GSB54

TASMANIA DEPARTMENT OF MINES

1972

GEOLOGICAL SURVEY
BULLETIN No. 54

A CENTURY OF TIN MINING AT MOUNT BISCHOFF, 1871 - 1971

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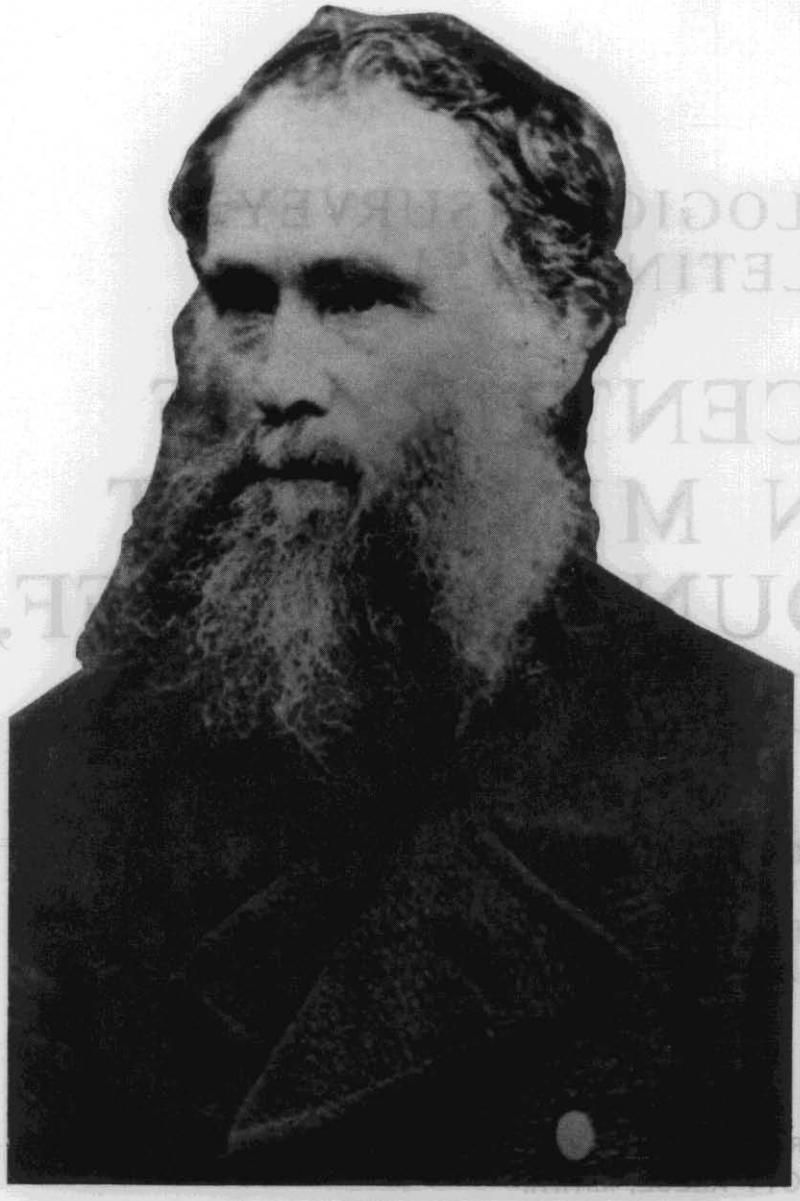

Frontispiece. *James 'Philosopher' Smith*

PREFACE

This Bulletin was prepared to commemorate the centenary of the epoch-making discovery of tin ore at Mt Bischoff in western Tasmania by 'Philosopher' Smith on 4 December 1871 and the commencement of mining operations in the following year.

The first discovery of coal in Australia was made on the southern coast of Tasmania by Labillardière in 1793 and gold was known to occur in Tasmania as far back as 1840, but the discovery of the rich tin deposits at Mt Bischoff awakened interest in the mineral wealth of the State and provided a mighty stimulus to mining in Tasmania, and indeed to the whole of Australia.

Following the discovery of Mt Bischoff, prospectors located the rich alluvial tin deposits of north-eastern Tasmania, and later the rich gold deposits of Beaconsfield.

The influence of Mt Bischoff on the development of western Tasmania was more decisive. Tin was discovered at Mt Heemskirk in 1876 and the discovery of the silver-lead deposits at Zeehan and the important deposits at Mt Lyell, Magnet, Renison Bell, Mt Read, Rosebery and Tullah followed.

In more recent years the Savage River iron deposits and the Cleveland tin ore bodies have been exploited.

It is interesting to note that the Broken Hill outcrop was of interest to the discoverer, Charles Rasp, because he knew of the deposit at Mt Bischoff and thought that the Broken Hill deposit was a tin orebody.

Thus the discovery of the Mt Bischoff tin orebody directly and indirectly made a tremendous contribution to the economic development of Tasmania and Australia.

This Bulletin gives an account of its discovery, mining and metallurgical history, and geology. It is principally concerned with the operations of the Mount Bischoff Tin Mining Company which, in its heyday in the late nineteenth century was world famous.

Information relating to the activities of the other companies and prospects in the area was given in Bulletin 34, *The Mount Bischoff Tin field*, by A.M. Reid, published in 1923.

J.G. SYMONS, Director of Mines

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ABBREVIATIONS

Symbols used in metallurgical reports

Product symbols

A	Anode	L	Liquid	S/D	Solid
C	Concentrate	M	Middling	S/K	Sink
C/E	Calcine	M/A	Magnetic	S/N	Solution
D	Discharge	N	Non-magnetic	T	Tailing
E	Extract	O/F	Overflow	U/F	Underflow
F	Fraction	O/S	Oversize	U/S	Undersize
F/D	Feed	P	Precipitate	#	Mesh (B.S.S.)
F/T	Float	R	Residue	µm	Microns
G	Gas	R/E	Raffinate	+	Retained on screen
H	Head	R/X	Reflux	-	Passed through screen
K	Cathode	S	Spigot		

Operation symbols

B	Batch	H/L	Heavy Liquid Separation
B/M	Ball Mill	H/M	Heavy Media Separation
C	Cyclone or Classifier	H/S	Hydrosizer
C/E	Cradle	I/S	Infrasizer
C/H	Crush	I/X	Ion Exchange
C/L	Column	J	Jig
C/N	Continuous	L	Leach
C/S	Cyclosizer	L/L	Liquid/Liquid Extraction
C/R	Condenser	M	Mixer
D	Tilting Deck or Strake	M/S	Magnetic Separation
D/N	Distillation	P	Panning
D/R	Dryer	P/N	Precipitation
E	Electrostatic Separation	R	Roasting
E/F	Elutriation Fraction	R/M	Rod Mill
E/S	Electrolysis	S	Screening or Sieving
F	Flotation	S/B	Stamp Battery
F/C	Filter or Centrifuge	T	Table
G	Grind	T/H	Thickener
		V	Vanner

Chemical symbols

The normal chemical symbols for the elements and formulae for compounds are used, with the addition of the following:

C.Sn	Cassiterite Tin
S.Sn	Acid soluble tin (Stannite Tin)
T.Sn (or Sn)	Total tin (C.Sn + S.Sn)
S.Fe	Acid (HCl) Soluble iron
T.Fe (or Fe)	Total Iron

Use of symbols

(1) Where there are a number of products from the one operation, *e.g.* magnetic products, spigot products, these are distinguished by numbers, *e.g.* M/A1, M/A2, M/A3, S3, S4 etc.

(2) Where there are a number of operations, these are numbered, *e.g.* Two jigging stages J1 and J2, flotation stages F2, F3.

(3) Products from operations are described by compounding operation and product symbols thus: F4C, C/S4 means the cyclosizer fraction 4 from the concentrate made in flotation stage 4.

Note: Research projects are given a number prefixed with R, *e.g.* R.606. The tests done in the project are numbered *N*1, *N*2 etc.

GENERAL INTRODUCTION

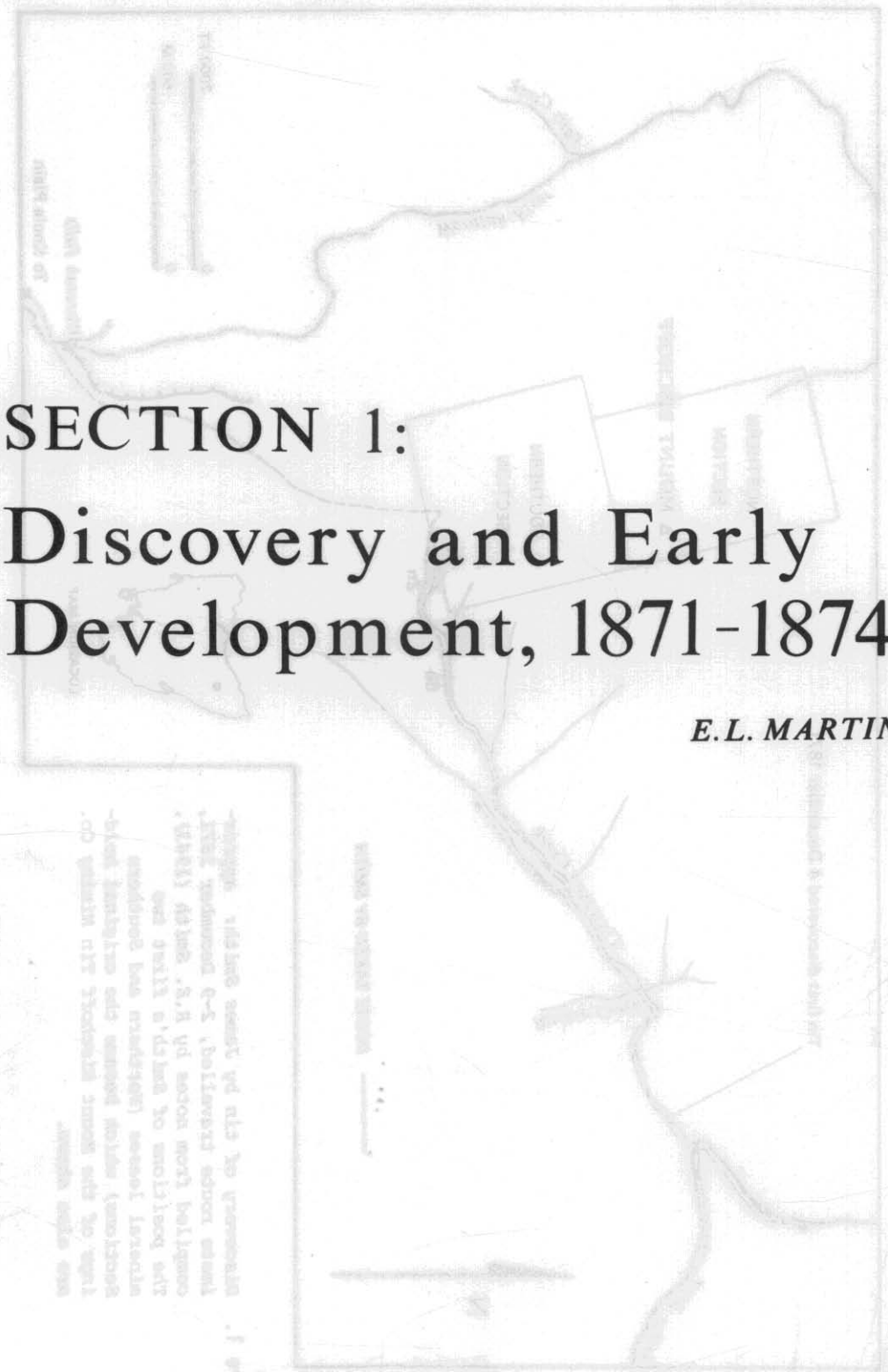
The discovery of tin at Mount Bischoff by James Smith in December 1871 was a significant event in the history of Tasmania. It stimulated interest in the mineral resources of the colony and the period of vigorous prospecting activity which followed led to the discovery and development of the mineral fields of western Tasmania and the tin deposits of the North-East. It was the main factor in bringing a period of prosperity to Tasmania following the prolonged economic depression. During the depression there were many people who favoured annexation by Victoria. Had it not been for the pioneering spirit of James Smith the mineral potential of Tasmania may not have been recognised for many years and the colony may have lost its separate identity.

Once James Smith had pegged his first two sections large numbers of prospectors and speculators visited the Mount. No less than 36 other leases had been applied for by the middle of 1873. Smith himself pegged three other sections along Tinstone Creek, the most westerly including the spot where he first found tin on 4 December 1871. He also purchased 88 acres of land in the township. Many of these sections were subsequently worked, the most profitable being bought by the Mt Bischoff, Bischoff Extended and other companies. Smith, Crosby and several other personalities associated with the early days at Mount Bischoff are commemorated by the street names of Waratah. From March until May 1881 Waratah had its own weekly newspaper, at first entitled *Waratah Weekly News* and later renamed *Mount Bischoff Mail*. The issues contain much information on the progress of work on the various leases.

The richness of the Mount Bischoff deposits and the large profits made by the Mount Bischoff Tin Mining Company were soon world famous. This Bulletin is concerned primarily with the activities of the Company from 1873-1942 during which time over 56,000 tons of metallic tin were produced (over 90% of the tin production in the Waratah area) and the subsequent period of government operations. Although the original rich deposits are worked out and only small quantities of tin ore are now being produced, geological prospecting and ore dressing research is continuing and the large reserves of low grade ore remaining at the Mount may yet be exploited.

The discovery of tin and the early development of the mine are described in Section 1. Subsequent operations are described in the following sections devoted to mining and metallurgy. Section 4 gives an account of the geology of the Mount Bischoff district, with special reference to the mine area. The final section lists the bibliographic references cited in the previous sections.

Acknowledgement is made to the State Library of Tasmania, Hobart, the Tasmanian Museum, Hobart, the Queen Victoria Museum, Launceston and Winter's Studio, Burnie for the use of historical photographs from their collections, and to the Lands Department for the use of aerial photographs.



SECTION 1:
Discovery and Early Development, 1871-1874

E.L. MARTIN

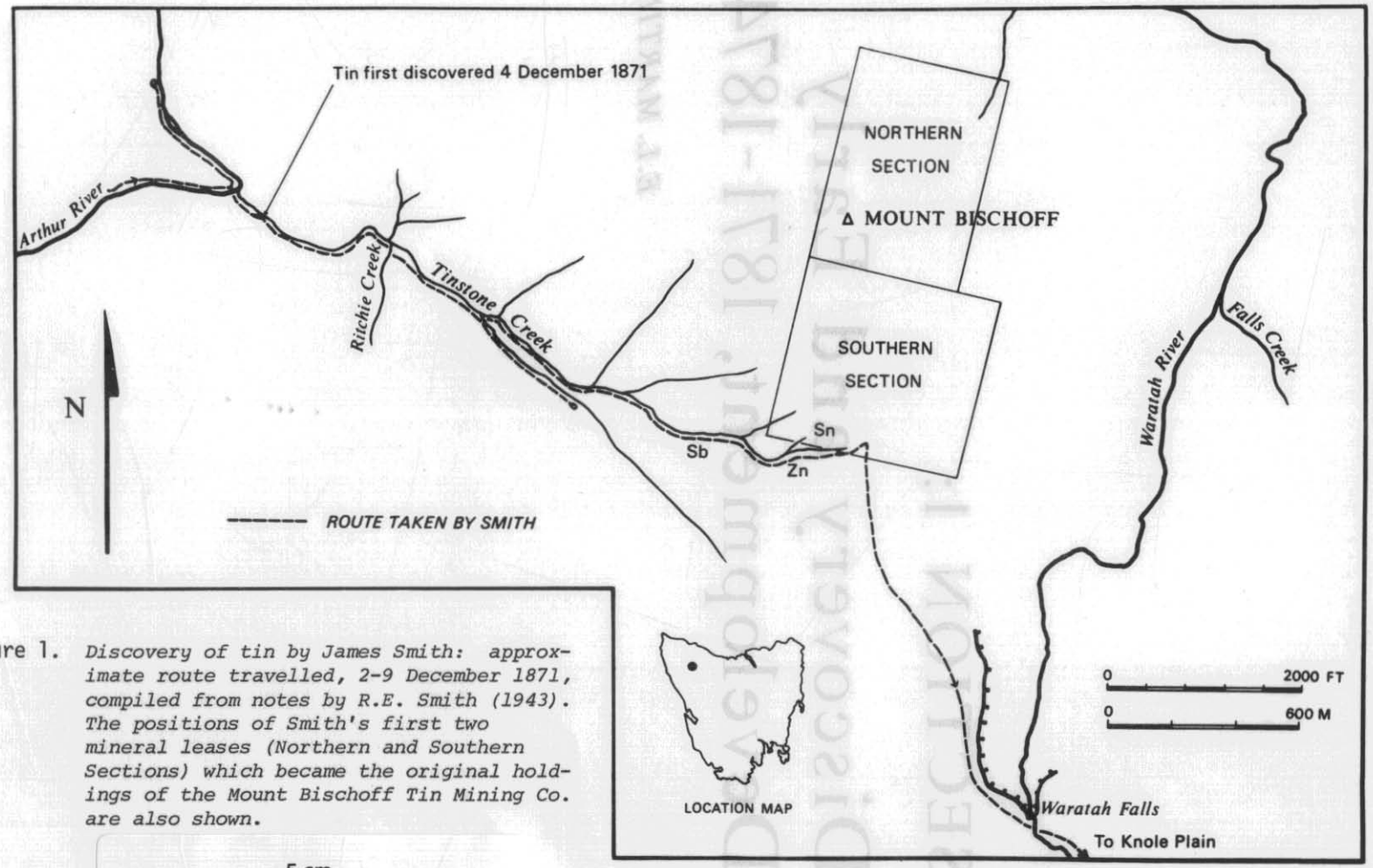


Figure 1. *Discovery of tin by James Smith: approximate route travelled, 2-9 December 1871, compiled from notes by R.E. Smith (1943). The positions of Smith's first two mineral leases (Northern and Southern Sections) which became the original holdings of the Mount Bischoff Tin Mining Co. are also shown.*

INTRODUCTION

Few publications on Mount Bischoff contain much information on the early years of the Mount Bischoff Tin Mining Company, and even less on James Smith's initial development of the mine. I have attempted, in the following pages, to give an account of this early period.

The account of the events leading to discovery of tin at Tinstone Creek by James Smith on 4 December 1871 is based on the manuscript written by his son, R.E. Smith (1943). Other information has been derived from the Smith family correspondence, preserved in the Archives section of the State Library of Tasmania in Hobart, and from contemporary newspaper accounts, mostly in the *Cornwall Chronicle*, *Launceston Examiner* and the *Tasmanian*. Although I have not been able to trace original copies of the early Half-Yearly Reports of the Company, many of the reports were reprinted in the local newspapers.

R.E. Smith (1943) has denied Kayser's (1893) claim that James Smith did not realise that it was tin ore which he had discovered and was disappointed to find that it was not silver. Although Smith was not an expert mineralogist, it seems unlikely that he would not have recognised cassiterite when he saw it.

Kayser also states (Kayser and Provis, 1895) that 'as the carriage of supplies over the existing rough bush-track cost from £24 to £30 a ton, a road to the nearest port, Emu Bay, became a necessity. In 1875 this work was completed.... (enabling) goods to be carted to the mine, in the dry season, at a cost of £6 to £8 per ton'. I have not been able to discover any factual basis for Kayser's figures. Even in 1873 Smith was able to get his ore carted for £4 per ton, and the Company paid £6-8 per ton during the summer of 1874. The maximum winter rates paid appear to have been £14 per ton for supplies and £10 per ton for ore in the winter of 1875. Certainly the track was bad and although improvements were being continually made to it, the increased amount of carting - each bullock dray carried a load of up to one ton - caused a great deal of damage especially during wet weather. Although drays could sometimes get up in the winter, at least as far as Knole Plain, pack horses were normally used. No carting was possible when the rivers were in flood, but the situation improved when bridges were built over the Wey (often referred to as the Wye in contemporary literature) and the Hellyer. Both bridges had been completed by the time of the Governor's visit in March 1875 and could be used by pack horses, although it was not until later in that year that the bridge approaches were finished and bullock teams could cross. The completion of the mine tramway early in 1875 was of great value in getting out the ore, since it replaced one of the worst sections of the track.

The close of 1875 marks the end of the pioneering stage of Mount Bischoff. The mine tramway had been completed, a water race, fed from a dam built across the Waratah, supplied water to the ore dressing sheds at the Falls, and despite the difficulties of transporting ore from the mine during the winter months the smelter at Launceston had had a successful first year. Work had commenced on the Emu Bay horse tramway, although this was not completed until 1878. Crosby had retired and Kayser, the new mine manager, had taken over and was soon to extend Crosby's use of water power at the Falls and to continue the mechanisation of ore-dressing operations.

The later history of the mine is continued in the following sections.

