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THE ANCHOR MINE

A RESOURCE ASSESSMENT

for

SPECTRUM RESOURCES PTY LTD

APRIL 1993

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RIDGLEY, TASMANIA

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1 INTRODUCTION

The Anchor Tin Mine lies 22 kilometres to the north-west of St Helens in Tasmania's north-east: see Figure 1. The mine is the largest of many mines which make up the Blue Tier Tinfield.

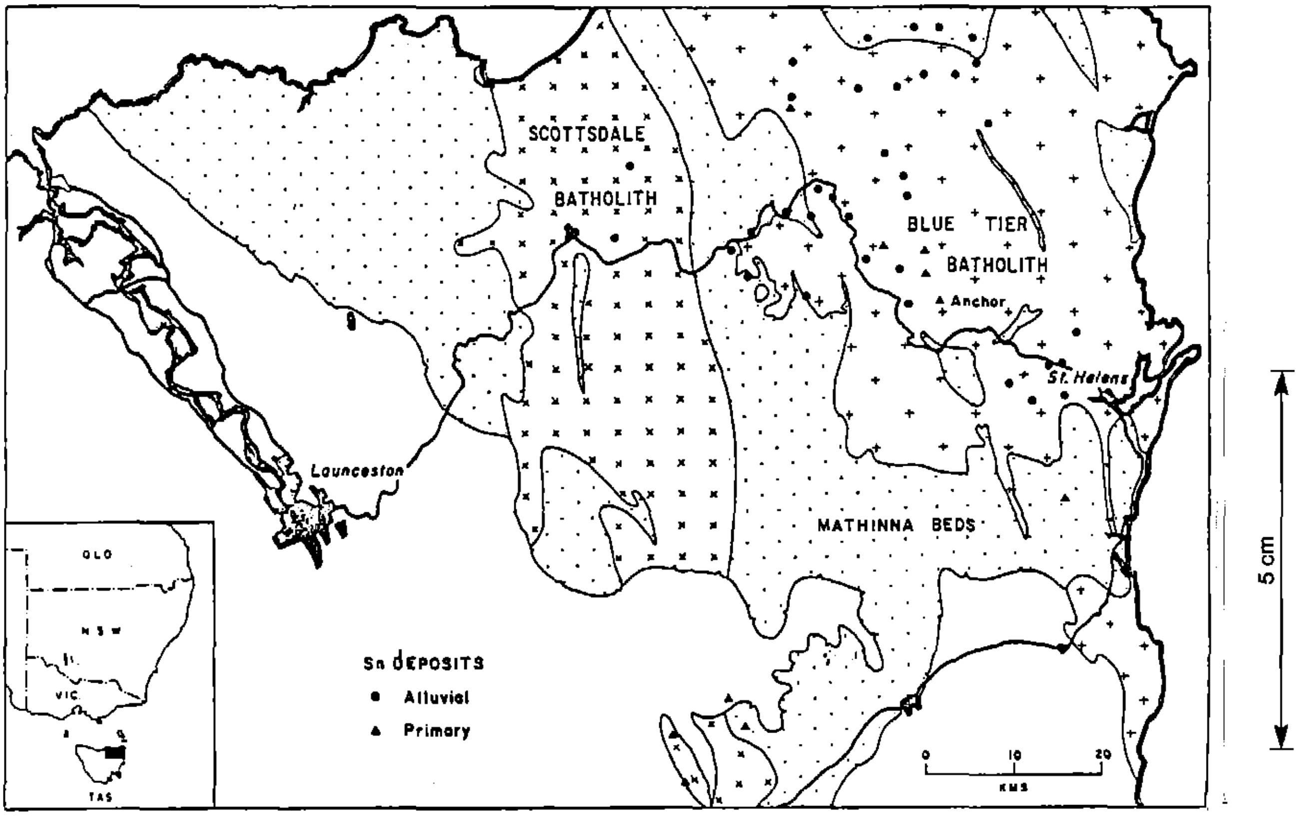

Figure 1. Geological Map of north-east Tasmania showing locations of principal tin deposits

The orebodies of the Anchor Mine were discovered during the burst of exploration for tin in the 1860's and 1870's. The Anchor Tin Mining Company Limited was floated on the Hobart Stock Exchange in 1882.

Alluvial operations eventually gave way to hard rock mining by opencut. Although the ore won from the opencut was low grade by the standards of the day, over-optimistic views of the economics of mining the orebody led to the installation of a 100 head stamp battery driven by a water wheel 66 feet in diameter. The chequered history of tin production over the years is outlined in Table 1.

TABLE 1

RECORDED PRODUCTION OF CASSITERITE CONCENTRATE
FROM THE ANCHOR MINE

to 1892	293 tonnes	Anchor Tin Mining Co
1898 to 1903	1 021 tonnes	Anchor Tin Mine Ltd
1898 to 1914	2 767 tonnes	Anchor Tin Mine Ltd
1901 to 1904	766 tonnes	Anchor Tin Mine Ltd
1903 to 1914	175 tonnes	Anchor Tin Mine Ltd
1907	229 tonnes	Anchor Tin Mine Ltd
1910	144 tonnes	Anchor Tin Mine Ltd
1911	163 tonnes	Anchor Tin Mine Ltd
1914	136 tonnes	Anchor Tin Mine Ltd
1914 to 1923	43 tonnes	Tributers
1928	6 tonnes	Tributers
1934 to 1935	113 tonnes	Anchor Tin Syndicate
1936 to 1942	250 tonnes	Tasman Tin N.L.
1989 to 1991	722 tonnes	Spectrum Resources Pty Ltd

Note: the early production records are incomplete; this table has been adapted and updated from Table 1.1 in Ross (1983).

The Aberfoyle Tin Development Partnership explored the Anchor area between 1963 and 1966. Thirty nine diamond drill holes were completed but the ore outlined did not meet their target objective of one million tonnes at 1% Sn.

In 1976, Hellyer Mining and Exploration Pty Ltd was granted Exploration Licence EL9/76 over 76 square kilometres which included the Anchor Mine. At about the same time, Renison Limited began a programme of exploration in the search for large, low grade, tin deposits in Tasmania. One of the prospects selected by Renison was the Blue Tier Tinfield including the Anchor Mine and, in December 1977, a joint venture agreement was entered into with Hellyer Mining and Exploration.

Renison drilled over one hundred diamond drill holes in the Anchor Mine area and identified a resource of 8.8 million tonnes at 0.18% Sn, based on a cut-off grade of 0.05% Sn. This was not considered an economically mineable resource (Ross, 1983).

Included in the Renison resource were 630,000 tonnes at 0.49% Sn, based on a cut-off grade of 0.2% Sn (Ross, 1983). This higher grade resource was located adjacent to the old opencut workings.

In 1988 Mike Baker, of Spectrum Resources, calculated a resource, in two lenses of ore adjacent to the old opencut workings, of 438,000 tonnes at 0.62% Sn, based on a cut-off grade of 0.5% Sn (Baker, 1988b). Spectrum Resources considered that there was potential for profitable extraction of this resource using a suitable underground mining method at the ruling tin price. Spectrum Resources developed an underground mine and gravity concentrator in 1988/89. Unfortunately tin prices fell dramatically from over US$10,000 in 1988 to less than US$6,000 in 1991. Due to low tin prices, the operation was placed on care and maintenance in December 1991.

2 GEOLOGICAL SETTING

The geology of the Anchor area was comprehensively described by Reid and Henderson in 1928. More recently, Ross (1983) gave details of the lithologies and alteration patterns in the Anchor Mine area. The development of the underground mine has generally confirmed these previous descriptions. Examination of the underground exposures has led to a better understanding of the greisen occurrence and distribution within the host rocks and, also, the possibility of ore extensions beyond the limits shown in previous ore reserve reports.

A detailed description of the geology of the mine area is beyond the scope of this report but a brief description is included to enable a sensible understanding of the ore resource assessment.

At the Anchor Mine there are two dominant rock types: the Poimena Adamellite and the Lottah Granite, both of late Devonian - early Carboniferous age. The Poimena Adamellite is a medium to coarse grained adamellite. The Lottah Granite is an alkali feldspar granite.

The Poimena Adamellite is the more extensive and older rock; it has been intruded by the younger Lottah Granite: see Figures 2 and 3.

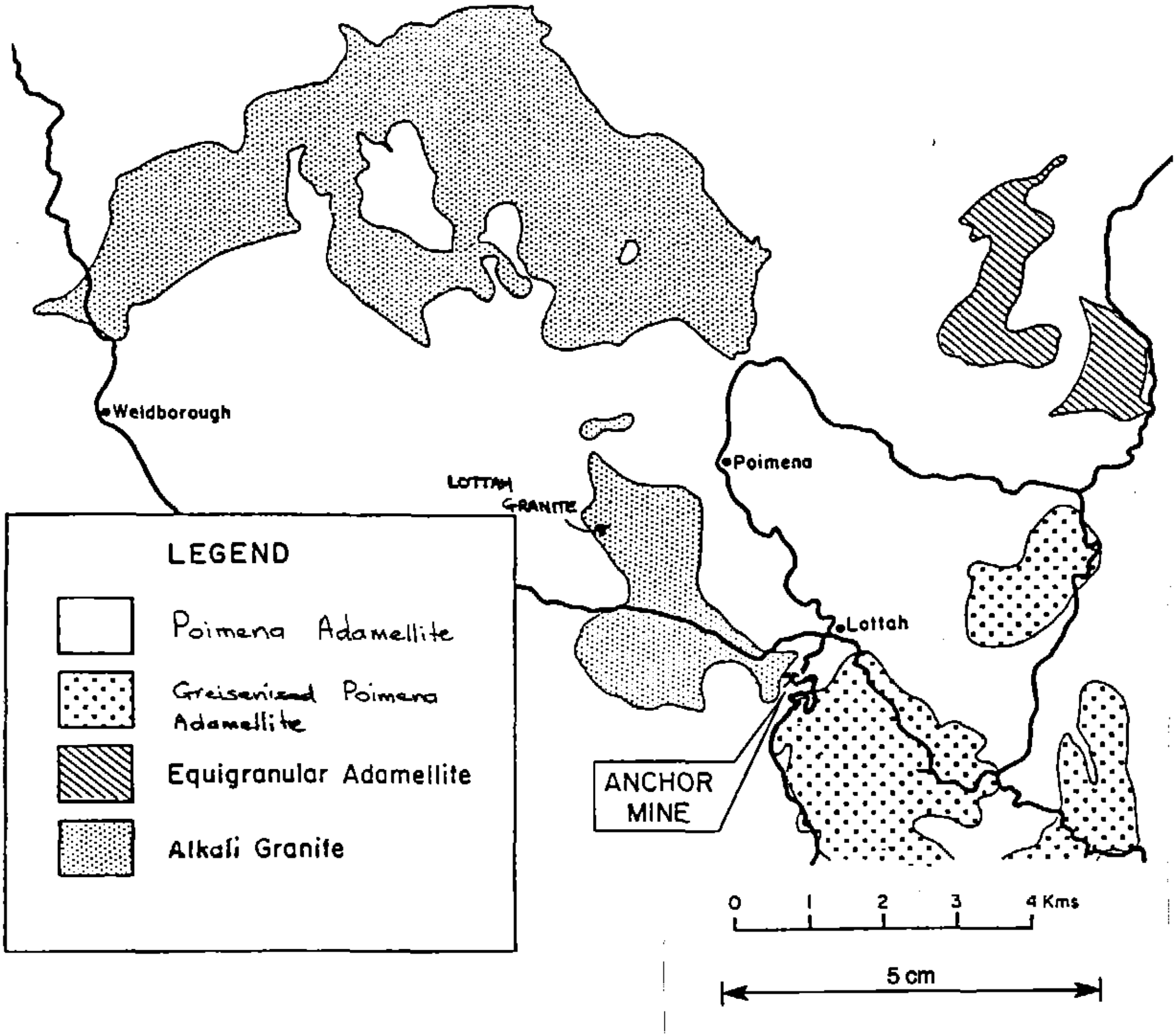

Figure 2. Geological map of the Central Blue Tier area

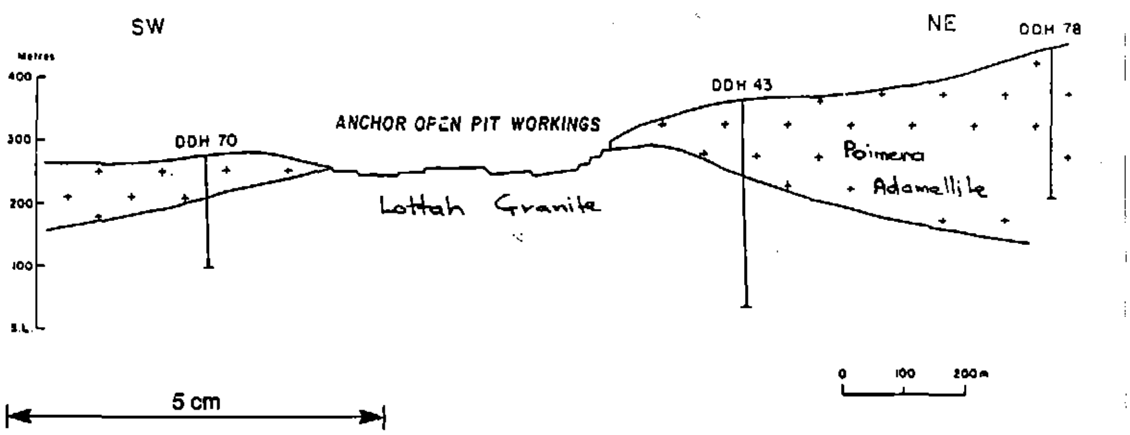

Figure 3. Cross section through the Anchor Opencut showing the relationship between the Poimena Adamellite and the Lottah Granite

An alteration zone occurs in and below the roof of the Lottah Granite and it is in this alteration zone that the Anchor orebodies occur. In places, the alteration has resulted in cassiterite bearing greisen which forms a technically typical deposit: see Appendix 2.

