

The Anchor Mine Operation

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ABSTRACT

This paper reviews the development and operation of a small underground tin mine in NE Tasmania. Tin mineralisation was discovered in the Lottah area in the 1880's with early alluvial and eluvial workings leading to the development of large low grade open pit operations. Apart from short lived activities operations ceased in 1915. Intensive exploration in 1978-81 outlined substantial tonnages of low grade tin mineralisation (0.2% Sn) of open pit material. An underground ore body was also located (800 000 tonnes 0.6% Sn). Spectrum Resources Australia Pty Limited acquired a 98 percent interest in the property in 1988. Mine and mill development began in early 1989 based on an underground mine utilising trackless mining using room and pillar with fill stoping. The milling circuit involves primary jaw crushing, Barmac secondary crushing to 1mm, Mogensen and DSM screens with jigs and tables to make a wet concentrate. Final tin concentrates are produced by high tension and magnetic separation. Planned production rates are 500 t/day with a mine life of 6-7 years.

INTRODUCTION

The Anchor mine is situated in the vicinity of Lottah on the flanks of the Blue Tiers range in NE Tasmania some 20km NW of St Helens along the Tasman highway and then 5km by gravel road to the mine site. Although in an area of moderate rainfall (100mm) the mine itself at an elevation of 300m is subject to heavy rainstorms as moisture laden westerly winds rise up over the higher ground. The location of the mine is shown in Figure 1.



FIG.1 Location Plan

Mining is believed to have commenced in the Blue Tier area in 1874 with exploitation of rich alluvial deposits. Later these extended from the river beds into in-situ detrital deposits overlaying primary tin bearing greisens. Large scale sluicing

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operations denuded the surface exposing the greisen ore bodies. Mining operations were then established to undertake hard rock mining by open pit techniques about 1884.

Operations around the Anchor mine were also undertaken at Crystal Hill, the Liberator and the Don and also in the Australia group of workings. The estimated hard rock production of the Anchor area is two million tonnes at a recovered grade of 0.2%Sn. The Anchor mine was the predominant producer of 1,750,000 tonnes. The period of greatest activity extended from 1895 to 1914 with sporadic tribute working and then a more sustained effort between 1939 and 1945.

The area attracted interest from time to time with Aberfoyle completing 39 short drill holes between 1964 and 1966. Renison carried out an intensive drilling programme between 1977 and 1981 comprising 71 exploration holes and 15 large core holes for metallurgical purposes.

Spectrum Resources Australia Pty Limited became interested in the property in 1986 with a view to developing higher grade zones by underground methods. A 98 percent interest was acquired in late 1988 with Nargun Pty Limited having the remaining interest. Operations are carried out by the Blue Tier J.V. and consist of an underground mine, gravity mill, and tailings disposal facility.

GEOLOGY

The tin mineralisation is hosted by an altered alkali granite which is overlain by an older coarser grained adamellite granite. A stanniferous zone of +500ppm Sn has been delineated by old workings and drilling. The zone trends northeast and has dimensions 800m by 200m by 40-60m thickness. A high grade tin zone lies immediately below an anticlinal or cupola structure of the alkali granite-adamellite granite contact and this zone forms the basis for the mining operation.

The ore zone averages 7-9m thickness with localised "root" zones extending into the footwall to give vertical intervals in excess of 20m. The hanging wall is often indicated by a band of pegmatite with the footwall being defined by assay cut off. A small secondary cupola is present where there is an enrichment of copper-zinc-silver, but only marginal tin grades.

The alkali granite has been altered to greisen in two phases. The first is believed to be late magmatic and involved the replacement of feldspars and primary micas by topaz and yellow-dark green siderophyllite accompanied by cassiterite, purple fluorite and trace apatite. The second phase is regarded as post-magmatic involving a low temperature hydrothermal phase where some topaz and siderophyllite was replaced by fine sericite, siderite, and sulphides introduced. Rare element deposition (Sn, W, Nb, Ta, Te, Be, Mo, REE) is consistent with deposits formed in the apical zones of the late stage intrusives from polyphase intrusive complexes. Ross (1981) examined the trace element distribution in the final report on the Renison exploration programme. Reid and Henderson (1928) have also reported on the Anchor deposit.

Orebody Structure

The orebody is shallow with cover being some 25m at the southern margin rising to 100m to the north under Goughs Hill. On the western flank in the southern portion ore outcrops to surface and to old open pit faces.

Previous exploration programmes resulted in a coverage of the orebody on 35-60m centres with a concentration of data on the southern and central zones. This enabled a general picture of the mineralisation to be built up and ore reserves calculations to be undertaken. A programme of infill drilling was carried out in early 1989 concurrently with mine and mill development. This concentrated on the southern half of the orebody. This reduced drill hole spacing to 15-20 m. The objective of this programme was to check results of old drilling, and enable the mining plan to be based on a more detailed prediction of the geology and cut off grades.

The results of the infill programme confirmed previous work, but did show that the orebody structure was more complex. The ore zone splits into two horizons towards the north with the upper horizon flexing upward and then flattening off again and continuing strongly northward. The lower zone is exhibiting a tendency to weaken and finger out to the north. The single horizon in the south has a thickness of 3-14 m, and averaging some 7 m. The upper horizon has 6-12 m thickness and averaging 9 m. The greisen between the two horizons is strongly mineralised in some areas giving vertical thicknesses of up to 24 m. This is shown in Figures 2 and 3.

It is strongly suspected that similar structures exist in the northern portion of the orebody which is subject to down flexing. This will be tested by a later infill drilling programme.

A small open pit resource of some 25,000 tonnes was outlined during the infill programme. The ore outcrops, or lies beneath incompetent overburden on the western side.

Ground conditions in the greisen were anticipated to be good and this has been confirmed by mining. Some concern exists where the orebody hanging wall forms the contact with the overlying coarse grained granite. Locally oxidation has altered this material to an almost incompetent sand particularly on the western margin. Apart from this excellent ground conditions are expected to persist.

MINING OPERATIONS

A mineable orebody some 300m by 90m was delineated with a long axis NE-SW. The north and south ends of the orebody warped down either by flexing or minor faulting or both. The central section being relatively flat lying as shown in Figure 4.

Ore resources at 0.3% Sn and 0.4% Sn cutoff were calculated for indicated and inferred categories, and a mining plan using room and pillar stopping with 6m rooms and 4m pillars assessed for mining reserves. This resulted in the mining reserve of 795,000 tonnes at 0.52% Sn at a cutoff of 0.3% Sn.

An extraction plan of 100,000t/yr rising to 125,000t/yr over a seven year life was drawn up with increasing tonnage partially compensating for a slowly falling grade over the mine life.

The orebody is accessed by two portals in old open pit faces and the minor development required was in low grade ore. This is shown in Figure 5.

