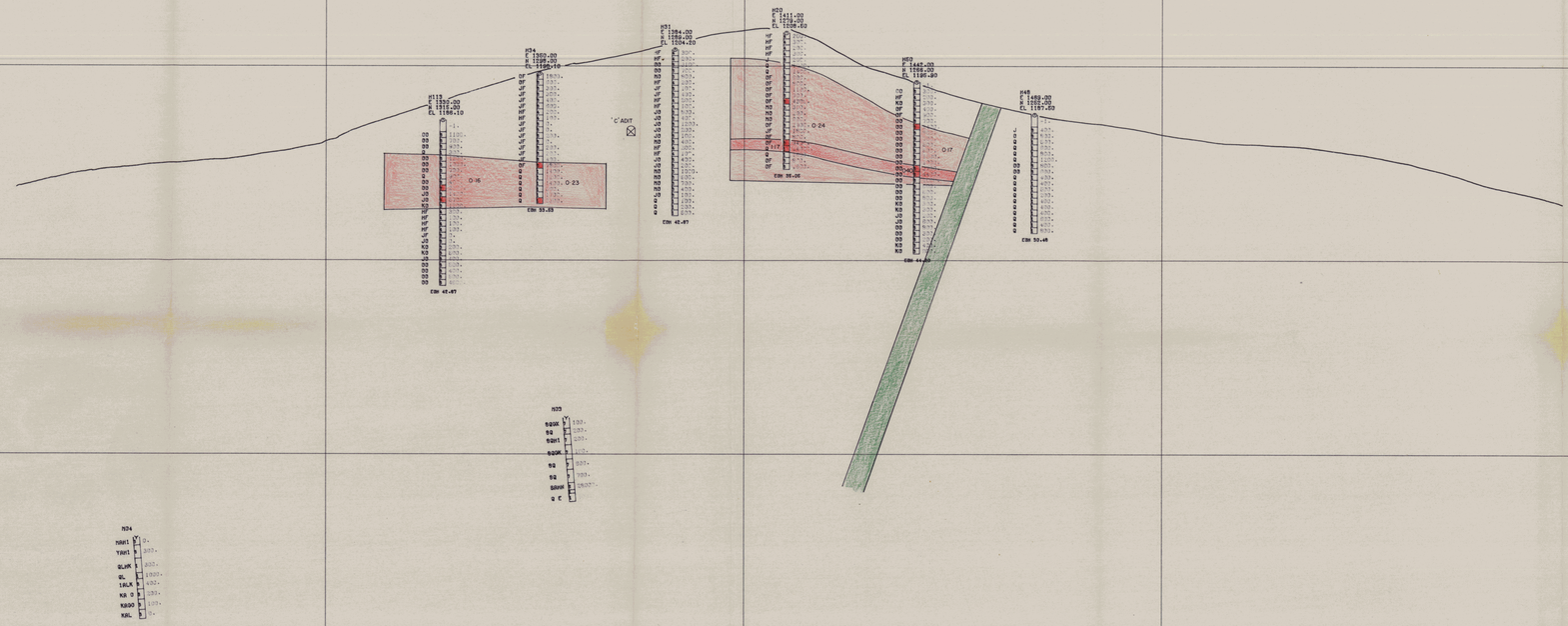
FIG.NO:	REPT.NO:
ENCL.NO:	DRG.NO: D/LH01/589
DATE:	AUTHOR:
DRAWN:	OFFICE:

1145.00 E
1145.00 N
1250.00 EL

1645.00 E
1182.00 N
1250.00 EL

1200.0 E | 1300.0 E | 1400.0 E | 1500.0 E | 1600.0 E

1200.0 EL
1150.0 EL
1100.0 EL
1050.0 EL
1000.0 EL
950.0 EL



5545

5 cm

SHELL COMPANY OF AUSTRALIA
METALS DIVISION

SHLMET ORE EVALUATION SYSTEM
DRILLHOLE SECTION PLOT

NUMERIC LEGEND

/ SN /

047126
LITHOLOGICAL DESCRIPTION CODES PLOTTED

LEGEND

- PROBABLE RESOURCE ≥ 0·1% Sn
- PROBABLE RESOURCE ≥ 0·2% Sn
- POTENTIAL RESOURCE

VERTICAL SCALE = 1 : 500
HORIZONTAL SCALE = 1 : 500

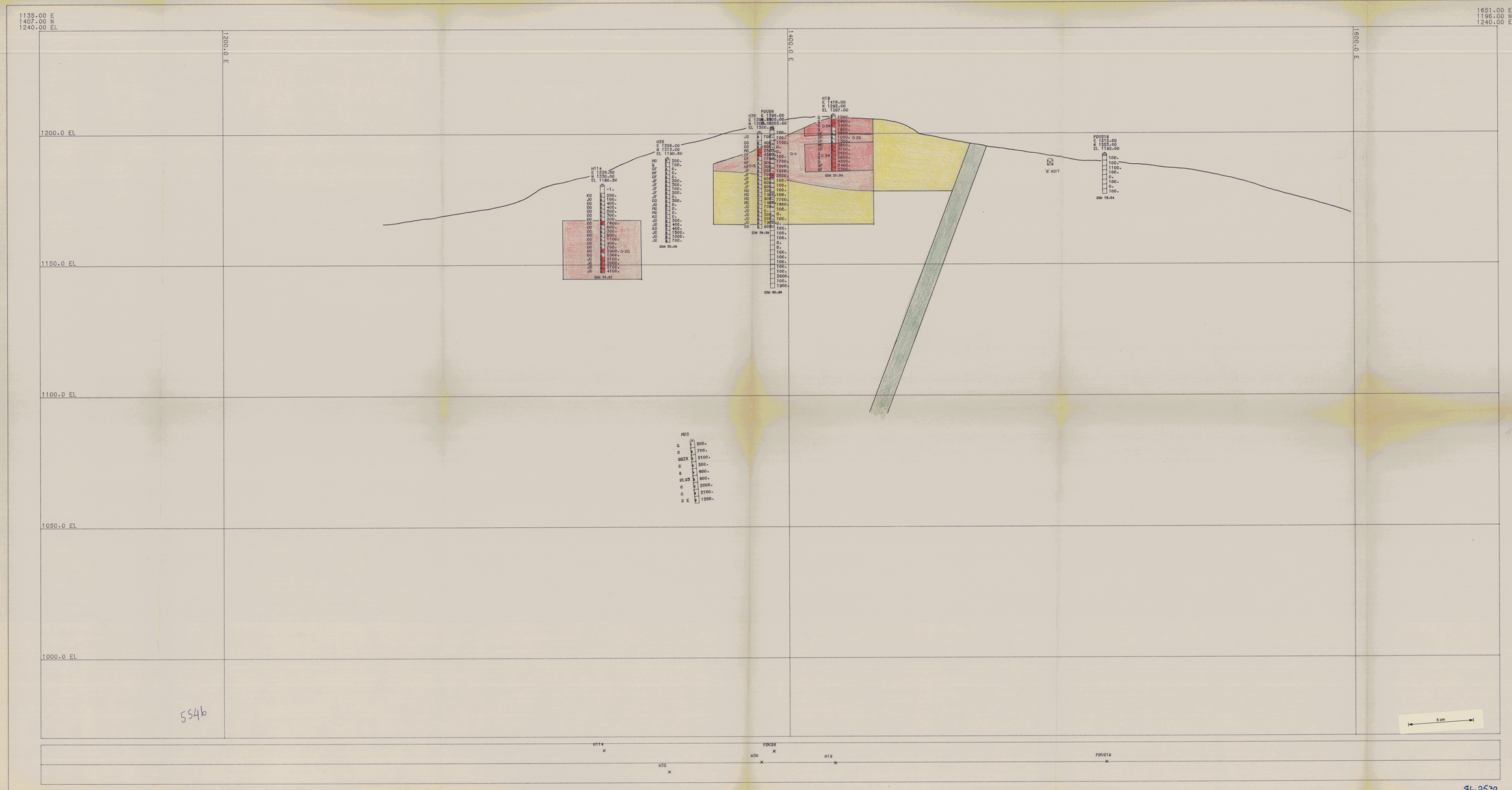
GREAT PYRAMID
SECTION 12 W-E
PRELIMINARY GEOLOGICAL
RESOURCE OUTLINES

FIG.NO:	REPT.NO:
ENCL.NO:	DRG.NO: D/LHO1/590
DATE:	AUTHOR:
DRAWN:	OFFICE:

5545

H113 | H34 | H31 | H30 | H50 | H48

86-3532



SHLMET ORE EVALUATION SYSTEM
DRILLHOLE SECTION PLOT

NUMERIC LEGEND

/ SN /

047177
LITHOLOGICAL DESCRIPTION CODES PLOTTED

LEGEND

PROBABLE RESOURCE ≥ 0·1% Sn

PROBABLE RESOURCE ≥ 0·2% Sn

POTENTIAL RESOURCE

VERTICAL SCALE = 1 : 500
HORIZONTAL SCALE = 1 : 500

SHELL COMPANY OF AUSTRALIA
METALS DIVISION

GREAT PYRAMID
SECTION 13 W-E
PRELIMINARY GEOLOGICAL
RESOURCE OUTLINES

5546

FIG.NO:	REPT.NO:
ENCL.NO:	DRG.NO: D/LH01/591
DATE:	AUTHOR:
DRAWN:	OFFICE:

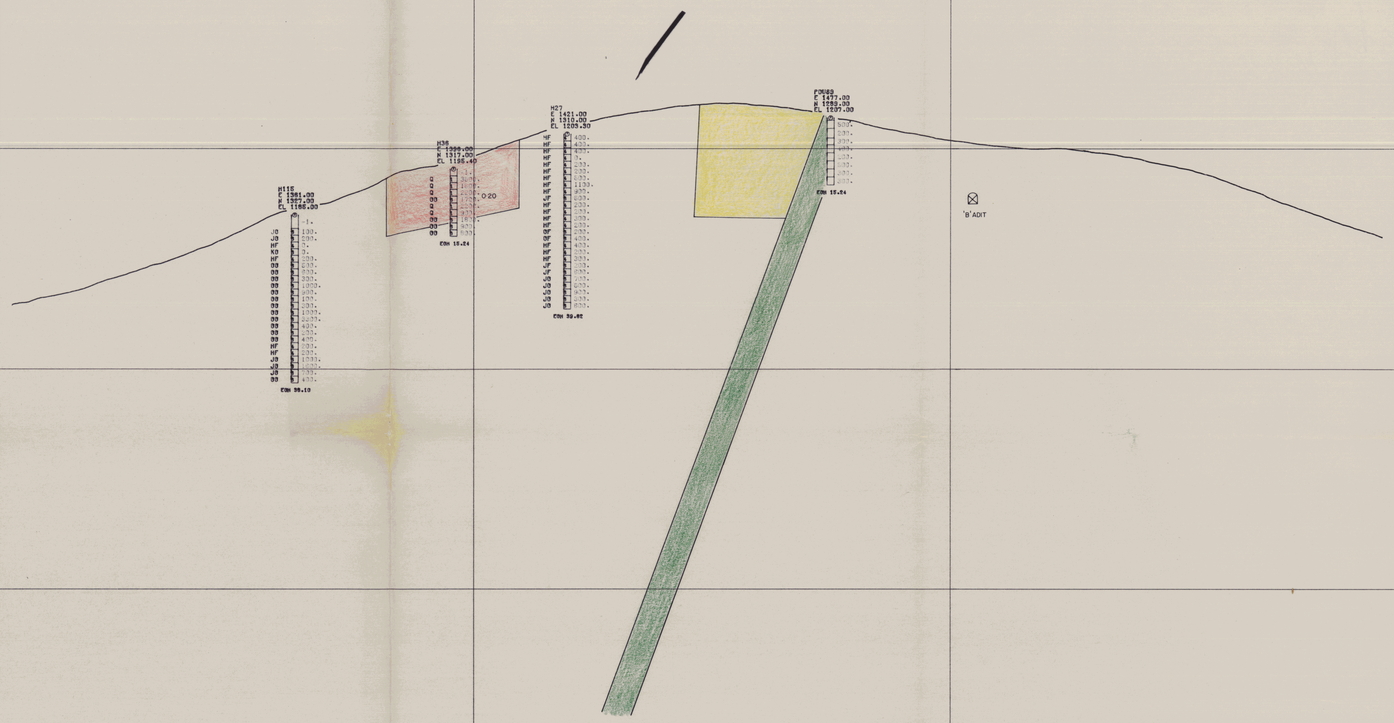
86-2532

1184.00 E
1400.00 N
1250.00 EL

1655.00 E
1209.00 N
1250.00 EL

1200.0 E
1300.0 E
1400.0 E
1500.0 E
1600.0 E

1200.0 EL
1150.0 EL
1100.0 EL
1050.0 EL
1000.0 EL
950.0 EL



H11E
H36
H27
FOU69

5 cm

SHELL COMPANY OF AUSTRALIA
METALS DIVISION

SHLMET ORE EVALUATION SYSTEM
DRILLHOLE SECTION PLOT

NUMERIC LEGEND

/ SN /
047178
LITHOLOGICAL DESCRIPTION CODES PLOTTED

LEGEND

- PROBABLE RESOURCE ≥ 0·1% Sn
- PROBABLE RESOURCE ≥ 0·2% Sn
- POTENTIAL RESOURCE

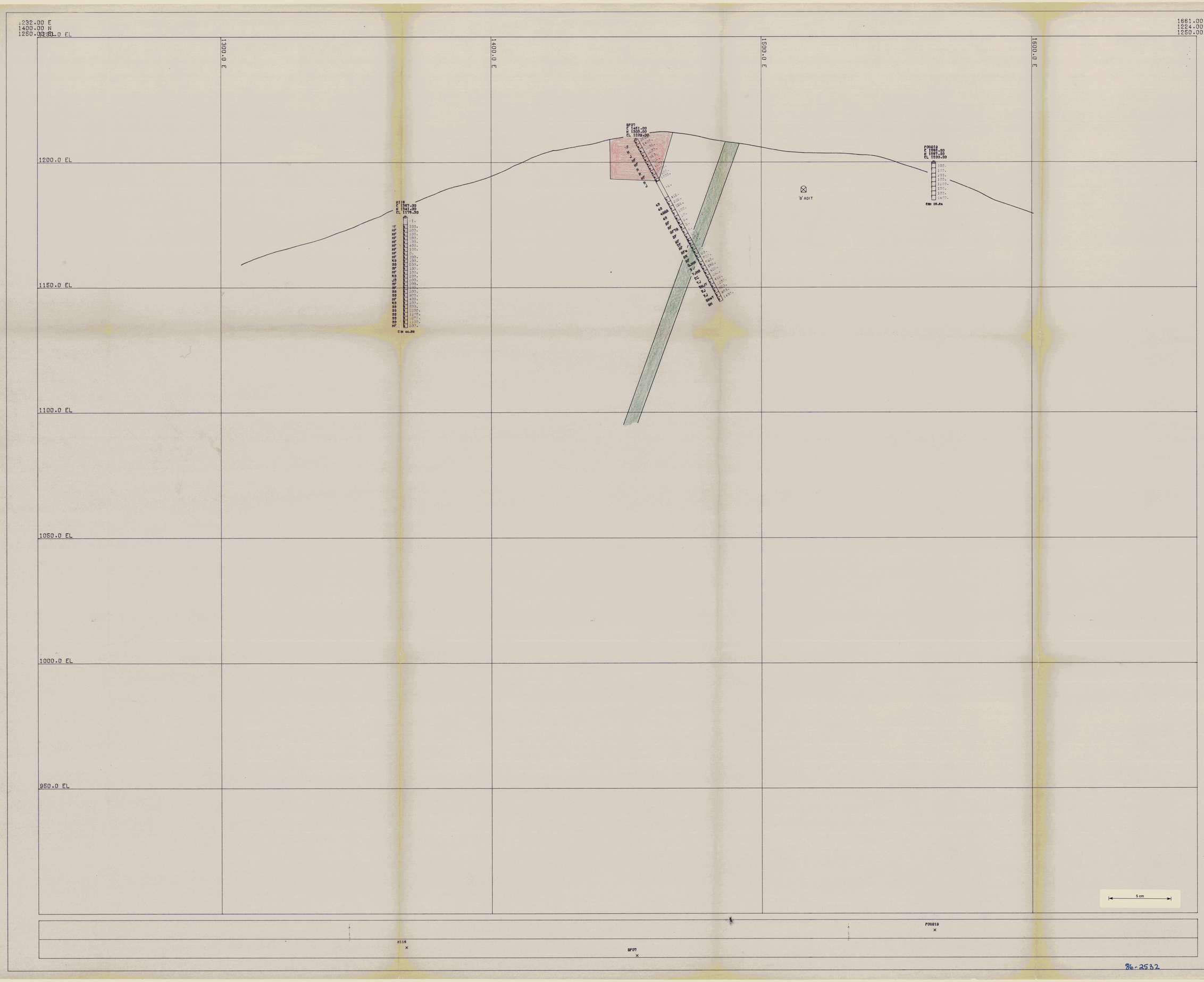
VERTICAL SCALE = 1 : 500
HORIZONTAL SCALE = 1 : 500

GREAT PYRAMID
SECTION 14 W-E
PRELIMINARY GEOLOGICAL
RESOURCE OUTLINES

FIG.NO:	REPT.NO:
ENCL.NO:	DRG.NO: D/LH01/592
DATE:	AUTHOR:
DRAWN:	OFFICE:

5547

86-2532



SHLMET ORE EVALUATION SYSTEM
DRILLHOLE SECTION PLOT

NUMERIC LEGEND

/ SN

LITHOLOGICAL DESCRIPTION CODES PLOTTED

LEGEND

PROBABLE RESOURCE ≥ 0·1% Sn

PROBABLE RESOURCE ≥ 0·2% Sn

POTENTIAL RESOURCE

VERTICAL SCALE = 1 : 500
HORIZONTAL SCALE = 1 : 500

047179

SHELL COMPANY OF AUSTRALIA
METALS DIVISION

GREAT PYRAMID
SECTION 15 W-E

PRELIMINARY GEOLOGICAL
RESOURCE OUTLINES

5548

FIG.NO:	REPT.NO:
ENCL.NO:	DRG.NO: D/LH01/593
DATE:	AUTHOR:
DRAWN:	OFFICE:

86-2532

1312.00 E
1400.00 N
1250.00 EL

1675.00 E
1250.00 N
1250.00 EL

1400.0 E

1500.0 E

1600.0 E

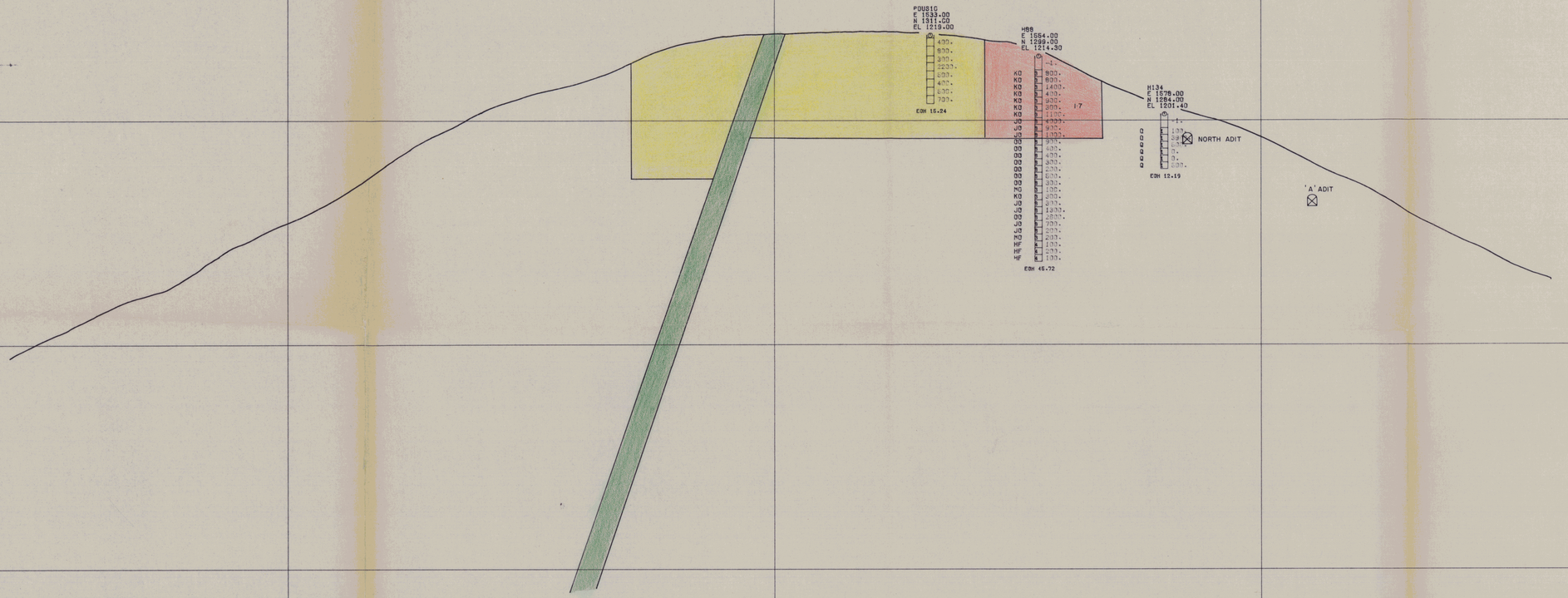
1200.0 EL

1150.0 EL

1100.0 EL

1050.0 E

1000.0 EL



POUS1G
HBB
H134

SHLMET ORE EVALUATION SYSTEM
DRILLHOLE SECTION PLOT

NUMERIC LEGEND

/ SN

047181
LITHOLOGICAL DESCRIPTION CODES PLOTTED

LEGEND

- PROBABLE RESOURCE ≥ 0·1% Sn
- PROBABLE RESOURCE ≥ 0·2% Sn
- POTENTIAL RESOURCE

VERTICAL SCALE = 1 : 500
HORIZONTAL SCALE = 1 : 500

SHELL COMPANY OF AUSTRALIA
METALS DIVISION

GREAT PYRAMID
SECTION 17 W-E
PRELIMINARY GEOLOGICAL
RESOURCE OUTLINES

FIG.NO:	REPT.NO:
ENCL.NO:	ORG.NO: D/LH01/595
DATE:	AUTHOR:
DRAWN:	OFFICE:

5550

86-2532

1271.00 E
1400.00 N
1250.00 EL

1670.00 E
1236.00 N
1250.00 EL

1300.0 E

1400.0 E

1500.0 E

1600.0 E

1200.0 EL

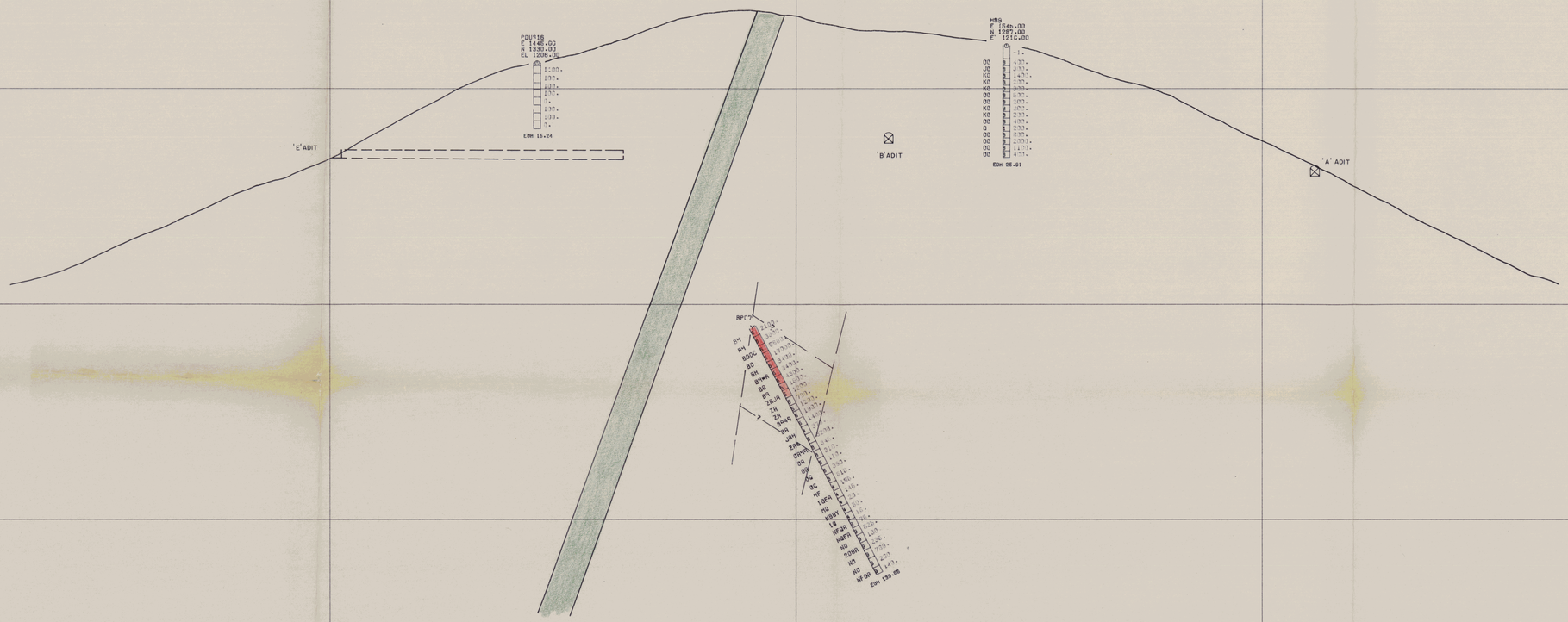
1150.0 E

1100.0 EL

1050.0 EL

1000.0 EL

950.0 EL



5 cm

*CUB16

HBS

SHLMET ORE EVALUATION SYSTEM
DRILLHOLE SECTION PLOT

NUMERIC LEGEND

/ SN /

047180

LITHOLOGICAL DESCRIPTION CODES PLOTTED

LEGEND

- PROBABLE RESOURCE ≥ 0·1% Sn
- PROBABLE RESOURCE ≥ 0·2% Sn
- POTENTIAL RESOURCE

VERTICAL SCALE = 1 : 500
HORIZONTAL SCALE = 1 : 500

SHELL COMPANY OF AUSTRALIA
METALS DIVISION

GREAT PYRAMID
SECTION 16 W-E
PRELIMINARY GEOLOGICAL
RESOURCE OUTLINES

FIG.NO:	REPT.NO:
ENCL.NO:	DRG.NO: D/LH01/594
DATE:	AUTHOR:
DRAWN:	OFFICE:

5549

86-2532

1352.00 E
1400.00 N
1250.00 EL

1680.00 E
1264.00 N
1250.00 EL

1400.0 E

1500.0 E

1600.0 E

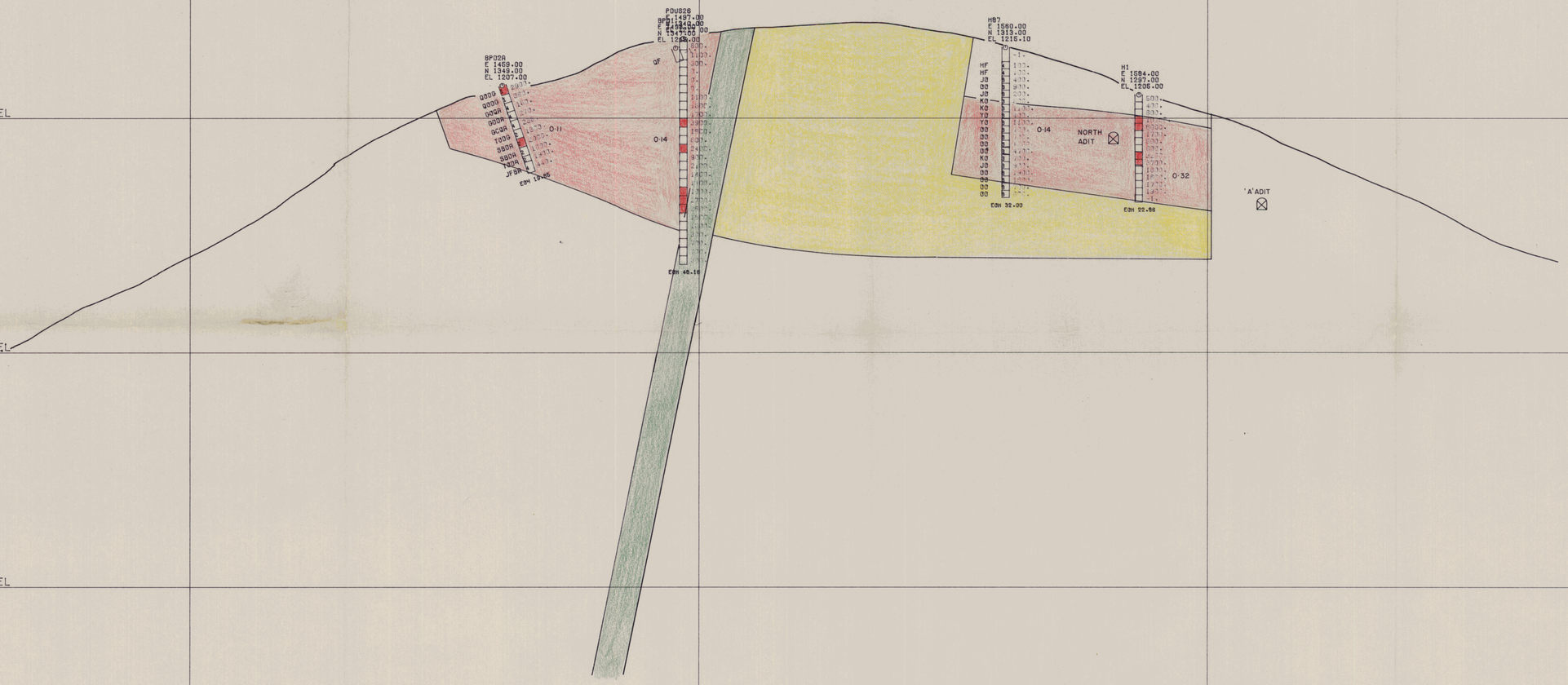
1200.0 EL

1150.0 EL

1100.0 EL

1050.0 EL

1000.0 EL



5 cm

SHLMET ORE EVALUATION SYSTEM
DRILLHOLE SECTION PLOT

NUMERIC LEGEND

/ SN

047182

LITHOLOGICAL DESCRIPTION CODES PLOTTED

LEGEND

- PROBABLE RESOURCE ≥ 0·1% Sn
- PROBABLE RESOURCE ≥ 0·2% Sn
- POTENTIAL RESOURCE

VERTICAL SCALE = 1 : 500
HORIZONTAL SCALE = 1 : 500

SHELL COMPANY OF AUSTRALIA
METALS DIVISION

GREAT PYRAMID
SECTION 18 W-E
PRELIMINARY GEOLOGICAL
RESOURCE OUTLINES

FIG.NO:	REPT.NO:
ENCL.NO:	DRG.NO: D/LH01/596
DATE:	AUTHOR:
DRAWN:	OFFICE:

5531

86-2532

1365.00 E
1411.00 N
1250.00 EL

1690.00 E
1272.00 N
1250.00 EL

1400.0 E

1500.0 E

1600.0 E

1250.0 EL

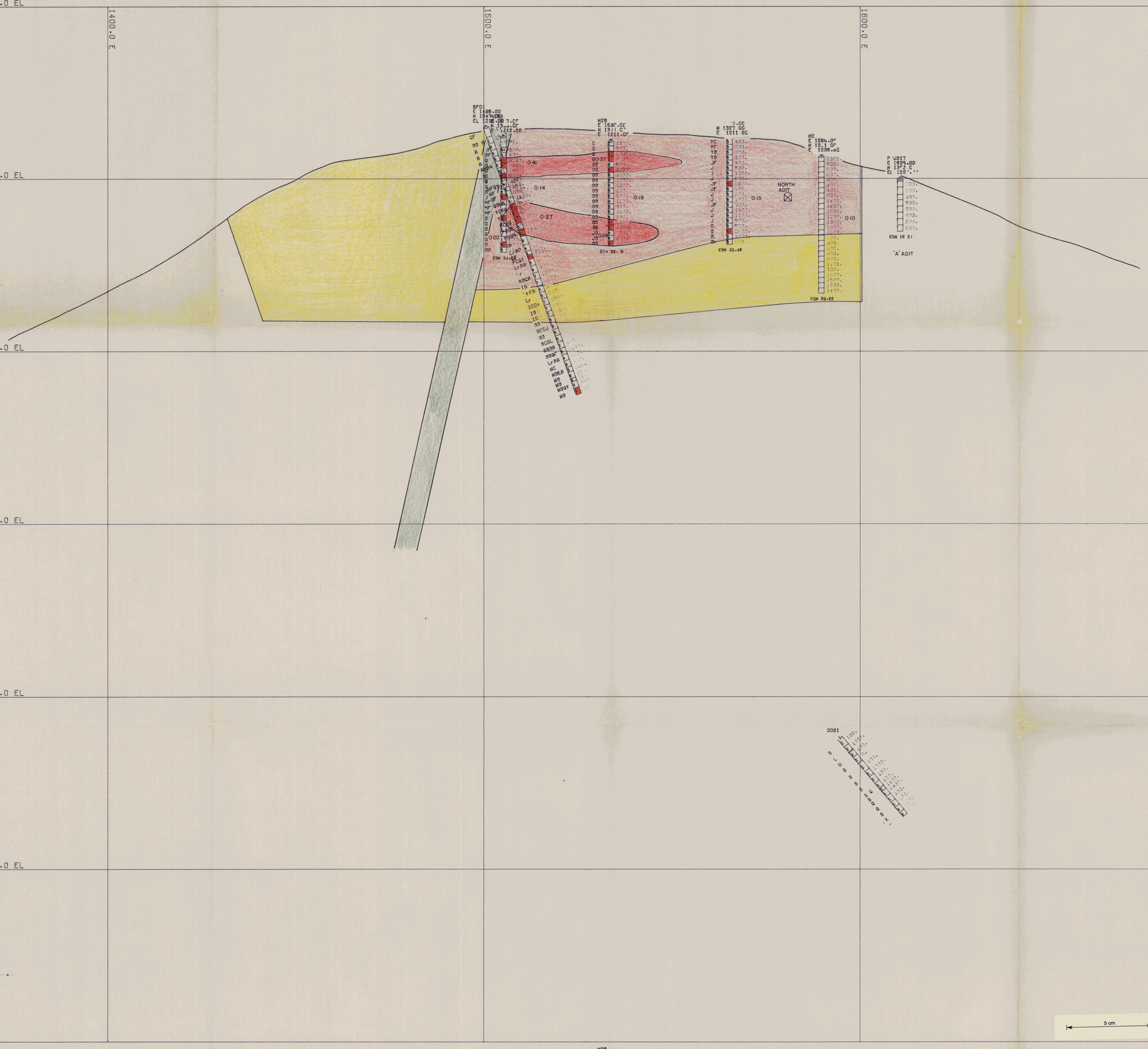
1200.0 EL

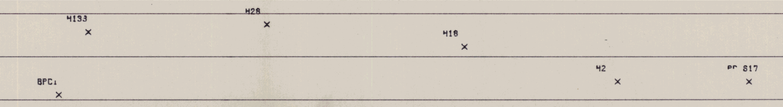
1150.0 EL

1100.0 EL

1050.0 EL

1000.0 EL





SHLMET ORE EVALUATION SYSTEM
DRILLHOLE SECTION PLOT

NUMERIC LEGEND

/ SN

047183

LITHOLOGICAL DESCRIPTION CODES PLOTTED

LEGEND

- PROBABLE RESOURCE ≥ 0·1% Sn
- PROBABLE RESOURCE ≥ 0·2% Sn
- POTENTIAL RESOURCE

VERTICAL SCALE = 1 : 500
HORIZONTAL SCALE = 1 : 500

SHELL COMPANY OF AUSTRALIA
METALS DIVISION

GREAT PYRAMID
SECTION 19 W-E
PRELIMINARY GEOLOGICAL
RESOURCE OUTLINES

FIG.NO:	REPT.NO:
ENCL.NO:	DRG.NO: D/LH01/597
DATE:	AUTHOR:
DRAWN.	OFFICE:

5552

86-2532

365.00 E
1426.00 N
1250.00 EL

1695.00 E
1292.00 N
1250.00 EL

1400.0 E

1500.0 E

1600.0 E

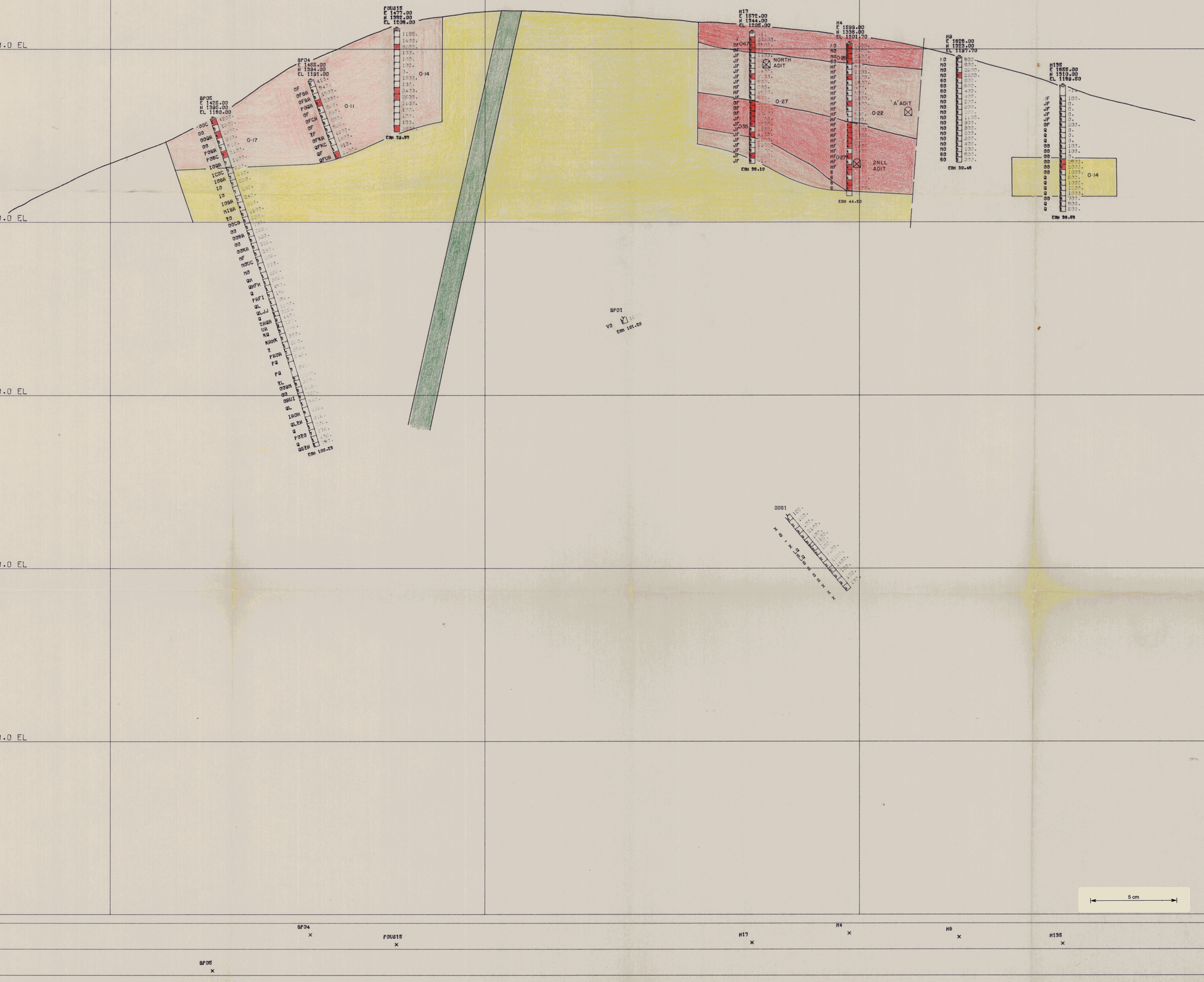
1200.0 EL

1150.0 EL

1100.0 EL

1050.0 EL

1000.0 EL



5 cm

SHLMET ORE EVALUATION SYSTEM
DRILLHOLE SECTION PLOT

NUMERIC LEGEND

/ SN

047184

LITHOLOGICAL DESCRIPTION CODES PLOTTED

LEGEND

- PROBABLE RESOURCE ≥ 0·1% Sn
- PROBABLE RESOURCE ≥ 0·2% Sn
- POTENTIAL RESOURCE

VERTICAL SCALE = 1 : 500
HORIZONTAL SCALE = 1 : 500

SHELL COMPANY OF AUSTRALIA
METALS DIVISION

GREAT PYRAMID
SECTION 20 W-E
PRELIMINARY GEOLOGICAL
RESOURCE OUTLINES

FIG.NO:	REPT.NO:
ENCL.NO:	ORG.NO: D/LH01/598
DATE:	AUTHOR:
DRAWN:	OFFICE:

5553

86-2532



SHLMET ORE EVALUATION SYSTEM
DRILLHOLE SECTION PLOT

NUMERIC LEGEND

/ SN

047185

LITHOLOGICAL DESCRIPTION CODES PLOTTED

LEGEND

PROBABLE RESOURCE ≥ 0·1% Sn

PROBABLE RESOURCE ≥ 0·2% Sn

POTENTIAL RESOURCE

VERTICAL SCALE = 1 : 500
HORIZONTAL SCALE = 1 : 500

SHELL COMPANY OF AUSTRALIA
METALS DIVISION

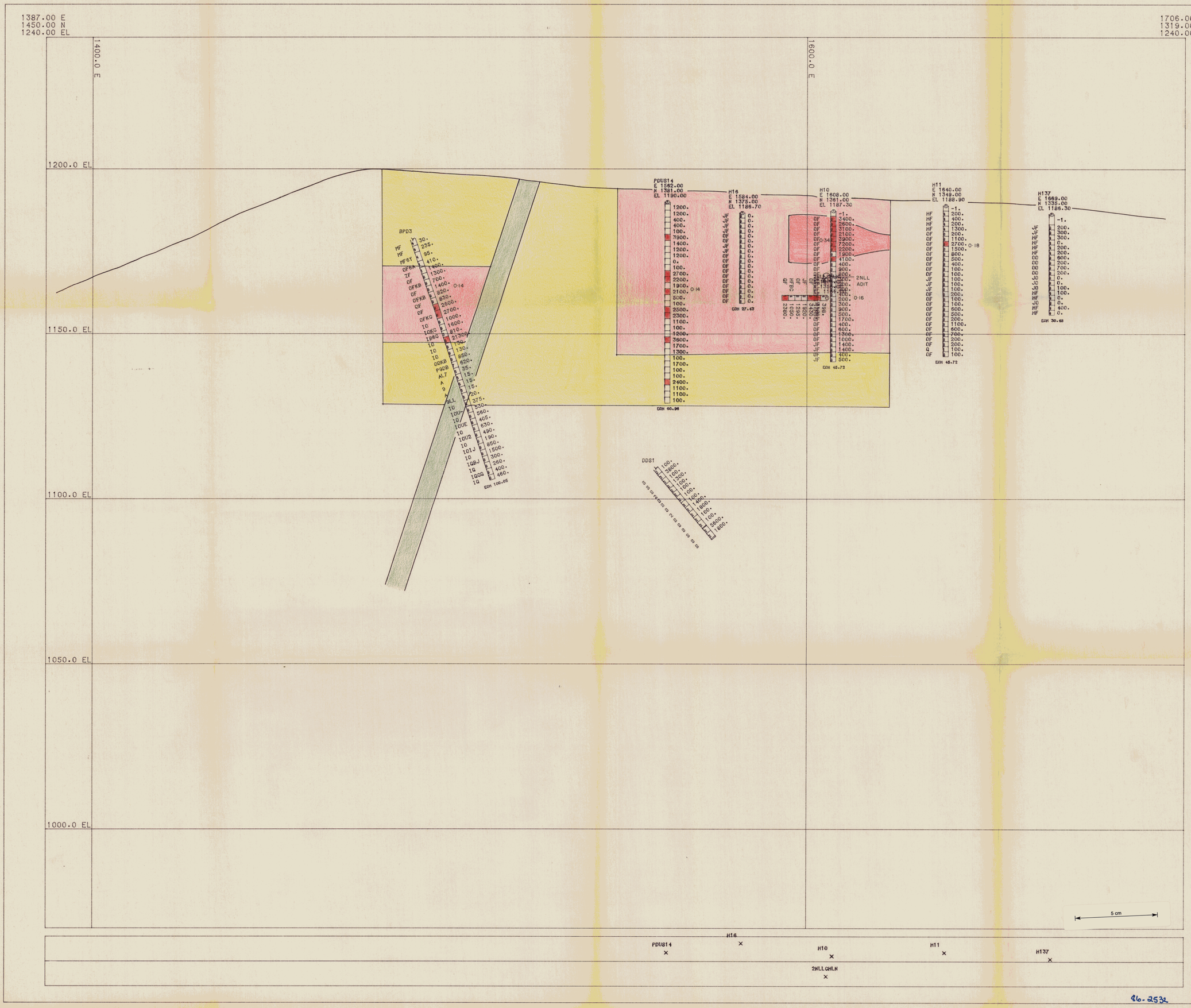
GREAT PYRAMID
SECTION 21 W-E

PRELIMINARY GEOLOGICAL
RESOURCE OUTLINES

FIG.NO:	REPT.NO:
ENCL.NO:	DRG.NO: D/LH01/599
DATE:	AUTHOR:
DRAWN:	OFFICE:

5554

86-2532



SHLMET ORE EVALUATION SYSTEM
DRILLHOLE SECTION PLOT

NUMERIC LEGEND

/ SN /

047186

LITHOLOGICAL DESCRIPTION CODES PLOTTED

LEGEND

PROBABLE RESOURCE ≥ 0·1% Sn

PROBABLE RESOURCE ≥ 0·2% Sn

POTENTIAL RESOURCE

VERTICAL SCALE = 1 : 500
HORIZONTAL SCALE = 1 : 500

SHELL COMPANY OF AUSTRALIA
METALS DIVISION

GREAT PYRAMID
SECTION 22 W-E

PRELIMINARY GEOLOGICAL
RESOURCE OUTLINES

5555

FIG.NO:	REPT.NO:
ENCL.NO:	DRG.NO: 600
DATE:	AUTHOR:
DRAWN:	OFFICE:

86-2532

1428.00 E
1450.00 N
1250.00 EL

1710.00 E
1332.00 N
1250.00 EL

1500.0 E

1600.0 E

1700.0 E

1250.0 EL

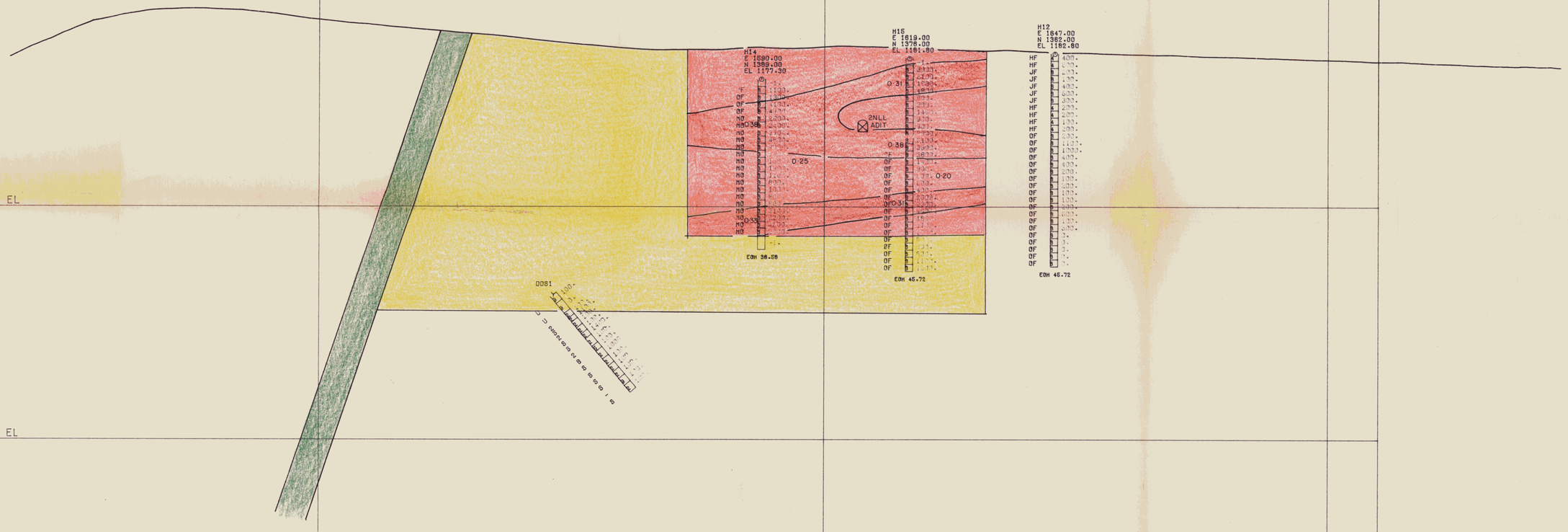
1200.0 EL

1150.0 EL

1100.0 EL

1050.0 EL

1000.0 EL



5 cm

H14 H15 H12

SHLMET ORE EVALUATION SYSTEM
DRILLHOLE SECTION PLOT

NUMERIC LEGEND

/ SN

047187

LITHOLOGICAL DESCRIPTION CODES PLOTTED

LEGEND

PROBABLE RESOURCE ≥ 0·1% Sn

PROBABLE RESOURCE ≥ 0·2% Sn

POTENTIAL RESOURCE

VERTICAL SCALE = 1 : 500
HORIZONTAL SCALE = 1 : 500

SHELL COMPANY OF AUSTRALIA
METALS DIVISION

GREAT PYRAMID

SECTION 23 W-E

PRELIMINARY GEOLOGICAL
RESOURCE OUTLINES

FIG.NO:	REPT.NO:
ENCL.NO:	DRG.NO: D/LH01/601
DATE:	AUTHOR:
DRAWN:	OFFICE:

5556

86-2532

1467.00 E
1450.00 N
1240.00 EL

1717.00 E
1347.00 N
1240.00 EL

1600.0 E

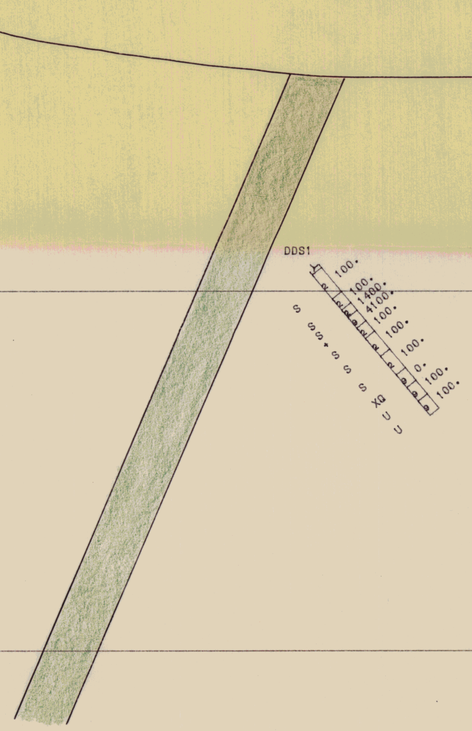
1200.0 EL

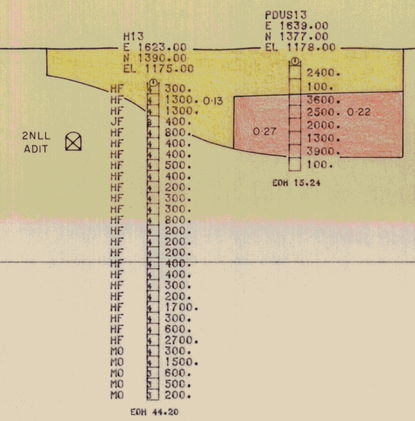
1150.0 EL

1100.0 EL

1050.0 EL

1000.0 EL





5 cm

H13
PDUS13

SHLMET ORE EVALUATION SYSTEM
DRILLHOLE SECTION PLOT

NUMERIC LEGEND

/ SN /

047188

LITHOLOGICAL DESCRIPTION CODES PLOTTED

LEGEND

PROBABLE RESOURCE ≥ 0·1% Sn

PROBABLE RESOURCE ≥ 0·2% Sn

POTENTIAL RESOURCE

VERTICAL SCALE = 1 : 500
HORIZONTAL SCALE = 1 : 500

SHELL COMPANY OF AUSTRALIA
METALS DIVISION

GREAT PYRAMID

SECTION 24 W-E

PRELIMINARY GEOLOGICAL
RESOURCE OUTLINES

FIG.NO:	REPT.NO:
ENCL.NO:	DRG.NO: D/LH01/602
DATE:	AUTHOR:
DRAWN:	OFFICE:

5557

86-2532