At the contact between the two granitic intrusions a narrow, extensive but non-continuous, zone of pegmatite has developed. Marginal pegmatites of this type are a feature of many greisen deposits and are known as stockscheiders.

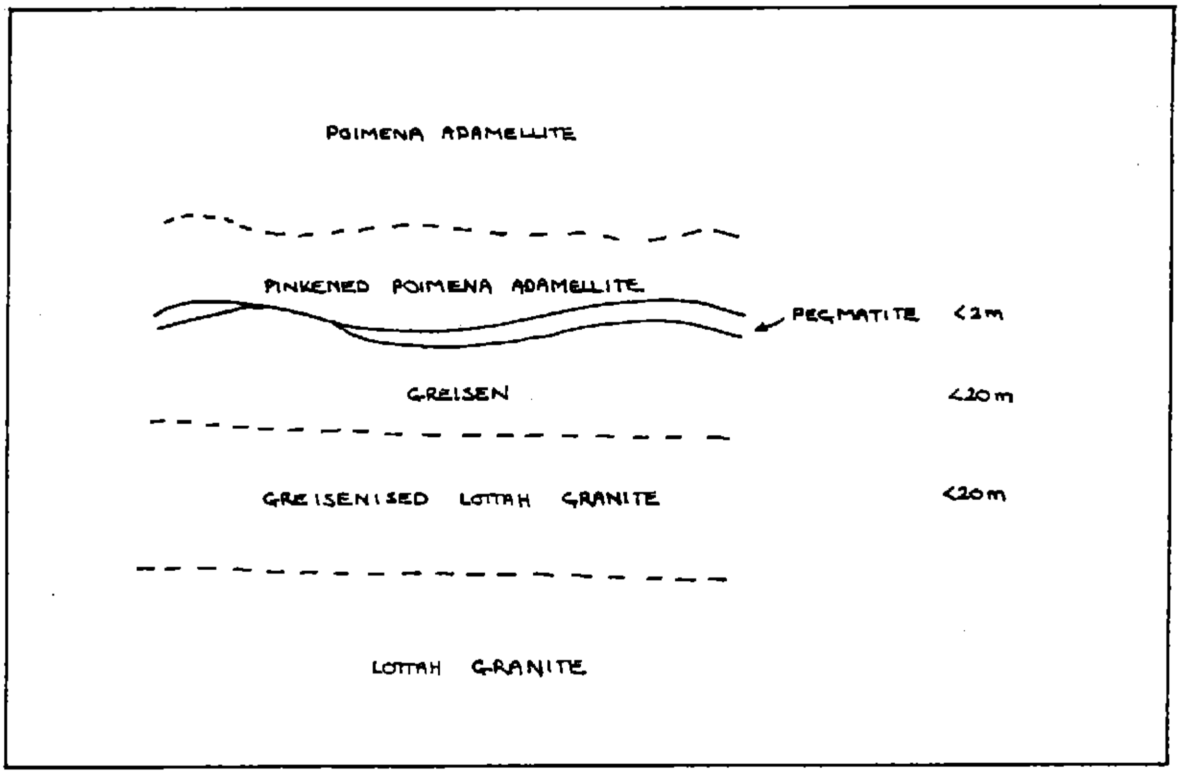

Figure 4. Sketch cross section of the contact between the Poimena Adamellite and the Lottah Granite showing the alteration styles near the contact; the broken lines represent diffuse contacts.

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3 TIN MINERALIZATION

Tin occurs as cassiterite within greisen zones which are developed within the Lottah Granite. The true greisen is a quartz-topaz-muscovite/sericite rock which has formed by alteration from the Lottah Granite. Ross (1983) describes two stages of alteration:

> "The first stage involved the replacement of magmatic minerals i.e. the replacement of feldspar and primary micas by topaz, yellow green to dark green siderophyllite and the introduction of cassiterite, accompanied by sporadic traces of fluorite. This phase may be regarded as greisenisation.
>
> "The second stage of alteration can be regarded as a low temperature hydrothermal stage in which some of the topaz and siderophyllite was replaced by fine sericite/muscovite, siderite and traces of sulphides."

Table 2 lists the most common minerals in paragenetic sequence.

TABLE 2

PARAGENETIC SEQUENCE OF ROCK FORMING MINERALS AT THE ANCHOR MINE

EARLY MAGMATIC MINERALS	quartz albite orthoclase pale biotite
LATE MAGMATIC MINERALS	muscovite topaz
GREISENISING MINERALS	siderophyllite topaz cassiterite fluorite apatite
HYDROTHERMAL MINERALS	sericite siderite sulphides

This table has been modified from Table 4.1 in Ross (1983).

The intensity of the alteration is reflected in the colour of the rocks. The unaltered Lottah Granite is a cream-white rock which, with alteration, progressively changes in colour through grey and light green in greisenised granite, to dark grey-green and black in the true greisen.

Traces of cassiterite occur within the greisen and greisenised granite. Not all the altered rocks contain cassiterite but the darker greisens tend to contain more cassiterite than the lighter coloured greisens and greisenised granites. The greisens also contain traces of sulphides and wolframite. See Table 3 for a list of the metal bearing minerals which have been identified.

TABLE 3

METALLIC MINERALS AT THE ANCHOR MINE

acanthite	galena
arsenopyrite	molybdenite
bismuthinite	proustite
bornite	pyrargyrite
cassiterite	pyrite
chalcocite	pyrrhotite
chalcopyrite	sphalerite
covellite	tetrahedrite
digenite	wolframite

Cassiterite generally occurs as disseminated grains and occasionally as larger crystals. The grains have been described as being greater than 40 microns in diameter and individual crystals up to 15 millimetres across have been observed in the underground workings.

Cassiterite is commonly intergrown with topaz, quartz and mica but rarely with sulphides. This pattern of intergrowths reflects the paragenetic sequence outlined in Table 3.

4 METALLURGICAL IMPLICATIONS OF THE ORE MINERALOGY

The mineralogy of the ores has direct implications for the production of a clean cassiterite concentrate. See Appendix 3 for a schematic drawing of the mill circuit.

The coarseness of the cassiterite grains means that high recoveries can be achieved with a relatively simple mill circuit. Fander (1977) describes the cassiterite crystals in diamond drill hole BT42 as a "metallurgist's delight". Ross (1983), following extensive metallurgical testwork by Renison Limited, stated that there is "significant liberation of cassiterite occurring above 400 microns." Good recoveries of cassiterite can be made by conventional gravity separation methods alone without recourse to more sophisticated recovery methods such as cassiterite flotation.

The presence of metal sulphides in the ore necessitates the use of sulphide flotation to remove them from the final cassiterite concentrate. This is achieved in the present circuit by a single sulphide float cell into which is fed the table concentrate.

The fact that the cassiterite and metal sulphide phases generally occur separately, i.e. there are few cassiterite/sulphide intergrowths, means that liberation produces very few composite cassiterite/sulphide grains. Consequently, losses of cassiterite during sulphide flotation should be low.

The mill circuit is not the same as the circuit installed in 1988/89. Several significant changes have been made e.g. the installation of a ball mill and extra Wilfley tables and the omission of the dry electrostatic and magnetic processes. No metallurgical mass balance was undertaken on this new circuit prior to the operation being placed on care and maintenance.

5 THE ORE LENSES

Spectrum Resources' underground mine was developed in what were thought to be two flat lying greisen lenses: an upper lens known as A Lens, and a lower B Lens. A Lens covers an area of approximately 40 000 square metres and is up to 20 metres in vertical thickness. B Lens lies at a lower level to the south of, and partly beneath, A Lens. B Lens covers an area of approximately 10 000 square metres and is up to 15 metres in vertical thickness. Views about the disposition and attitude of the ore lenses have changed as mining has proceeded.

In an internal report in 1991 McKeown wrote:

> "When the original mine plan was developed it was thought that both A Lens and B Lens were relatively flat, but uniformly dipping, tabular lenses. This was shown on the cross sections available at the time. Since then, underground exposures, further diamond drilling, structure contouring of the available data and reference to old reports have revealed that:
>
> - the A Lens and B Lens footwalls strike obliquely to the room and pillar layout;
>
> - the orebodies probably consist of antiform and synform structures developed parallel to the strike i.e. the orebody is undulose;
>
> - the hangingwall of A Lens tends to be clear cut and is marked by a pegmatite zone (the "stockscheider");
>
> - the footwall of A Lens tends to be diffuse but is probably controlled by flatly dipping joints;
>
> - the footwall and hangingwall of B Lens are both diffuse;
>
> - the large thickness of ore intersected in BT42 is probably part of a steeply dipping pipe." (McKeown, 1991a)

Not all of these are completely new ideas. In 1928, Reid and Henderson pointed out the undulose nature of the Anchor greisen zones: see Figure 5. Others, for example Lindsay Newnham (pers. comm.), have noted the existence of root zones beneath the flat lying greisens.

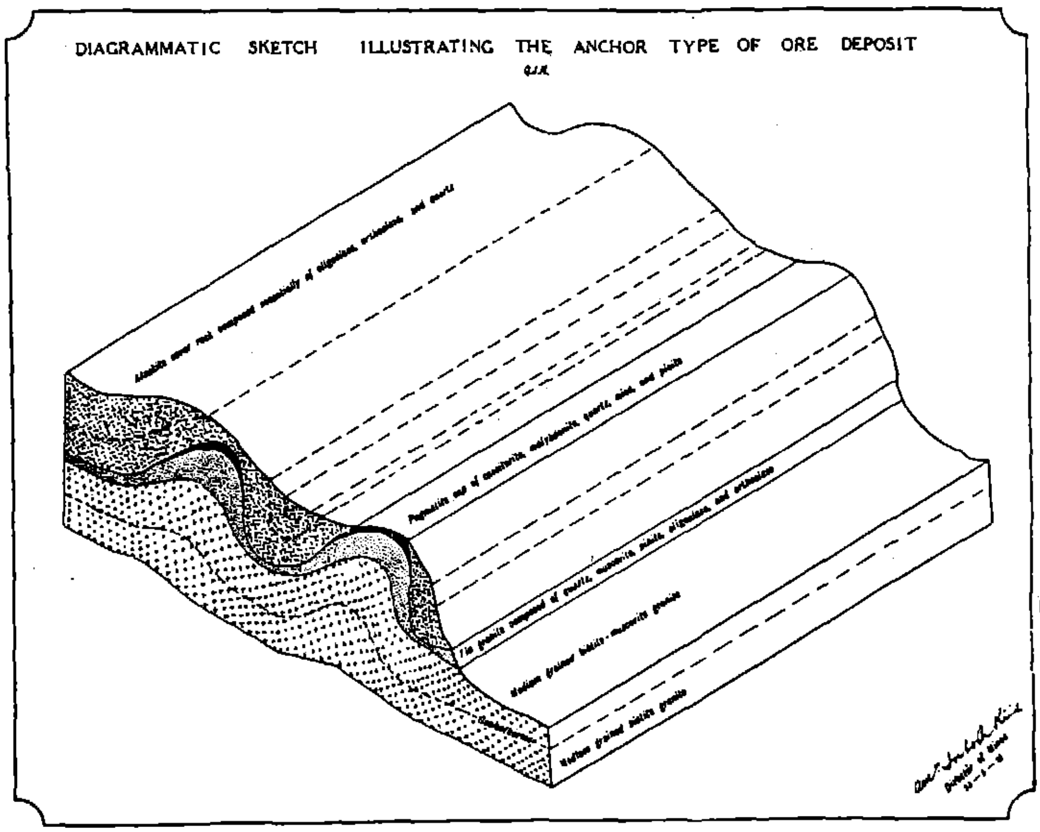

Figure 5. Isometric view of the "folded" nature of the granite intrusions at the Anchor Mine (Reid and Henderson, 1928).

This ore reserve estimate takes this newer interpretation of the geological structure into account. The interpretation used is shown on the 1:250 scale plans and sections kept in the Anchor Mine office. Several drawings are included to illustrate key points of the interpretation.

- the general shape of A Lens can be seen in the cross section along 5040E: see Figure 6.

- the location of the postulated pipe can be seen on the cross section along T drive: Figure 7. The interpretation of the drill holes on which the pipe is based is shown in an isometric view: Figure 8.