Mining Method

The relatively flat lying orebody and ease of access from the old open pit area favoured a high extraction with a mechanised stopping system. The method selected was the "post pillar" system which is basically a room and pillar system with fill. The method was evolved by Falconbridge in the early nineteen seventies, and was introduced into the Dolphin Mine at King Island in the early

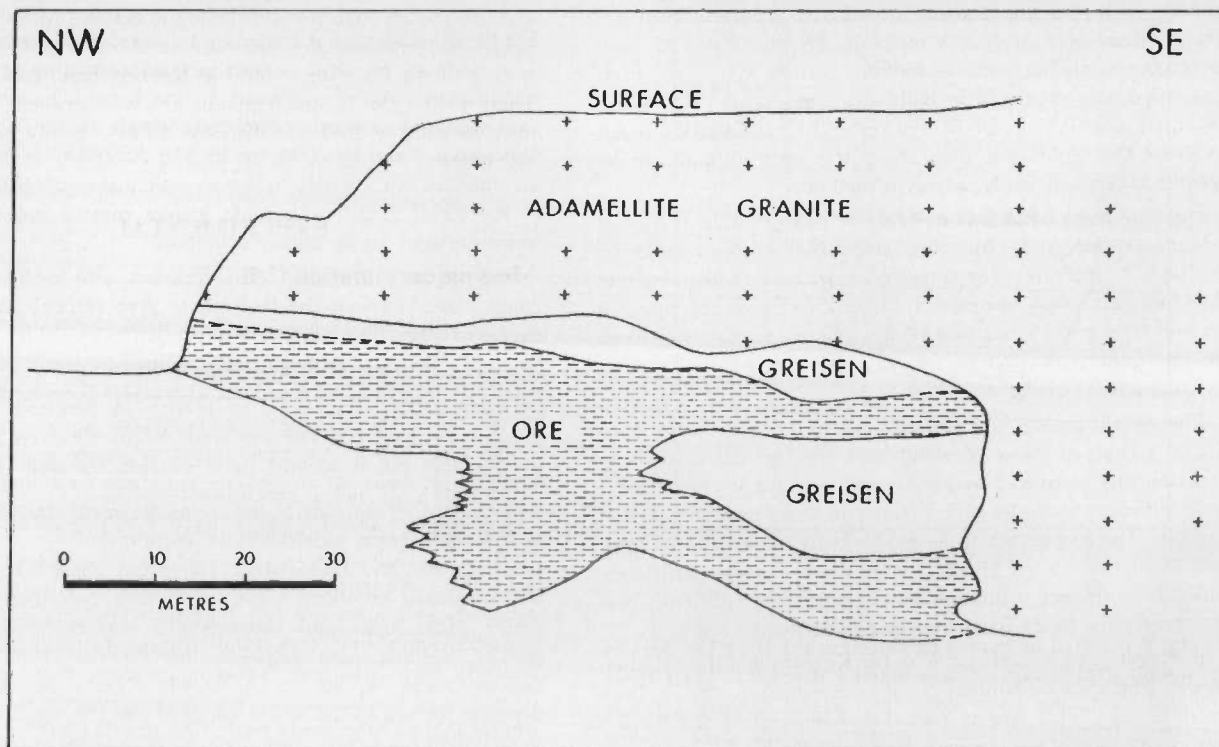


FIG. 2. Detailed Cross-Section

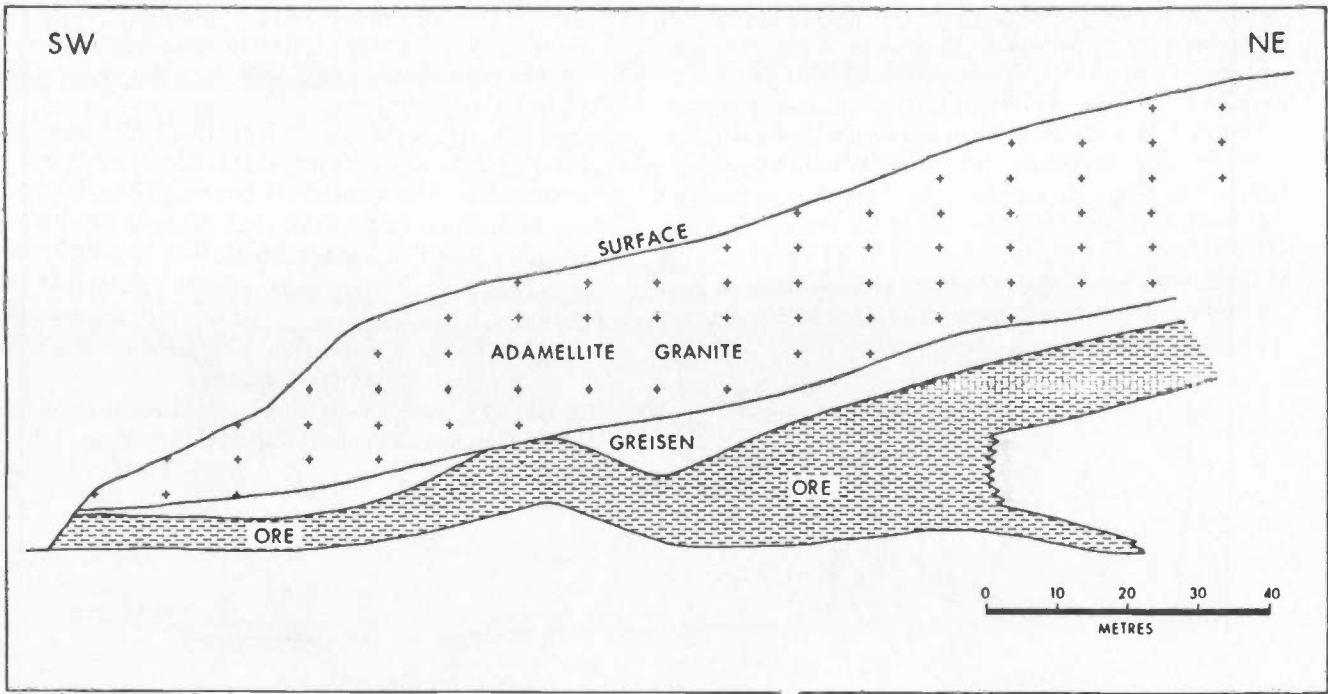


FIG. 3. Detailed Long-Section

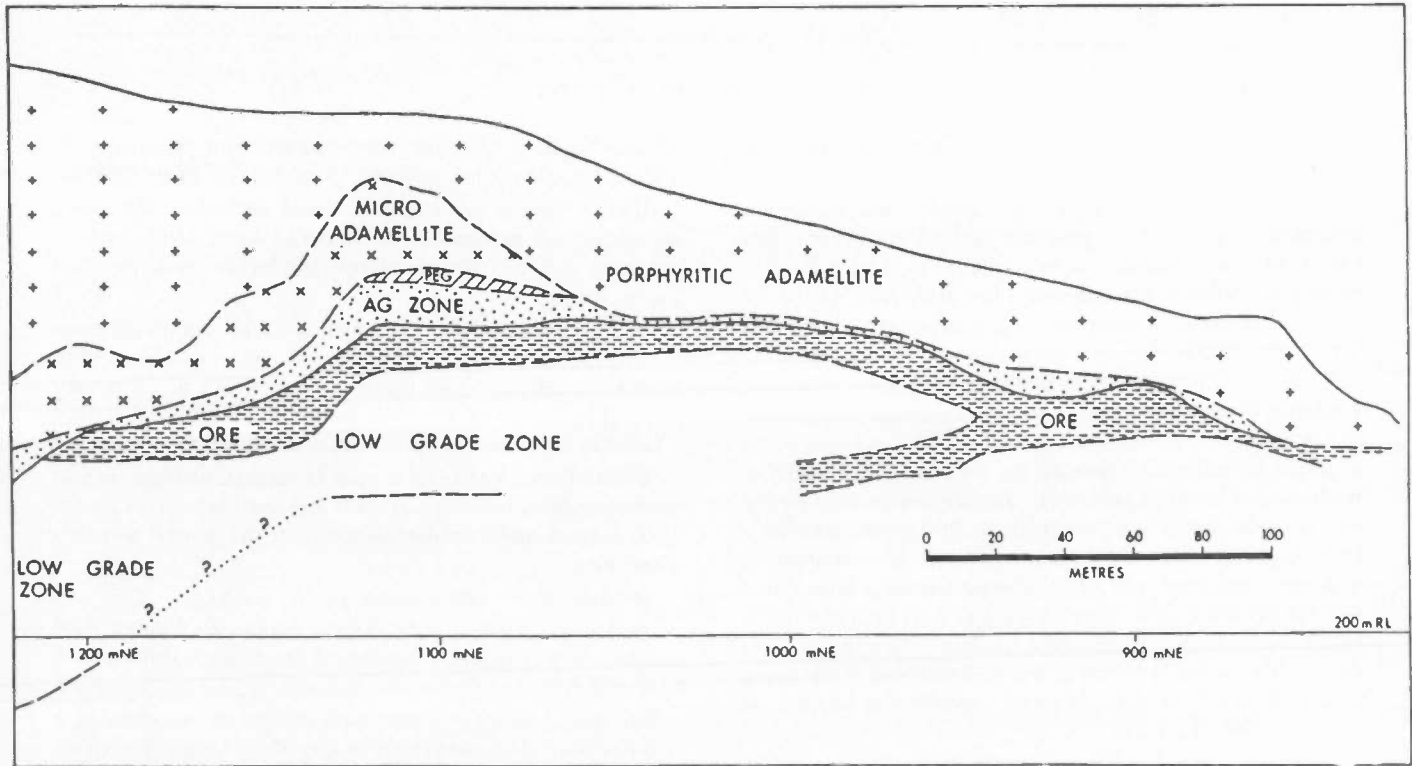


FIG. 4. Anchor Mine Long-Section

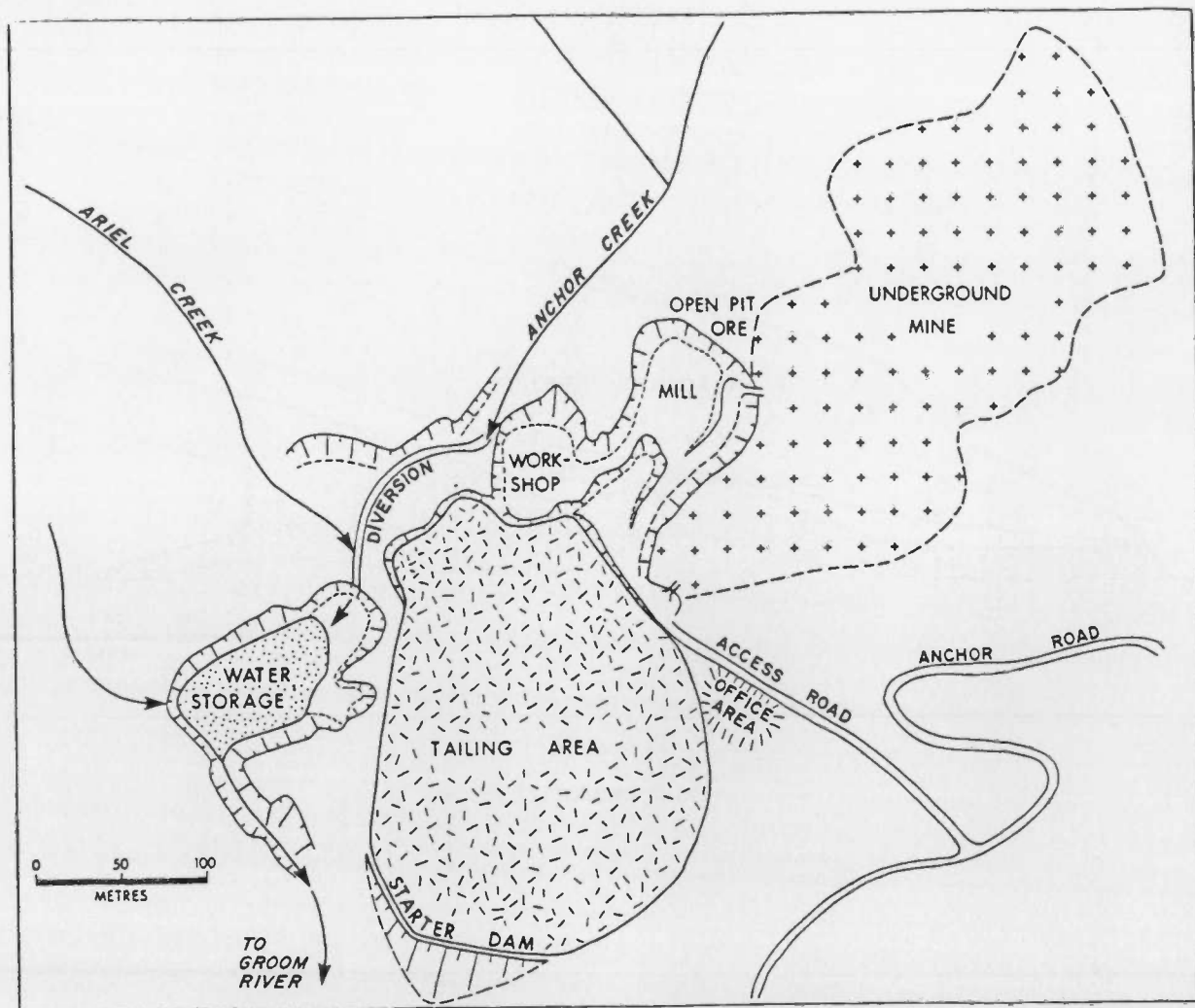


FIG. 5. Anchor Mine General Layout

years of production, and is also used with run of mine waste fill at Renison Mine.

The method can be highly mechanised, and requires a minimum of labour. High productivities and low mining costs can be attained. Dilution can be controlled to low levels. The method is flexible in that barren and low grade areas can be left and small ore pods can be followed. A disadvantage is that pillars are not recoverable.

The layout selected was 6 m rooms and 4 m pillars which gives a nominal 84 percent recovery. An initial undercut was planned at 3 m height with subsequent 2 m flat back lifts. A height of 1.5 m would be maintained between the back and the fill floor to facilitate movement