JAMES 'PHILOSOPHER' SMITH

James Smith was born at George Town on 1 July 1827. As a youth he worked in one of the Launceston flour mills belonging to his guardian John Guillan. He left Launceston for the Victorian goldfields in 1851, but returned to Tasmania to settle at the Forth in 1853 where he was a neighbour of James Fenton. When he first acquired the nickname of 'Philosopher' is not known but doubtless it helped to distinguish him from another James Smith who had a nearby property.

Smith undertook numerous prospecting trips in many parts of the North West Coast. He discovered gold in 1858 at Golden Point on the Forth River, a few miles below the crossing of the V.D.L. Company's track. In 1860 a government exploration party led by Ronald Gunn examined this and other mineral occurrences discovered by Smith. Smith also discovered rutile on his own land at the Clayton Rivulet.

Later, he cut his way through the bush to the Dove River in search of pines, and successfully floated logs down the Forth to Leith where they were sawn into planks. Apart from his prospecting and farming activities, he did a considerable amount of timber-getting; a profitable undertaking at the time due to the large quantities of timber being exported to Victoria.

He was, for a time, collector of the Road Rate for the district of Devon.

Smith found copper and argentiferous galena at Penguin Creek. The Devon Mining Company was formed in June 1869 to exploit this prospect and assay figures of 80-100 oz/ton were quoted. The company appears to have been still-born and operations were later commenced by the Penguin Silver Mines Company which was registered in August 1870. It had a capital of £60,000 in £1 shares and Smith, with 5,563 shares, was the largest shareholder. The operations of the company were unsuccessful.

Later, Smith came to an agreement with the V.D.L. Company under which he would undertake to prospect for minerals on their land and would receive in return 5% of the net proceeds deriving from the exploitation of minerals within 0.25 miles of any of his discoveries. His discovery of tin at Mt Bischoff in 1871 was only a few miles from the boundary of the V.D.L. Company's Surrey Hills Block.

Although Smith was actively involved in the Mt Bischoff Company's affairs during the early development of the mine he still continued to explore and prospect in other areas, although failing health caused him to curtail these activities.

He married Mary Jane Love, née Pleas, the widow of Captain James Love, in 1874.

Following his disagreement with the majority of members of the Board of Directors over Kayser in 1876, Smith severed his connection with the Company, and disposed of his shares before the first dividend was declared in 1878.

In July 1877 a public meeting was held at the Mechanics' Institute in Launceston with the object of giving public recognition of Smith's services

to the colony*. This resulted in the presentation of a silver salver and 250 sovereigns, made by Governor Weld on 8 February 1878. The Governor read the following address on behalf of the subscribers:

'Dear Sir,

For many years you have given yourself up to the work of exploring the mineral resources of Tasmania; struggling with hunger, cold, and weariness; facing with a manful courage all kinds of difficulties and disappointments, until, to use your own words, "death or victory should settle the question."

Such quiet heroism, apart from all questions of success, would have merited and won our approval; but we are glad to be able to congratulate you as the first discoverer, on Monday, the 4th day of December, 1871, of a payable tin mine in Tasmania, thus preparing the way for other discoveries of great value.

It is not saying too much when we declare our firm conviction that to you, more than to any living man, our colony owes its present state of prosperity. As the fair result of your discoveries property has increased in value, commerce has developed, the tide of immigration has turned to our shores, and all classes of the population have been benefited.

As a community we owe you much honour, because we know that in your long search you were more anxious to benefit the colony than yourself; and as a small proof of our sincere gratitude to you as a public benefactor, and our high appreciation of your character as a man, we respectfully beg your acceptance of the accompanying purse of 250 sovereigns.

Wishing for Mrs Smith, yourself, and children long life and much happiness,

We are, dear Sir, on behalf of subscribers,

WILLIAM LAW, *Chairman*
WILLIAM RITCHIE, *Hon. Secretary*
E.D. HARROP, *Hon. Treasurer*'

and continued 'Mr Smith, in making this presentation I wish to add a few words of my own. I accepted the invitation of the Committee with great pleasure, because I felt that the tribute to be paid to you was thoroughly well deserved; and it is one of the highest privileges pertaining to the office of a Governor to assist in rewarding merit. Your past services have been remarkable, not only for the inflexible determination with which they have been carried out, and the privations that attended them, but also for the great success by which they have been crowned. But no success can be deemed quite complete unless it has with it some touch of self-sacrifice. In my early days I was often engaged in exploring in New Zealand, and I know something of the hardships that have to be encountered in such an occupation. I am well aware that your privations and exertions have been far greater than mine; but you will be able to look back upon them with greater pleasure, and to feel the satisfaction that, as the result of your labours, happiness has been bestowed upon others, and numbers of families have been comfortably brought up and educated. As Governor of the colony I rejoice at its prosperity, and I shall always feel that in great part it is owing to your exertions.'

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* A similar proposal had been made in 1868 by John Regan, but apparently James Smith was against the idea at that time.

The recognition of his services by Parliament brought him a life pension of £200 per annum.

According to Ramsay (1958) Smith operated a brick and pipe works at the Forth during the early 1880s.

Smith represented Mersey in the Legislative Council, from 1886 until 1888. He died at Launceston on 15 June 1897.

WILLIAM MORGAN CROSBY

Crosby emigrated from Canada and settled at the Don in 1853. His early years in Tasmania were spent in the timber industry. He built the first bridge over the Forth in 1859. With Anthony Raymond he made an epic trip up the Forth River by boat in December 1868, and succeeded in reaching the 'Gates' about 40 miles upstream, but although they continued on foot and managed to reach the V.D.L. Company's crossing they saw no pines. Early in 1869 they reached the River Dove by an overland route using the recently cut Penguin track and were able to repeat James Smith's feat of sending logs down to the Forth.

Although Crosby had no mining experience, his age and disposition, his bent for exploration and his experience with timber made him an obvious choice when Smith was looking for someone to take charge of the development of his mining leases.

Following a period of recuperation after his retirement from the Mount Bischoff Company, Crosby was engaged by the Tasmanian Bismuth and Gold Mining Company to report on water facilities at Mt Ramsay. Later he prospected for tin in north-eastern Tasmania. He later retired to the Forth and died in 1916, aged 80.

DISCOVERY OF TIN

James Smith left the Forth in October 1871 on a prospecting expedition and reached Knole Plain late in November. He then followed the partly overgrown track cut by Gordon Burgess in 1864, reached the Arthur River and examined the country to the west towards Mt Cleveland. On 1 December he decided to return to the Arthur and on the following day reached what he believed, from the old inaccurate map he carried, to be the junction of the Waratah and the Arthur. He camped there and rested until Monday, 4 December 1871, and then proceeded a short distance up the 'Waratah' where he obtained from the bed of the stream a very small quantity of what appeared to be, in the dim light of the forest, cassiterite. He returned to his camp and then descended the Arthur; on 6 December he reached a small sunlit clearing and was able to examine his tin sample more closely. Some of the cassiterite was very little waterworn so that he assumed the source of the ore could not be far distant. He returned to the 'Waratah' and found slate and schist bedrock and porphyry boulders at the place where he had obtained the cassiterite. On 8 December he started to trace the porphyry boulders to their source; he followed the stream until late afternoon when he made camp. He could find no tin near his camp although he did find some galena. On the following day he retraced his steps to a small tributary stream flowing from the north-east. He ascended this stream, sampling with his prospecting dish as he went, and at one spot obtained 0.5 lb of tin ore in a few minutes; further up the stream he traced the tin to its source in what was to become his southern section.

As he was short of provisions he attempted to reach Knole Plain by walking south-eastwards. Because of the dense forest cover he had to detour to the east and reached a point where a large stream fell over a basalt cliff. He realised that this stream must be the Waratah, and the stream which he had previously believed to be the Waratah he decided to call Tinstone Creek.

He reached Knole Plain on the evening of Saturday, 9 December and rested until Monday morning. He now had very little food left, but decided to return to Tinstone Creek to do further prospecting, before making for a stockman's hut some ten miles away to replenish his food supplies.

In March 1872 he applied to the Minister for Lands and Works for two 80 acre sections*. He was asked to provide further information regarding the location of his claims: presumably there was some doubt as to whether they were on Crown land or on land belonging to the Van Dieman's Land Company. The nearest V.D.L. land was later shown to be only a few miles away.

He set out for Mt Bischoff again in May 1872 and was joined by E.B.E. Walker and Maurice Duncan at Hampshire Hills. They cut a foot track from Knole Plain to Tinstone Creek. With the information gained from this visit, Smith applied again for his mineral sections. The southern section (no. 7) was surveyed by Charles Sprent in September and the lease was dated 1 November 1872. The northern section was surveyed early in 1873 and the lease dated 1 April.

Once the leases had been issued Smith could commence mining operations.

MINING OPERATIONS BEGIN

Although Smith's aim was to form a company to work his sections he had first to prove their value. Not only was there much prospecting work to be done, but access to the leases had to be improved.

In August he accompanied Charles Sprent's survey party, clearing fallen trees from the track to Knole Plain to enable the bullock dray to pass. The foot track to Tinstone Creek was improved. Later, Smith engaged two workmen to widen this track and make it passable for drays; the track was cleared to the west side of the Waratah by 13 December 1872. Mining and prospecting operations, under the supervision of William Morgan Crosby, commenced on the following day.

At first a number of trial holes were dug with pick and shovel to sample the washdirt and to try and locate the course of the lodes. By the end of January 1873 about a ton of tin ore had been carted to Emu Bay and the ore was being produced at the rate of 3 cwt per day, presumably by panning.

It is not clear exactly when cradling was introduced, but by 12 March when Charles Meredith, the Minister for Lands and Works, visited the mines Smith had 16 men at work and 3 cradles were in use. About 3 tons of ore had been produced, some had been sold in Melbourne for £80 per ton and the remainder transhipped to London for sale there. The absence of a jetty at Emu Bay meant that the S.S. *Pioneer*, which provided a regular service between Launceston and the North West Coast, could only call if the weather conditions were suitable. A jetty was under construction, and although part

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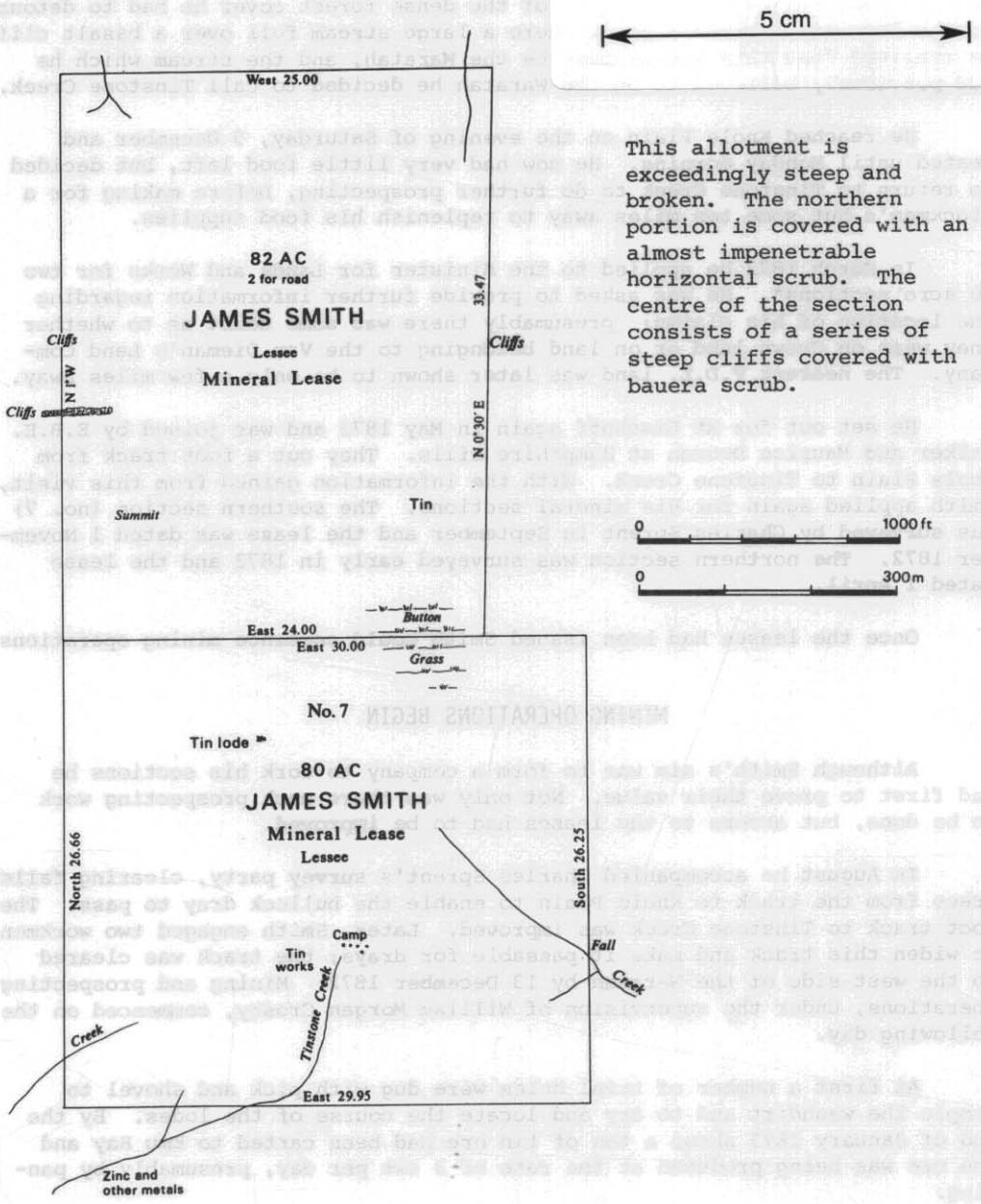
* 80 acres was the largest area that could be applied for under the Mineral Leases Act, 1870 for the purpose of mining minerals other than coal.



This allotment is exceedingly steep and broken. The northern portion is covered with an almost impenetrable horizontal scrub. The centre of the section consists of a series of steep cliffs covered with bauera scrub.

The above lot is covered with a dense scrub of horizontal, tea tree, cutting grass and bauera. The timber is celery topped pine, myrtle and a few sassafras and stunted white gum.

Figure 2. *Surveys of James Smith's first two mineral sections, redrawn from Charles Sprent's original survey diagrams of 1872-1873, and with Sprent's comments on the terrain.*

of it was swept away by rough seas it was completed towards the end of 1873, a tramroad laid along it and a wooden crane erected.

Smith had raised about 7 tons of ore by the middle of April, but the roads had suffered due to the passage of the drays which carried loads of up to 1 ton, and heavy rains made the road impassable to teams during much of May and June, although pack horses could still reach the mines when the fords at the Wey and Hellyer rivers were passable.

About 10 or 11 tons of tin had been raised by 1 August 1873, which was the official start of the Company's operations although effective control was not exercised until September.

Smith paid £4 per ton for the carting of his ore.

FORMATION OF THE COMPANY

Once mining operations commenced there was a constant stream of visitors to the mine. These visitors seem to have been impressed with the richness of the mine and some of them applied for leases adjoining those of James Smith, others wanted to purchase an interest in the mine. One such offer was made by Captain Noyes on behalf of a Melbourne syndicate and although this was declined by Smith, Noyes urged him to work the mine in a more extensive way and he would 'find the money'. Smith also rejected offers to work the mine on tribute.

Smith's aim was to float a Company and he went to Launceston in April for this purpose; whilst he was there he had a good offer from a Victorian, which led to an invitation to go to Melbourne. The negotiations were successful and the Waratah Tin Mining Company was formed. Smith arrived in Launceston on 6 May en route for the mines. He was accompanied by several shareholders and a 'practical miner who has been appointed by the Company to thoroughly test the ground'. If their report was favourable the company would be registered and mining operations would commence at once. Although there could have been no doubt of the abundance of tin on Smith's leases the Victorians seem to have been put off by the remoteness of the region, the difficulties of transporting the tin ore over the rough bush tracks, and the absence of port facilities at Emu Bay.

Smith was still determined to float a Company, although he had been offered an overdraft by the Union Bank to enable him to continue operations on his own account.

In July 1873 Smith induced William Ritchie, a prominent Launceston lawyer, Colonel Crawford and James Cowle to accompany him to the mines. Ritchie had an interest in several of the recently-formed Tasmanian gold mining companies, and he had paid a visit to Mount Bischoff in January 1873 shortly after mining operations commenced. Their visit led to the issuing of the following prospectus (see *e.g. Launceston Examiner,* 9 August 1873):

PROSPECTUS OF THE MOUNT BISCHOFF TIN MINING COMPANY, TASMANIA

To be registered under *'The Mining Companies Limited Liability Act, 1869'*

CAPITAL: £60,000 in 12,000 shares of £5 each. Four thousand four hundred to be accounted fully paid up and reserved for the proprietor, 7,600 to be allotted on the following terms. Five shillings per share to be paid on application, 5s on allotment and a further 15s per share at four months

from 1 August 1873. After the sum of 15s per share has been expended, the profits of the company shall be applicable to the construction of works and the purchase of plant and machinery. The residue shall be liable to call, but so that only one call may be made at a time; that no call shall exceed 2s6d per share; that there shall be an interval of at least one month between calls; and that after £1 per share shall have been called up there shall be no further call except by consent of an Extraordinary Meeting of Shareholders.

The mining property of the Company will consist of the two mineral sections of 80 acres each at Mount Bischoff held by Mr James Smith under the Mineral Leases Act, 1870, and upon which mining operations are now being carried out.

The property has been recently visited by Colonel Crawford and Messrs James Cowle (late Deputy Surveyor-General of Western Australia) and William Ritchie, who satisfied themselves by careful inspection as to the existence of tin in unusual abundance in numerous parts of both sections, and of the presence of a lode of tin of great magnitude. Samples of both may be seen at the Secretary's office. Samples of the alluvial tin, assayed by Mr Cosmo Newbery, Government Analytical Chemist of Victoria, yielded 64% of tin; and a lump from the lode assayed by Mr Charles Watt, Assayer, of Sydney, yielded 69.2%.

The lessee is prepared to dispose of his interest in the sections referred to in consideration of receiving 4,400 paid up shares above mentioned and £1,500 in cash out of the capital of the Company. He will retain the ore raised up to 1 August 1873.

Mr Smith makes the following stipulations in order to protect the large interest he will retain in the property, *viz:*

(1) That there shall be seven Directors, of whom he shall be one so long as he shall continue to hold 1,000 shares in the Company and that while a Director he shall have the power to nominate another who shall be the holder of at least 100 shares.

(2) That each shareholder shall be entitled to a vote in respect of each share held by him up to 2,000 shares.

The Company shall be considered formed as soon as 5,000 shares are subscribed for. Three thousand shares have already been applied for and will be deemed allotted. Two thousand shares have been reserved for disposal in Victoria. There remain therefore only 2,600 shares for which applications can at present be received.

Applications will be received by the Secretary up to 15 August 1873.

Henry Ritchie

A special general meeting of shareholders was held on 29 August and the few remaining unallotted shares were quickly taken up at the end of the meeting. Henry Ritchie, a solicitor and estate agent, was appointed Legal Manager of the Company and William Crosby was appointed Mining Manager.

The Company was registered in October 1873.

COMPANY OPERATIONS, 1873-1874

Although much time had to be expended in repairing the roads, cutting tracks and clearing scrub, Crosby was able to start sluicing operations near the camp. The camp at this time consisted of a windowless hut, 32 ft x 10 ft, built of split fern trees and celery top pine, with a roof of waterproofed calico to admit the light. In addition to providing accommodation for the miners it was also used to store the bags of tin ore.

At the site of the original workings where washdirt was dug from a cutting about 200 ft long and 4 ft deep, two 11 ft long sluice boxes with perforated false bottoms, and a buddle were supplied from a 30 ft water race. Crosby's 'buddle' appears to have been a simple box or strake into which the washdirt was shovelled and raked whilst a stream of water was allowed to play over it to remove the clay and silt. The ore was removed from the 'buddle' and then dried in an oven before being bagged.

The new workings were at the head of a gully about 30 yards from the hut, where 'stream tin' was found in abundance at depths of from 1-9 ft. A small reservoir was dug at the top of the gully and this was connected by a water race about 450 ft in length to a small stream, to provide water for sluicing operations.

Work was also proceeding on the North Lode (in the northern section near the summit of Mt Bischoff), following the cutting of a track from the southern section, and the South Lode (half-way up the mountain). A huge boulder, dug from below the South Lode provided 4 cwt of ore which assayed 80% tin. The first recorded use of blasting powder was on 13 August 1873 at the North Lode where 0.75 tons of ore were broken, including 3 'nuggets' weighing a total of 164 lb. Trial shafts were being sunk between the two lodes.