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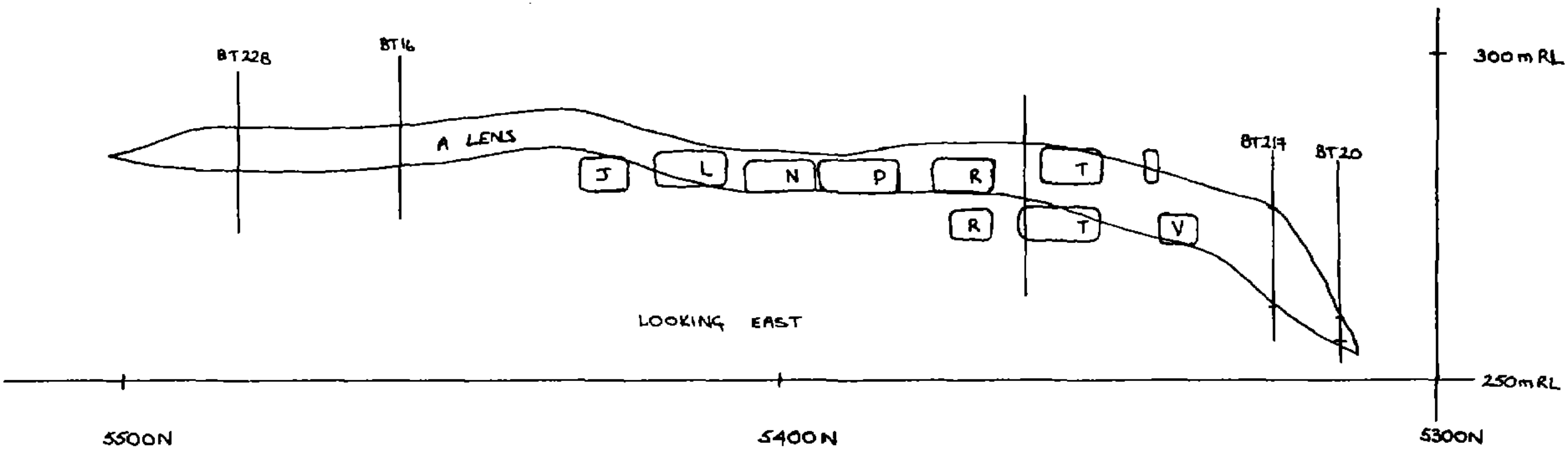

Figure 6. Cross section through A Lens along 5040E

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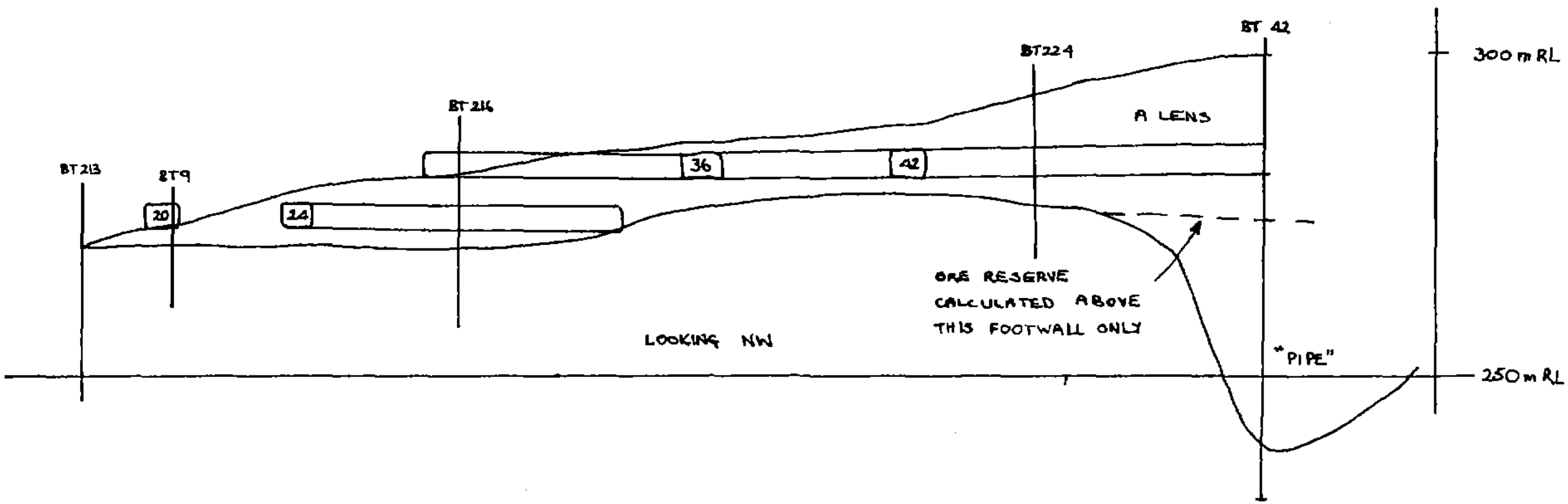

Figure 7. Cross section through A Lens along T drive

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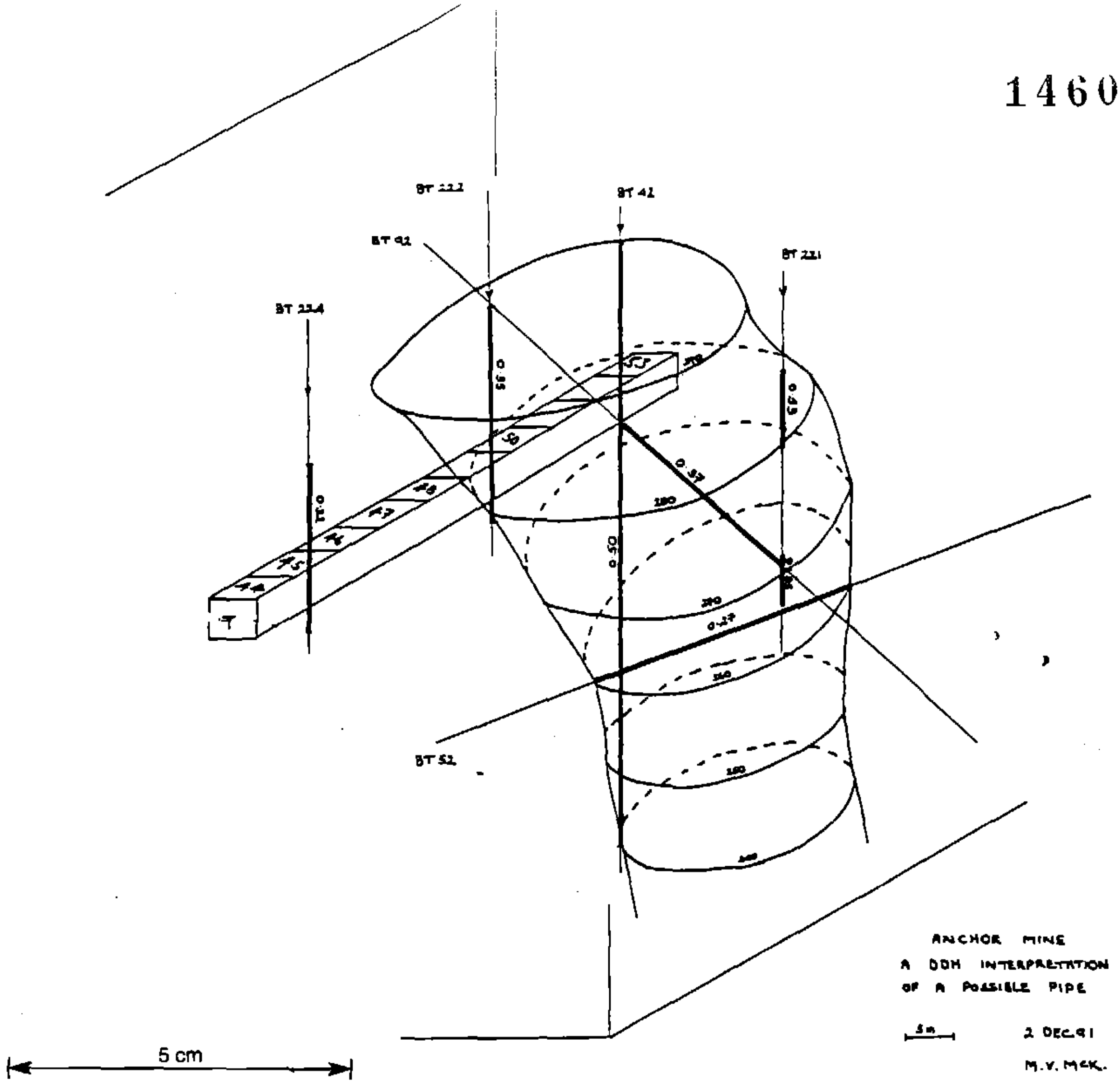

Figure 8. Isometric view of the interpreted pipe near BT42.

- the position of the underground workings relative to A Lens can be seen in the cross sections along 5040E and along T drive and R drive: Figures 5,8 an 9.

NOTE: Each development heading is identified by an alphanumeric code. The codes are based on a square grid. The convention is shown on Plan 1. The mining sequence is such that initial development is along drives with letter allocations e.g. F drive and V drive. Crosscutting, which forms the pillars, is identified by numbers e.g. 36 drive and 42 drive. Using this system, all 5m X 5m ore blocks can be identified by a combination e.g. V36 and F54.

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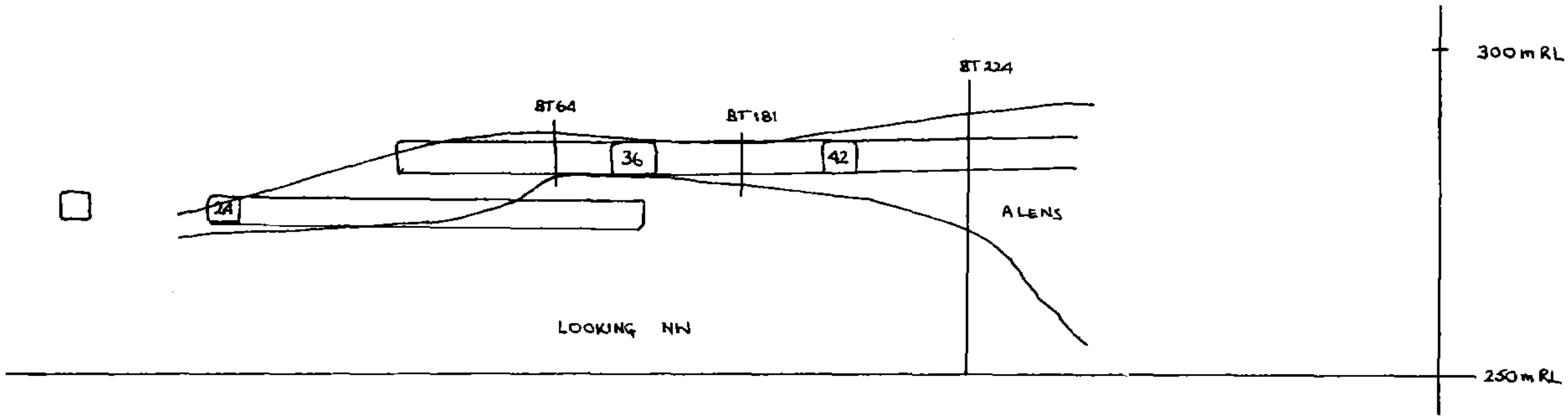

Figure 9. Cross section through A Lens along R drive

6 MINING

Mining has been by room and pillar. Rooms and pillars are planned to be 5m X 5m on a square grid: see e.g. Figure 13. Without pillar recovery, this leaves 25% of the ore in the ground as pillars. The method has proved workable and safe to date. The mining method was outlined by Mike Baker in The Anchor Mine Project Feasibility Report (1988a) as follows:

> " The relatively flatly lying ore body, and ease of access from the old open pit area indicates that a highly mechanised stoping system with very limited development is applicable.
>
> "The method selected is room and pillar with fill which becomes the post pillar mining system. The method is very flexible and barren or low grade areas can be left in situ. The method has been used successfully in other Tasmanian mines notably Renison and Dolphin Mine at King Island. The method involves a room and pillar floor plan with an undercut being made on the footwall. Fill is then introduced, and forms the floor for the next stoping level. A further stoping cut is made in the roof, ore extracted, and another level of fill introduced. This is continued until the hangingwall is reached."

The ore reserve tonnage calculations for this assessment have assumed the continuation of this mining method.

The original underground workings were developed in B Lens at a back RL of about 260m. An examination of the B Lens workings (McKeown, 1991b) revealed that the B Lens workings are, for the most part, at or near the base of B Lens. The B Lens greisen which is exposed, is patchy in its development and grade distribution. This patchiness may be a function of the level at which the greisen has been exposed i.e. close to the footwall. Figure 10 shows the places in the B Lens workings where greisen or greisenised granite is visible and also the footwall contours of the B Lens mineralisation, based on diamond drill hole intersections.

The original plan was to mine B Lens completely before moving to A Lens. The B Lens workings would have been filled and then mining of A Lens would take place. To mine A Lens first would have placed the mining of B Lens in jeopardy: the parts of A Lens workings which would lie above B Lens would have the potential to induce unsafe back conditions in B Lens. However, the grade of ore from B Lens was not profitable and mining moved to A Lens.