of personnel. Stopping was initially carried out with 5 m rooms and 5 m pillars so that ground conditions could be better assessed. Back conditions have been excellent with only occasional split set bolts being required. Pillars were well cut and stable. The larger early pillars have been trimmed to final size, and the 6 m room is now standard. The pillar layout in the lower undercut is shown in Figure 6 and that in the upper zone in Figure 7. It was also found convenient to increase the undercut height to 3.5m.

Drilling is carried out by a single boom electrohydraulic Mercury 14 Eimco-Secoma jumbo mounting a Hydrastar 300 and using 4.1 m steels and 45mm button bits. Up to six 76mm cut holes are drilled for the cut and 50 holes for the round. Average

advance has been 4.0 m per round with a drilling productivity of 1.15 m/min. Bit life has averaged 350 m and drill steels 1,500 m.

Blasting utilises non-electrical Nonel detonators and Isanol explosives at 60 percent ANFO. About 125kg of ANFO are used per blast and 190 tonnes of ore are broken with excellent fragmentation.

An Elphinstone R1500 LHD with a 5.3 m³ bucket performs loading duties. A larger than standard bucket could be fitted to maintain an efficient load factor as the ore S.G. of 2.6 is very light. The LHD haul from the face to the crusher bin is very efficient at the planned one way haul of under 200 m.