Crosby's report for October (see *Cornwall Chronicle*, 5 November 1873) states that 21 men were employed: 5 on opening the North Lode, 2 repairing the road from Knole Plain to the mines, 5 carpenters, 3 clearing ground and 'scrubbing', 2 cutting water races and preparing ground for sluicing purposes, 1 blacksmithing and charcoal burning, 1 carrying supplies from Knole Plain, 1 splitting timber and 1 cooking. At the North Lode a face was being cut along the crest of the hill and was 30 ft long and reached a maximum of 10 ft in height. The face consisted of a mass of boulders, soft tin-bearing stone and washdirt. The richest boulders were at the top. There was no sign yet of a 'solid, defined, lode'. About 150 tons of stone had been broken out and divided into 3 lots (Nos. 1, 2 and 3 ore). Sluicing operations had been discontinued due to sickness, preparing accommodation for the men and other urgent work. Two sets of sluice boxes had been prepared, one set 95 ft in length, and a further set was in course of construction. The sluices were set at different elevations so that they could be operated by the same stream of water. Even though these were supplied by the water race they could only be worked during wet weather. In dry weather the men employed on sluicing were set to work on the adjoining South Lode. A hut for Crosby was built from celery top pine boards sawn at the mine and was roofed with waterproofed calico.

During November 1875 work continued on the North Lode and a trench proved tin-bearing stone to a depth of 50 ft below the crest of the hill; below this, in a tea tree swamp, 3 ft of rich washdirt overlay the slate bottom. About 300 tons of stone was taken out during the month including 40 tons of No. 1 ore and 45 tons of No. 2 ore. A small gang was employed opening out the South Lode and had cut a face about 20 ft long and about

20 ft high. About 150 tons of stone were taken out including 3 tons of No. 1 ore. Crosby states that the South Lode seemed to be solid and irregularly defined and was situated about 325 yd from the North Lode. Four shafts were being sunk, and some of the spoil from these was yielding up to 1 lb to the dish.

A dam, 180 ft in length, was constructed during December across the outlet from the swamp in the northern section in order to retain water for sluicing. Work on the shafts continued and some of the spoil was yielding as much as 3 lb to the dish.

By the end of the year 60-70 tons of No. 1 ore had been won, also several hundred tons of 'inferior quality' ore. However only a little over 6 tons had been carted to Emu Bay. At the report of the meeting of shareholders in January 1874 the Directors were strongly criticised for not getting more tin ore carted, especially as A.M. Walker was able to cart 20 tons from his claim. Although Walker had his own drays it seems that the Company were not willing to pay competitive rates for carting. The rate they normally paid was £5 per ton for carting provisions from Emu Bay to the mines and £7 per ton for carting tin ore to the coast. As much as £10 per ton was paid during the winter season. An agreement was reached between the Mt Bischoff Company, A.M. Walker and Cummings, and Henry & Company for the joint construction of 1.25 miles of tramway from the mines to the Waratah Falls.

The mine workers were paid from 2s to 4s2d per day and received rations equivalent to 13s per week.

Towards the end of 1873 the company purchased a 15-head stamp battery from the Shamrock Gold Mining Company for £300. Although it was never erected it proved to be a good investment for it was later sold for £650 when Ulrich reported that the erection of a crushing plant was premature. The Company also purchased 2,000 ore bags from the ill-fated Penguin Silver Mines Company.

Following the Company meeting in January 1874 the Directors decided to obtain the services of a consulting mining geologist and engineer, George H. F. Ulrich of Melbourne. Ulrich visited the mine in March and his report was published in April 1874.

Before Ulrich visited the mine there was an important change in the method of working. Now that the dam across the outlet from the 'flat' had been completed the new reservoir could supply water for sluicing. Since the reservoir had a limited catchment a method of working was adopted to enable the water to be re-used. A single sluice was put into operation about the middle of January: it was set up behind the dam and fed with surface material from the floor of the reservoir area. The water was supplied by means of a hand pump worked by one man. The waste water was allowed to flow back into a pit which had been dug behind the dam wall. About 0.5 tons of tin ore were produced each day. By early February another sluice was in operation and production was increased to 1 ton per day. The sluice boxes were arranged alongside each other and emptied on to the same ripple box or table. A 5 hp vertical steam engine and a centrifugal pump had been ordered and were being installed at the time of Ulrich's visit.

Ulrich gives the following description: 'The working of the drift in the basin consists simply in excavating the drift off the slate bottom and removing it in wheelbarrows to two sluices, from which the ore saved is transferred to several strakes or ties for final cleaning. The water required for these appliances is pumped by several men from a small reservoir excavated

in the slate bottom, and dammed across the outlet of the flat....although the principal part of the tailings is removed in ditches before [the water] re-enters the reservoir, its still muddy character, barring other considerations, is no doubt detrimental to the saving of the fine tin ore - in part real ore-slime'.

Ulrich's special recommendations were:

'(1) Not, as already advised to resume working on the north and south lodes at least for some time, nor to institute special search for similar or other kinds of original deposits of the tin ore, for the reasons that the great extent and thickness of richly stanniferous drift presents to the Company a less expensive and more certainly profitable field for working, and that during this work the baring of the rock surface will easily lead to the discovery of any original deposits existing.

(2) Not to spend any money in the premature erection of crushing machinery, but rather:

(3) To take steps towards improving the washing process and appliances, by sending Captain Crosby to Victoria, where, at Eldorado and other parts of the Beechworth gold field, he can see the best methods of sluicing stanniferous drifts and cleaning the ore, and where in the Melbourne Technological Museum I should gladly show and explain to him models of classifiers, jiggers, buddles, etc., applicable to the concentration of tin-ore slimes.

(4) To commence smelting of the tin-ore at the mine as soon as possible, partly on account of saving carriage, partly because I apprehend that neither the washed ore, nor much less that obtained from the lodes, would on account of color and general outward appearance command a ready sale in the English market, at least not at prices commensurate with their intrinsic value. Clays suitable for the manufacture of bricks for furnaces do, I believe exist in the locality. And Captain Crosby visiting Eldorado could there also see a simple and inexpensive kind of tin furnace in actual work.

(5) To lose no time in properly housing in the washing place in order to avoid stoppages of the work during the rough weather frequently experienced on the Mount.

(6) To purchase one or two teams of bullocks or several horses and the necessary drays and harness for use at the mine, in order to prevent for the future such serious delays as have already occurred in the supply of necessary mine-material, &c., by private carters.'

At the request of the other Directors, James Smith visited the mainland in February and March 1874. He left Launceston for Melbourne on board the '*Derwent*' on 16 February and he later visited Clunes, probably to see the Port Phillip Company's Works at which some of the latest types of ore-dressing machinery could be seen. He then left for Sydney and went to the Pyrmont Smelting Company's works where 5 tons of Mt Bischoff ore were being smelted. The results of this smelting were satisfactory, although not quite as good as hoped for, possibly because the method of smelting was not ideally suited to the Tasmanian ore.

When the steam engine was installed at the beginning of April it did not work well and could only lift water for about 15 minutes at a time. Two furnaces were built, one at each side of the boiler. Crosby had a machine

made 'for cleaning the coarse nuggety tin' which is said to have worked well.

Following Ulrich's report the Directors agreed that Crosby should go to Victoria to visit the Eldorado mine and also to see Ulrich at the Technological Museum in Melbourne. He went to Victoria in early May, leaving Pearson in charge of mining operations. He was instructed to make full enquiries about the cost of erecting a smelter.

By the time Crosby returned from Victoria the steam engine was working well - although it cost twice as much as when hand power was used. A bulkhead was built around the water intake for the pump so that the cleanest water could be used. The average height of water in the dam was raised to 5.5 ft and the reservoir occupied nearly 1.5 acres.

During June 50 men were employed and 5 sluices were in operation, producing about 2 tons of ore per day.

Much attention had been given to improving the Emu Bay road. The old road had been improved and a new, shorter road was being cut. The cost of this work (January-June 1874) was £925 which was shared between the Mt Bischoff Company, the V.D.L. Company, Walker & Beecraft, Cummings, Henry & Co. and J.H. Munce. The Government had let a contract to Boland for the erection of bridges over the Wey and the Hellyer rivers.

The Company was still having difficulty in getting the ore from the mine to the coast. They were apparently unable to purchase any bullocks or drays, and were unwilling to pay competitive rates for cartage. They made an agreement with Charles Adams of Pipers River to cart 100 tons of ore during the coming summer season at £6 per ton (£4 per ton for supplies). Adams stipulated that the Company must erect stables at Waratah, Emu Bay and at some intermediate point. The Company purchased 504 acres of grazing land at Knole Plain for £1 per acre, agreed to rent Rutherfords farm of 130 acres (4 miles from Emu Bay) for 7 years and to lease 200 acres at Ridgley, 12 miles from Emu Bay. A tramway from Emu Bay to Mt Bischoff to be built by the V.D.L. Company was first mooted during 1874.

Following Ulrich's suggestion the Company applied for a site at the Waratah Falls for washing and for the erection of furnaces. Apparently the Directors had second thoughts regarding the location of the smelter, which they decided to build at Launceston. This decision was strongly criticised by many shareholders. Although the Directors said it would cost more to bring coal to the mine than to bring the ore to Launceston, it would have been possible to use charcoal, as was done at the Stanhope Company's smelter which was later erected at Waratah. There was apparently a move to form a tin smelting company in Launceston at that time and I suspect this was the key factor which influenced the Directors. The Directors' choice of location was a fortunate one for it ensured the success of the smelter, enabling it to purchase alluvial ore from the new mines in the North-East and which could be satisfactorily blended with the Mt Bischoff ore.

Wages paid to men at the mine ranged from 20s to 30s per week, the highest rates were paid to the best men working the sluices. Boys were paid at lower rates.

By the end of July 1874 a tramway had been constructed to bring the washdirt over the heads of the sluice boxes. The trucks were built at the mine. Fluming, about 200 yd in length, led from the lower end of the sluice boxes, over the dam and continued down the creek leading from the flat. When water could be spared some of that lifted by the pump was diverted in

to the fluming and the heaps of sandy tailings were shovelled in and run to waste. Some tin was caught in the fluming and recovered, and a set of sluice boxes which had been constructed at the lower end of the fluming was now ready for use. A little prospecting work was being done from a 34 ft stope at the bottom of the hill and a 9 ft shaft near the North Lode, to test for the continuation of the rich sandy wash dirt. New wheelbarrows were made at the mine and the iron wheels on the existing wheelbarrows were being replaced by wooden ones since they 'gave out very quick'.

With the assistance of the manager for Cummings, Henry & Co., Crosby selected a site for a dam on the Waratah and levelled through for a water race. The costs of cutting the race were to be apportioned between the two companies. For ease of construction the race was to be on the east side of the Waratah, although this would mean that it would have to be carried over the river at the falls.

Winter brought the usual transport difficulties. Although over 30 tons of ore were produced during July, only 1.9 tons could be carted and nearly 130 tons were on hand at the mine.

Crosby tendered his resignation on 4 August, the reasons for this are not clear, but presumably there must have been some criticism of his management of the mine. He was persuaded to continue, however, following Henry Ritchie's assurance that 'the Directors are quite satisfied with your management and hope that you will not think of leaving their employment'.

Crosby went to Launceston in September to make arrangements for the construction of ore-dressing machinery. His plans for 'a revolving cylinder, double troughs, a jigging machine with percussion tables' were approved by the Directors.

Since the Company had received no tenders for the construction of the mine tramway it was decided to commence the work using day labour. By the end of the year the 1.25 mile route, which included a 180 yd long self-acting incline in the Company's southern section, had been cleared and most of it graded. Nearly half the wooden rails and ties had been cut. The costs were to be apportioned as follows:

(1) For the section from the Falls to the Company's south line:

Mt Bischoff Tin Mining Co.	One half
Walker & Beecraft	One quarter
Cummings, Henry & Co.	One quarter

(2) For the section from the Mt Bischoff Company's southern boundary to Walker & Beecraft's boundary:

Mt Bischoff Tin Mining Co.	Two thirds
Walker & Beecraft	One third

In December, James Smith paid a visit to the mine with W.L. Jenkin who had been appointed manager of the smelter. Jenkin appears to have been very impressed with what he saw there.

Production of ore for the period July-December 1874 was 154.5 tons, bringing the total produced by the Company to 324.1 tons. At the end of December 246.5 tons were at hand at the mine; a mere 9.6 tons having been carted since July.

By the end of 1874 the survey of the route for the Emu Bay - Mt Bischoff tramway was under way and the Company had made an agreement with the

V.D.L. Company for the carriage of ore at £4 per ton. The dressing sheds at the Falls were under construction, a store 20 ft x 30 ft and 3 cottages, 15 ft x 12 ft had been erected and foundations put in for two more cottages. The water race had been cut from the dam site to the falls, part of it had been blasted through solid basalt. The race was 5 ft wide at the bottom and from 3-7 ft in depth. A dam on the Waratah 100 ft wide and 13 ft high was under construction to turn the water into the race.

THE FIRST SMELTER

Once the decision to erect the smelter had been made, a site at the corner of William and Tamar Streets was leased from William Hart. W.L. Jenkin who appears to have been an experienced smelter from Sydney arrived at Launceston on Board the '*Tamar*' on 10 October 1874. Meanwhile bricks had been selected in Melbourne. By the beginning of November the foundations for the two furnaces were complete and a small assay furnace had been erected in the store.

At the end of 1874 the smelter was virtually complete and was described by Jenkin (*Cornwall Courier*, 28 December 1874) as follows:

'The works for the present consist of two furnaces (reverberatory), and a stack or chimney of sufficient capacity to produce enough draught and consequently heat, to work both. This is connected with the furnaces by means of a flue and culvert carried from the reverberator of the furnaces up to above 6 ft 6 in from the base of the stack. The furnaces, overall, are 17 ft 3 in long and 9 ft 4 in in extreme width and capable of smelting from 25 up to 30 cwt of tin ore in each charge, the quantity varying according to the quality of the mineral to be operated on. The stack, now completed, is 65 ft in height. Attached to each furnace are two refining pots, which also serve as receivers when tapping the charge of metallic tin. Under these pots is fitted a small grate on which a little fire is kept to raise the temperature sufficiently to prevent the pots from cracking when tapping into them molten metal. The furnaces are covered in by a main shed 50 ft x 30 ft with a verandah extending 12 ft on one side and at each end of the building, and roofed with galvanized iron. In connection with the works are the various outbuildings, consisting of a large galvanized iron store for the reception of the tin ore, metallic tin ingots, iron for smelting, tools etc., also a laboratory and blacksmiths shop'. Jenkin also gives a general outline of the smelting process.

At first only one furnace was used. This was first charged at 7.30a.m. on Monday, 4 January 1875. Three charges of ore, of about 1.25 tons each, were made on that day, and 4 tons of fuel were used for all purposes (smelting, heating the refining pots, etc.). The yield of metallic tin assaying about 99.7%, was about 70%.

On 7 January the furnace was charged with ore from Walker and Beecraft's claim. Before this ore could be smelted it had first to be crushed by hand. The second furnace was started on 25 January.

Soon tin ore from the mines in the North-East was being brought to the smelter, it was mostly carried by small coastal schooners which loaded the ore at the nearest river and landed it at the wharf near the smelter.

The barque '*Araunah*' left Launceston for London on 16 February carrying 751 ingots (27 tons) valued at £2,160: this was the first direct shipment of Mt Bischoff tin to England. Soon afterwards the barque '*Grasmere*'

loaded 22 tons the the *'Fugitive'* 16 tons. By the middle of February a total of 70 tons of tin had been produced. One furnace was used for Mt Bischoff ore, the other for Walker & Beecraft's ore.

The fourth report of the Company (July 1875) as reported in the press, contains no reference to the operations of the smelter, although a detailed statement of ore received and smelted since the beginning of operations was given in the *Launceston Examiner* of 7 August 1875:

	tons	*cwt*	*qr*	*lb*
ORE:				
Received from Mt Bischoff Co.	311	16	1	22
Smelted	283	10	0	22
On hand	28	6	1	0
Purchased from N.E. companies	13	17	0	16
Smelted	9	2	2	26
On hand	4	14	1	18
Ore received from Walker & Co.	171	7	3	21
METALLIC TIN:				
Shipped for Mt Bischoff Co.	177	6	0	0
On hand	19	7	0	0
	196	13	0	0
Shipped for Walker & Co.	88	10	2	18
On hand about -	1	9	0	0
	89	19	2	18
Shipped since opening of works	286	12	2	18

With the arrival of winter the ore supplies to the smelter dwindled and continuous operation was not possible and smelting ceased on 20 May. During the last half of 1875 one furnace was in operation for only 12 weeks, producing 139.8 tons of metallic tin from 216 tons of ore. Less than half this ore came from Mt Bischoff.

By the end of 1875 a third furnace was completed and a fourth was half finished. Whilst smelting ceased the opportunity was taken to lengthen the brick wall on Tamar Street by 112 ft 7 in and extend the smelting shed by 70 ft 6 in. Much of the yard was 'made up and substantially fenced around'.

During 1876 a Chilian mill, engine and boiler were erected. The Chilian mill was used for crushing ore and slag and also old firebricks. The crushed bricks yielded 'a superior fire clay, particularly well suited for setting furnace bottoms'. The crushed slag yielded 9 tons of tin. By the end of 1876 five furnaces were operational, another almost completed, and a new chimney built. The smelting capacity (with 6 furnaces in operation) would be 80 tons per week.

MINING OPERATIONS, 1875

The continued strenuous work involved in opening up the mine and the miserable weather conditions had combined to affect Crosby's health, and he was forced to retire. His resignation took effect from 1 November 1875. H.W.F. Kayser was appointed as the new Mine Manager on the recommendation of Ulrich. The year was also marked by the visit of Governor Weld to the mines on 3 March.

The great expense incurred in developing the mine made it necessary for the Company to obtain more money. An Extraordinary Meeting of Shareholders was held on 14 May and the Directors were authorised to make a call of £2 per share and to borrow up to £15,000 from the bank. The call does not appear to have been made.

During the first half of the year sluicing operations were mainly confined to the southern section, partly because the steam engine was frequently breaking down. There were 3 sluices at different elevations using water obtained from the reservoir. The face at the upper sluice was 10 ft high with pipe clay at the bottom, the washdirt was very rich. The middle face was 22 ft high and not bottomed, the washdirt was very rich but contained a large number of boulders; much labour was needed to remove them. The lower face consisted of a yellow clay and had not yet reached the rich washdirt.

Ten men were employed at each sluice. The daily output of ore from the sluices was about 2 tons, almost all of this being from the middle and upper sluices. The ore was packed to the tramway and then conveyed in trucks to the Falls where it was dressed. The ore-dressing machinery was driven from a 20 ft diameter waterwheel. The trucks from the sluices were discharged into a kind of hopper built of logs and with a floor sloping at an angle of 8 or 9°. The ore in the hopper was kept loose by a stream of water, so that it could be fed automatically to a launder. The launder, which was lined with sheet iron, was 20 ft in length and had a fall of 1 in 18. The launder discharged into a revolving cylindrical trommel, 8 ft in length and 3 ft in diameter. The trommel had 3 sieves for sizing the feed. The trommel was stated to have been driven at 50-60 rph, however this seems extremely slow for this type of trommel and it may be that the figure given is in error.

Gravel and coarse dirt screened from the trommel was conveyed to a sluice box* 1.75 ft wide and sluiced with a strong current of water which carried off the light particles, leaving the heavy behind at the head of the box. The head of the box was cleaned up when necessary and passed to a hand jigging machine for final cleaning. The 'skimpings' or scrapings from the top of the jigger were stacked on one side to await crushing. The stones turned out of the trommel are picked by hand, since many were tin 'nuggets'. The fine material containing ore (sand and slime) passed from the finer sieves of the trommel into a 20 ft long launder where the 'shotty' or granular tin was saved. At the foot end of the launder there were five strips to catch the fine tin. Preparations were being made for the construction of slime- or settling-pits into which water from the strips, buddles and tyes would be conveyed before being allowed to escape.

The granular tin saved in the long launder was transferred to 2 tyes where most of the coarse grains of porphyry were removed. It was then buddled and finally sluiced in a compact sluice box; the tailings from the sluice were run into 2 catch pits (each 8 ft x 4.5 ft x 2.5 ft deep). These pits are recorded as saving 'a great quantity of tin ore' which was re-buddled and then kieved. The kieves were 2.5 ft deep, from 2.5-3.5 ft in diameter, and shaped like a tub. One man shovelled the dry material to be treated into the kieve whilst another man kept it stirred with a long-handled shovel. When the kieve was sufficiently full, its sides were tapped with a piece of wood so that the heavier material would be concentrated at the bottom. This operation was repeated until the ore was sufficiently clean.

The dressing sheds measured 60 ft x 40 ft and contained 5 buddles, 4 strips, 2 tyes, a hydraulic separator, several sluice boxes, a hand-jigging

*In another account a hydraulic separator (spitzlutte) is mentioned: presumably this was also used ahead of the jig.

machine and several other minor ore dressing appliances.