Mining in A Lens proceeded in the lower workings close to the A Lens footwall. This was the procedure originally proposed by Mike Baker (1988a) and would have resulted in the maximum ore

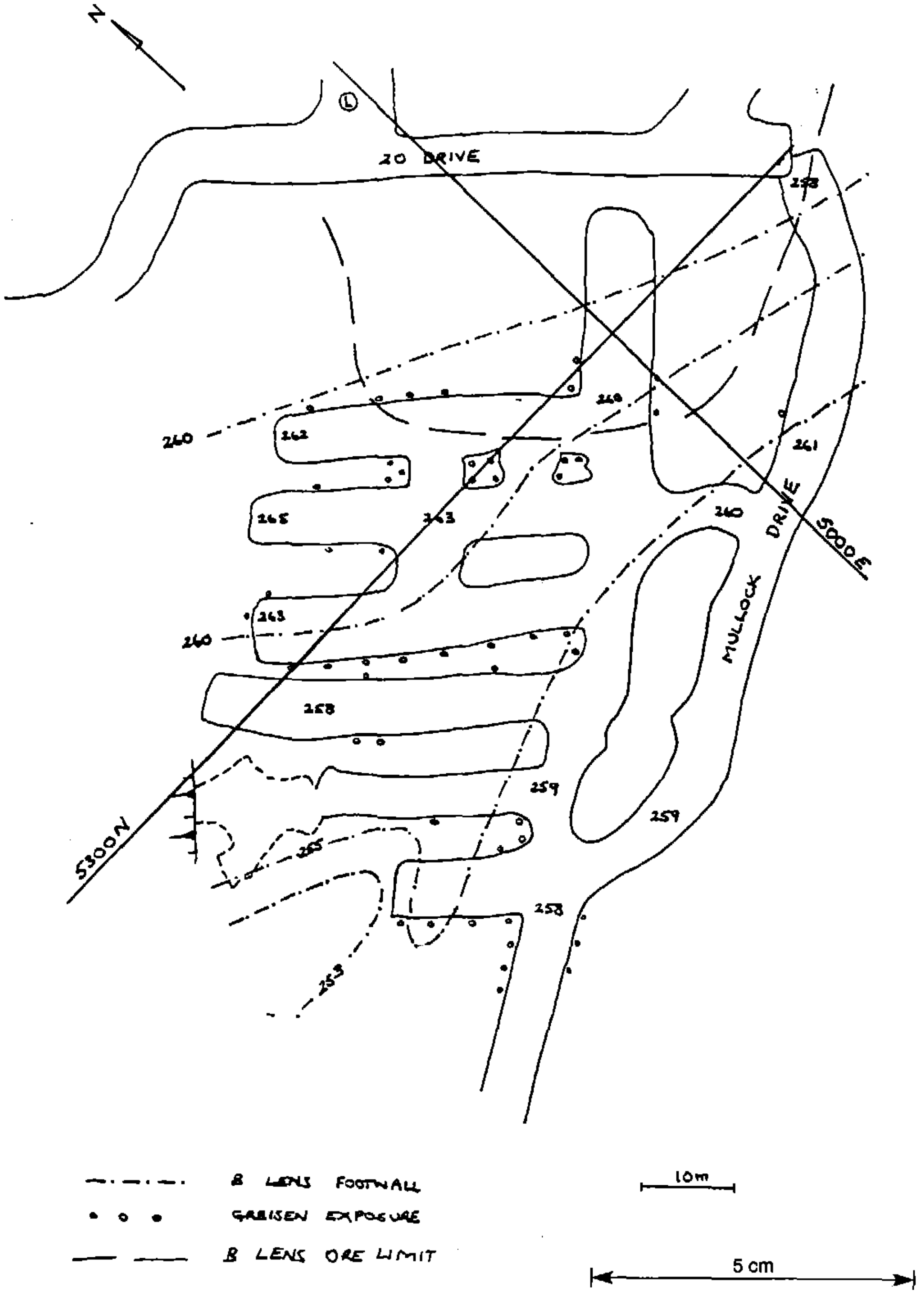

Figure 10. Plan of the B Lens workings with structure contours of the B Lens footwall.

extraction possible using room and pillar mining. Emplacement of fill into the workings was to take place before a stoping cut was made in the backs. Unfortunately, the grade of ore was again not proving profitable. The grade of ore from the lower workings was being diluted by the mining of greisenised granite beneath the footwall of A Lens: see, for example, Figure 6.

When mining of A Lens in the upper workings commenced the workings were totally within A Lens and mine head grade and tin output improved. As mining proceeded up-dip, some drives again passed through the footwall of A Lens into lower grade greisenised granite, and head grade fell.

A short diamond drilling programme and geological examination of the workings were undertaken in early 1991. As a result, changes were made to the location and priorities of development headings. With room and pillar mining, there is no scope to vary the plan positions of the development. The vertical location of the development can be varied, and as is explained above, this is a critical factor in maintaining acceptable tin grades.

Two important changes were made to enable a satisfactory head grade to be achieved. Firstly, the backs of the drives which were low relative to the ore footwall were stripped and the gradient of the drives was increased to conform with the footwall. The drives involved were those on the west side of the workings i.e. H, J and K drives.

Secondly, mining priority was given to the development on the east side of the mine where it was becoming obvious that tin grades were higher.

7 RESOURCE ESTIMATES

The method used for estimating the reserves and resources is described in this section. The reliability of these estimates depends on the variability of tin grades and the data on which the calculations are based. A brief discussion of the tin grade distribution is included and the criteria on which the estimates are based are summarised.

For this report, a new estimate of the tonnage and grade in A Lens was made. The B Lens tonnage and grade is taken from Lindsay Newnham's 1989 ore reserve estimate.

7.1 TIN GRADE DISTRIBUTION

Results of diamond drilling suggested that the distribution of tin grades through the orebodies was erratic: see, for example, Figure 11 which is taken from Ross (1983). However, a down hole variogram based on assays in BT52 reveals that there is some structure to the distribution of tin grades: see Figure 12. BT52 was drilled sub-horizontally at a bearing of approximately 065 degrees AMG in an attempt to test the

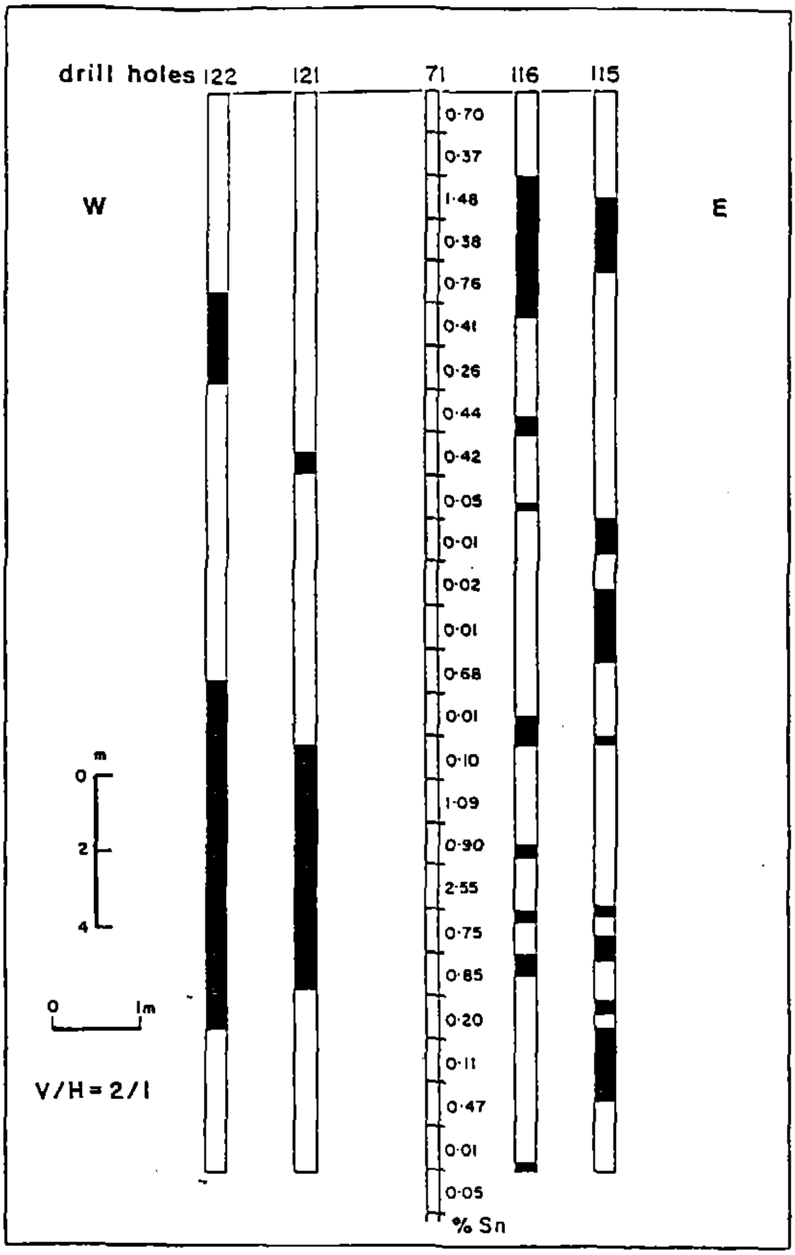

Figure 11. Cross section through closely spaced diamond drill holes illustrating the erratic distribution of cassiterite. BT71 is annotated with tin assays; the other holes are shaded where cassiterite was present (from Ross, 1983).

mineralisation intersected in vertical hole BT42. The hole passed beneath A Lens where it has now been mined: see Figure 8.

The implication from the variogram is that, along this bearing at least, diamond drill holes should be spaced at about 15 metre intervals if a good grade estimate is to be made for the orebody. In practice, Spectrum Resources has attempted to close up the drilling pattern to about 15 metre centres before mining of an area of A Lens. This not only gives a better estimate of the grade to be expected but, more importantly, a more reliable estimate of the position of the ore.

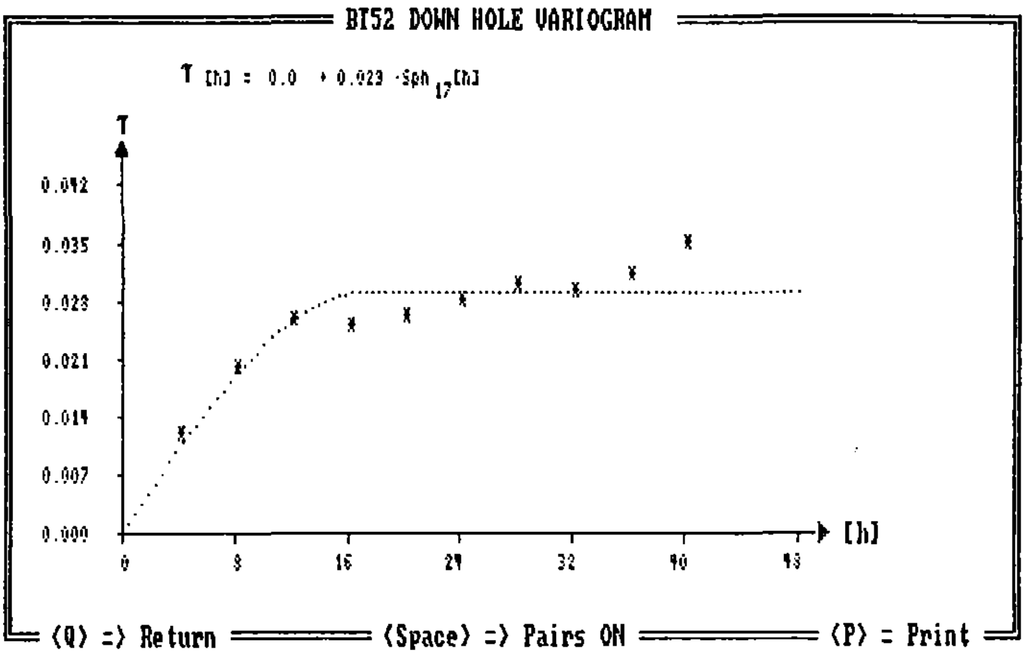

Figure 12. Down hole variogram along diamond drill hole BT52.

The lack of information regarding grade continuity should not affect the global grade estimate for the resource. The method used here for the estimation of the grade of ore reserve blocks has been used in the past at many mines e.g. Renison Ltd and Mt Lyell, and for previous resource estimates at the Anchor Mine. The method is explained in the next section of the report.

During mining it was the practice to take a sludge sample from blastholes. The results of the sludge sample assays and a discussion of the results is included in Appendix 4.

7.2 THE RESOURCE ESTIMATION METHOD

This resource estimate was calculated using the principles of the isoline method. This method was used for previous estimates by L.A. Newnham (1989). McKeown (1991c) described the basic steps in the isoline method as:

> "- plotting a plan showing the mid-points of all the diamond drill hole intersections annotated with the vertical thickness and grade of each intersection,
> - contouring the thickness and the grade of diamond drill hole intersections,
> - dividing the plan into regular blocks (at the Anchor mine the blocks are 20 m X 20 m),
> - assigning a grade and thickness for each block based on the grade and thickness contours,
> - calculating the tonnage for each block by multiplying the block area by the density of the ore ... ,
> - calculating the overall grade for an orebody by weighting the individual block grades by block tonnage."

The calculations here were complicated by the presence of the mine openings. The grade for each block was estimated as described above. The vertical thickness for each block was not. It was necessary to estimate up to four vertical thicknesses for each block:

- the thickness of the orebody itself, not including the ore in the pipe: see Figure 7.

- the thickness of ore remaining above mine development,

- the thickness of ore remaining below mine development, and

- the thickness of ore remaining between upper and lower A Lens mine development.

Each thickness was calculated by averaging thicknesses measured off the 1:250 cross sections drawn along the mid-lines of the mine drives. For each block, thicknesses were measured on the cross section in question and the cross sections adjacent to each side, and the three measurements were averaged. The thicknesses are shown on the calculation sheets in Appendix 1.

The resource estimate has been classified in the first instance into seven parts based on this procedure:

- resource tonnage: ore beyond the extent of the mine workings;

- existing pillar tonnage: ore within existing pillar outlines, from the footwall of the ore to the hangingwall;

- planned pillar tonnage: ore which lies within planned pillar outlines, from the footwall of the ore to the hangingwall, based on the present room and pillar mining method;

- remnant tonnage: ore which lies between existing or planned pillars, from the footwall of the ore to the hangingwall;

- tonnage above development openings: ore between the backs of drives and crosscuts and the hangingwall;

- tonnage below development openings: ore between the floor of drives and crosscuts and the footwall;

- tonnage between development openings: ore between the floor of the upper workings and the backs of the lower workings.

In addition to the estimate for the flatly dipping, generally tabular part of A Lens, an estimate of the resource based on the pipe has also been made. Figure 7 illustrates the lower limit of the A Lens tabular estimates; this is the upper limit of the A Lens pipe estimate.

For the estimate of tonnage in the pipe, the plan areas of the pipe from 240m RL to 270m RL were measured at 10 metre intervals and averaged. The average area was multiplied by the depth of the pipe and the specific gravity of the ore to arrive at a tonnage. Insufficient data exists for a grade estimate to be made for the mineralisation in the pipe.