A small Eimco 921 LHD is used for general services such as service platform, building sandwalls and road cleanup. A Toyota 4WD is fitted up for explosives transport and general materials distribution.

An Atlas ROC 601 crawler rig is available. This was purchased to undertake rock bolting duties, but has not been needed. It will probably be utilised for flatback drilling as a backup unit.

Two mining crews of 4 men each operate on two shifts on a five day week with production budget of 500t/day. The crews operate on a do-everything basis and receive maintenance backup from a diesel mechanic-fitter.

Mine fill will consist of deslimed jig and table tailing pumped directly from the mill at some 65 percent solids. The coarse

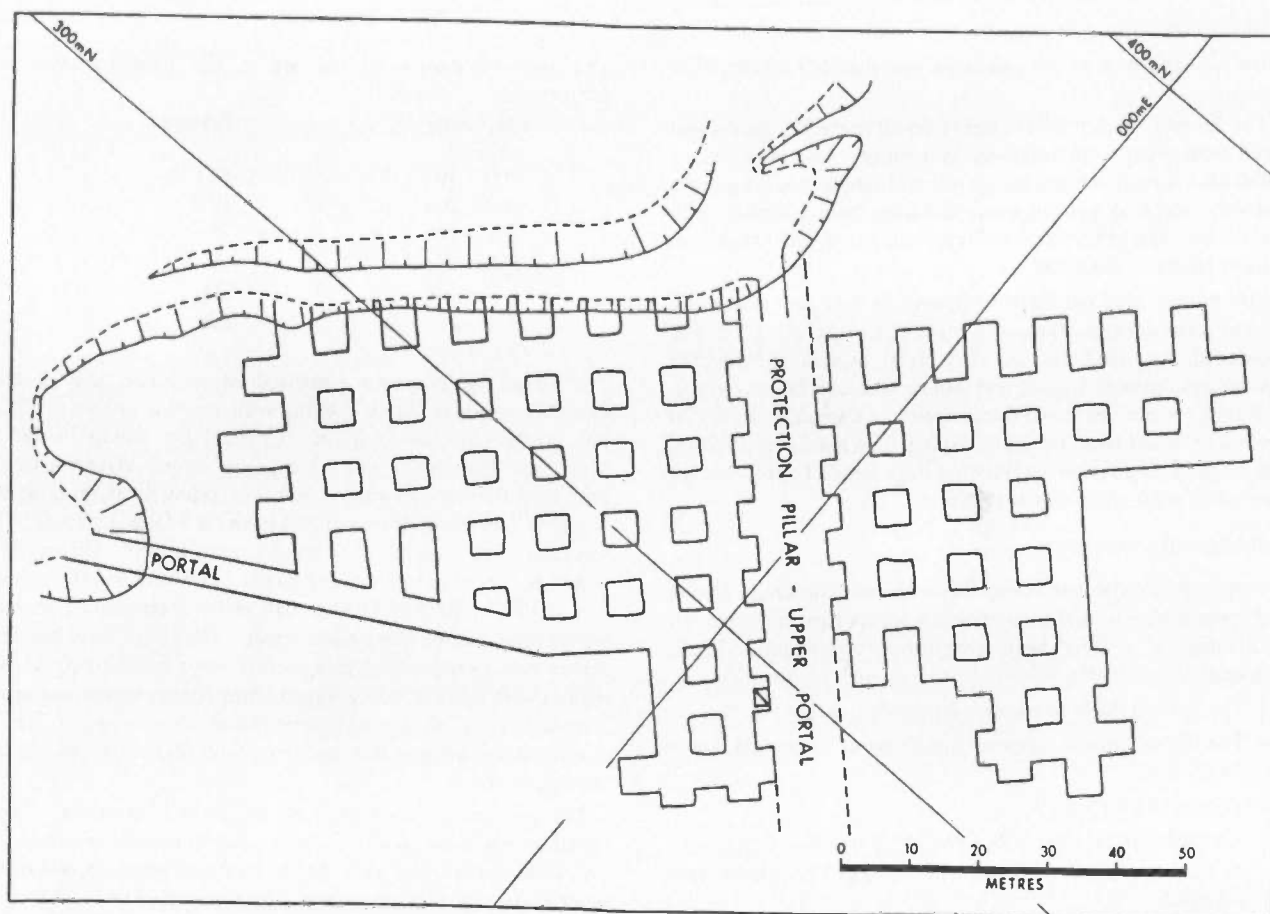


FIG. 6. Pillar layout Lower Undercut before Pillar Stripping

tailing at some 65 percent plus 200 micron consolidates and dewater very rapidly. Fill drainage is not anticipated to present problems particularly as only a minor tonnage of ore lies below the lower portal elevation so the mine will be self draining.

Grade control is heavily dependent on close space diamond drilling from surface to delineate hanging and footwall cutoffs based on chemical assay. This data forms the basis of the mining plan where reasonable continuity of grade is assumed (10-20 m). Grade can change abruptly within a metre from high grade to barren. Sampling of drill cuttings is being undertaken and samples are subjected to a direct read out neutron scanning analyser. Not unsample grainsize affects the results, but reasonable correlations are being obtained. Improved accuracy can be obtained by pulverising the sample and this step will be necessary for acceptable results.

Servicing of power, water and ventilation are via the two portals or by breakthroughs back to surface. A separate emergency access development was not required as stope breakthroughs fulfilled this function and will continue to do so during most of the mine life.

MILLING OPERATIONS

Metallurgical Background

Reports were available on previous operations at the Anchor Mine, and Renison Limited carried out metallurgical investigations as part of studies in 1979-81. The emphasis was on bulk low grade operations in the range of 0.2-0.3% Sn. Certain observations included comments that the greisen was soft and granular and easily crushed and ground, ore became sticky

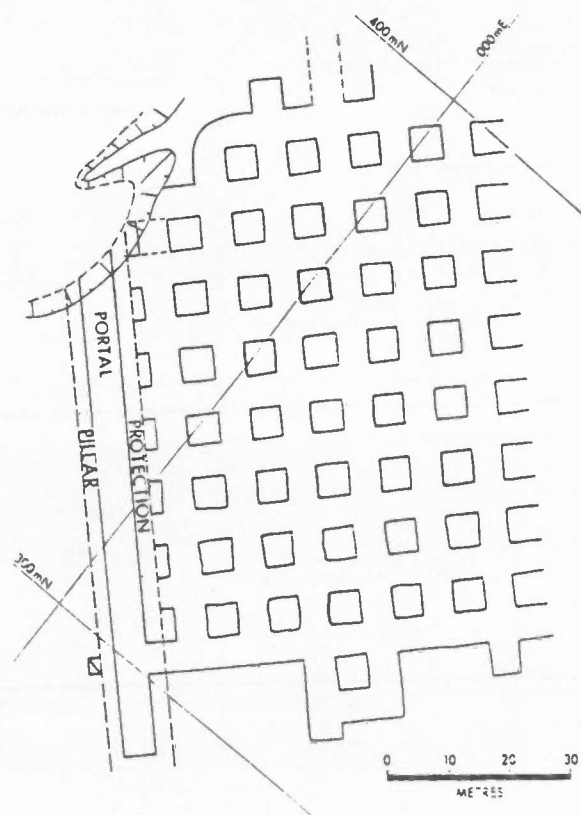


FIG.7. Pillar Layout Upper Zone Before Pillar Stripping

when wet, and bulk of the cassiterite was plus 147 micron (100 mesh).