A blacksmiths shop and cottage was built at the Mount, and 6 cottages and a stable built at the Falls.

The double line of tramway and self-acting incline from the Mount to the Falls was in working order, and the grading bridging and trestlework for the 2.25 mile single line tramway to the open country was complete and most of the track had been laid. The dam on the Waratah (later known as the Town Dam) was complete and the water race and fluming complete except for a short section at the approach to the trestle bridge above the Falls. The tramway ran along the top of the trestle and the fluming of the water race was placed at a lower level.

The bridges over the Wey and the Hellyer had been completed by Boland, but since the approaches had not been completed the bridges could only be used by pack horses and the drays still had to ford the rivers.

The summer had been dry and the carting season a successful one. Over 294 tons were carted, whereas from the start of the Company operations until the end of 1874 only 78 tons had been carted. During the first half of 1875 126 tons of ore were obtained, bringing the total amount of ore raised by the Company to 440 tons.

Thomas Rowlandson visited the mine in July to give a demonstration of his system of hydraulic sluicing. It is reported to have been only partially successful. However the use of the hose was evidently effective in breaking down the face - also in removing the trousers of one of the bystanders! Hydraluic mining appears to have been adopted for some time, although there are very few references to it. The water was led from the old reservoir at the Mount to a hose made of cotton canvas. The hose was tapered until it was 6 inches in diameter at the nozzle. The connection between the hose and the nozzle appears to have given trouble and frequent breakages occurred. One man using the hose for half a day could break from the deep face as much washdirt as 8 men working 3 days with picks and shovels. The hose was also useful in washing the porphyritic boulders free of adhering tin ore, which was very difficult to do by hand.

During the second half of the year sluicing operations were again confined to the southern section. A sluice was used at each of the three faces. When water was abundant the tailings were re-worked in a fourth sluice and a tye. A branch tramway was laid from the bottom sluice to the main tramway at the bottom of the self-acting incline, so that the ore could be trammed through to the Falls, and so avoid the use of pack horses. From August to December some washdirt was also sent direct to the Falls 'to keep the men at the sheds employed and to increase the yield of ore'.

By the end of the year the dam across the river was full and the water race completed. According to Kayser the dam could supply the present water requirements of the three companies. The dam at the Mount was raised by 6 ft, making necessary the removal of 7 huts. The remaining portion of the tramway had been finished and the line was now open from the Mount through to Rouse's Camp. Carpenters and blacksmiths shops were built at Waratah.

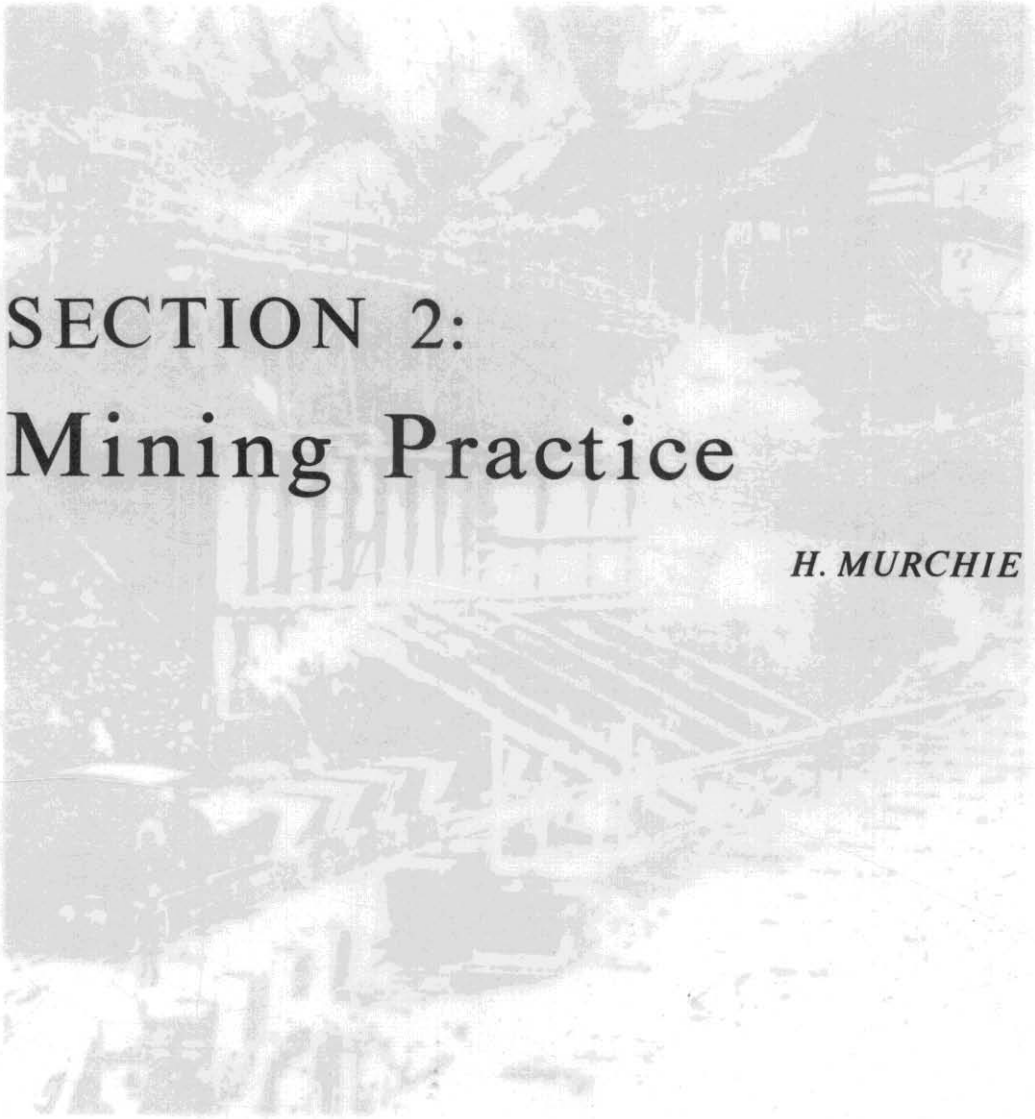
Three hundred tons of ore were mined during the last six months of 1875 (twice the amount raised during the equivalent period in the preceding year), bringing the total obtained since the formation of the Company to 740 tons.

According to Kayser, who took charge of the mine on 16 November 1875, the washing sheds were not what he expected to find. With the exception of the trommel everything was done by hand. Eddy, who had had experience in tin mines in Cornwall and in New South Wales, and had been mining manager for the Waratah Tin Mining Co., was in charge of tin dressing operations. Judging from letters by Crosby and Kayser to James Smith, Eddy was not an easy man to get on with. Kayser evidently had a poor opinion of Eddy and a few weeks after his arrival at the mine in mid-November wrote to Smith: 'I have been greatly surprised by the machinery of your great oredresser Mr Eddy. I thought by what I had heard in Launceston, I could learn something, but in this I find a great disappointment, in fact so much so that I shall dispense with his services very soon altogether.... The whole arrangement he has got in the shed reminds me of what you could see in Cornwall a hundred years ago.... I believe Mr Crosby would have done better without him altogether'.

Although, according to Henry Ritchie, Kayser 'created a very favourable impression on the Directors', his autocratic nature did not endear him to his subordinates. Kayser's early months were very difficult ones: his initial plans were at variance with those of Crosby. He advocated putting a steam-power crusher at the Mount and thought that it would not pay to tram the washdirt to the Falls. Norton Smith accused him of not sorting the low grade ore from the spoil. His attempt to lower the rate paid for carting resulted in the carters' strike in February 1876 and meant that more ore would have to be carted in the winter at £10 per ton, instead of at the summer rate of £5 per ton. Smith accused Kayser of prevarication.

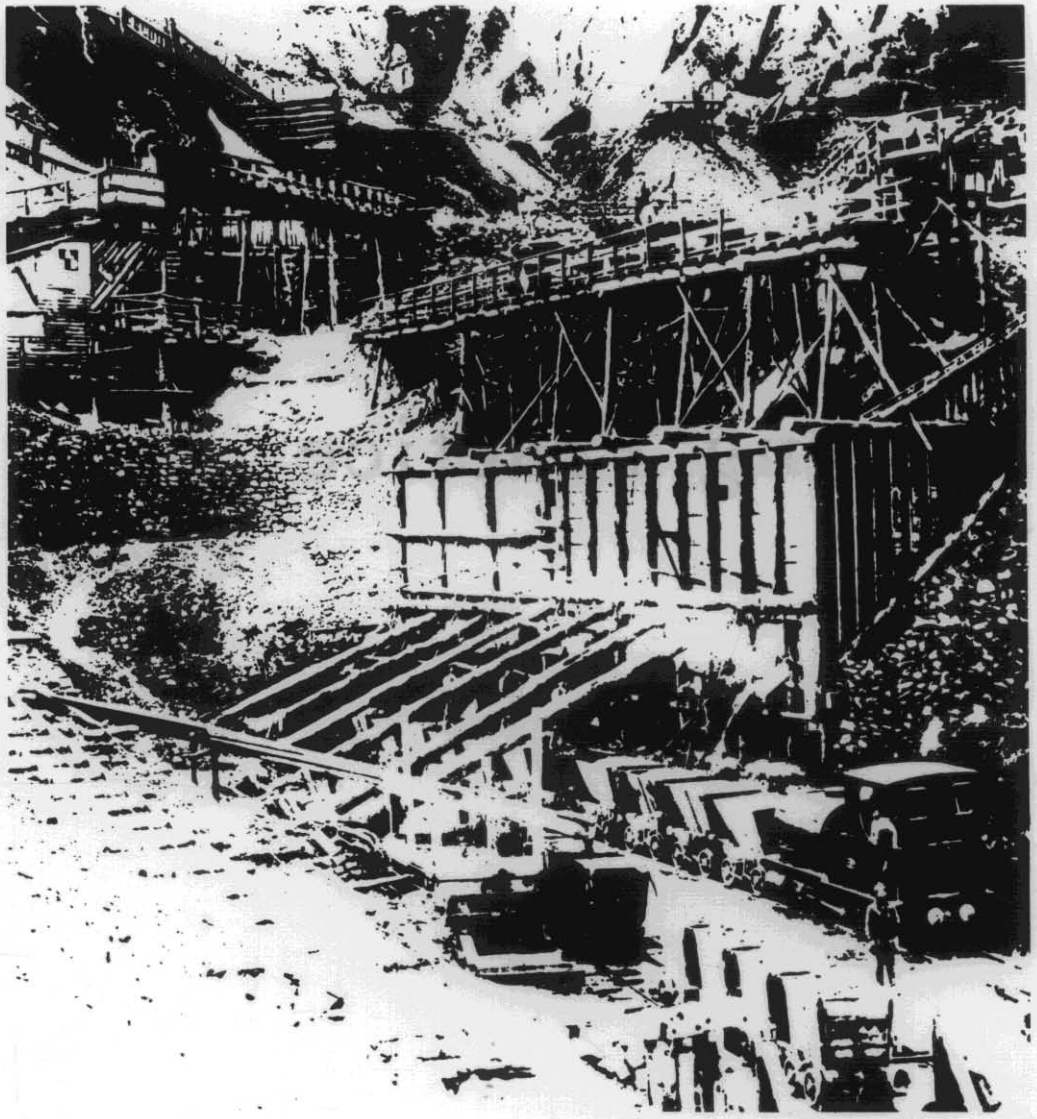
Crosby and Smith wrote a joint letter to the Board demanding Kayser's dismissal, but were unable to carry the majority of the Directors with them. Although the Board appears to have had doubts about Kayser, and William Ritchie stated that 'we shall need another manager before we can get much success from the mine', it seems that they thought it best to give Kayser time to prove his worth.

Despite Kayser's earlier statements he soon erected a 6-head stamp battery at the Falls and was able to record a four-fold increase in the amount of ore and washdirt trammed with no increase in working expenses. A second ore dressing floor was built at the sheds. Kayser had weathered the storm and, despite some false moves at the start, production was greatly increased and the Company paid its first dividend in 1878, the year of completion of the Emu Bay tramway. As a result of his disagreement with Kayser, Smith gradually severed his connections with the Company.



SECTION 2:
Mining Practice

H. MURCHIE



INTRODUCTION

Published records of Mount Bischoff are numerous and similar, in that all give detailed accounts of the geology of the deposit but little or no information of the mining operations. This is not surprising as geologically Mount Bischoff was and is remarkable, whereas the mining methods employed were quite unremarkable.

The Mount Bischoff Tin Mining Company produced over 56,000 tons of tin metal from 5,500,000 tons of ore treated, and paid dividends of £2,550,000 on paid up capital of £29,600. From the records of the Company the impression is one of strong profit motivation and in this the Mount Bischoff Company was outstanding and became a legend in its time. The obsession with profit however may explain why mining methods, prospecting, grade control, and mine planning generally, appear to have been neglected during the first 35 years of the mine's life. The story of Mount Bischoff presents a lesson to today's mining engineers and this may well be the value of examining the history of the mine.

HISTORICAL SUMMARY

The Mount Bischoff Tin Mining Company was formed in August 1873 and company operations commenced shortly afterwards with W.M. Crosby as mining manager. Earlier, in 1872 Crosby had opened up a track to the mine with a party of workmen and mining was commenced in a small scale in December 1872.

The first published record of mining at Mount Bischoff is contained in an article by Charles Gould (1875).

Gould stated that the ore has been found 'in large masses in situ'. However, when he visited the lease, active mining operations were only commencing, 'and in the absence of mining experience on the part of those engaged, coupled with the numerous obstacles to the progress of pioneers, presented by a rough and exceptionally difficult country, some time may be expected to elapse before its full importance is developed'. 'Large quantities ...of stream-tin have been obtained, and the amount will be increased when an effective water-supply has been provided and the primitive mode of washing at present employed, abandoned'. The problems associated with remote location, clearing of dense scrub, and lack of water were sufficient to prevent anything other than a crude form of mining. The horizontal scrub was impenetrable and no estimate could be made of the exposed area of stanniferous rock, although it did not appear to exceed a few hundred acres. This proved to be an accurate estimate as the bulk of the tin ore was mined from an area no greater than 160 acres, the original two blocks of 80 acres pegged by James Smith.

In 1875 Crosby retired due to ill health and was succeeded by H.W.F. Kayser. Kayser tacked the basic problems by erecting a second dam on the Waratah River, another on Falls Creek and extended the water races. Work on the tramways was soon completed and goods could now be transported from the mine to Rowses camp.

For the first seven years the mine was worked mainly as an alluvial mine, from which masses of tin oxide weighing as much as a ton, were obtained.

Thereafter a limited amount of underground prospecting and mining was done, however it can be claimed that underground operations were neglected whilst the surface operations were highly productive and profitable, producing an average of 200 tons of tin oxide per month for 20 years from 1878 until 1898. (Figure shows the annual production and the steady decline from 1900 onwards).

An interesting comparison between the profitability of a noted tin mine in Cornwall, the Dalcoath and Mount Bischoff for the year 1901 was made by D. Clark (1904, p. 205):

Dalcoath	£ s d		*Mount Bischoff*	£ s d
Authorised capital	350,000. 0. 0			60,000. 0. 0
Paid up	303,667.18.11			29,000. 0. 0
Balance from 30 June 1901	750.12. 0			46,803. 0. 2
Profit half-year to 31 December 1901	16,273.18. 8			31,671. 0. 2
			Interest	378. 0. 8
	17,024.10. 8			78,852. 1. 0
Less written off as under:				
Depreciation of machinery and plant	2,333.13. 2		Plant	3,585. 9. 1
Loose plant	445. 2. 9		Races and dams	411.10. 9
Development of mine	6,153. 7. 3		Horses, etc.	66. 2. 3
	8,932. 3. 2			4,063. 2. 1
Difference	8,092. 7. 6			74,789.11.11
Less Dividend 6d (2½%)	7,481. 6. 3		Dividends (114.14%)	33,000. 0. 0
				41,789.11.11
			Income Tax	1,650. 0. 0
Balance forward	611. 1. 3			40,139.11.11
Black tin sold, 1 July-31 December 1901 1033.65 tons	£71,133.14. 1		732.7 tons (@ £120 per ton)	£55,800. 0. 0

Both mines were producing about the same grade at 2% tin.

'There is certainly a vast difference in the returns. Mount Bischoff made a profit of £31,670 on an output of £55,800, while Dalcoath only made £16,270 on £71,133. The comparison is interesting, so far as figures go, but the difference in cost of production would be found in underground work and hauling, as against surface quarrying. The hardness of the rock is also an important factor in milling. The duty per stamp of the Bischoff plant

would only be a little more than half of what it is now were hard rock alone being crushed. Independent of this the cost of production is remarkably low, as is attested by the following figures' - of production costs per ton mined at Mt Bischoff in 1901 (Clark, 1904, p. 206):

	s	*d*
Mining, including new works and maintenance	2	8.221
Filling, hauling, and emptying trucks	0	6.026
Crushing, dressing, and maintenance of plant	0	9.600
Slime sheds	0	1.228
Ringtail sheds	0	2.072
Management and supervision	0	6.370
Plant, including all machinery	0	3.075
Development and progressive works	0	2.063
Waterworks	0	.305
Ore bagging	0	.374
Sundries	0	.868
Stores	0	5.226
	5	9.428

When Millen took over as General Manager in 1907 he succeeded in increasing tin production by increasing the tonnage mined. However the ore reserve grade had declined, and when the tin price fell in 1914, the mine became a marginal operation. The Company continued mining until 1929 when the mine was let to tributors who operated until 1942. From 1943 until 1945 it was worked by the Commonwealth Government, and from 1945 to 1947 jointly by the Commonwealth and State Governments. Since then tributors have worked on a small scale.

Authorities to Prospect were granted to Mt Costigan Mines Ltd early in 1963, and in 1964 they were joined by Broken Hill South Ltd, and Comstaff Pty Ltd. Active exploration work has been carried out and is continuing. It is hoped that their efforts will be successful and the mining industry again established at Mt Bischoff.

MINING METHODS

Gould's description of the mining operation is vague, but it can be assumed that the early miners were high-grading, selectively working the richer patches by digging and washing. The tin washing would be done by panning. A pan is a circular dish with sloping sides, 10-16 in in diameter, 2-2.5 in deep, and with the sides sloping at an angle of 35-40°. The pan is filled with gravel and placed in water. The gravel is thoroughly wetted and stirred by hand to break up lumps of clay, and the larger stones are picked out. The pan, still under water, is then given a shaking or gyratory motion, which brings the light material to the surface and allows heavy particles to settle; at intervals the pan is tilted and the surface material washed off. These processes alternate until nothing but tin and a little heavy sand is left. Panning is slow, back-breaking work and it is estimated than an experienced man can pan carefully about 10 pans of average gravel per hour, amounting to roughly 1 cubic yard a day. Panning appears to have been extensively used in the early prospecting operations on the Mount.

It is also likely that simple methods used in the Australian Gold Diggings of the 1850s were employed. This would involve puddling and cradling. Puddling is done by filling a large tub with gravel, adding water and chopping the mixture with a spade to free the ore from the adhesive soil.