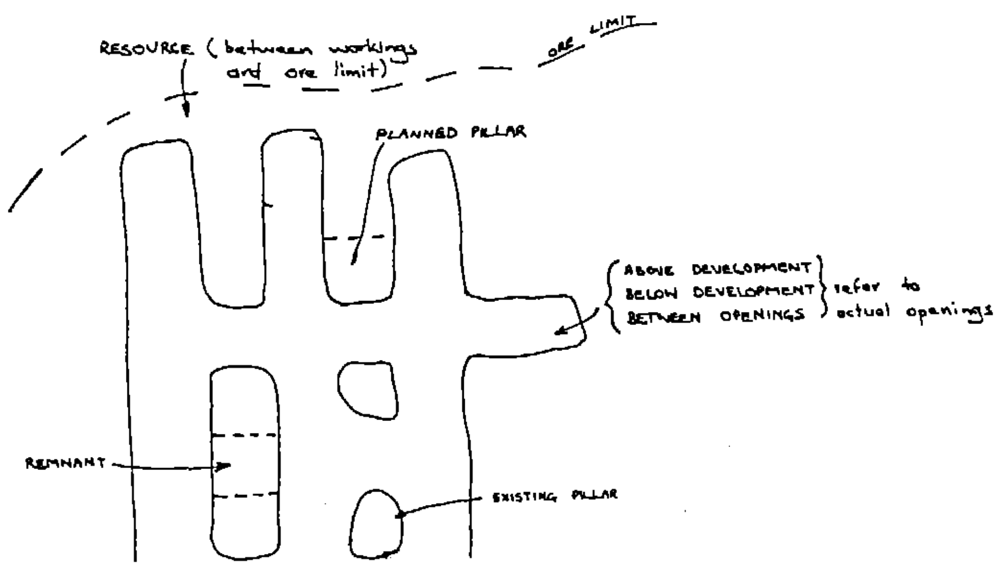

Figure 13. Sketch plan showing the classifications used for this ore resource statement

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7.3 THE RESOURCE ASSESSMENT CRITERIA

Data density: in a plan view of the orezones, diamond drill holes occur at centres about 15 to 20 metres apart in the area of and near the underground workings; ahead of the workings the hole spacing is about 50 metres.

Accuracy of location of samples: all drill hole collars are located by survey in AMG coordinates; all holes from BT40 onwards have been surveyed by down hole camera to determine drill hole deviation.

Accuracy of location of mine workings: the mine wrokings were surveyed on a regular basis during the operation of the mine. Unfortunately, a completely up to date survey of the mine is not available. The last survey of the mine was plotted on the 5th Nobember 1991, 6 weeks before operations were suspended (McGregor, 1991). The unsurveyed workings were sketched onto the sections. An accurate survey of the workings may result in slight changes to the allocation of tonnage above and below the existing workings.

Drilling technique: all holes used for the resource assessment were diamond drill holes: holes up to, and including, BT39 were drilled AXT size; holes from BT40 to BT180 were mostly drilled BQ size; holes from BT181 on were drilled NQ size.

Sampling technique: the sampling technique for holes BT1 to BT39 is not known; holes BT40 onwards were sampled by sawing the core in half lengthways.

Core recovery: core recovery in fresh rock has normally been in excess of 95%, with 100% recoveries generally being achieved in fresh rock in all holes from BT182 onwards. There is occasionally some core loss in the porphyry zone at the contact between the Poimena Adamellite and the Lottah Granite due to the occasionally clayey nature of the rock. However, this zone is narrow and any contribution to errors in grade estimation should, consequently, be small. The core loss in the porphyry zone is one indication that there is the potential for bad ground in parts of the mine: see Section 7.4 below.

Tonnage factors: the tonnage factor used for fresh ore is 2.65 tonnes per cubic metre and for weathered ore 2.50 tonnes per cubic metre; these are the densities used for assessments in 1988 (Baker, 1988b) and 1989 (Newnham, 1989).

Quality of assay data: most of the assays were carried out by Renison Limited in their laboratories at Renison Bell with the balance being undertaken at the Tasmanian Government Mines Department laboratories.

Quality of data description: core has been logged by Aberfoyle Limited and Renison Limited geologists and, during the Spectrum operations by Lindsay Newnham and Mick McKeown. All core has been logged in detail.

Geological interpretation: the geological interpretation has been described in sections 2,3 and 5 above.

Estimation technique: an isoline estimation technique has been used and is described in section 7.2 above.

Cut-off grades: strictly speaking a cut-off grade was not used; geological boundaries were identified in each drill hole based on the geological descriptions of the drill core, the assays of the drill core and comparisons with the underground exposures.

7.4 COMMENTS ON THE RESOURCE ESTIMATION

The isoline method gives a good estimate of the tonnage and grade of the global ore resource. It should not be used to select ore for mining based on a cut-off grade. The grade contours shown on the ore reserve plans are not a guide to the location of ore at any particular grade.

The ore reserve estimate is based on continuation of mining by room and pillar. The ore reserve estimate can be increased by using a different mining method or by modifications to the existing method.

A small amount of ore can be mined by opencut methods. This ore occurs immediately to the north of the mill site. The hangingwall of the ore has been exposed and sampled. The ore blocks which refer to this ore are highlighted in Appendix 1.

In general, in both A and B Lens workings, ground conditions have been such that little artificial support of the rock mass has been necessary so far. Ground conditions in the A Lens workings have never been bad enough to preclude mining although there are some specific, but restricted, areas where back conditions required some remedial action. The bad ground which is exposed in the mine occurs as a result of:

- weathering, where the mine is close to the surface;

- faulting associated with water, e.g. in F Drive and in the lower A Lens workings: small rock falls from clay zones along faults have been occurred and bolts, mesh and W strap have been installed;

- the occurrence of clay in the stockscheider (zone of porphyry) on the hangingwall of A Lens. Care needs to be exercised wherever the hangingwall is close to the backs of the mine. Special precautions will be required where water is present in the stockscheider.

No allowance has been made for ore which may not be mined due to poor ground conditions.

7.5 TONNAGES AND GRADES

The tonnages and grades for A lens have been calculated anew for this report. The tonnage and grade for B Lens has been taken from Lindsay Newnham's report of 25th August 1989.

TABLE 4

ANCHOR MINE ORE RESERVES, ORE RESOURCES
AND OTHER ORE OCCURENCES
AT 30 APRIL 1993

		tonnes	%Sn
ORE RESERVES			
A LENS ORE RESERVE			
PROVED	remnants	7 000	0.48
PROVED	between workings	4 000	0.62
PROBABLE	above workings	67 000	0.51
PROBABLE	below workings	35 000	0.51
TOTAL ORE RESERVES		113 000	0.51
ORE RESOURCES			
A LENS ORE RESOURCE			
INDICATED	beyond mine workings - u'ground	162 000	0.46
MEASURED	beyond mine workings - opencut	17 000	0.54
B LENS ORE RESOURCE			
INDICATED	beyond mine workings - u'ground	76 000	0.44
TOTAL ORE RESOURCES		255 000	0.46
OTHER ORE			
A LENS OTHER ORE			
	existing pillars	58 000	0.57
	planned pillars	6 000	0.45
TOTAL OTHER ORE		64 000	0.56
OTHER MINERALISATION			
A LENS OTHER MINERALISATION			
	A Lens pipe	85 000	not known

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APPENDIX 1

SPREADSHEETS OF TONNAGES AND GRADES

These spreadsheets refer to A Lens only. See Appendix 5 for B Lens estimates.

Each 20m X 20m ore block is identified by reference to the 5m X 5m ore block at its centre: see section 5 for an explanation of this nomenclature.

The blocks which may be mineable by opencut are identified by an asterisk.

146029

RESOURCE

		%Sn	thickness m	area square m	tonnes
AV*	30	0.5	5.0	100	1250
AV*	26	0.4	4.0	200	2000
AV*	22	0.4	4.0	200	2000
AZ*	30	0.5	4.5	300	3375
AZ*	26	0.7	4.0	400	4000
AZ*	22	0.6	4.0	300	3000
AZ	58	0.4	1.5	50	199
AZ	54	0.5	3.0	280	2226
AZ	50	0.1	5.0	400	5300
AZ	46	0.1	2.0	400	2120
AZ	42	0.2	3.0	250	1988
D	66	0.3	3.0	320	2544
D	62	0.4	3.0	320	2544
D	58	0.4	3.0	400	3180
D	54	0.6	4.5	400	4770
D	50	0.3	5.5	400	5830
D	46	0.3	6.0	400	6360
D	42	0.3	3.0	300	2385
D	38	0.2	2.0	330	1749
D	34	0.6	3.0	220	1749
D	30	0.8	5.0	320	4240
D	26	0.6	8.5	300	6758
D	22	0.6	8.5	320	7208
H	66	0.3	3.0	180	1431
H	62	0.3	3.0	400	3180
H	58	0.3	2.0	60	318
H	54	0.4	2.0	250	1325
H	50	0.4	4.7	400	4982
H	46	0.4	5.0	320	4240
H	42	0.5	3.5	100	928
H	38	0.3	7.0	5	93
H	34	0.6	9.5		0
H	30	0.9	6.0		0
H	26	0.8	8.0		0
H	22	0.6	9.0	330	7871
L	54	0.4	6.5	0	0
L	50	0.4	6.0	130	2067
L	46	0.4	5.0		0
L	42	0.7	6.5		0
L	38	0.6	7.0		0
L	34	0.8	8.0		0
L	30	0.8	8.0		0
L	26	0.6	10.0		0
L	22	0.3	7.0	100	1855
P	54	0.4	11.0	230	6705
P	50	0.4	10.0		0
P	46	0.5	9.0		0
P	42	0.7	7.5		0

P	38	0.9	6.0		0
P	34	0.5	6.5		0
P	30	0.6	10.0	230	6095
P	26	0.5	6.0		0
P	22	0.4	3.0	360	2862
T	54	0.4	17.0	250	11263
T	50	0.3	21.0	25	1391
T	46	0.3	15.5		0
T	42	0.5	10.5		0
T	38	0.7	9.0		0
T	34	0.6	11.0		0
T	30	0.7	10.5		0
T	26	0.4	8.5		0
X	54	0.5	8.0	50	1060
X	50	0.5	8.0	225	4770
X	46	0.4	8.5	185	4167
X	42	0.3	10.0	130	3445
X	38	0.6	14.0	140	5194
X	34	0.6	16.0	160	6784
X	30	0.6	16.5	300	13118
X	26	0.7	9.0	200	4770
X	22	0.5	3.0	200	1590
TOTAL		0.47			178275

REMNANTS

		%Sn	thickness m	square m	tonnes
AV*	30	0.5	5.0		
AV*	26	0.4	4.0		
AV*	22	0.4	4.0		
AZ*	30	0.5	4.5		
AZ*	26	0.7	4.0		
AZ*	22	0.6	4.0		
AZ	58	0.4	1.5		
AZ	54	0.5	3.0		
AZ	50	0.1	5.0		
AZ	46	0.1	2.0		
AZ	42	0.2	3.0		
D	66	0.3	3.0		
D	62	0.4	3.0		
D	58	0.4	3.0		
D	54	0.6	4.5		
D	50	0.3	5.5		
D	46	0.3	6.0		
D	42	0.3	3.0		
D	38	0.2	2.0		
D	34	0.6	3.0		
D	30	0.8	5.0		
D	26	0.6	8.5		
D	22	0.6	8.5		
H	66	0.3	3.0		
H	62	0.3	3.0		
H	58	0.3	2.0		
H	54	0.4	2.0		
H	50	0.4	4.7		
H	46	0.4	5.0	10	133
H	42	0.5	3.5	15	139
H	38	0.3	7.0		
H	34	0.6	9.5		
H	30	0.9	6.0		
H	26	0.8	8.0		
H	22	0.6	9.0		
L	54	0.4	6.5		
L	50	0.4	6.0	40	636
L	46	0.4	5.0	50	663
L	42	0.7	6.5	25	431
L	38	0.6	7.0		
L	34	0.8	8.0	20	424
L	30	0.8	8.0		
L	26	0.6	10.0	10	265
L	22	0.3	7.0		
P	54	0.4	11.0		
P	50	0.4	10.0	20	530
P	46	0.5	9.0		
P	42	0.7	7.5		

P	38	0.9	6.0		
P	34	0.5	6.5	60	1034
P	30	0.6	10.0		
P	26	0.5	6.0		
P	22	0.4	3.0		
T	54	0.4	17.0	25	1126
T	50	0.3	21.0	10	557
T	46	0.3	15.5		
T	42	0.5	10.5		
T	38	0.7	9.0		
T	34	0.6	11.0	25	729
T	30	0.7	10.5		
T	26	0.4	8.5		
X	54	0.5	8.0		
X	50	0.5	8.0		
X	46	0.4	8.5		
X	42	0.3	10.0		
X	38	0.6	14.0		
X	34	0.6	16.0		
X	30	0.6	16.5		
X	26	0.7	9.0		
X	22	0.5	3.0		
TOTAL		0.48			6665