The Renison work involved heavy liquid testwork, jigging and spiral pilot plant work followed by tabling. The expectations based on a circuit comprising spirals and tables was 80 percent recovery into a 55 percent tin concentrate from a feed of 0.27 percent tin. The grind size was 80 percent passing 350 micron to achieve adequate liberation.

Spectrum carried out further testwork on old core examining jig and table concentration, and concentrate upgrading. It was concluded that the liberation size of 80 percent passing 350 micron was correct. Jigging and tabling was an efficient method of gravity concentration and that concentrate upgrading to plus 70 percent tin could be achieved by drying, high tension separation, and magnetic separation. A flotation stage would be required for removal of small quantities of sulphide.

Conceptual Flowsheet

A simple and inexpensive circuit was required to minimise capital and operating costs, and be operated by inexperienced personnel. A number of criteria were considered significant. These included:

- The ore was likely to become sticky when wet.
- The slimes content appeared significant at 33 percent minus 74 micron.
- The ore crushed easily.
- Overgrinding of cassiterite should be avoided.
- A liberation size of 80 percent passing 350 micron was accepted.

The size distribution of the ore at this liberation size was estimated as:

Size Range Micron	Percent
- 500 + 300	17
- 300 + 200	17
- 200 + 100	25
- 100 + 74	8
- 74	<u>33</u>
	100

The circuit as proposed consisted of a 24x36 jaw crusher, vibrating screen at 50mm opening with oversize to a small 12x20 jaw crusher in closed circuit. Crushed ore would be fed to Mogensen screens for wet screening at 1mm. Oversize would pass to a Barnac Duopactor with this product returning to the screens. The minus 1mm product from the Mogensens was to be screened at 500 micron over a sieve bend screen. The oversize would be jigged to remove free coarse cassiterite and the tailings ball milled in an 8x4 Denver ball mill. The ground product would return to the sieve bend screen. Undersize from this first screen was then pumped to a second sieve bend sizing at 200 micron with oversize being jigged. The 200 undersize was again screened over a 74 micron sieve bend and oversize jigged. The 74 micron undersize was then cycloned with underflow joining the third jig feed.

The jig concentrates would be fed to Wilfley tables. Table flotation was to be used to remove small quantities of sulphides. The table concentrates were then to be dried, screened, and batch treated on a high tension separator for removal of non-conductors particularly topaz, and finally passed over a magnetic separator for removal of magnetic minerals including wolfram.

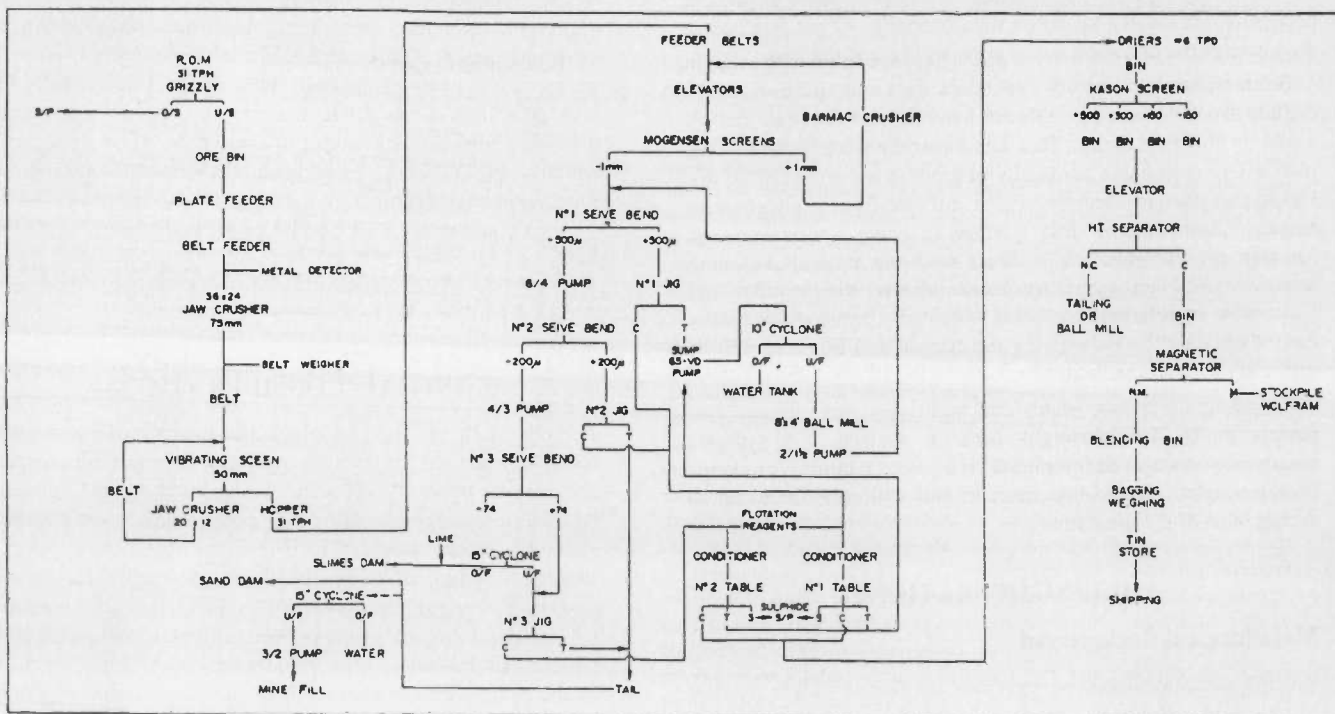


FIG. 8. Anchor Mill Flowsheet

The plant capacity was a nominal 31 TPH to treat 500 t/day on two shifts. An annual production of 730 tonnes of 70 percent concentrates containing 510 tonnes of tin was planned.

The Barmac crusher is now used extensively in the quarrying industry and has demonstrated low operating costs. Tests showed that the ore was easily crushed and the desired 1mm product could be attained. The great attraction was that the unit could accept a 50mm top size and produce a 1mm product without the need for a tertiary crusher and rod mill to treat all the feed. The small ball mill was introduced to reduce the +500 micron tailings to pass 500 micron which was the indicated liberation size.

THE INSTALLED FLOWSHEET

Primary Crushing

The primary crushing arrangement consists of a 25 tonne coarse ore bin with a fixed grizzly at 450mm aperture being fed by a 5.3 m³ capacity LHD. A hydraulic plate feeder delivers to a 600mm feed belt to a 36x24 single toggle jaw crusher of Chinese manufacture. The crusher is protected by a metal detector.

Screening

The primary jaw crusher product is delivered by conveyor to a double deck vibrating screen screening at 50mm. The oversize is crushed by a Parker 20x12 jaw crusher in closed circuit. The final product falls into a 15 m³ capacity crushed ore bin. Provision is made for a bypass for crushing road metal.

Secondary Crushing

The Barmac unit was purchased as an Aggtech 2000 plant package from the Barmac manufacturers in New Zealand. This was a used plant in good condition that had been used for quarry crushing demonstrations. The plant was overhauled by the suppliers before shipment to Tasmania. The plant consisted of a crushed ore bin and feeder belts to twin elevators which deliver to a pair of 1.5 m Mogensen screens. These are fitted with relieving screens and a final 1mm screen. Wash water sprays are fitted to the screens. Oversize falls directly to a Barmac MKII Duopactor, and the undersize to No.1 sieve bend. The Barmac product is returned by belt to the crushed ore bin.