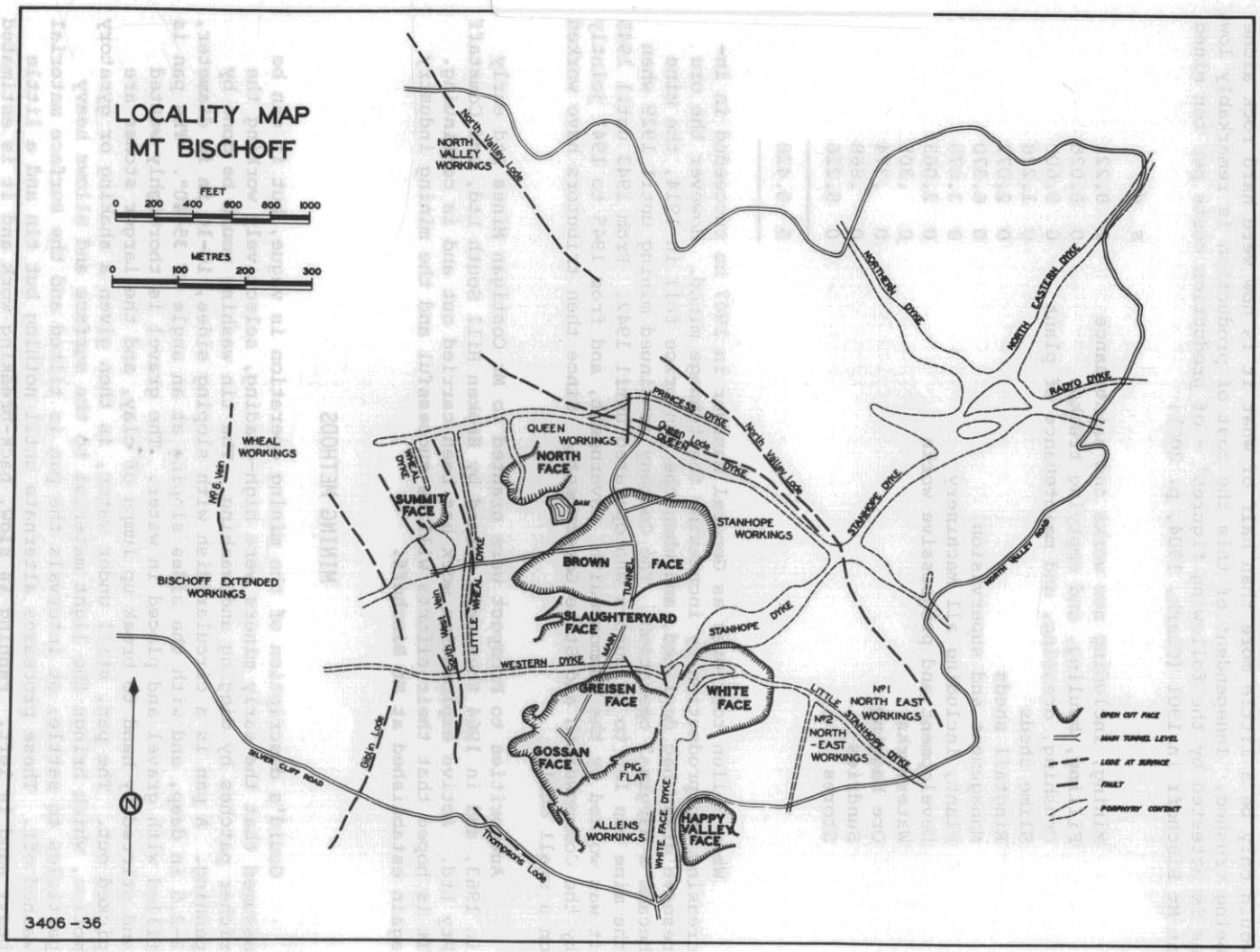

5 cm
LOCALITY MAP
MT BISCHOFF
FEET
0 200 400 600 800 1000
METRES
0 100 200 300
NORTH VALLEY WORKINGS
North Valley Lode
NORTHERN DYKE
NORTH EASTERN DYKE
RADIO DYKE
PRINCESS DYKE
QUEEN WORKINGS
Queen Lode
QUEEN DYKE
North Valley Lode
NORTH FACE
WHEAL DYKE
SUMMIT FACE
WHEAL WORKINGS
No 6 Vein
BISCHOFF EXTENDED WORKINGS
STANHOPE DYKE
STANHOPE WORKINGS
NORTH VALLEY ROAD
BROWN FACE
TUNNEL
Vein
LITTLE WHEAL DYKE
SLAUGHTERYARD FACE
STANHOPE DYKE
WESTERN DYKE
MAIN
GREISEN FACE
WHITE FACE
No1 NORTH EAST WORKINGS
LITTLE STANHOPE DYKE
No2 NORTH-EAST WORKINGS
GOSSAN FACE
PIG FLAT
Gibbin Lode
SILVER CLIFF ROAD
ALLENS WORKINGS
Thompsons Lode
WHITE FACE DYKE
HAPPY VALLEY FACE
OPEN CUT FACE
MAIN TUNNEL LEVEL
LODE AT SURFACE
FAULT
PORPHYRY CONTACT
3406-36

Figure 3.

Cradling is then carried out by using a rocker in which the washed dirt is placed in a perforated hopper at the head of the rocker, and while the rocker is rocked the dirt is washed with water. The tin is caught on a wooden shelf below the hopper, but any which is carried down into the bottom of the cradle is removed by panning. The rocker is normally a prospector's tool but would work at Mt Bischoff until a water supply was organised, especially as it is estimated that a rocker can be operated on 4-6 barrels of water. Two men rocking a cradle can treat from 3-5 cubic yards per day.

Sluicing commenced towards the end of 1873 using stream water, and early in 1874 a small dam was built from which the water was drawn and then re-used. Later water from the Waratah and Falls Creek dams was used, permitting sluicing operations on a much larger scale. The ground was opened up by stripping off the peaty surface. The water-worn boulders which underlie the peat were rich in tin and were sent direct to the mill by horse tramway. The loose dirt was shovelled into sluice boxes and the semi-washed product sent to the mill. The sluice boxes were simply wooden launders with a constant water flow sited close to the face where the men were working and provided with riffles set on the floor of the boxes to catch the heavier materials including tin. The ground was opened up in a series of benches or steps. The material was so soft that it could be shovelled. Bars were driven in at the top of the workings and ropes hung down to the men below to enable them to dress and clean down the faces and escape in case of landslips. Due to the scarcity of water, the flow was directed down through a series of sluice boxes until it reached the bottom of the workings. With this method it was not possible to save all the fine ore, but as the recovery grade was over 3% it was considered best to conserve water and work the maximum number of sluices.

By 1878 there were 21 sluices at work, and at the peak of the sluicing operations 32 sluices were employed. The tailings escaped to the river, but in 1883 the Waratah Tin Mining Company's lease was purchased and the Ringtail sheds erected to treat the tailings before entering the river. Between 1883 and 1891, 1,116 tons of tin oxide were recovered from the treatment of tailings.

Explosives were only used to break up lode material. The first shot in August 1873 broke out 0.75 tons of ore. Kayser records in 1878 that a charge of 150 lb of gunpowder dislodged hundreds of tons of earth and rock.

Two self-acting tramways were employed to transport the ore from the mountain to the battery using descending loaded cars to raise the empty cars. Wooden rails were used and replaced by iron rails in 1879.

In 1879 after 6 years of mainly alluvial working the first extensive underground prospecting was started. An adit into the hillside at the foot of the self-acting incline was driven 455 ft under contract labour. This was considered to be expensive and was discontinued when hard ground was struck to await delivery of a boring machine driven by compressed air. The contract rate for driving the adit had been 7s6d per foot. It is not recorded whether the contractors provided their own tools in addition to labour. Without drilling machines it would be very slow back-breaking work. In the same year the first shaft was sunk to a depth of 45 ft and then abandoned owing to the influx of surface water.

The boring machine arrived in 1881; only 4 ft of advance in the adit was achieved for the first two weeks. The rate of advance was increased to 14 ft in a fortnight but hard ground and poor ventilation made progress difficult, the adit was stopped and cross cutting undertaken in softer ground.

The fear of water escaping from an old reservoir above the adit through soft porphyry ground also influenced the decision to stop the adit.

A set of boring rods was manufactured on the mine to prospect the shaft abandoned due to water problems. However due to the water the ground caved in the hole and the hole was cased using tube, this is an interesting record of a technique widely used today. About this time, 1883, the ventilation problem was acute in the prospect cross cut and it was decided to sink a 3-in borehole from surface to intersect the tunnel and assist natural ventilation. After six months the hole was abandoned due to difficulty in boring hard ground and a rise was put up from the cross cut to intersect the borehole. This was not successful, possibly due to hole deviation which could be expected in hard ground. Finally on 15 February 1884 the holing was effected, ventilation improved and it is recorded that the total length of the adit was 766 ft, and it was hoped to extend the adit to a position beneath the present Brown Face workings before the end of the year. Good values had been obtained in this prospect, one section sampling 2 lb of tin to a dish, roughly equivalent to a grade of 200 lb to the cubic yard.

The alluvial workings continued to be developed with tin being won from the three faces that were to be the main areas of production throughout the life of the mine. These were the White Face, the Brown Face and the Slaughteryard Face. All of these faces were rich in tin and they were being steadily excavated by hand, benching where necessary and operating the sluices as the water supply permitted. In 1880 a 5 cwt solid nugget of tin ore was recovered in the Slaughteryard Face. In 1879 the primary crushing plant was greatly expanded and the tailings were stacked in the Brown Face workings for treatment at the Battery. Kayser records that the wash dirt varied from 6-78 ft in depth on the Brown Face, where 18 sluices were working, all connected by tramways to the face. Water shortage continued to be the main drawback to the operation especially in the summer months when there was no alternative but to cease sluicing. During these enforced shutdowns the opportunity was taken to clean the reservoirs, repair the tramroads and carry out other necessary maintenance work. In 1881 the iron tramway from the mine to the battery was repaired to convert from horses to a steam locomotive and ensure an adequate supply of stone to the crushers. This changeover was effectively done by the end of the year and increased tonnage by 50%.

Telephones were installed in 1882, and electric lighting in 1883 although it was not until the end of 1884 that the lighting was recorded as being successful. About this time a new lathe was purchased for the workshops to turn and repair locomotive wheels; a necessary item to ensure self-sufficiency in such an isolated location. New rail trucks were imported to the mine in 1885, but proved to be unsatisfactory, and it was decided to manufacture their own trucks thereafter. A stonebreaker was set up at the mine in 1884 to reduce hand spalling and this proved to be a valuable saving as in addition it supplied the battery with easier crushing dirt. The steam boiler driving the compressor was found to be unsafe and replaced by a multi-tubular boiler in 1885. Corrosion was still a problem due to the dirty water supply and a water tank was built at the edge of the Brown Face to provide a clean water supply to the boilers through a 1-in pipeline. Both the air compressor and the stonebreaker were steam driven. In 1887 a compressed air operated diamond drill was used for the first time on prospecting operations. It is recorded as the Government diamond drill and prospecting was done south of the Mount. A series of bores were put down without success and the project abandoned. The engineering facilities were extended by establishing a foundry which is recorded as a particularly useful addition to the mine's repair and manufacturing capability.

The first record of productive underground mining is contained in the Company's half-yearly report for December 1888. Details are given of a lode containing 15% to 20% tin ore in a quartz vein 3 ft thick, with feldspathic clay and porphyry. A winze was sunk on the lode to connect with the main adit beneath the Brown Face. The strike was E-W with competent walls which permitted open overhand stoping up dip from the adit to surface. It was found that the air compressor was unable to supply more than one rock-drill, due to the distance to the stope face resulting in high friction losses. A decision was made to install an additional compressor worked by water power; an incredible decision in view of the problems associated with water shortage. By 1892 wet drilling with rock-drills had been introduced and a tank of 50,000 gallons capacity had been erected at the top of the self-acting tramway to feed the drill water underground. Prospecting had begun in the North Valley and the No. 2 Adit was abandoned due to the poor ventilation affecting the health of the miners. Kayser considered it vital to ventilate the mine more effectively and commenced sinking a 300-ft shaft to intersect the adit and provide through ventilation. The attitude of the Company is not wholly reflected in the above action as in 1893 they report that the ventilation of the mine was perfect without the aid of the air compressor. The hope was expressed that 5-in pipes ordered from England would soon arive, as the pipes being used did not allow the air to pass so freely as the larger ones would. By the end of the year the air compressors were idle although underground development work and stoping continued. Apparently, where possible, they preferred hand working to the use of rock-drills and natural ventilation (or none at all) to the use of compressed air with the disadvantage of fitting up the necessary pipework. In the Queen lode during 1893 a stope was being worked by the underhand rill method which resulted in only slow progress due to the large amount of stripping necessary, with rich patches being located in the stope walls. No information is available on the method of support used in the above stope. Underground prospecting in the Waratah section struck very changeable ground which was so broken that it was found impossible to use a rock-drill resulting in hand work and the employment of close timbering. Steam driven winding engines were employed at the mine at the turn of the century as it is recorded in 1899 that the winding engine for the underground workings in the Stanhope Section was to be removed to the Brown Face. Stoping in the Queen lode produced 1,043 tons of ore in 6 months during 1901. Production in these stopes totalled 380 tons of ore in 1904. In 1907 a rise was started in the Queen lode dyke from the North Cross Cut off the main tunnel, which exposed some very rich patches of tin ore. This rise was 15 ft wide by 4 ft high and was divided into three compartments, one for mullock, one for ore and the third a ladderway compartment. The contract price for the rise was £2 per foot.

Underground prospecting at the Wheal Bischoff section from Nos. 2 and 3 adits was stated to be encouraging and mining would commence when a system of transporting the ore was provided. Several schemes were examined with a tramway round the western side of the Mount being preferred to an aerial ropeway. In the following year J.S. Millen took over as General Manager on the retirement of H.W.F. Kayser. Millen, in his first report to the Directors in January 1908 stated: 'An expenditure of both time and money will be required to bring the work up to the position it would have been in had prospecting been carried on for say the last 20 years, along with other mining work; also had this been done, the property would have been on a sounder footing'. He claimed that the rise in the Queen lode dyke was of faulty design and had to be remodelled. The prospecting work at Wheal Bischoff was suspended as the tramway system was crude and costly. This brief summary of the underground prospecting and mining in the years between 1779 and 1908 gives the impression that Millen's claims were justified, and that the Company did not pursue the exploitation of the deposit as vigorously as it

should whilst the mine was in its heyday. The year 1908 marks the beginning of the rapid decline in the Company's fortunes.

Continuing the history of the surface workings, Kayser originally estimated that the White Face would last a total of five years however, by 1883 he expressed himself confident that there were sufficient resources to continue for many years. By 1890 the surface workings had become large excavations and slope stability was becoming a problem. The first recorded major landslip was reported in the Slaughteryard Face where 4,000-5,000 tons of wash were dislodged with some material running as far as the bottom of the Brown Face. At first Kayser believed that a leakage from the reservoir through the clay had weakened the ground but subsequent events proved that the face must have been weakened long before this by rainwater percolating through surface cracks and fissures, and gone unnoticed until shortly before the incident. However, once the ground began to move it did so rapidly and he records that 'the whole hillside for 200 ft in length is working and it is only a question of time before the whole top section will slip to the bottom of the face'. He considered that no harm had been done beyond weakening one side of the upper reservoir and believed that this was offset by the expense which would have been incurred to break such a large tonnage of wash by hand. He appears to have viewed the incident with less concern than it would be by today's pit operators.

By 1893 the Company was critically examining its sluicing operation and the report of June 1893 records that 'it is considered necessary to alter the system of working the mine'. The alluvial ground was largely worked out, and it was not considered practical to continue sluicing, which is an operation designed to produce cleaner material for the treatment plant by removing the clay and soil from the wash gravel, in addition to its main function of retaining tin concentrates on the riffles. The sluicing operations were therefore discontinued: the broken ore from the faces being trucked direct to the batteries; with the end result of an ample water supply for the life of the mine. By 1895 the total storage capacity at the mine was 490 million gallons. The three major sources of supply were the Waratah rivulet, Fossy rivulet and Falls Creek on each of which extensive reservoirs had been constructed. The water from the Fossy rivulet was diverted to the Falls Creek reservoirs by a water race 4.5 miles long with a fall of 2 ft per mile.

Kayser and Provis (1895) gave the following details on the method of mining: 'The three races [Brown, Slaughteryard and White] together with such portions of the adjacent porphyry dykes as are worth removing, are worked as open quarries. Ninety-one men are employed in getting and moving crush-dirt from the faces, thirty being miners receiving 8s per day and the rest truckers and labourers at 7s6d or 7s per day. In the softer ground each miner excavates about 12 tons of "stuff" per day; but in the hard porphyry dyke only 4 tons. The total cost of mining and delivering the material to the dressing sheds is 3s2½d per ton'.

Land slips continued to be a problem and in the Brown Face in 1894 several slips buried payable ore with barren ground and as no tips were available to dump the mullock it was necessary to sort the large lumps of slate into heaps on the benches and send the fine dirt to the batteries. The tramroads on the benches were not completely laid out and they record the intention of 'laying permanent double tramroads to the East, West and Middle sections of the Brown Face which will permit the extension of the open cut system and the use of horses for heavy tramming'. It was now becoming necessary to blast the faces as they were in harder ground and the old hand working methods could no longer be employed. About this time the mine transport system

was being rationalised and the underground adits were used as main haulage roads with ore passes to the surface connecting them to the tramroads. This had the additional advantage of some storage capacity being available in the ore passes permitting a steadier flow of ore to the batteries. At the White Face the two ore passes were roughly 70 ft in length and the adit was being driven under the Brown Face to provide it with an ore pass system. In 1896 it was decided to replace the small trucks with ½-ton trucks and the wooden track on the tramroads with steel rails.

A further major slip occurred on the Brown Face in March 1898 with clay and slate from the North Wall burying a large stockpile of crush dirt on the benches. The West Wall then began to move and over 2,000 tons of debris engulfed the workings although this was only a small portion of the ground involved in the slip. A total of more than 10,000 tons collapsed and as there was no mullock tip where it could be dumped it had all to be washed, sent to the crushers and treated through the batteries. The difficulty of the reclamation work is indicated by the fact that 9 months later only half the debris had been cleared away. A rise was put up from the main tunnel to drain the slip area and prevent further landslips. This was not successful and the Company finally realised that until the slope of the face was reduced to 45° the problem of slips would not be solved. Work was done to strip off the top of the faces as rapidly as possible but it was not undertaken soon enough and further falls occurred. Benching was attempted and on the White Face by 1902, three major benches had been established, the Upper or No. 3 bench being 51 ft high, Middle or No. 2 bench 40 ft high and the No. 1 bench 13 ft high. After 12 months work the old face at the bottom was still idle, as the benches above were not yet advanced sufficiently to allow safe work below. The problem continued and in March and May of 1902 it was reported that the opening of a new stope in the Brown Face section was delayed due to slips. In 1904 a slip at the Brown Face buried a large stockpile of concentrated ore collected in a temporary dam at the bottom of the face carrying it away and scattering it over the quarry floor. The above series of mishaps would have been sufficient to ruin any mine working on average grade, and the fact that this did not happen indicates the value of the ore body. However no mine can carry such setbacks, and the production of tin dropped from 2,256 tons in 1898 to 1,291 tons in 1902 and eventually to 730 tons in 1907.

Any complacency that existed amongst the directors and shareholders of the Company after 35 years of profitable operation was now dispelled and the following measures were taken to improve the Company's fortunes.

Firstly the Goldfields Diamond Drilling Company was engaged to drill a series of prospect holes in the Slaughteryard Face section. In 1904 three holes were drilled, the deepest to 895 ft, all without success. The third hole struck soft ground with numerous vughs and two months were lost reaming and cementing. This hole cost over £1 per ft at £833 for a total of 747 ft of drilling; more than half the cost of the 15-ft x 4-ft rise that was being developed at the same time. Millen was very critical of this work and stated in his first report to the directors that as the boreholes were not put down systematically, the work was valueless and in his opinion the results obtained by this drilling proved nothing. The positions of the holes were not selected according to any recognised law, but purely in a haphazard manner and nothing of any value was struck.

Secondly a hydro-electric power scheme was evaluated and in 1905 a contract was let to Noyes Bros, Melbourne to supply the equipment and build the power station. It was considered that by converting the crushing and treatment plant to electric power, using electric locomotives rather than

steam, and providing sufficient capacity to operate additional crushing plant, greater economy could be achieved. The Waratah river has a rapid drop of 600 ft in a little over a mile just below the battery. This fact combined with a neavy rainfall of over 100 inches annually made possible the development of cheap hydro-electric power. The power station contained four generators, two 345 kW A.E.G. and two 145 kW Westinghouse, all 2,200 volts. The motors, numbering 50 were 550 volt, and lighting service 110 volt. Three-phase current was used for everything, except the main line railway, which was supplied with direct current from a rotary transformer. The power plant outlived the mine and was scrapped about 1950.

Thirdly the assistant Mine Manager, H. Herman was instructed to study and report on

(1) the economic handling of the ore, and

(2) the economy of handling an increased output.

He estimated that £2,888 per annum could be saved by laying a 3-ft gauge track through the main tunnel to continue the main line from Waratah to the mine. Ore passes to feed ore from the Gossan and Brown Faces directly to the main tunnel were recommended. This would have saved excessive hand trucking and eliminated handling from three to four times before delivery to the mill. The main saving was to have been in labour with a reduction of 24 men employed on trucking, and the horse used in the main tunnel replaced by an electric locomotive and large trucks. By this time the main tunnel had been driven through Mt Bischoff and was below all the surface workings and readily accessible to them by an ore-pass system making it the obvious solution to the first problem of economic handling of ore. Regarding the second problem he suggested that the mine, that a topographical survey be done of the mine and that systematic sampling and assaying be adopted for the faces of the mine. The survey was considered necessary for forward planning, especially regarding haulage layout and it is surprising that it had not been considered earlier. In 1907 an assay office was set up at the mine and in 1908 the survey was completed, the mapping being co-ordinated with an assay plan for grade control. The new haulage system was completed in 1908.