BETWEEN DRIVES

		%Sn	thickness m	m	square m	tonnes
AV*	30	0.5	5.0			
AV*	26	0.4	4.0			
AV*	22	0.4	4.0			
AZ*	30	0.5	4.5			
AZ*	26	0.7	4.0			
AZ*	22	0.6	4.0			
AZ	58	0.4	1.5			
AZ	54	0.5	3.0			
AZ	50	0.1	5.0			
AZ	46	0.1	2.0			
AZ	42	0.2	3.0			
D	66	0.3	3.0			
D	62	0.4	3.0			
D	58	0.4	3.0			
D	54	0.6	4.5			
D	50	0.3	5.5			
D	46	0.3	6.0			
D	42	0.3	3.0			
D	38	0.2	2.0			
D	34	0.6	3.0			
D	30	0.8	5.0			
D	26	0.6	8.5			
D	22	0.6	8.5			
H	66	0.3	3.0			
H	62	0.3	3.0			
H	58	0.3	2.0			
H	54	0.4	2.0			
H	50	0.4	4.7			
H	46	0.4	5.0			
H	42	0.5	3.5			
H	38	0.3	7.0			
H	34	0.6	9.5			
H	30	0.9	6.0			
H	26	0.8	8.0			
H	22	0.6	9.0			
L	54	0.4	6.5			
L	50	0.4	6.0			
L	46	0.4	5.0			
L	42	0.7	6.5			
L	38	0.6	7.0			
L	34	0.8	8.0			
L	30	0.8	8.0			
L	26	0.6	10.0			
L	22	0.3	7.0			
P	54	0.4	11.0			
P	50	0.4	10.0			
P	46	0.5	9.0			
P	42	0.7	7.5			

P	38	0.9	6.0			
P	34	0.5	6.5			
P	30	0.6	10.0			
P	26	0.5	6.0			
P	22	0.4	3.0			
T	54	0.4	17.0			
T	50	0.3	21.0			
T	46	0.3	15.5			
T	42	0.5	10.5			
T	38	0.7	9.0			
T	34	0.6	11.0	4.0	300	3180
T	30	0.7	10.5	4.0	60	636
T	26	0.4	8.5			
X	54	0.5	8.0			
X	50	0.5	8.0			
X	46	0.4	8.5			
X	42	0.3	10.0			
X	38	0.6	14.0			
X	34	0.6	16.0			
X	30	0.6	16.5			
X	26	0.7	7.0			
X	22	0.5	3.0			
TOTAL		0.62				3816

ABOVE DRIVES

		%Sn	thickness m	m	square m	tonnes
AV*	30	0.5	5.0			
AV*	26	0.4	4.0			
AV*	22	0.4	4.0			
AZ*	30	0.5	4.5			
AZ*	26	0.7	4.0			
AZ*	22	0.6	4.0			
AZ	58	0.4	1.5			
AZ	54	0.5	3.0			
AZ	50	0.1	5.0			
AZ	46	0.1	2.0			
AZ	42	0.2	3.0			
D	66	0.3	3.0			
D	62	0.4	3.0			
D	58	0.4	3.0			
D	54	0.6	4.5			
D	50	0.3	5.5			
D	46	0.3	6.0			
D	42	0.3	3.0			
D	38	0.2	2.0	3.0	70	557
D	34	0.6	3.0	2.5	180	1193
D	30	0.8	5.0	4.0	80	848
D	26	0.6	8.5	6.0	100	1590
D	22	0.6	8.5	7	80	1484
H	66	0.3	3.0			
H	62	0.3	3.0			
H	58	0.3	2.0			
H	54	0.4	2.0			
H	50	0.4	4.7			
H	46	0.4	5.0	4.0	70	742
H	42	0.5	3.5	2.5	260	1723
H	38	0.3	7.0	1.0	295	782
H	34	0.6	9.5	2.5	310	2054
H	30	0.9	6.0	4.0	175	1855
H	26	0.8	8.0		220	
H	22	0.6	9.0	4.0	70	742
L	54	0.4	6.5	2.0	35	186
L	50	0.4	6.0	3.0	170	1352
L	46	0.4	5.0	2.5	280	1855
L	42	0.7	6.5	3.0	295	2345
L	38	0.6	7.0	3.0	310	2465
L	34	0.8	8.0	3.0	305	2425
L	30	0.8	8.0	2.5	300	1988
L	26	0.6	10.0	4.0	300	3180
L	22	0.3	7.0	4.5	200	2385
P	54	0.4	11.0	4.0	145	1537
P	50	0.4	10.0	4.0	290	3074
P	46	0.5	9.0	2.5	315	2087
P	42	0.7	7.5	1.5	315	1252

P	38	0.9	6.0	1.0	310	822
P	34	0.5	6.5	1.0	285	755
P	30	0.6	10.0	3.5	145	1345
P	26	0.5	6.0	3.0	300	2385
P	22	0.4	3.0	2.0	40	212
T	54	0.4	17.0	9.5	125	3147
T	50	0.3	21.0	9.0	265	6320
T	46	0.3	15.5	5.5	300	4373
T	42	0.5	10.5	2.5	310	2054
T	38	0.7	9.0	1.5	270	1153
T	34	0.6	11.0	0.5	300	398
T	30	0.7	10.5			
T	26	0.4	8.5			
X	54	0.5	8.0			
X	50	0.5	8.0	3.5	50	464
X	46	0.4	8.5	3.0	165	1312
X	42	0.3	10.0	2.0	220	1166
X	38	0.6	14.0	2.0	210	1113
X	34	0.6	16.0			
X	30	0.6	16.5			
X	26	0.7	9.0			
X	22	0.5	3.0			

TOTAL 0.51 66714

146037

BELOW DRIVES

		%Sn	thickness m	m	square m	tonnes
AV*	30	0.5	5.0			
AV*	26	0.4	4.0			
AV*	22	0.4	4.0			
AZ*	30	0.5	4.5			
AZ*	26	0.7	4.0			
AZ*	22	0.6	4.0			
AZ	58	0.4	1.5			
AZ	54	0.5	3.0			
AZ	50	0.1	5.0			
AZ	46	0.1	2.0			
AZ	42	0.2	3.0			
D	66	0.3	3.0			
D	62	0.4	3.0			
D	58	0.4	3.0			
D	54	0.6	4.5			
D	50	0.3	5.5			
D	46	0.3	6.0			
D	42	0.3	3.0			
D	38	0.2	2.0			
D	34	0.6	3.0			
D	30	0.8	5.0			
D	26	0.6	8.5			
D	22	0.6	8.5			
H	66	0.3	3.0			
H	62	0.3	3.0			
H	58	0.3	2.0			
H	54	0.4	2.0			
H	50	0.4	4.7			
H	46	0.4	5.0			
H	42	0.5	3.5			
H	38	0.3	7.0			
H	34	0.6	9.5			
H	30	0.9	6.0			
H	26	0.8	8.0			
H	22	0.6	9.0	1.0	70	186
L	54	0.4	6.5			
L	50	0.4	6.0			
L	46	0.4	5.0			
L	42	0.7	6.5			
L	38	0.6	7.0			
L	34	0.8	8.0			
L	30	0.8	8.0			
L	26	0.6	10.0			
L	22	0.3	7.0			
P	54	0.4	11.0	3.0	145	1153
P	50	0.4	10.0	2.0	290	1537
P	46	0.5	9.0	2.0	315	1670
P	42	0.7	7.5	1.5	315	1252

P	38	0.9	6.0	0.5	310	411
P	34	0.5	6.5	1.0	285	755
P	30	0.6	10.0			
P	26	0.5	6.0			
P	22	0.4	3.0			
T	54	0.4	17.0	5.5	125	1822
T	50	0.3	21.0	5.5	265	3862
T	46	0.3	15.5	5.5	300	4373
T	42	0.5	10.5	4.0	310	3286
T	38	0.7	9.0	3.0	290	2306
T	34	0.6	11.0			
T	30	0.7	10.5	2.5	330	2186
T	26	0.4	8.5			
X	54	0.5	8.0			
X	50	0.5	8.0			
X	46	0.4	8.5			
X	42	0.3	10.0			
X	38	0.6	14.0	7.5	210	4174
X	34	0.6	16.0	12.0	195	6201
X	30	0.6	16.5			
X	26	0.7	9.0			
X	22	0.5	3.0			
TOTAL		0.51				35172

			thickness	EXISTING PILLAR	
		%Sn	m	square m	tonnes
AV*	30	0.5	5.0		
AV*	26	0.4	4.0		
AV*	22	0.4	4.0		
AZ*	30	0.5	4.5		
AZ*	26	0.7	4.0		
AZ*	22	0.6	4.0		
AZ	58	0.4	1.5		
AZ	54	0.5	3.0		
AZ	50	0.1	5.0		
AZ	46	0.1	2.0		
AZ	42	0.2	3.0		
D	66	0.3	3.0		
D	62	0.4	3.0		
D	58	0.4	3.0		
D	54	0.6	4.5		
D	50	0.3	5.5		
D	46	0.3	6.0		
D	42	0.3	3.0		
D	38	0.2	2.0		
D	34	0.6	3.0		
D	30	0.8	5.0		
D	26	0.6	8.5		
D	22	0.6	8.5		
H	66	0.3	3.0		
H	62	0.3	3.0		
H	58	0.3	2.0		
H	54	0.4	2.0		
H	50	0.4	4.7		
H	46	0.4	5.0		
H	42	0.5	3.5	25	232
H	38	0.3	7.0	100	1855
H	34	0.6	9.5	90	2266
H	30	0.9	6.0	225	3578
H	26	0.8	8.0	180	3816
H	22	0.6	9.0		
L	54	0.4	6.5		
L	50	0.4	6.0		
L	46	0.4	5.0		
L	42	0.7	6.5	80	1378
L	38	0.6	7.0	90	1670
L	34	0.8	8.0	75	1590
L	30	0.8	8.0	100	2120
L	26	0.6	10.0	90	2385
L	22	0.3	7.0	100	1855
P	54	0.4	11.0		0
P	50	0.4	10.0	65	1723
P	46	0.5	9.0	85	2027
P	42	0.7	7.5	85	1689

P	38	0.9	6.0	90	1431
P	34	0.5	6.5	60	1034
P	30	0.6	10.0		
P	26	0.5	6.0	100	1590
P	22	0.4	3.0		
T	54	0.4	17.0		
T	50	0.3	21.0	100	5565
T	46	0.3	15.5	100	4108
T	42	0.5	10.5	90	2504
T	38	0.7	9.0	110	2624
T	34	0.6	11.0	75	2186
T	30	0.7	10.5	70	1948
T	26	0.4	8.5		
X	54	0.5	8.0		
X	50	0.5	8.0	25	530
X	46	0.4	8.5	50	1126
X	42	0.3	10.0	50	1325
X	38	0.6	14.0	50	1855
X	34	0.6	16.0	45	1908
X	30	0.6	16.5		
X	26	0.7	9.0		
X	22	0.5	3.0		
TOTAL		0.57			57916

			PLANNED PILLAR	
	%Sn	thickness m	square m	tonnes
AV* 30	0.5	5.0		
AV* 26	0.4	4.0		
AV* 22	0.4	4.0		
AZ* 30	0.5	4.5		
AZ* 26	0.7	4.0		
AZ* 22	0.6	4.0		
AZ 58	0.4	1.5		
AZ 54	0.5	3.0		
AZ 50	0.1	5.0		
AZ 46	0.1	2.0		
AZ 42	0.2	3.0		
D 66	0.3	3.0		
D 62	0.4	3.0		
D 58	0.4	3.0		
D 54	0.6	4.5		
D 50	0.3	5.5		
D 46	0.3	6.0		
D 42	0.3	3.0		
D 38	0.2	2.0		
D 34	0.6	3.0		
D 30	0.8	5.0		
D 26	0.6	8.5		
D 22	0.6	8.5		
H 66	0.3	3.0		
H 62	0.3	3.0		
H 58	0.3	2.0		
H 54	0.4	2.0		
H 50	0.4	4.7		
H 46	0.4	5.0		
H 42	0.5	3.5		
H 38	0.3	7.0		
H 34	0.6	9.5		
H 30	0.9	6.0		
H 26	0.8	8.0		
H 22	0.6	9.0		
L 54	0.4	6.5		
L 50	0.4	6.0	60	954
L 46	0.4	5.0	70	928
L 42	0.7	6.5		
L 38	0.6	7.0		
L 34	0.8	8.0		
L 30	0.8	8.0		
L 26	0.6	10.0		
L 22	0.3	7.0		
P 54	0.4	11.0	25	729
P 50	0.4	10.0	25	663
P 46	0.5	9.0		
P 42	0.7	7.5		

146042

P	38	0.9	6.0		
P	34	0.5	6.5	25	431
P	30	0.6	10.0	25	663
P	26	0.5	6.0		
P	22	0.4	3.0		
T	54	0.4	17.0	25	1126
T	50	0.3	21.0		
T	46	0.3	15.5		
T	42	0.5	10.5		
T	38	0.7	9.0		
T	34	0.6	11.0	25	729
T	30	0.7	10.5		
T	26	0.4	8.5		
X	54	0.5	8.0		
X	50	0.5	8.0		
X	46	0.4	8.5		
X	42	0.3	10.0		
X	38	0.6	14.0		
X	34	0.6	16.0		
X	30	0.6	16.5		
X	26	0.7	9.0		
X	22	0.5	3.0		
TOTAL		0.45			6221

146043

APPENDIX 2

DESCRIPTIVE MODEL OF TIN GREISEN DEPOSITS

from

United States Geological Survey Bulletin No. 1693: Mineral Deposit Models, D.P. Cox & D.A. Singer (eds), 1986.

DESCRIPTIVE MODEL OF Sn GREISEN DEPOSITS

146044

By Bruce L. Reed

DESCRIPTION Disseminated cassiterite, and cassiterite-bearing veinlets, stockworks, lenses, pipes, and breccia in greisenized granite (see fig. 44).

GENERAL REFERENCE Scherba (1970), Taylor (1979), Reed (1982), Tischendorf (1977).

GEOLOGICAL ENVIRONMENT

Rock Types Specialized biotite and(or) muscovite leucogranite (S-type); distinctive accessory minerals include topaz, fluorite, tourmaline, and beryl. Tin greisens are generally post-magmatic and associated with late fractionated melt.