Fine Screening

The Mogensen undersize passes via a launder to the No.1 sieve bend with a cut size of 500 micron. The oversize passes to No.1 jig. The undersize falls to a pump sump and is pumped by a 6/4 Warman pump to No.2 sieve bend with a cut size of 200 micron. Oversize is jiggled on No. 2 jig. The undersize is pumped to No. 3 sieve bend screening at 74 micron fitted with vibrators with oversize passing to No. 3 jig. The screen undersize is pumped to a cyclone with underflow joining the No.3 jig feed. Cyclone overflow goes to waste directly to the rear of the tailings dam.

Jigging and Tabling

UHF jigs are used each with two compartments and central diaphragm. The hutch concentrates from No.1 and No.2 jigs are combined and tabled on a Wilfley table, and the No.3 jig concentrate tabled on a second Wilfley table. The jig concentrates were to be conditioned and table flotation used to remove minor sulphides. The No.1 jig tailing and all table tailings were to be pumped to a 250mm cyclone for dewatering before ball milling.

Ball Milling

An 8x4 Denver ball mill takes the No.1 jig tailings and table tails and reduces these to pass 500 micron. The ball mill product being pumped back to No.1 sieve bend.

Dry Separation

Wet concentrates are taken in bins by forklift and tipped into oil fired Chinese driers. Dry concentrate is shovelled into self tipping bins and taken to a hopper feeding a Kason screen fitted with 300, 150 and 74 micron screens. Products are stored in bins and then batch treated through a Mineral Deposits three roll high tension separator. This removes the principal topaz contaminant and any other non-conductor minerals. Conductors are then treated on a Readings crossbelt magnetic separator for removal of abraded iron, magnetite, siderite, any biotite mica, and wolfram. Cassiterite is blended, weighed into 2 tonne bulker bags and trucked to Burnie for shipment to Penang in 20 tonne lots. Wolfram will be stock piled for future sale. Tin concentrates are plus 70 percent Sn.

Mill Control

A neutron analyser is used on site for grade determination together with burette assay and zinc block checks on concentrates. Final tin assays are made by outside laboratories. The installed flow sheet is shown in Figure 8.

INSTALLATION AND COMMISSIONING

The mill installation was anticipated to be a relatively straightforward matter involving simple circuitry. However, installation and commissioning was to be long drawn out and expensive. The reasons for this could be classified as:

- Weather and site problems.
- Design and installation shortcomings by the contractor.
- Ore characteristics different from those anticipated.
- Equipment design shortcomings.
- Equipment failure.

Weather conditions over the 1989 winter were extremely bad with one of the wettest winters ever recorded. Torrential storms resulted in the loss of the access road on one occasion with over half a kilometre having to be rebuilt. Roads deteriorated in days under heavy vehicular traffic as the coarse grained granite that formed most of the roading ballast turned to ooze. Imported river gravels held the roads together somewhat until good quality rock became available from underground development. The very wet conditions prevented installation work from taking place for days at a time.

The design and installation contractor did not perform well and had to be replaced. This subsequently led to modifications to belt drives, chutes and launders, and strengthening of structural members.

Once commissioning commenced problems became more serious. The first of these was a difference in crushed ore characteristics becoming apparent. The ore crushed easily being very friable which was anticipated, but the Barmac product had a much coarser size distribution than predicted. Well over half the No.1 sieve bend feed was plus 500 micron. This overloaded the No.1 jig and the ball mill as these tailings were being recirculated. The ball mill was taken out of circuit temporarily.

It became evident that the Barmac crushing behaviour was preferentially separating the different minerals in the ore by splitting along grain boundaries rather than crushing mineral grains. This resulted in effective liberation of cassiterite at about 1mm instead of 500 micron as previously established. This will have benefits in recovery and costs. It is also a behaviour that may be applicable to other ore types.

The changing crushed ore characteristics resulting in a coarser product also resulted in greatly reduced fines. Again this has future benefits for tailings disposal and mine fill, but it was found that there was insufficient feed to operate No.3 jig.

The coarser tailings led to sanding up of tailings lines. This was corrected by installing a 100mm line on a surveyed grade line. Power drawdown on the pump has been reduced at the same time.

Major problems existed with the Barmac crushed product. There had always been a question mark over wet feed in the Barmac as its success had been in the quarrying industry with dry feeds. The Barmac would repeatedly bog down necessitating shutdown of the plant. The crushed ore at a certain moisture content assumes the characteristics of shotcrete and would cake up in the discharge chutes. The crusher then had to be sluiced out by high pressure hoses. Morale tended to suffer when this occurred two or three times a shift. Solutions were eventually arrived at by trial and error. The water sprays on the Mogensen screens were restricted to the lower decks, and the spray nozzles replaced by nozzles with small orifice. The whole discharge structure of the Barmac crusher was modified to prevent hang up. The steeply inclined product belt back to the crushed ore bin aggravated the bogging problem with fines slipping back under the Barmac discharge chutes. Fitting a ribbed belt effected some improvement.

On the mechanical side troubles had existed running the unit up to speed from the starter motor. Eventually after frequent retensioning the drive belts from the 200 HP drive motor to the Barmac burned off. After removing the guards it was found that the six belt pulley was installed with one belt short to the maindrive. Fitting the corrective five belts made a marked improvement in the crusher rotor speed. This significantly enhanced the crushing behaviour, and decreased wear. A check on the energy transfer showed that a five belt drive was marginal and eight belt pulleys have now been fitted.

The worst mechanical failures have been the Mogensen screens. After only a few weeks of operation first one and then the other cracked both the transverse structural members, and then the side frames. These were serious failures and resulted in new screens being ordered from Mogensen Australia.

The bucket elevators have given some problems in tracking. This has been resolved by modifications to the drive pulleys. Bogging still occasionally takes place.

Circuit Changes

Sampling has indicated that the present circuit operating at 60-70 percent capacity is achieving over 90 percent recovery of cassiterite.

Changes to achieve the 500 t/day rate and improve reliability apart from replacement of the Mogensen screens are redistributing the feed between the jigs by changing the sieve bend screens to 750 micron, and 350 micron and removal of the No.3 sieve bend. This will be replaced by a 375mm cyclone to feed No.3 jig at plus 74 micron feed. No.1 and No.2 jigs will feed to their own tables No.1 and No.2, and No.3 jig will feed to

two half sized tables. This change should improve table capacity and recovery of fine cassiterite. Table tailings will go to tails with a small middling being recirculated. The No.1 jig tailings will flow to tails and the ball mill removed from circuit. The table concentrates will be batch treated in a float cell instead of using table flotation.

While the commissioning phase has been drawn out due to operational and equipment problems the conceptual flowsheet has been shown to be sound with recoveries in excess of the 85 percent planned.

TAILINGS DISPOSAL AND SURFACE WATER MANAGEMENT

Tailings disposal formed the major consideration in the Environmental Management Plan prepared for the Department of the Environment to support the application for a licence to operate. The tailings facility operation was integrated into the site water management programme with the objective of minimising adverse impacts on the existing environment. The EMP was prepared by John Miedecke and Partners Pty Limited.

It was proposed to site the tailings dam immediately below the mine workings and the mill. It would occupy an area of old mining operations consisting of open pit workings and ground sluicing occupying an area of some 5 ha and have a capacity in excess of 400,000 tonnes of tailings. Since mining ceased some seventy years ago apart from sporadic tribute workings there had been vigorous regeneration of regrowth scrub/closed forest.

The Groom River flows through the lease area, but to the south of the proposed tailings dam. Anchor Creek forms a tributary to the Groom River and flowed through the tailings dam area. The second smaller Ariel Creek flows into a small dam which was a deep area of open pit workings. This dam then overflowed through an adit to eventually join the Groom River. A schematic arrangement of water management is shown in Figure 9.