The mine entered a period of redevelopment in an effort to make it a viable operation. The tonnage treated was increased, new plant and methods introduced and strenuous efforts made to increase the underground production and ore reserves. The programme was successful and tin production increased from 730 tons in 1907 to 1,206 tons in 1913, however the recovery grade had now dropped to below 0.5% tin. The surface faces were still being quarried and expanded underground stoping operations helped to increase production. Stopes were worked by open over-hand methods using timber and ore pillars for support. With the increased development necessary for underground prospecting the waste rock was used to fill the stope excavations improving the general support of the mine. Spontaneous combustion in the underground workings of the Brown Face deposit became a problem due to the rapid oxidation of the marcasite component of the ore. In order to retard this action, the stopes were filled with mullock and low grade gossan, and carefully packed. Despite these precautions fires broke out frequently and sections of the workings had to be sealed off to prevent the fires spreading. This added to the Company's troubles making operations costly and slowing down production.

By 1914 Company losses had continued to increase and during the months of July, August and September of that year, when tin became an unmarketable product, it was decided to close the mine down completely. This action how-

ever created such hardship to the employees that it was decided to discuss the matter with the Minister for Mines. The Minister was unavailable, but the directors suggested to W. Twelvetrees, the Government's representative, that the mine be re-opened on a co-operative basis with the workers. The latter were to be given three-fourths of the tin output, on the condition that they paid wages, stores, freight, insurance and smelting charges, while the Company was to pay the manager, administration and office expenses. This arrangement was to be for a six month period with either party given the right of refusal or renewal with one month's notice. The Company was to start 30 heads of the battery and would require only 100 men out of the 430 formerly employed. This arrangement was finally accepted by the men and work was resumed on 15 September 1914, on a very reduced scale. Prospecting had uncovered further alluvial deposits in the North Valley, which proved both extensive and payable.

The co-operative system was renewed on 15 March 1915 and the production of concentrates increased from 194 tons for the period ending December 1914 to 221 tons for the same period in the following year. The rise in freight prices, together with the heavy war insurance risk and the imposition of an export whafage duty however continued to cause concern, as did the increasingly difficult conditions regarding the extraction and reduction of the low grade ores of the southern slopes. A haulage for the North Valley alluvial deposits was completed with an eight hour capacity of 300 tons. The loading station at North Valley was destroyed by a bush fire in 1920.

The co-operative system of working was terminated on 15 March 1915 and the Company resumed operations on its own account, but still at a greatly reduced level, only 132 men being employed. Employment increased to 240 men in 1916 and remained at about that level until 1921.

Surface operations continued on the main faces where a milling system of mining was introduced to replace the benching system of quarrying. In this system the quarry floor was worked rather than the quarry faces. An adit was driven under the quarry floor and several cross cuts taken off it. Vertical rises were put up from the cross cuts to hole in the quarry floor, the rise height being normally 50 feet. Sloping faces were worked from the top of the rises giving a cone shaped excavation. Broken ore was drawn off from the bottom of the rise. This system has advantages over face working due to cheaper breaking and handling of the ore, with the chief disadvantage being the danger and trouble of hung-up prices.

In the year 1921 operations were greatly curtailed due to the slump in tin prices and the uncertainty of obtaining sales for the metal produced. Employment dropped from 250 men in January to 40 men by October of that year. Production was then gradually increased until 1926 when a further decline set in. By 1929 the Company found it uneconomic to work the leases and again tributors were permitted to carry out the operations with the Company maintaining the offices and concentrating mill.

An investigation was made of the alluvial deposits of the Waratah river indicating that a sluicing or dredging operation was feasible. The area was test sampled and a road cut round the North Valley to the river flats. It was realised from the sampling that although the wash contained rich patches, it also contained numerous large boulders, difficult to mine and handle. A pontoon dredge was set up in 1927 and the necessary pipe lines and water races completed. As anticipated the mining operation proved difficult and although the wash returned 2 lb of tin to the cubic yard, the plant was continually breaking down causing delays which increased the operating cost. The wash was broken up by hydraulic sluicing and then dredged by the pontoon

dredge for gravity concentration. Once the first paddock was mined out the area was used to dump the large boulders although the pontoon dredge was never successfully able to handle these boulders. In 1933 these workings were let to the Walsh Brothers from the East Coast who put in their own elevating plant. A heavy flood in 1934 caused extensive damage to the alluvial workings depositing tailings and poor ground on top of the wash. This material had to be removed before tin mining could resume. Dredging was never completely successful and the tributors altered their method to ground sluicing by the late 1930s. In 1937 the river was diverted to the eastern side of the flats to expose river bed wash for ground sluicing.

Underground prospecting continued in the hope that rich ore could be proved at depth. However the necessary equipment was not available, and when in 1927 development in the Stanhope Section appeared promising it could not be continued as compressed air and rock-drills were needed to open up the hard ground. The tributors persevered with underground prospecting, development and stoping, and installed air-compressing plant for pumping, hoisting and rock-drilling. Their efforts could only be on a small scale, were not co-ordinated, and no ore bodies of any consequence were found. However their operations were profitable, provided employment and prolonged the life of the mine.

The Company continued on this basis for 13 years until 1942 when it was found to be impossible to continue without financial assistance. The production of tin was essential for war purposes and the mine was resumed by the Minerals Production Division of the Commonwealth Department of Supply and Shipping in 1943. Tin production was to be stepped up and to achieve this new plant and equipment was installed. On the southern slopes a ¾-cubic yard electric shovel was installed to load low-grade ore from the benches into main line trucks. Twenty new 2-ton ore trucks were made, and 13 old trucks reconditioned. A 6-ton 30 hp International petrol locomotive was used to haul the trucks from the mine to the mill and the main line track was reconditioned. A tippler was built and the trucks could be emptied at the rate of 1 per minute. A 500 cubic feet air compressor was purchased and a new air main provided to the underground workings. A jack-bit grinder was purchased and jack-bits were sharpened and used at the mine.

Because of the financial losses involved in the operation of the mine the Commonwealth Government decided to close the mine in 1945, however to avoid hardship to the town of Waratah and in the hope of a viable operation the Commonwealth and State Governments agreed to continue work at the mine and a Board of Management was appointed. During this period production was increased from the surface and underground deposits but the ore produced was of very low grade and the tin return uneconomic. The Commonwealth Government withdrew its support and the mine ceased operations in August 1947.

SECTION 3:
Metallurgical History

H.K. WELLINGTON

INTRODUCTION

When one thinks of Mt Bischoff one thinks of the Mt Bischoff Tin Mining Company Registered whose production and dividends made the Mount world famous. Although the work of other companies in the Bischoff field is insignificant in comparison, when considered only in terms of production and productivity, the Mount Bischoff Extended Tin Mining Company, about the year 1910, appears to have been foremost in the technical field of metallurgy.

In addition to its oredressing activities at Waratah, the Mt Bischoff Company conducted a custom tin smelter in Launceston, the history of which has been included.

The primary sources consulted were the Mount Bischoff Company's Half-Yearly and later Annual Reports and the Mines Department Annual Reports. As some of the Company reports were not available I have listed those consulted on pages 163-164 so that if further Bischoff records are found a ready reference to those used here is available. I would have liked to have consulted the Board minutes and the correspondence that passed between Waratah and Launceston but none was available. This would have firmly established the relationship between the Board and its operating staff at Waratah.

In order to answer questions that could not be explained from purely metallurgical considerations I have had to digress into the Company's financial affairs.

The period of governmental operation, while insignificant with regard to overall tin production from Mt Bischoff, does represent a time when money for investigational work was made available on a scale as never before. The Federal information has not been consulted but as the State was in partnership with the Commonwealth during the last twenty-one months of this phase the State files give a comprehensive view of the metallurgical activity.

The opportunity has been taken to review here hitherto unpublished Department of Mines technical reports on Mt Bischoff.

No tin recovery figures were published except for a period of sixteen months between December 1945 and March 1947. Prior to 1907 no one kept such figures but recovered head grades were sometimes given. Recovered tin figures have been calculated from the tin produced.

In arranging this history I have first presented that of Mt Bischoff Extended (or the West Bischoff) then that of the Mt Bischoff Company and its successors, the government operation, and present activity. The smelting operations are then considered. In each of these sections the recorded information is first presented and is followed by a discussion and review; the metallurgy of Mt Bischoff as a whole is then reviewed.

A list of abbreviations used in the text is given on page 14.

MT BISCHOFF EXTENDED MILL (FORMERLY WEST BISCHOFF)

The mine and its mill were situated on the western slopes of Mt Bischoff above Tinstone Creek.

Historical Outline

The West Bischoff Mining Company was floated in 1878 to exploit this area which was thought to contain much alluvial tin; although this supposition was not correct, prospecting lead to the discovery of the Giblin lode in 1879.

Construction of the first treatment plant, a 15-head battery driven by an overshot water wheel, began in 1881. Ore was brought from the mine, which was 1.5 miles away, by a self-acting tram and a horse tramway.

The mill which had a throughput of 100 tons per week began operating on 3 March 1882 and ran until 1884 with the following results:

Ore crushed	7,102 tons
Concentrate produced	306.60 tons @ 70.2% Sn
Recovered head grade	3.02% Sn

Production was valued at £18,850 but the cost of production was £19,209 excluding expenditure on the plant, hence the Company ceased operations.

A tributor named White produced 110 tons of concentrate during the period 1885-1886. He was followed by another tributor, Roots, who produced 34 tons of concentrate.

The Company was reorganised as the New West Bischoff Company which began milling towards the end of 1892 and continued until 1894. The results of their operations were:

Ore Crushed	3,040 tons
Concentrate produced	120 tons @ 69.9% Sn
Recovered head grade	2.76% Sn

Operations were uneconomic; the Company got into debt and was sold to the Commercial Bank for £50.

At this time mining companies were reluctant to treat pyritic ore (Clark, 1904) and little work was done until the Mount Bischoff Extended Tin Mining Company acquired the mine in 1907 and began building a new steam-driven mill at the mine site.

This plant, which began operating in 1908, was added to and modified frequently. Herman (1914) wrote an account of the milling operations. The company ceased operations in 1931.

Grade and Type of Ore

While one may infer that the early production was from the oxidised zone and hence sulphides were not a problem, by 1910 the quantity of sulphides was sufficient to warrant an initial concentration stage to make a sulphide concentrate which was then roasted.

Herman (1914) gives the head as 0.95% Sn with 11% pyrite and sulphides

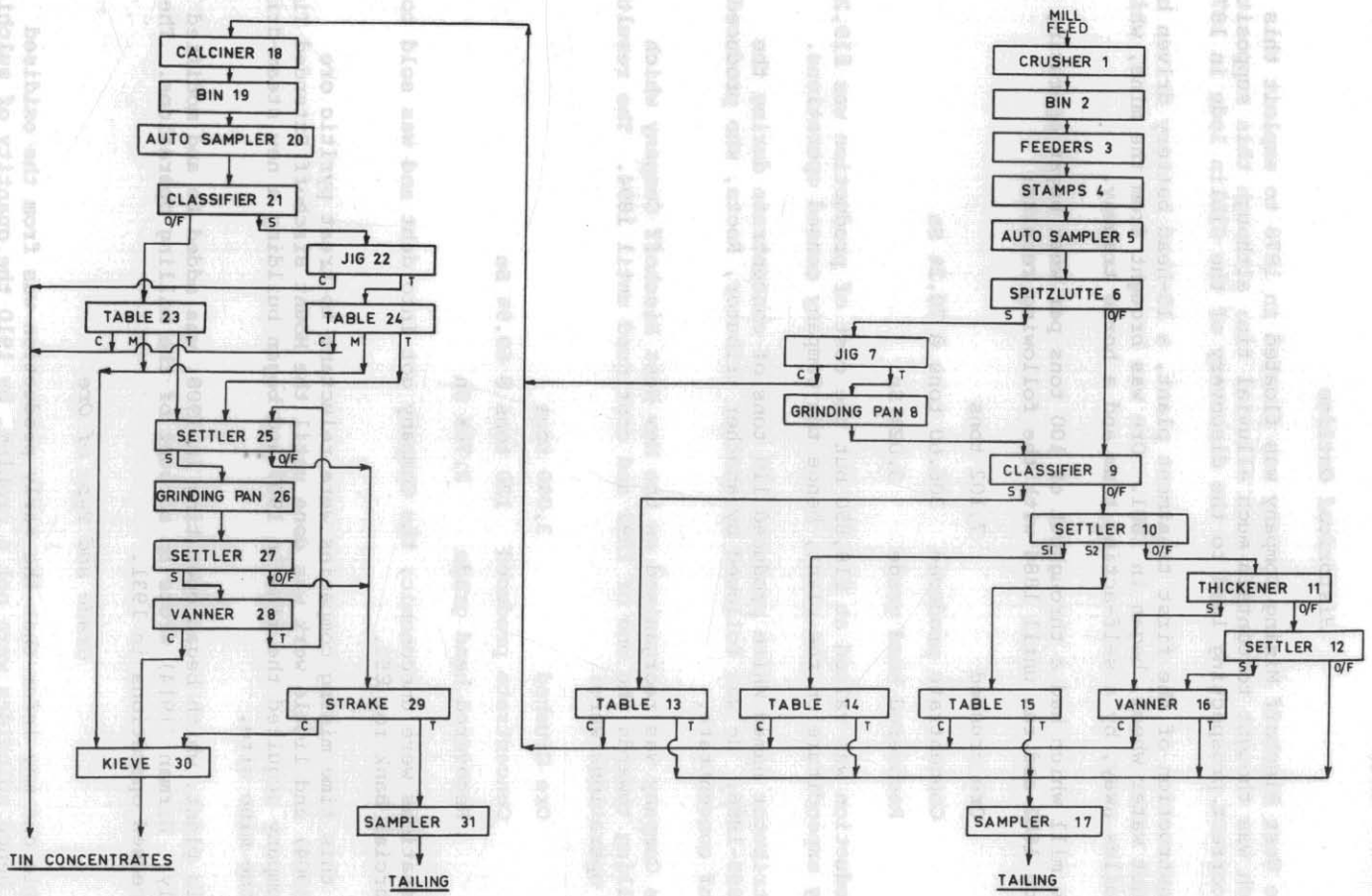


Figure 4. *Mt Bischoff Extended Tin Mining Co., mill flowsheet, 1914 [Herman 1914]*

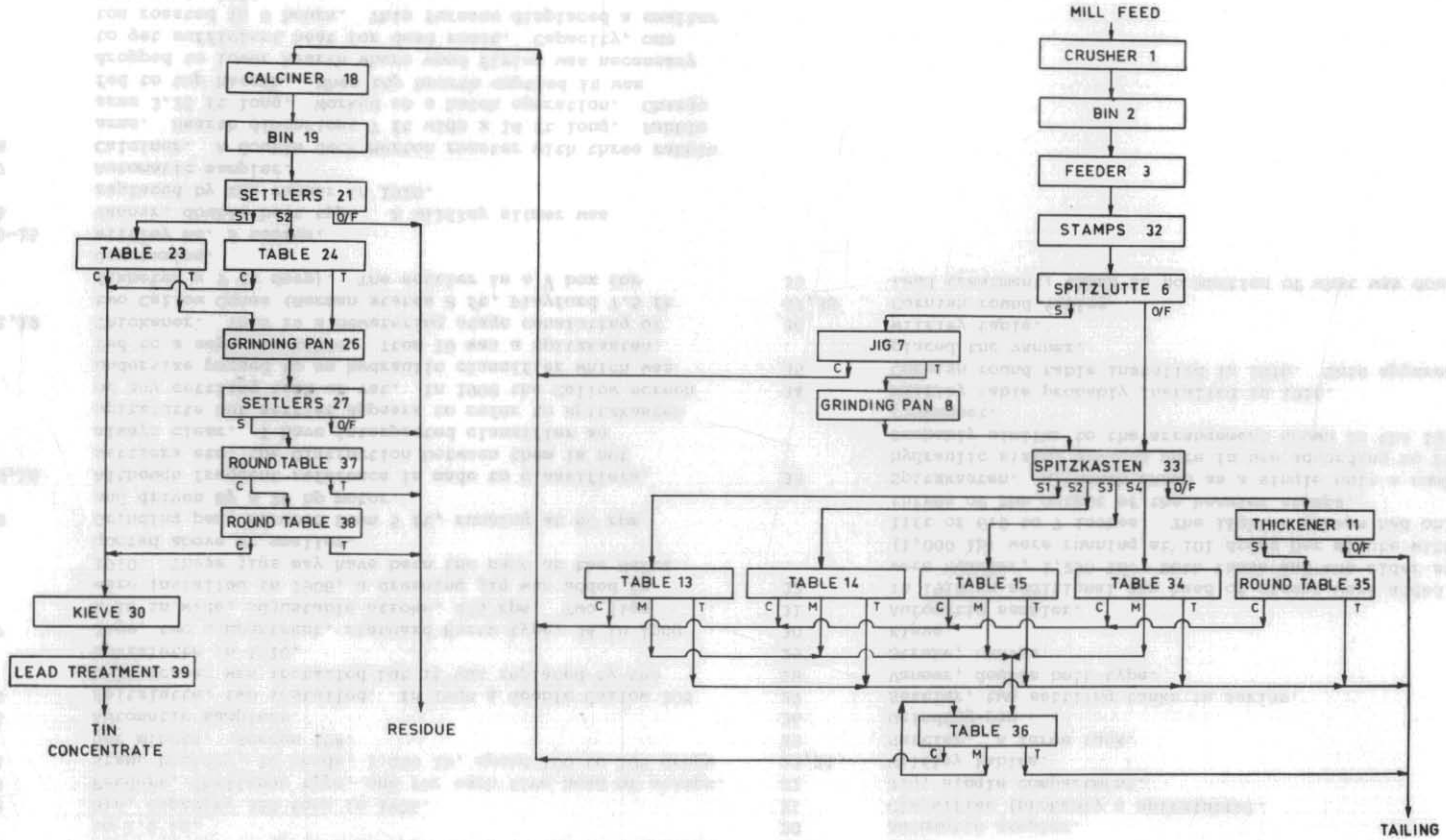


Figure 5. *Mt Bischoff Extended Tin Mining Co., mill flowsheet 1926 [Playford, 1928]*

Notes on Figures 4 and 5. *Items in Mt Bischoff Extended Mill Flowsheets*

Where the same item appears to be common to both flowsheets the same item number has been used.

Item No.	*Description*
1	Jaw crusher, Jacques/Blake type 14 in x 7 in set to break to 1.5 in.
2	Bin, capacity 350 tons in 1926.
3	Feeders, Challenge type, one for each five head of stamps.
4	Stamp Battery, 10 heads, 1,050 lb, speed 100 to 105 drops per minute. Screen 10#.
5	Automatic samplers.
6	Spitzlutte, two installed. In 1908 a double Callow 30# belt screen was installed but it was replaced by the spitzlutte in 1910.
7	Jigs, two compartment, standard Hartz type, 34 in long x 16 in wide, adjustable stroke, 235 rpm. Two jigs were installed in 1908, a dressing jig was added in 1910. These jigs may have been the same as the Hartz quoted above or smaller.
8	Grinding pan, Forwood Down 5 ft, running at 60 rpm and driven by a 20 hp motor.
9,10	Although frequent reference is made to classifiers, settlers etc. the distinction between them is not always clear. I have interpreted classifier as spitzlutte but settler appears to refer to spitzkasten or any settling tank or vat. In 1908 the Callow screen undersize passed to an hydraulic classifier which was fed to a separate table. Item 10 was a spitzkasten.
11,12	Thickener. This is a dewatering stage consisting of two Callow Cones (Herman states 8 ft, Playford 7.5 ft diameter x 9 ft deep). The settler is a V box for dewatering.
13-15	Wilfley No. 5 tables.
16	Vanner, double belt type. A Wilfley slimer was replaced by the vanner in 1910.
17	Automatic sampler.
18	Calciner. A double deck Merton roaster with three rabble arms. Hearth dimensions 7 ft wide x 14 ft long. Rabble arms 3.25 ft long. Worked as a batch operation. Charge fed to top hearth. When top hearth emptied it was dropped to lower hearth where wood firing was necessary to get sufficient heat for dead roast. Capacity, one ton roasted in 8 hours. This furnace displaced a smaller roaster in 1910.
19	Bin.
20	Automatic sampler.
21	Classifier (probably a spitzlutte).
22	Jig, single compartment.
23,24	Wilfley tables.
25	Settler. A large tank.
26	Grinding pan.
27	Settler, two settling tanks in series.
28	Vanner, double belt type.
29	Strake, canvas.
30	Kieve.
31	Automatic sampler.
32	In 1918 an additional ten head of stamps were added. These were heavier, 1,250 lb. Both these and the older stamps (1,000 lb) were running at 101 drops per minute with a cam lift of 6.5 to 7 inches. The lighter stamps had only two-thirds of the output of the heavier stamps.
33	Spitzkasten. Although shown as a single unit a number of hydraulic sizing devices were in use according to Playford. Probably similar to the arrangement shown in the 1914 flowsheet.
34	Wilfley table probably installed in 1916.
35	Cornish round table installed in 1916. This apparently displaced the vanner.
36	Wilfley table.
37,38	Cornish round tables.
39	Lead treatment; there is no mention of what was done.

of lead, antimony, and zinc and also wolframite.