Textures Common plutonic rock textures, miarolitic cavities may be common; generally nonfoliated; equigranular textures may be more evolved (Hudson and Arth, 1983); aplitic and porphyritic textures common.

Age Range May be any age; tin mineralization temporally related to later stages of granitoid emplacement.

Depositional Environment Mesozonal plutonic to deep volcanic environment.

Tectonic Setting(s) Foldbelts of thick sediments ± volcanic rocks deposited on stable cratonic shield; accreted margins; granitoids generally postdate major folding.

Associated Deposit Types Quartz-cassiterite sulfide lodes, quartz-cassiterite ± molybdenite stockworks, late complex tin-silver-sulfide veins.

DEPOSIT DESCRIPTION

Mineralogy General zonal development of cassiterite + molybdenite, cassiterite + molybdenite + arsenopyrite + beryl, wolframite + beryl + arsenopyrite + bismuthinite, Cu-Pb-Zn sulfide minerals + sulphostannates, quartz veins ± fluorite, calcite, pyrite.

Texture/Structure Exceedingly varied, the most common being disseminated cassiterite in massive greisen, and quartz veinlets and stockworks (in cupolas or in overlying wallrocks); less common are pipes, lenses, and tectonic breccia.

Alteration Incipient greisen (granite): muscovite ± chlorite, tourmaline, and fluorite. Greisenized granite: quartz-muscovite-topaz-fluorite, ± tourmaline (original texture of granites retained). Massive greisen: quartz-muscovite-topaz ± fluorite ± tourmaline (typically no original texture preserved). Tourmaline can be ubiquitous as disseminations, concentrated or diffuse clots, or late fracture fillings. Greisen may form in any wallrock environment, typical assemblages developed in aluminosilicates.

Ore Controls Greisen lodes located in or near cupolas and ridges developed on the roof or along margins of granitoids; faults and fractures may be important ore controls.

Weathering Granite may be "reddened" close to greisen veins. Although massive greisen may not be economic as lodes, rich placer deposits form by weathering and erosion.

Geochemical Signature Cassiterite, topaz, and tourmaline in streams that drain exposed tin-rich greisens. Specialized granites may have high contents of SiO (>73 percent) and K_2O (>4 percent), and are depleted in CaO, TiO_2, MgO, and total FeO. They are enriched in Sn, F, Rb, Li, Be, W, Mo, Pb, B, Nb, Cs, U, Th, Hf, Ta, and most REE, and impoverished in Ni, Cu, Cr, Co, V, Sc, Sr, La, and Ba.

EXAMPLES

Lost River, USAK	(Dobson, 1982; Sainsbury, 1964)
Anchor Mine, AUTS	(Groves and Taylor, 1973)
Erzgebirge, CZCL	(Janecka and Stemprok, 1967)

146045

GRADE AND TONNAGE MODEL OF Sn GREISEN DEPOSITS

By W. David Menzie and Bruce L. Reed

COMMENTS See figs. 45, 46.

DEPOSITS

Name	Country	Name	Country
Altenberg	GRME	Coal Creek	USAK
Anchor	AUTS	E. Kempville	CNNS
Archer	AUTS	Hub	CZCL
Cinovec	CZCL	Potosi	BRZL
Cista	CZCL	Prebuz	CZCL

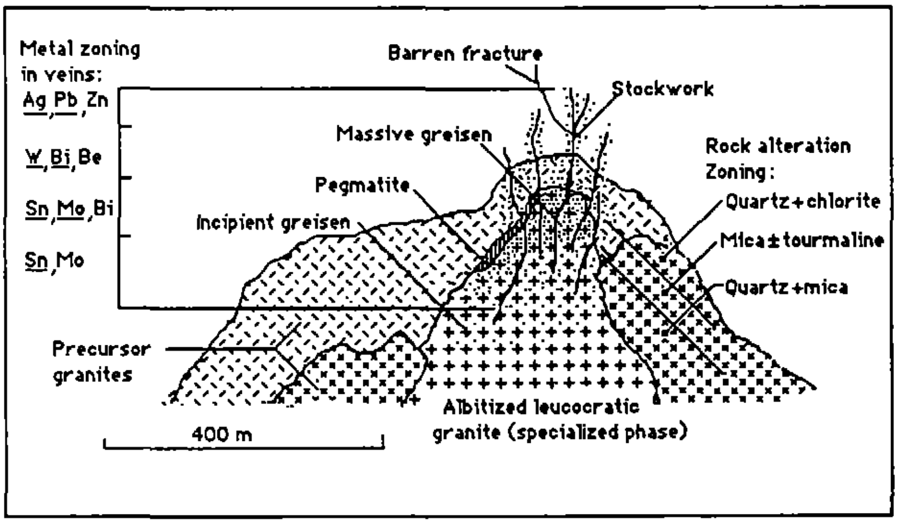
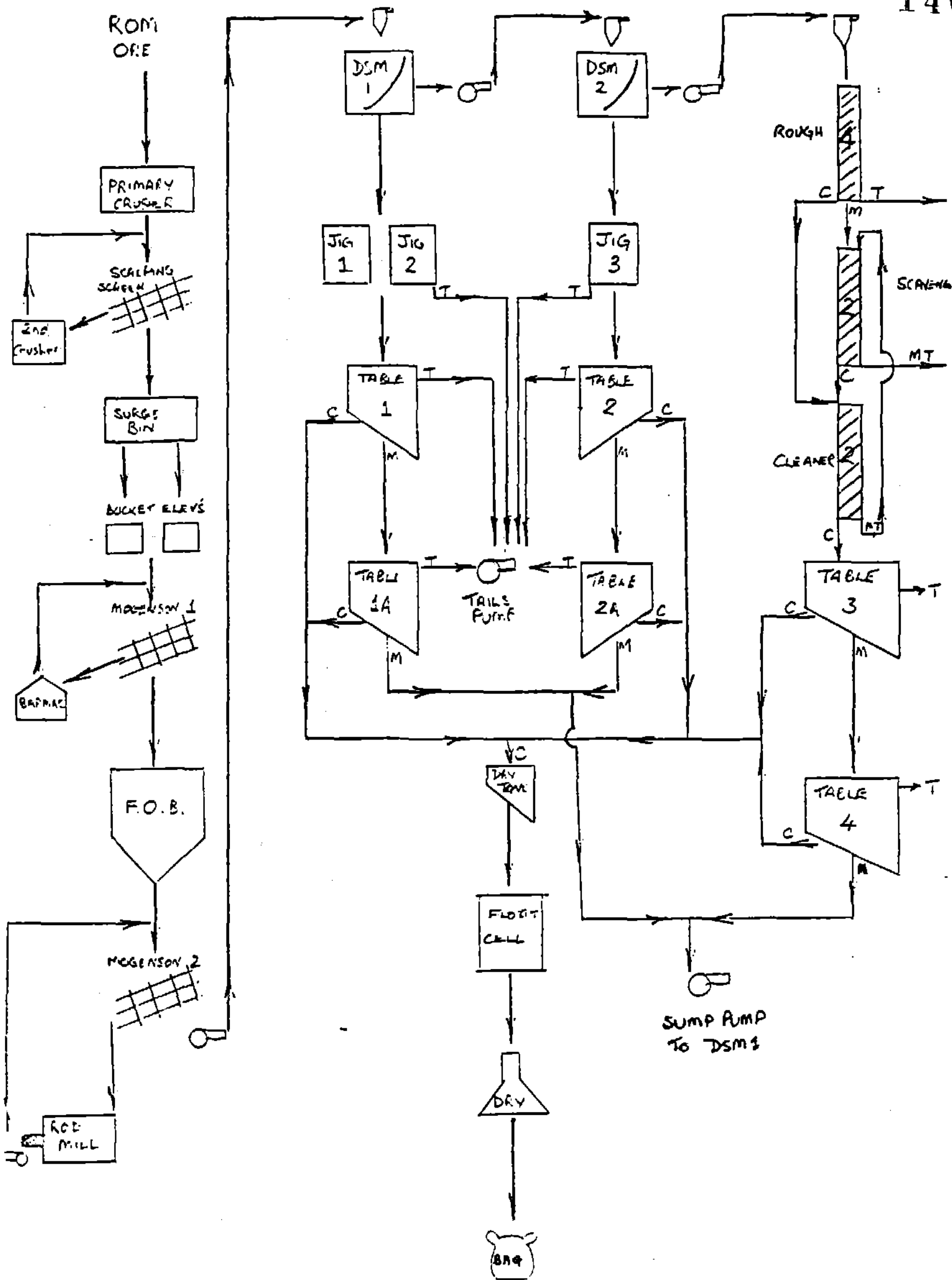


Figure 44. Cartoon cross section of a Sn greisen.

5 cm

APPENDIX 3

SCHEMATIC MILL CIRCUIT AT 24TH OCTOBER 1991



Schematic mill circuit at 24th October 1991

APPENDIX 4

DEVELOPMENT SLUDGE SAMPLES

DEVELOPMENT SLUDGE SAMPLES

During mining it ws the practice to take a sludge sample from blastholes. The sludge samples were dried, split, pulverised and analysed at the mine site using a bench top XRF analyser. See below for a brief description of the bench top analyser. Multi-directional variogram analysis of these samples showed that along a bearing of 015 degrees there was some correlation of tin grades: see Figure 1.

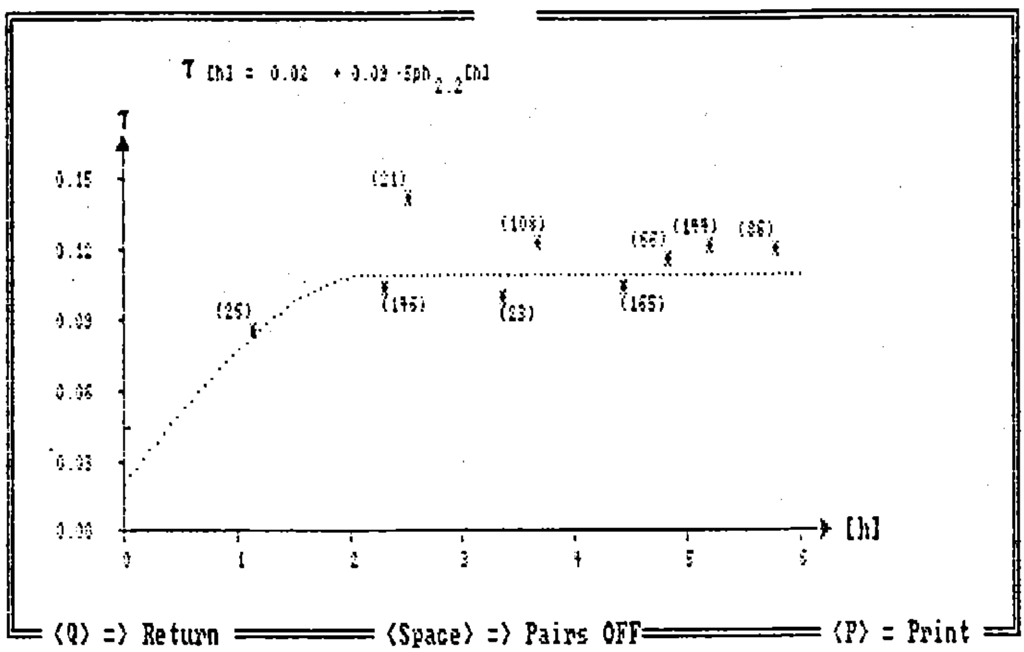


Figure 1. Variogram of sludge samples

The experimental variogram also had a low nugget effect: see Figure 1. The low nugget effect suggests that the sampling error due to sample collection and preparation was low i.e. the samples were 'good' samples. However, at the time the mine closed the reliability of sludge sampling as a guide to mine head grade had not been established and was a topic of much discussion. This is one facet of the operation which requires future examination.

Assays of the sludge samples are plotted on the schematic mine plans shown in Figures 2 and 3.

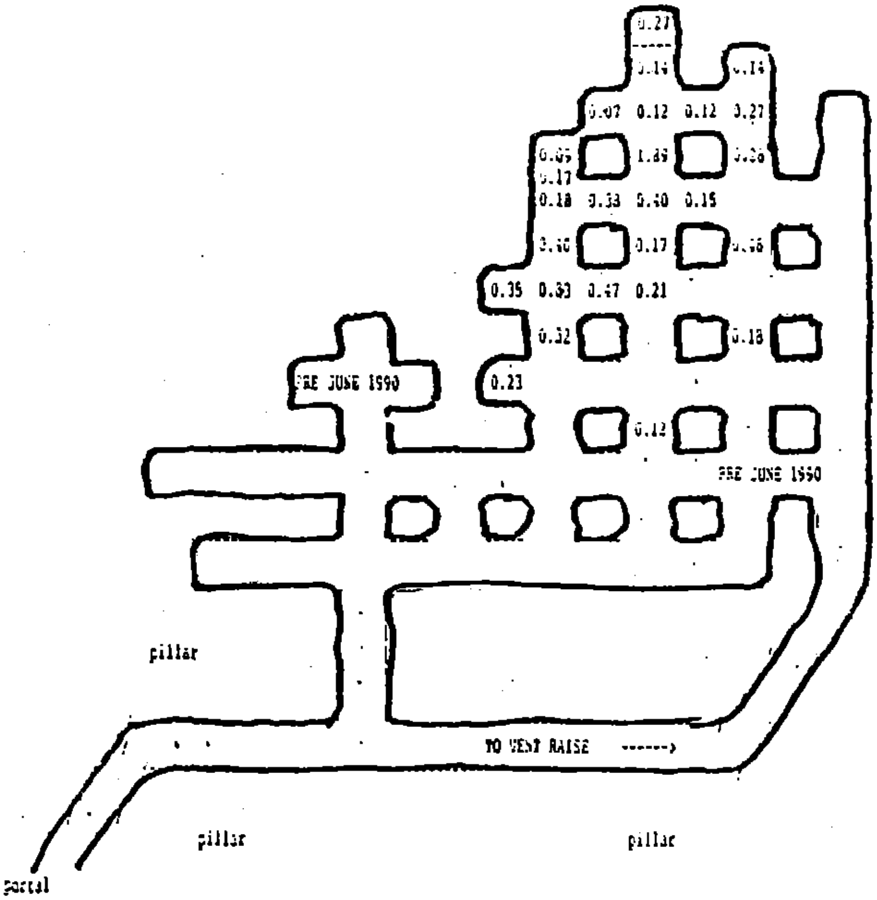


Figure2. Sketch plan of tin assays from development sludge samples collected in A Lens lower workings.