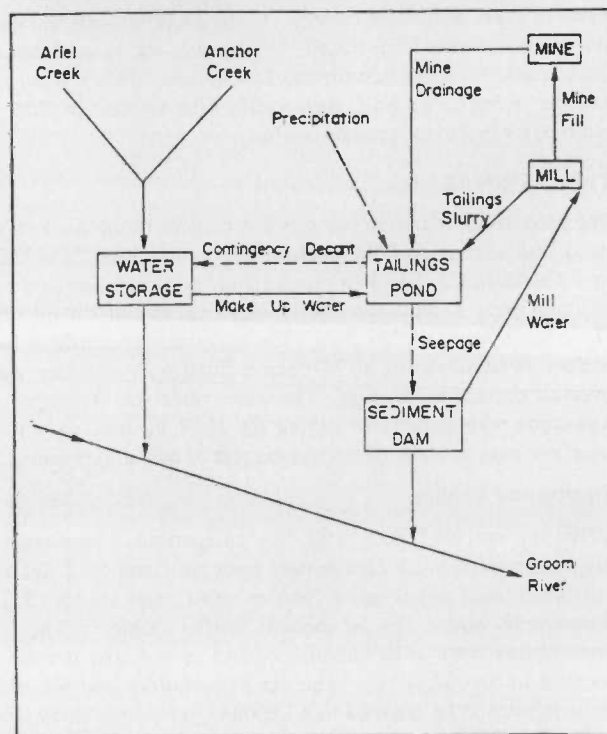


FIG.9 Water Management Schematic

The Anchor Creek had to be diverted from its existing bed by a diversion channel high on the side of the open pit workings to flow into the existing dam. This was to be a permanent diversion. The dam adit was to be plugged to increase water storage and a new overflow channel created.

The tailings dam would impound tailings and process water. It would also receive mine drainage water. The tailings dam wall was to be designed as a flow through structure with drainage water being collected in a small sediment dam downstream of the main wall. Water would then discharge over a permanent weir to eventually meet the Groom River. This weir would be the normal discharge point of water from operations. Provision was made for pumping to the mill header tanks from the sediment dam. A decant system was designed into the tailings dam for storm condition contingencies. This would decant to the sediment dam, or at a later date to the existing water dam.

The general water management plan achieved several objectives:-

- It freed Anchor Creek for tailings storage.
- Natural waterflows were maintained by short permanent diversions.
- Water storage was created for plant make up water.
- All water from mining activities was directed to the tailings pond.
- A single operational discharge point was established.
- Diversions, storages and discharges are of a permanent nature and planned for long term abandonment.

Tailings Dam Design

The tailings dam design had to accommodate a number of factors, the most important being:-

- Final abandonment considerations.
- Integrity during the operation phase.
- Method of tailings deposition.
- Water management.
- High intensity storm events.
- Drainage of the tailings.
- Tailings characterisation.
- Revegetation.

The basis of the design was to plan for abandonment at the outset. In its simplest terms this required the structure to be self draining to avoid any build up of the phreatic surface where the tailings become saturated and lose their cohesiveness. This could be achieved by adequate drainage under the base of the tailings structure, and drainage from the surface of the tailings area during high rainfall events. Similar considerations apply during the operational phase.

The structural stability is improved by appropriate tailings deposition techniques, so as to increase the density of the material and reduce the percentage of pore water. This reduces the chances of liquefaction under shock loading from seismic events, and also under intense rainfall.

Water management required that excess storm water and runoff was appropriately channelled to discharge ditches or decant structures to avoid increasing the saturation level of the tailings, or overtopping of the dam.

The character of the tailings is of considerable importance. An assessment of the size grading will influence the height to which the dam could be raised and the slope angle of the face. The

chemical composition would affect the quality of the discharge waters.

Tailings Characterisation

The predicted tailings grading from crushing tests and the required liberation size was estimated as;

Size Range Micron	Per cent
500 + 300	16.7
- 300 + 200	16.7
- 200 + 100	25.0
- 100 + 74	8.3
- 74	33.3
	<u>100.0</u>

This was in the relatively coarse range for tailings disposal and storage. Samples of the -74 micron fraction from the crushing tests were tested for settling behaviour as it was suspected that as the site water was near neutral pH slimes could be reluctant to flocculate and settle. This was found to be the case. Treatment with alum and lime was found to be effective in altering the pH and effecting precipitation. Alum was not well regarded as this creates an acidic environment. Lime treatment at an addition of 150 ppm gave effective flocculation and precipitation with sedimentation rates of 2.5m/h and zero solids in the supernatant after one hour. Final pulp densities of 58 percent solids were achieved after 15 minutes. These tests indicated that excellent depositional behaviour could be expected from the fine fraction. Admixture with the coarse fractions and beach deposition would further improve the storage characteristics by giving higher densities.

The operational results have demonstrated that liberation can be achieved at a coarser size of 1mm together with a greatly reduced proportion of minus 74 micron material. This has given superior settling characteristics than predicted.

The chemical characteristics of the ore are notable for the low levels of sulphide and base metals. Copper and zinc do not exceed 0.1% combined. There are low levels of other metals and some pyrite. The sulphide minerals would concentrate with the tin concentrate, and then be removed by flotation. This would make the tailings virtually sulphide free, and there would be little opportunity for acid leachates containing heavy metals to be created.

Investigations were carried out on samples of ore, process waters and tailings solids that were produced by the Tasmanian Department of Mines during the metallurgical studies. Investigations were carried out to determine:

- the chemical composition of the liquid and solid fractions, the acid forming potential of the solids, and
- the leaching characteristics of the ore and tailings.

The chemical composition of the process waters and solid samples were determined by commercial laboratories using ICP and AAS methods. Total sulphur and the acid neutralising capacity (ANC) were determined on all solids to assist the potential for acid generation. Saturation extracts were prepared on the ore and tailings samples to assess the composition of easily soluble constituents. All investigations were carried out by Stuart Millar and Associates.

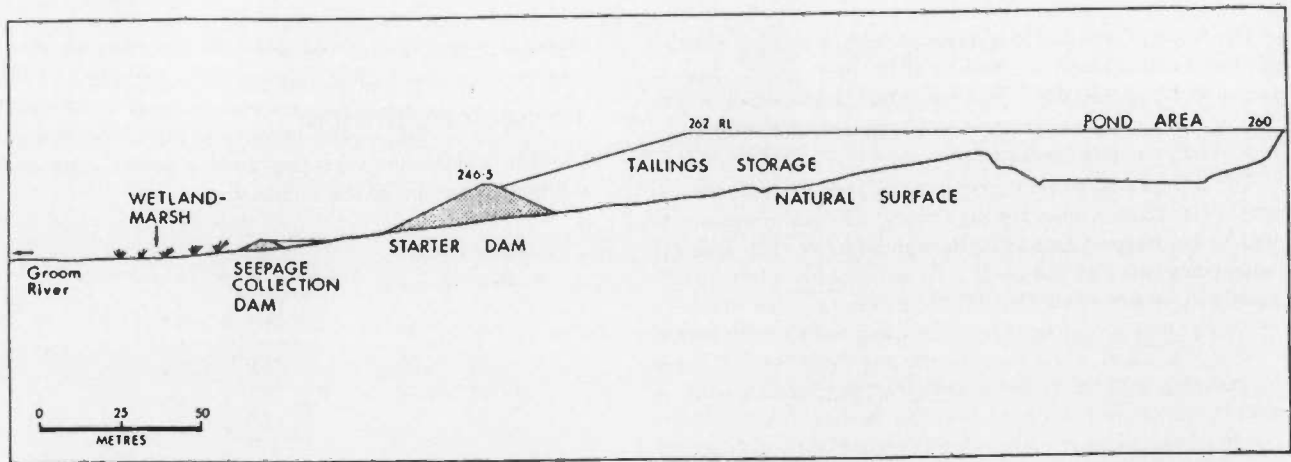


FIG. 10. Longitudinal Section Through Tailing Dam

Column leaching tests were carried out to examine the leaching behaviour of various constituents and analysed for a range of elements and pH, EC, and SO_4 . All materials had an inherent acid neutralising capacity and the calculated NAPP indicated that the ore and tailings could be classified as non-acid forming. The natural pH of the materials was approximately 8 and the solids had low salinity.