Gudgeon (1919) listed the minerals observed in the lode but states that only lead, sulphur and arsenic were in sufficient quantity to affect milling. To remove these elements roasting and lixiviation with soda solution were employed in addition to the normal oredressing methods.

Gudgeon gives the production for the period 1908-1918 as:

Ore crushed	130,768 tons
Concentrate production	1,741.92 tons @ 70.5% Sn
Recovered head grade	0.94% Sn

The average concentrate analysis for the four years to 1918 is given by Gudgeon as:

Sn	70.75%	S	0.34%
Pb	0.37%	As	0.11%

Gudgeon states that the enriched sulphide zone contained up to 40% of sulphides but the primary ore contained only 3-5% sulphides.

Playford (1928) gives a grade of 0.75% Sn in 1926 but qualifies this by saying '...but this may be after vanning the weighed assay sample...'. The recovered head shown by published figures from 1918 to 1929 was 0.53% Sn.

Milling Methods

The earliest treatment consisted of battery crushing followed by the recovery of tin by jigging and the use of round tables.

When the new mill was built in 1908 the old mill was to be used for treating calcine when required. By 1910 the milling aim became firstly to produce a pyritic concentrate which was roasted and then the calcine was tabled to produce a final concentrate. This was the method Playford described in 1926.

1910 Plant

The plant was completed in 1908 and additions were made in 1909. Further extensive alterations were made in 1910 when:

(1) the 30# Callow belt screens were replaced by spitzlutte (fig. 4 , item 6);

(2) two double belt vanners were installed (fig. 4 , items 16, 28), item 16 replaced a Wilfley slimer;

(3) a second grinding pan was installed (presumably fig. 4 , item 26);

(4) a larger calciner was built (fig. 4 , item 18); and

(5) another jig was installed (presumably fig. 4 , item 22).

1914 Practice

Herman described the milling operations in April 1914. His flowsheet is given as Figure 4 . The result of a 10# screen on the battery box is a coarser product (fig. 6) than obtained later when 12# was used. However, at this coarse grind the tin was not liberated as can be seen from the

concentrate sizing. From the sizing analyses (fig. 6) and Playford's distribution from the spitzlutte one can guess that in 1914 the jig received half of the battery discharge. Again using Playford as a guide one would guess about a third of the battery discharge passed through the grinding pan (fig. 4 , item 8).

With further reduction of grain size in the roaster the recovery in the jig (fig. 4 , item 22) would not be large but may have been necessary to prevent large grains disturbing the smaller grains on the tables.

While the additions of a grinding pan, jigs and a larger roaster were made to enlarge the capacity of the calcine treatment now that it was necessary to produce a pyritic concentrate, the removal of the Callow screens is not easily explained as a screened product should produce better results in a gravity plant than hydraulically sized material. The inefficiency of the screening machine and perhaps corrosion due to the sulphureous nature of the ore probably accounts for the replacement of screening by hydraulic sizing.

The grinding pan (fig. 4 , item 26) is shown treating table tailings but presumably the kieve (fig. 4 , item 30) tailings would be ground to liberate tin from the table middlings.

The pyritic concentrate sent to the calciner averaged 17.5% Sn while the final concentrate averaged 69.5% Sn. No recovery figures are given although with the samplers shown in the flowsheet these could have been determined. From Playford's figures and allowing for a coarser battery discharge the mill capacity would have been about 40 tons per 24 hours.

1914-1927 Practice

In 1916 we read of the installation of a 'fifth' Wilfley table and of a round slime table (five Wilfley tables are shown in 1914, fig. 4). In 1918 there was a three month shutdown while a new ten-head battery was installed. This according to Playford would have more than doubled crushing capacity. This installation was accompanied by additional boilers, water supply, a new crusher and an ore bin.

The plant ran spasmodically in the post-war period until steam power was discarded in favour of less costly electricity purchased from the Mt Bischoff Company. The mill began operating on electric power on 24 March 1925. From this time until 1927 the mill treated about 19,000 tons or ore per annum for a recovered head grade of 0.4% Sn.

As the 1918 ore reserve was 41,000 tons at 1.01% Sn the inferred recovery is less than half.

1926 Practice

In August 1926 Playford (1928) visited the mill. He sampled various mill products over an eight hour period. Based on this sampling he wrote his report of the operations.

Playford's report and flowsheet show a number of changes from those given by Herman (1914).

The battery screens which were formerly 10# were now 12# with a consequent finer discharge. Playford remarks that the jig does poor work on the +600 µm material, presumably because it is unfreed material.

The spitzlutte (fig. 5 , item 6) appear to be running in 1926 on a

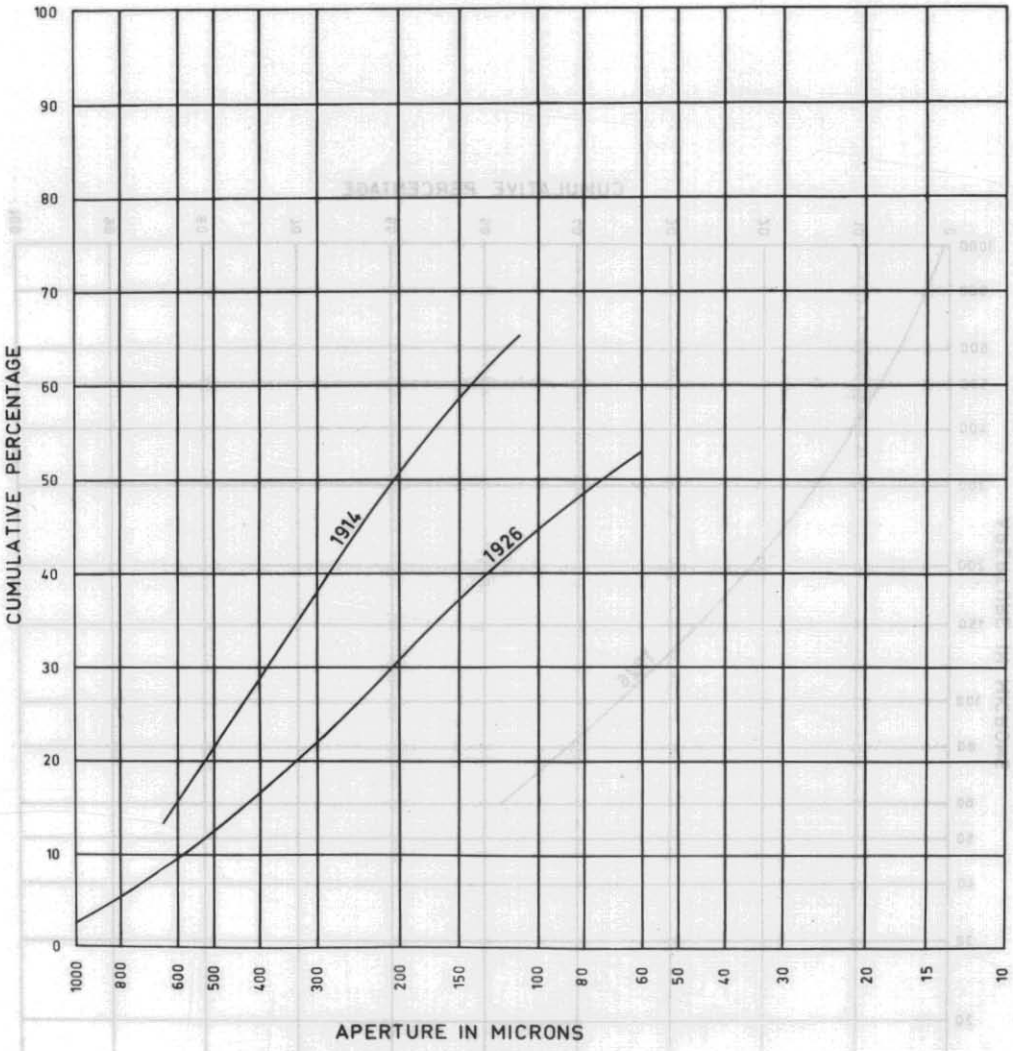


Figure 6. *Mt Bischoff Extended Tin Mining Co. Sizing analyses of battery discharge.*

5 cm

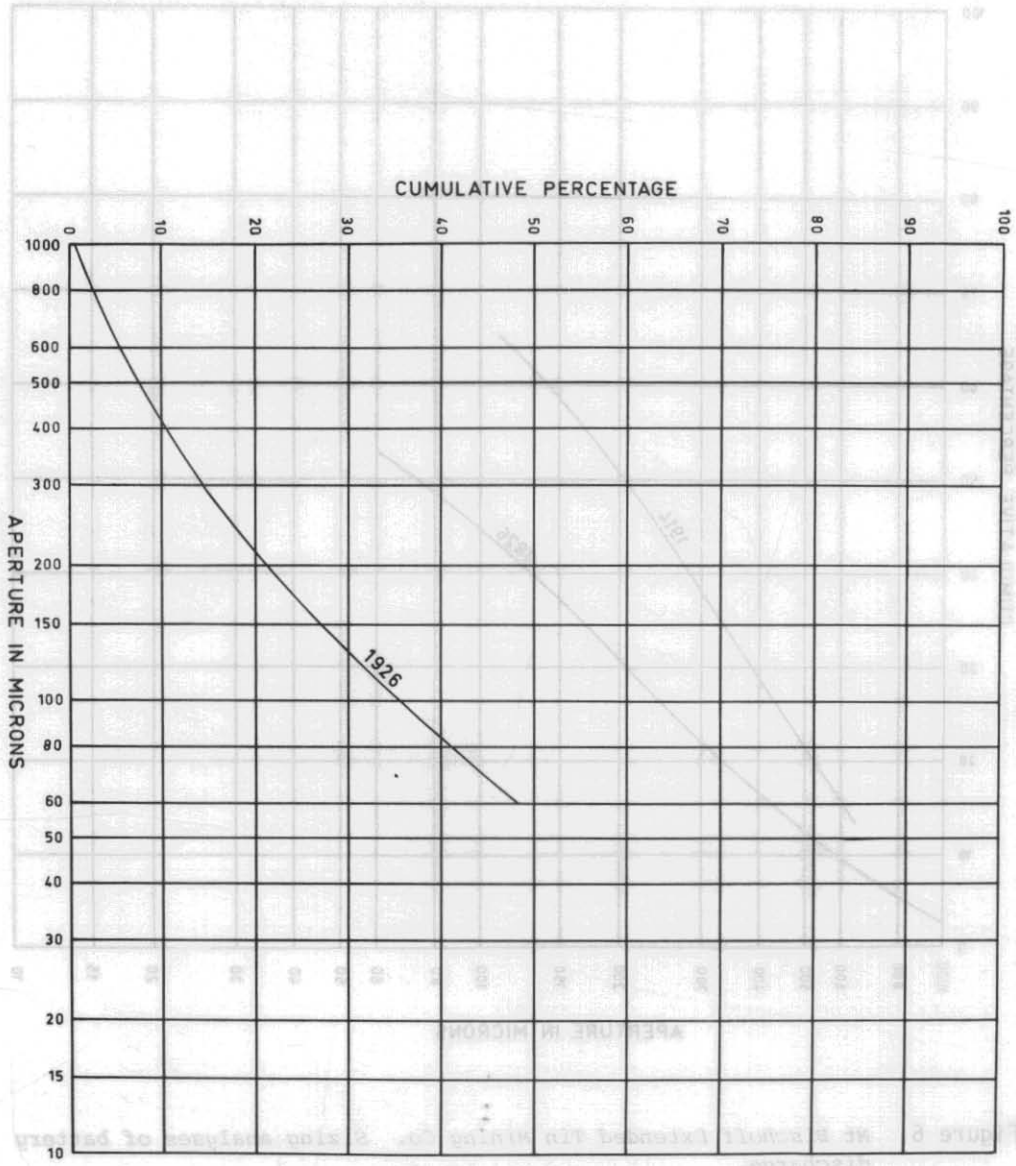
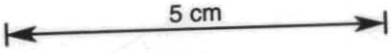


Figure 7. *Mt Bischoff Extended Mill. Tin distribution in battery discharge, 1926.*



similar setting to that of 1914 in spite of finer feed. This in turn means less feed to the jig (fig. 5 , item 7) and consequently leads Playford to comment on the virtual redundancy of the grinding pan (fig. 5 , item 8).

Playford remarks on the poor jig performance outside the size range 100-600 µm yet half the jig feed is inside this size range.

The sizing arrangements ahead of the tables appear to have changed little between 1914 and 1926 except for the treating of the finest fraction where a Cornish round table (fig. 5 , item 35) has replaced the vanner (fig. 4 , item 16).

Playford comments on the performance of the tabling section where less than half the tin in the table feed was recovered. The main loss was in the -60 µm material which contained two-thirds of the tin in the table feed. Only 38% of this tin was recovered. He would have liked to have compared the results from each table and in particular to have compared the Wilfley and round table performances but the set-up precluded such an analysis. However, he concludes that while the very fine cassiterite probably never reaches the tables, even if it did it would not be recovered as table performance below 60 µm is poor. In the calcine treatment circuit the jig (fig. 4 , item 22) in the 1926 circuit has been removed and the grinding pan (fig. 4 , item 26) although still in the circuit, is shown by Playford to be redundant.

Two Wilfley tables (items 23, 24) are still the primary concentrators but the finest tin is recovered by two stages of round tables (fig. 5 , items 37, 38) in place of the vanner (fig. 4 , item 28). The recovery of this section where 93% of the tin passes 60 µm is given as 96% which is contrasted with the poor recovery on the primary tables. Playford suggests reasons for the difference, amongst which the poor classification of the primary table feed is probably the most significant. The calcine being all quite fine is probably of more uniform size than any of the primary table feeds.

Playford (1928) calculated the overall tin recovery at 54.3%. He mentions that flotation of a pyrite concentrate has not been attempted on the Bischoff field "...since the arsenic is present in the form of mispickel and the lead as galena, these minerals would be floated with the pyrite". This quite clearly shows that Playford contemplated floating the sulphides and cassiterite together.

After 1927

During 1927 operations were suspended due to low tin prices and breakages of mill equipment. From then until the plant closed in 1929 about 2,500 tons were treated for a recovered grade of 0.4% Sn. The Company ceased work in 1931.

During 1937 and 1938 tributors worked the old company dumps. These activities recovered 5.8 tons of tin in concentrates assaying 69% Sn. The recovered grade of the residues treated was 0.12% Sn.

Discussion

Although the two early operations treated high grade ore (about 3% Sn) which was probably low in sulphide and hence should have been easily treated neither was viable. The reason probably lies in the attempt to use water power derived from a small and unreliable stream. In order to get sufficient water the plant was located away from the mine thus incurring considerable

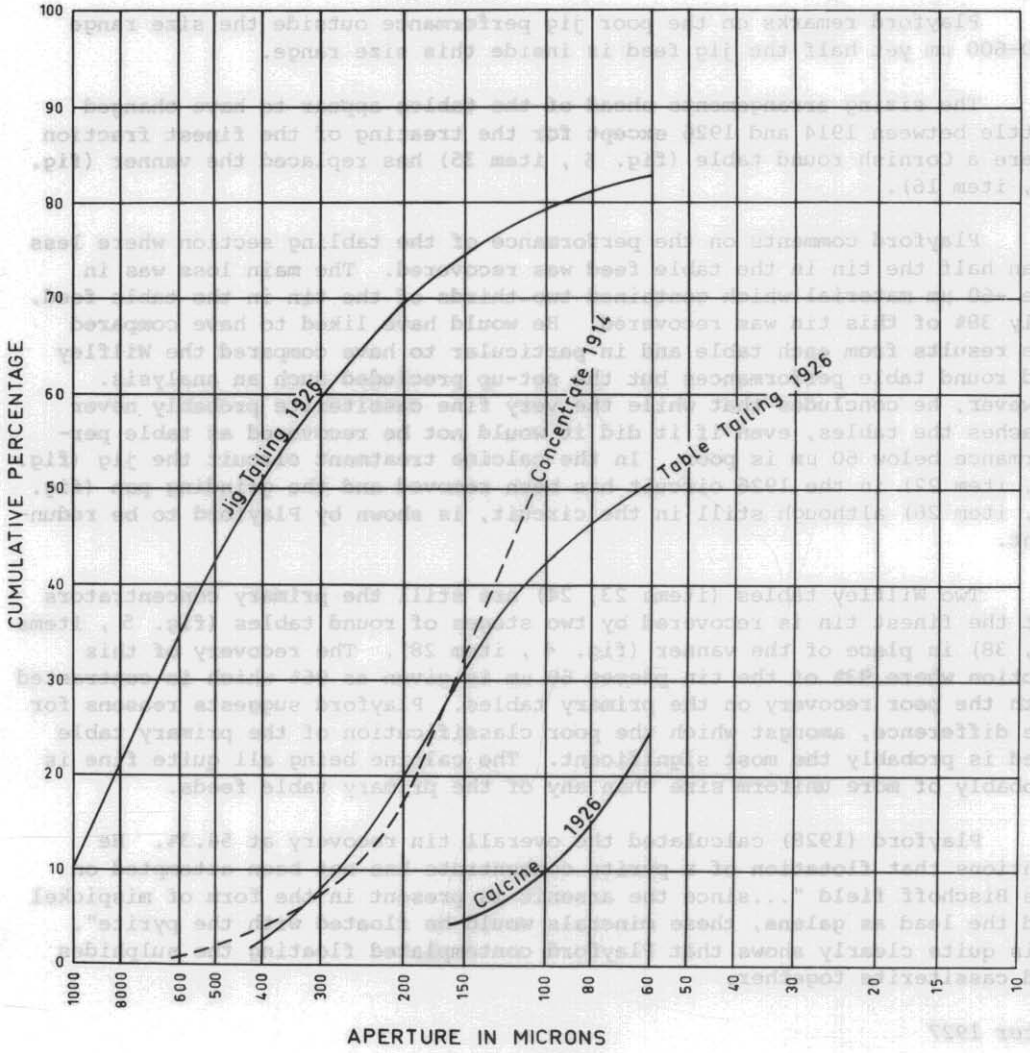
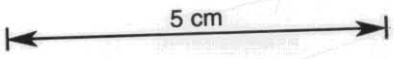


Figure 8. *Mt Bischoff Extended Tin Mining Co. Miscellaneous sizing analyses.*



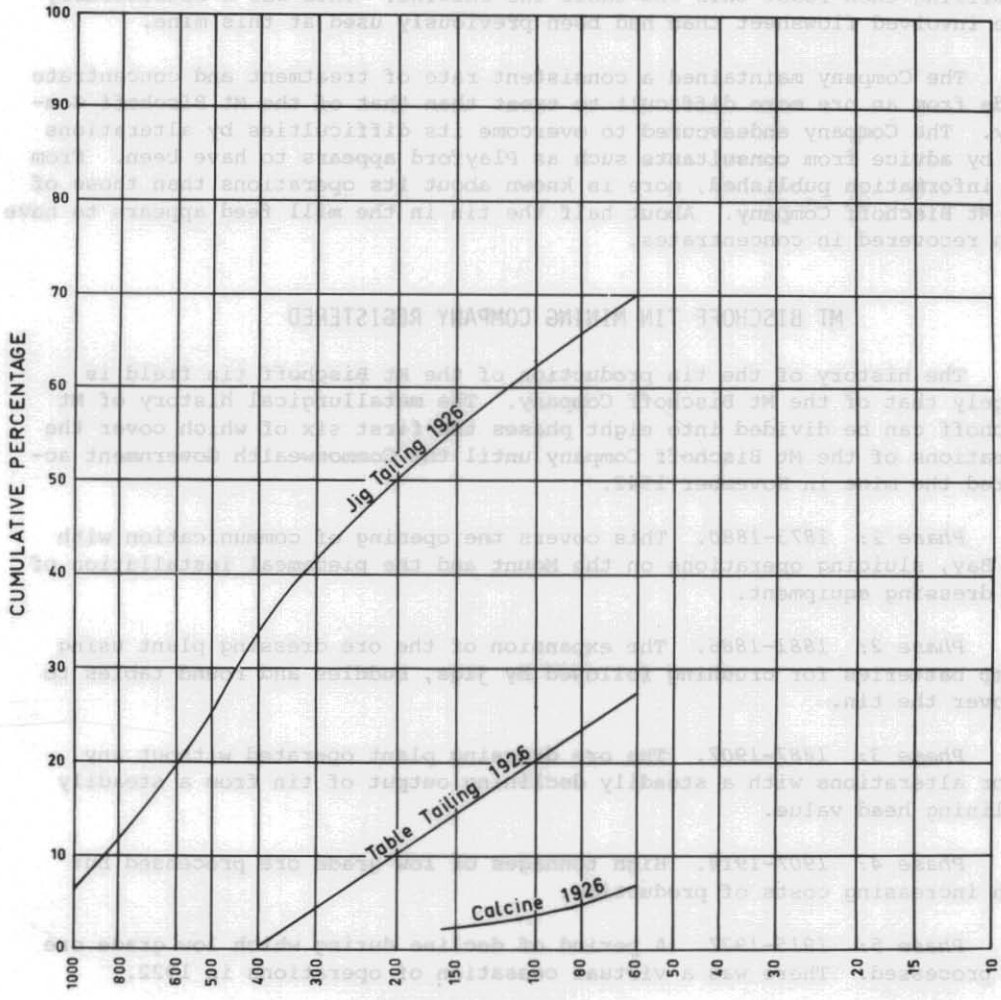


Figure 9. *Mt Bischoff Extended Tin Mining Co. Miscellaneous tin distribution curves.*

transport costs. These costs together with shutdowns due to lack of water, were probably the causes of failure, although in 1894 the presence of sulphides would have become a factor as well.