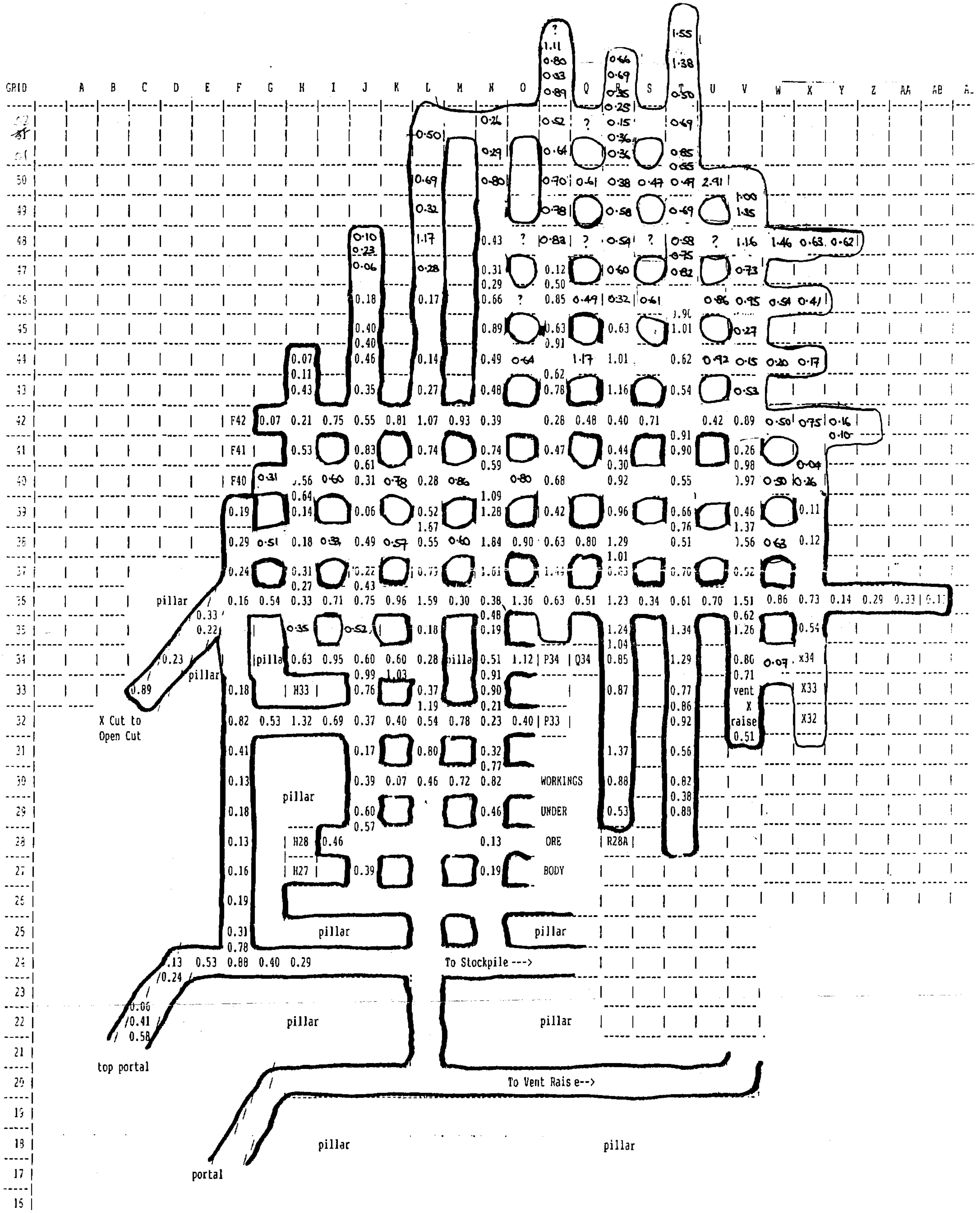


Figure 3. Sketch plan of tin assays from development sludge samples collected in A Lens upper workings.

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INTRODUCTION

The MCI Tin Analyser is designed to determine the tin content in drill sludge.

The function of the instrument is controlled primarily by the three position access key at the front of the unit, which controls the degree of access that the operator has to the instrument.

All aspects of the operation of the instrument are controlled by a microprocessor which receives its instructions from the keypad on the front panel.

Tin Determination

When used for tin determination in pulverised samples, the analyser operates on the principle of x-ray fluorescence.

The sample is exposed to gamma rays from a 10mCi Americium-241 source. Gamma rays from the source excite tin K_α x-rays from the sample. In addition, the incident gamma rays are backscattered and detected simultaneously with the tin x-rays. These two types of x-rays are at different energies and are resolved into two x-ray channels in the analyser. Channel one gives the count-rate of tin x-rays, and channel two the backscattered gamma rays which are used for matrix correction.

Tin Determination (cont)

In operation the pulverised sample is packed into the sample holders supplied. The sample holder is placed on the sample tray and rotated into the x-ray beam.

Upon initiation of the analysis, by pressing the RUN button, x-rays from the sample are detected and counted for a preset time period. (15 - 20 sec)

When the count is complete the percentage tin content is automatically calculated from a pre-determined calibration equation, and displayed on a liquid crystal display as percentage by weight.

146054

APPENDIX 5

B LENS ORE RESERVE

from

a fax from L.A. Newnham to M. Baker, 22nd September 1988.

146055

SPECTRUM RESOURCES AUSTRALIA PTY. LIMITED

FAX: (004) 242036
PH: (004) 245929
ADDRESS: P.O. Box 1057,
DEVONPORT. TAS 7310

FROM: LINDSAY NEWNHAM

TO: MIKE BAKER. FAX:

DATE: 22 SEP 89 :

PAGES (Including this one): 1 +

Mike:

① Anchor Ore Reserves:

When I recently re-estimated the Reserves of the Western half of the main Anchor ore zone, the mathematically more precise figures I obtained were:

	Tonnes	% Sn.
Open-Cut :	29,132	0.59
Upper or Main Zone :	419,495	0.46
Lower or Root Zone :	75,207	0.44
TOTAL	523,834	Average: 0.46.

I also wrote on 25th Aug. a memo. outlining my re-interpretat[ion] of the geology of the deposit used as a forerunner to estimating this

14C01C

In my August activity report, I rounded these figures as follows

Open-cut :	25,000 t.	0.5 – 0.6 Sn
Main Body :	420,000	0.4 – 0.5 Sn
Root Zones :	80,000	0.4 – 0.5. Sn

It is now accepted practice to attempt to indicate the reliability or confidence in the estimate by expressing the assay grade as either ± a certain percentage or ~~lying~~ within a range – same thing.

With a view to all the now existing data at Anchor, I would choose to express the more mathematically precise data above as

523,834 Tonnes at 0.46% Sn ± 10%.

A few points need to be made :

(a) The reserve is an in-situ estimate which conforms to the mine layout plan. However I have NOT subtracted a pillar support figure, nor have I added on a dilution figure

(b) Undoubtedly in a body such as this one where we are attempting to mine high grade zones within a much lower grade body, there will be significant dilution, but it is important to remember that the dilution will also carry significant tin, probably of an average grade of around 0.2-0..

(c) With further drilling, additional reserves will be defined. There are already drill indicated resources adjacent to the reserve detailed, but they have not been classified as reserves because I do not consider the data density sufficient to do so. These additional tonnes are likely to come from two sources viz. lateral extensions to the North and East of the main zone, and additional lower ~~&~~ root zones North-East of the already defined one. No mine plan has yet been devised for these additional resources, but a simple extension of the current layout together with appropriate ventilation systems seems appropriate and straightforward.

(d) A core drilling program within the 89-90 budget has been designed to test these additional resources, but I would not anticipate the grade to be significantly different to that of the existing reserves.

(e) The grade of 89-90 production can be lifted above that of the average grade by doing two things:

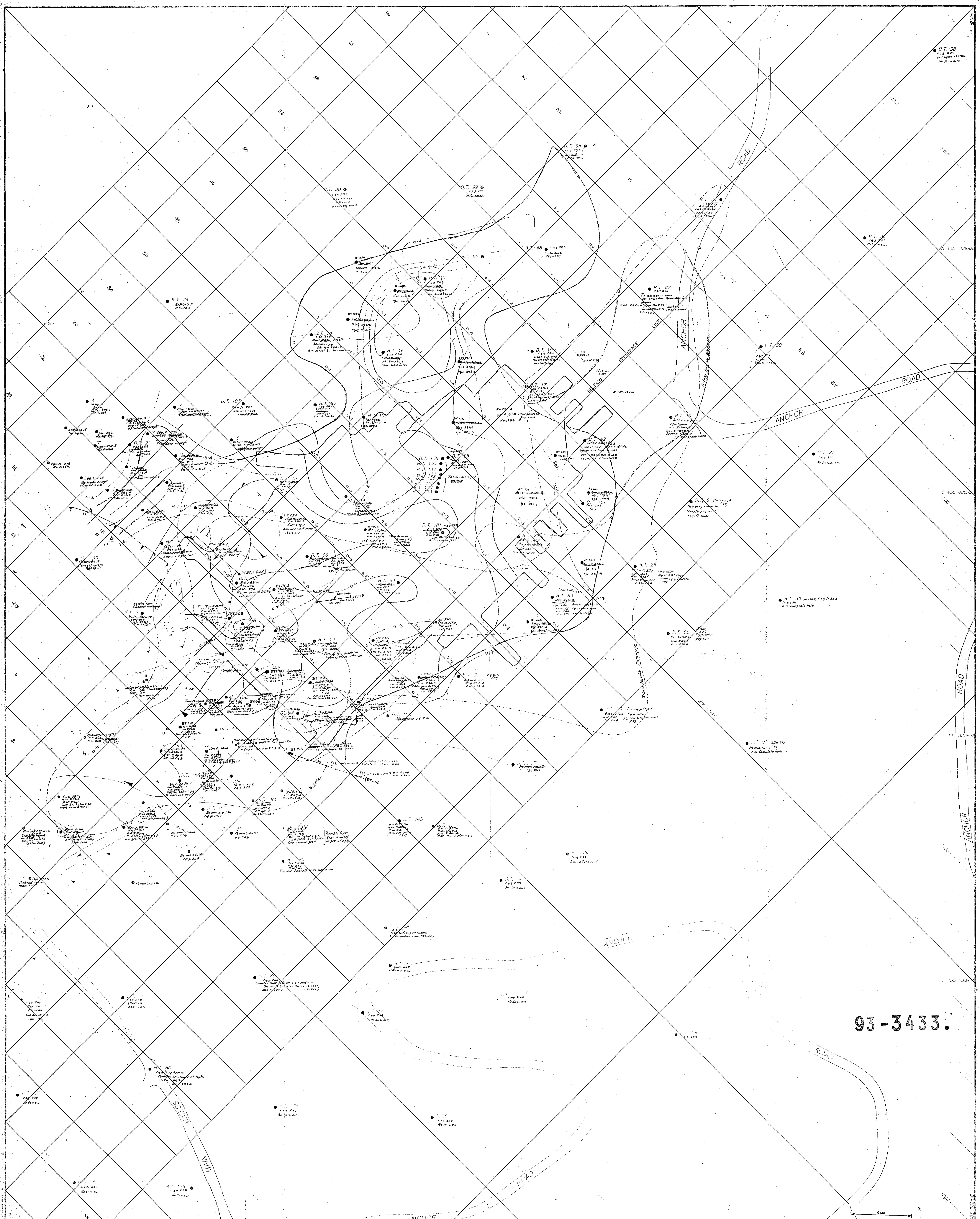
(i) accelerate development from the upper portal.
(ii) mine the open-cut reserve.

(i) Work on the upper portal is progressing well but has by necessity to be reasonably slow in order to take account of the not very good ground conditions around the portal and to avoid damage to the mill.

I have laid out a design for the first leg or "mill access" from this portal and discussed it with Pete and Ian. It is essentially as per the original mine plan but with a few gradient modifications to take account of the new drilling data.

(ii) I cannot see the open cut producing before Summer. Additional timber will have to be removed by Forestry Commission (high value veneer logs) and it will take a while to develop an appropriate underground ore pass system off the upper portal sill drive. Further the Anchor area is predictably wet until late November - December, and removal of that waterlogged sand overburden will be impossib in anything but the driest weather.

(f) If you on the Board require it, I can write a full and comprehens ore-reserve and resource report in accordance with the new AIM reporting code. That will take some time, and will be done at the expense of other work. Please advise if you require such a report



SPECTRUM RESOURCES AUSTRALIA PTY. LTD.

ANCHOR MINE

ORE RESERVE APRIL 93

93-3433.

DRAWN BY :	L.A.N.
DRAFTSMAN:	T.G.D.S.
DATE	APR 93
REVISIONS	
FILE No.	
FIG.	

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