The concentration of significant constituents in leachates was low except for fluoride. Comparison with the Environmental (Water Pollution) Regulations suggested that compliance was readily achieved with the exception of F.

The higher levels of F were puzzling as although there was relatively abundant topaz in the ore its solubility is extremely low. It was eventually realised that water in the Mines Department laboratory came from Launceston town supply. A check revealed that fluoridation of the supply was carried out at a level of 1ppm and that fluoride was not likely to pose a problem in leachates.

Drainage of the Tailings Structure

The depth of tailing on completion of the project would not exceed 20 m above original surface. The shallow depth and coarse grading of the tailing would create good permeability and drainage to the base of the structure. Barratt Fuller and Partners Limited were retained to advise on the dam design and undertake investigations of the existing ground conditions. A longitudinal section through the tailings dam is given in Figure 10.

The underlying land surface was mainly solid open pit benches with a shallow cover of silt and vegetation. The Anchor Creek bed was progressively filled with deeper layers of rubble and overlying fine material when proceeding downstream. This had resulted from old sluicing and rock stacking operations. As sluicing proceeded upstream fines were deposited over the coarse cobbly material. This created excellent drainage and soil profiles that contributed to rapid regeneration of the area.

Advantage was taken of the natural site conditions to enhance the base drainage of the dam and create a flow through structure for eventual abandonment. The objective was to encourage vertical drainage through the tailings to the base. Water would then travel through the permeable base.

A rock starter dam was constructed to establish the tailings dam. The base of the dam was excavated to remove all fines and down to solid ground by means of a cut off trench. The trench was lined with a water permeable liner and back filled with rock.

The starter dam was then constructed. It was intended that the upstream face of the dam would not be sealed other than by deposition of coarse tailings. The objective was to encourage horizontal drainage beneath the dam and minimise any build up in the phreatic surface. In practice the run of quarry rock was rather coarse and piping took place. This was rectified by placing 1-1.5 m of crushed rock on the upstream face to fill any voids between large rocks.

Tailings Deposition

The tailings consist of relatively coarse jig and table tailings and a dilute cyclone overflow. The coarse tailings are pumped to the starter dam wall and deposited so that a beach is built up allowing the water to drain to a pond toward the rear of the tailings structure. The beach will be built up in thin layers so that sand density and dam stability can be improved by air drying and better drainage. The discharge point will be moved around the wall as necessary to create the beach and to maintain the desired downstream slope angle of IV:2.5H as the height of the dam face is raised. The fine fraction will initially be discharged at the rear of the dam behind a low bund. This will give an opportunity for the coarse well draining sand beach to become established at the starter dam. When this is established the fines can be mixed with the coarse sands to avoid a slimes pond situation.

Re-vegetation and Abandonment

The design and management of the tailings dam during the operational life is to permit abandonment to take place smoothly and with minimal additional work. The tailings dam structure is designed for abandonment at any time with storm water decants or channels in place. The total structure acts as a flow through unit with permanent vertical and horizontal drainage in place, and the sediment dam in place with a spillway.

The starter dam rock face can be progressively vegetated and extended upwards on to the coarse tailings sand face. The outer edge of the face will have layers of straw and biodegradable textile netting incorporated to reduce erosion and provide a foothold for vegetation. This will be planted in the first instance with fast spreading ground cover which in turn will provide a medium for planted and naturally introduced species. Hydromulch and fertilisers may also be helpful.

On abandonment the surface of the tailings dam will require stabilisers to prevent erosion, and provide a growing medium.

The surface area will be roughly oval measuring 210 m by 170 m and giving an area of some 3.5 ha for rehabilitation.

The area is already sloped for drainage but some contouring may be required to establish drainage lines which can be surface reinforced for permanent drainage by rock lining and other materials. The surface would then be mulched and seeded for a fast growing surface cover for stabilisation and planted with native species of plants and shrubs. The ponding area will probably be dry in summer and reasonably wet in winter as the tailings depth will be shallow. This will become a seasonal wet land area and the appropriate plantings can be made. Two years will probably suffice for development of vegetation to a point where formal abandonment handover can be requested.

Sediment Collection Dam

A small dam was created downstream of the starter dam. Its function is to intercept water draining from the tailings dam and to collect any fines eroding from the tailings dam face as the dam is increased in height. It also functioned as a sediment trap during site construction. This is a water holding dam and is constructed accordingly. Discharge is via a concrete spillway which is the main discharge from the site. The spillway discharges into a marsh area before joining the Groom River. Monitoring of water quality takes place at the marsh discharge.

A 70HP submersible pump is mounted on a floating barge tethered in the centre of the dam. Water is pumped to header tanks above the mill through a 150mm polyline.

Some difficulties were encountered during construction. Trenching had indicated a solid base according to ability to penetrate with a hydraulic digger bucket. Excavation exposed abutments of large granite blocks but also heavy alteration on some joints. Additional excavation was necessary. The dam was overtopped twice during construction when rainstorms of 275mm and 235mm in a day lashed the site. Damage was minimal.

On abandonment this dam will become a permanent water retaining structure. Natural regrowth around the margins is already evident and this will become a habitat for native fauna.

Water Storage Area

A dam had been created by old open pit mining in the form of a deep pit some 2.5 ha in area and a depth of 5 m of water. The water volume was doubled by increasing the depth. This was achieved by blocking an old adit which drained the dam and cutting a new overflow channel through an existing gully.

Ariel Creek and the diverted Anchor Creek both flow into the dam. Natural water flows are all diverted around the project area so minimising any degradation of water quality. The dam is available for emergency storage at times of abnormally dry summers and also acts as an emergency silt catchment should the

tailings emergency decant ever overflow. It also enhances the long term abandonment security for water management.

SITE FACILITIES

Maintenance

A maintenance building with washdown pad is on the site and supervised by a diesel mechanic/fitter. Mine equipment is serviced by the diesel mechanic and operators on a daily basis with steam cleaning being carried out on weekends. Major services will be carried out by the suppliers service team from Burnie and exchange units carried.

Mill maintenance consists of daily servicing at end of shift and weekend maintenance. Most of the mill operators have fitting and boilermaking skills and are capable of carrying out most maintenance work.

Administration buildings consists of office block and changeroom facilities. A small store and a sample preparation laboratory are attached to the maintenance building. Power is supplied by the HEC.

PERSONNEL

A small team of multi-skilled people mainly locally recruited carry out all operations. The team consists of:

- Mine Manager
- Manager's Technical Assistant
- Secretary
- 8 Miners
- 4 Mill operators
- Diesel mechanic/fitter

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