The Mt Bischoff Extended Company sought to overcome the power problem by using steam but this proved expensive. The Company also encountered the enriched sulphides hence it had to make a pyrite concentrate as a first step in milling then roast this and table the calcine. This was a considerably more involved flowsheet than had been previously used at this mine.

The Company maintained a consistent rate of treatment and concentrate grade from an ore more difficult to treat than that of the Mt Bischoff Company. The Company endeavoured to overcome its difficulties by alterations and by advice from consultants such as Playford appears to have been. From the information published, more is known about its operations than those of the Mt Bischoff Company. About half the tin in the mill feed appears to have been recovered in concentrates.

MT BISCHOFF TIN MINING COMPANY REGISTERED

The history of the tin production of the Mt Bischoff tin field is largely that of the Mt Bischoff Company. The metallurgical history of Mt Bischoff can be divided into eight phases the first six of which cover the operations of the Mt Bischoff Company until the Commonwealth Government acquired the mine in November 1942.

Phase 1: 1873-1880. This covers the opening of communication with Emu Bay, sluicing operations on the Mount and the piecemeal installation of ore dressing equipment.

Phase 2: 1881-1886. The expansion of the ore dressing plant using stamp batteries for crushing followed by jigs, buddles and round tables to recover the tin.

Phase 3: 1887-1907. The ore dressing plant operated without any major alterations with a steadily declining output of tin from a steadily declining head value.

Phase 4: 1907-1914. High tonnages of low grade ore processed but with increasing costs of production.

Phase 5: 1915-1927. A period of decline during which low grade ore was processed. There was a virtual cessation of operations in 1922.

Phase 6: 1928-1942. A period of disintegration during which alluvial and old tailings were retreated. From a central mill treating all production the Company set up around the Mount small milling plants which were let to tributors. Their gravity concentrates were roasted and the calcine tabled in a central Company plant.

Phase 7: 1943-1947. A period of investigation and operation by the Commonwealth and State Governments instigated by the shortage of tin caused by the war.

Phase 8: 1948 onwards. Operations by small parties with small milling plants with little tin production.

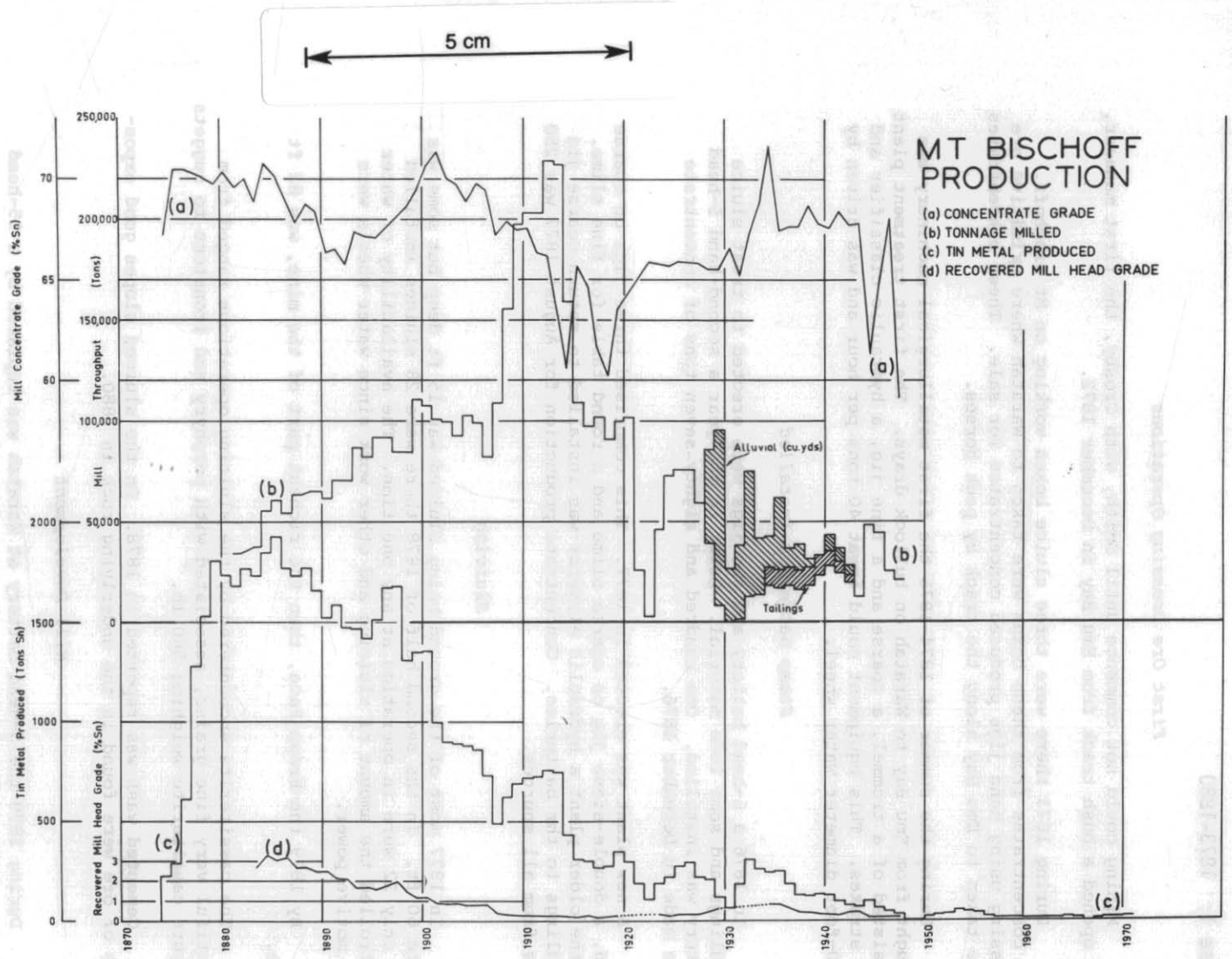

5 cm
MT BISCHOFF PRODUCTION
(a) CONCENTRATE GRADE
(b) TONNAGE MILLED
(c) TIN METAL PRODUCED
(d) RECOVERED MILL HEAD GRADE
Mill Concentrate Grade (%Sn)
Mill Throughput (Tons)
Tin Metal Produced (Tons Sn)
Recovered Mill Head Grade (%Sn)
Alluvial (cu.yds)
Tailings
(a)
(b)
(c)
(d)
250,000
200,000
150,000
100,000
50,000
0
70
65
60
2000
1500
1000
500
0
3
2
1
0
1870
1880
1890
1900
1910
1920
1930
1940
1950
1960
1970

Figure 10.

Phase 1: 1873-1880

First Ore Dressing Operations

Mining could not commence until Smith, with Crosby, the first manager, had opened a bush track from Emu Bay in December 1872.

During 1873 there were three sluice boxes working on Mt Bischoff. The concentrates from these boxes were taken to Waratah where primitive ore dressing using hand jigs produced concentrates for sale. These concentrates were taken to Emu Bay along the track by pack horses.

During the summer of 1874-1875 the first metallurgical machinery was brought from Emu Bay to Waratah on bullock drays. The first treatment plant consisted of a trommel, a coarse and a fine jig, a hydraulic classifier and two strakes. This equipment could treat 40 tons per hour and was driven by a 20-foot diameter water wheel.

Stamp Batteries Installed

In 1876 a 6-head battery and two jigs were erected to treat sluice middlings and some lode material. Later in the year a second-hand 5-head battery was installed. One hundred and eighty-seven tons of concentrate were made in December 1876.

A new plant was erected in 1877. This comprised three jigs on coarse sand, a double-sieve jig on coarse slime and a round table for fine slime. In the older plant a hydraulic elevator was installed to return coarse jig middlings to the batteries. Concentrate production for August 1877 was 250 tons from all sources.

Sluicing

In 1877 most of the ground being sluiced was 15 ft deep but some was up to 60 ft. In the second half of 1878 there were 28 sluices installed but only 22 were in operation at any one time. The availability of water controlled the amount of sluicing and other work since water wheels were the motive power.

By 1879 the Brown Face, then the richest part of the mine, was 82 ft high.

The cassiterite encountered by the sluicing operations ranged from plentiful very fine grains, associated with porphyry and ironstone to nuggets of pure cassiterite weighing 560 lb.

Cemented wash was reported in 1878. On the sluiced slopes good exposures of ore were found in the underlying rock in 1880.

Mill Development

During 1879 the 5-head battery at Waratah was replaced by a 15-head battery. Work began simultaneously on a 40-head battery.

Loss of tin from the sluice boxes and the Waratah mill resulted in steps to recover this tin.

On Mt Bischoff itself, probably in Happy Valley below the sluices, a small plant similar to but smaller than that at Waratah was built late in

1878. However due to lack of water and the fineness of the cassiterite this plant did not recover much tin. Its tailings were apparently rich for in 1881 a ditch was dug to prevent them running onto the East Bischoff Company's lease. This plant closed in August 1883 when the Ringtail plant was commissioned.

At Waratah, work began in 1879 on a plant to treat the tailings from the jigs. Two Munday buddles were installed for this purpose.

In June 1879 it was intended to apply for a five acre tailing lease so that tailings could be stockpiled until the price of tin rose and hence retreatment became profitable.

The 40-head battery, which was driven by a 40-foot diameter water wheel, ran for the first time on 1 October 1880.

However once this greatly enlarged milling capacity became available the horse tram from the mine to the mill could not convey enough ore to keep the mill at full capacity.

Services

New workshops for the carpenters and blacksmiths were built in 1880 together with a house for the mine manager. In 1887 an iron and brass foundry was installed.

Review of Phase 1

Before the first dividend was paid in February 1878 over £100,000 had been spent developing Mt Bischoff. Almost all of this money came from revenue. The paid up capital of £29,600 was made up as follows:

	£
Subscribed capital (7,600 £5 shares paid to £1)	7,600
Fully paid shares given Smith in payment for mine (4,400 @ £5)	22,000

The vital factor in this development was the opening of the road from Emu Bay in 1875. Once the tin could be moved outwards, the money generated provided the capital to finance further development. This ready production of tin without much equipment was only possible because of the extraordinary richness of the surface ground at Mt Bischoff.

This richness and ease of production is well illustrated by the fact that in 1879 over 1,800 tons of contained tin metal were produced (this was the third highest annual yield ever achieved) from between twenty and thirty sluice boxes and some primitive milling and concentrating equipment. In producing this quantity of tin, recovery must have been low because tailings were being stockpiled and a plant was being built to treat mill tailings.

By the end of 1880 the site of the main concentrator had been fixed, namely just below the waterfall in Waratah where there was sufficient fall to allow the use of water wheels to drive the plant.

Phase 2: 1881-1886

Whilst this phase of operations began with newly built enlarged capacity at the mill, the weight of ore treated varied greatly (Table 1). However despite this variation the amount of tin produced remained at a constantly high level.

Table 1. HALF-YEAR PRODUCTION FIGURES

Half-Year ended	40-Head Battery (Tons)	15-Head Battery (Tons)	Sand from Sluices (Tons)	Total (Tons)
December 1882	14,402	7,204	3,118	24,724
June 1883	7,961	4,710	2,012	14,683
December 1883	14,101	7,254	2,370	23,725
June 1884	14,854	6,788	1,992	23,634
December 1884	20,936	7,980	3,460	31,976
June 1885	19,364	9,623	2,860	31,847
December 1885	19,826	7,411	2,538	29,775
June 1886	14,744	7,785	1,270	23,798
December 1886	20,468	8,299	2,558	31,325

Transport of Ore from Mine

Although the Waratah mill had 55 stamps installed during the first half of 1881 an average of only 27 was employed because of insufficient ore. The horse tramway then in use over the mile and a quarter of track between mine and mill could not cope with the increased capacity.

Later in 1881 a steam locomotive replaced the horses. Not only did this keep the mill supplied but it did so at half the cost.

Shortage of Water

With the treatment methods dependant on water the tonnage milled was greatly affected by dry summers like that of 1882-1883.

In the early 1880's four dams were built on Falls Creek. Their total capacity was 165 million gallons. In 1894 the capacity of these dams was raised to 490 million gallons. Additional water was obtained from the Waratah River and by a 4.5 mile long race from the Fossy River.

For sluicing there were three dams on Mt Bischoff.

Sluicing

Those parts of the Mount amenable to sluicing were so treated until such material was exhausted in June 1893 when sluicing was discontinued.

The sluices produced a concentrate assaying 15% Sn which was sent to the 15-head battery which was reserved for treating high grade material. In 1881 five head of stamps were employed on this sluice concentrate.

During the dry summer of 1882-1883 sluicing ceased for a considerable time.

Date	*Number of Sluices Working*
1 January 1883	16
11 January 1883	12
20 January 1883	None
c. 14 April 1883	21

However such was the nature of the Mt Bischoff deposit that during this dry period sufficient high grade 'wash dirt' could be mined and milled that the tin production was maintained.

In 1884 the Brown Face was feeding 14 sluices which accounted for half the mine output of tin

The maximum number of sluice boxes installed was 32 in 1886.

Battery Operation

The practice was adopted of treating the highest grade of ore in the 15-head battery along with the sluice box concentrates. Thus while this battery treated only about a third of the tonnage milled it produced more than half the final concentrate.

Early in 1884 a jaw crusher was installed at the mine. This eliminated the need for hand spalling of the battery feed.

In December 1882 it is reported that the batteries, while now in good order, would need maintenance soon. In December 1884 maintenance was being delayed because there was ample water to maintain operations. In June 1885 the floor of the 15-head battery had to be renewed. Water storage during the dry summer of 1882-1883 reduced the total number of stamps in operation to 30.

Operating Costs

In December 1881 the average cost of crushing and dressing when the machinery was fully employed was 1s4d per ton.

For the half-year ended 31 December 1884 the cost of operating the 40-head battery was £1,311.3.4. By valuing the concentrate made at £42 per ton this gave a profit of £17,975.4.8 which was one penny per ton less than it was three years before.

Additions to the Plant

In this phase the plant evolved the general flowsheet that was to remain unaltered from 1886 to 1909. The major additions were:

(1) In 1881 work on the Slime Sheds to treat jig tailings had progressed to the stage where part of the plant could be run. However, it was not until 1884 that production from the Slime Sheds is regularly reported. In 1886 additions began in order to handle the additional tailings from the new 20-head battery.

(2) A tailings treatment plant known as the Ringtail was built about a mile downstream from the Waratah plant, at a point after the run-off from the Mt Bischoff sluicing joined the mill tailings. Construction began in 1882 and operations commenced in September 1883.

(3) In 1884 construction of a further 20-head stamp battery commenced. This battery was nearing completion at the end of 1886.

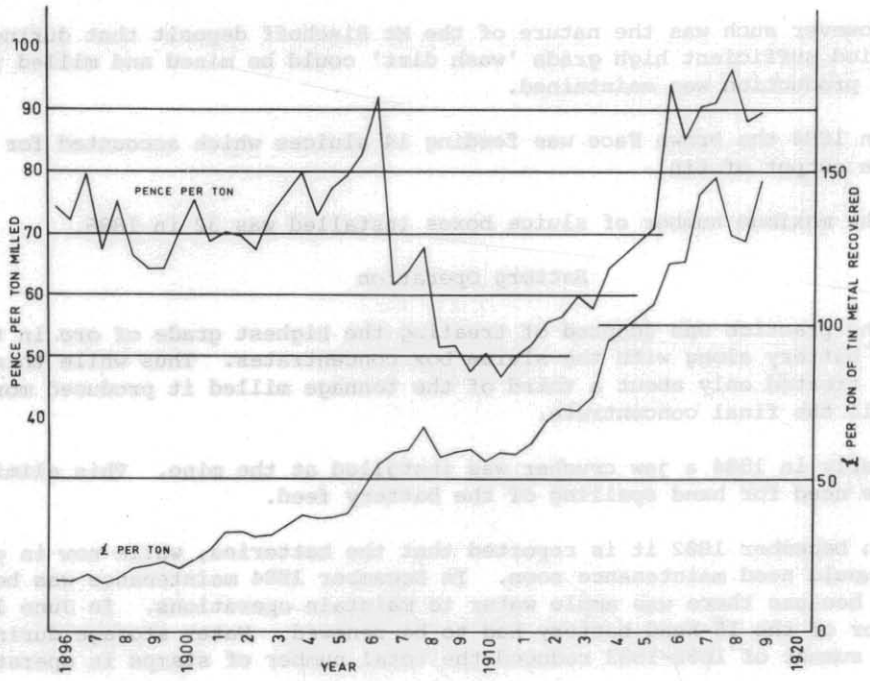


Figure 11. *Mining and milling costs.*

Electric Lighting

A direct current dynamo was installed in June 1883 but it did not work satisfactorily until late in 1884. Electric lighting displaced kerosene lamps, the maintenance cost of which was much higher than that of electric lamps. This dynamo in the Waratah mill was driven from the main water wheel shaft so that when the machinery stopped the lights went out.

The initial dynamo supplied power for 50 lamps in the mill, workshop, offices and store. When the plant was extended these extensions were electrically lit by power from a 100 lamp dynamo installed in 1886. Accumulators were added to the system in 1889 to improve the lighting. Further dynamos were installed in 1897 and 1902.

The dynamo installed at the Ringtail plant in 1897 had its own Pelton wheel drive so that the lights were kept burning when the machinery had been stopped. Although this was a great advantage, arrangements in the Waratah mill precluded its adoption there.

These direct current generators were superseded by the alternating current hydro-electric scheme installed in 1907.

Slime Sheds

The treatment of the mill tailings when intiated proved very profitable. In the half-year ended December 1884, 67.5 tons of concentrate were recovered at a cost of £164.2.8d.

5 cm

A Chilian mill and additional buddles were installed in 1885.

Concentrate production figures were:

Half-year Ended	*Concentrate Produced* (tons)
June 1884	46.6
December 1884	67.5
June 1885	99
December 1885	106.4
June 1886	?
December 1886	76.7

Ringtail Plant

Although operations started in 1883 the installation was not completed until 1884. Apparently floods caused losses of tin because in June 1885 it was reported that new storage dams for mine waste and plant feed had eliminated the risk of flood losses.

For the half-year ended December 1884, 103 tons of concentrates valued at £4,326 were made at a cost of £516 so that the profit was £3,810 for the period.

Concentrate production figures were:

Half-year Ended	*Concentrate Produced* (tons)	*Remarks*
December 1883	50	Sept.-Dec. only
June 1884	92.5	
December 1884	103	
June 1885	123	
December 1885	94	
June 1886	76.7	Lower grade feed

Catch'em Plant

Early in 1886 work began on yet another tailings retreatment plant which was sited downstream from the Ringtail plant near the company's boundary.

Review of Phase 2

During this period the output averaged just under 1,800 tons of metallic tin in concentrates per annum, the minimum being 1,680 tons in 1881 and the maximum 1,925 tons in 1885 which was the highest production for any year.

This high rate of production was maintained despite two occurrences that would have severely curtailed production of an ordinary mine. The first of these was the inability to transport ore from the mine to the mill in 1881 when for six months there was only sufficient ore to keep less than half the stamps working. Secondly, in the drought of early 1883, the tonnage milled was nearly cut by half.

The manager's reports refer to the highly profitable nature of the operations including those recovering tin from previously discarded tailings. Such tailings point to the inefficiency of the tin recovery processes employed.

The conclusion is that no matter how inefficiently the mine was run the extraordinary richness of the Mt Bischoff ore body would allow a substantial profit to be made.

By the end of 1886 the Waratah concentrator had assumed the form which it was to keep for the next twenty years.

Phase 3: 1887-1907

The plant installed during Phase 2 operated throughout this period with additions to equipment but only minor amendments to the circuit. A decline both in metal produced and the grade of the ore treated is a feature of this period which culminated in the crisis of 1906.

Ore Treatment

The ore was treated by the following:

Crushers	Portable units which were moved around the mine workings.
Waratah Mill	Batteries of 15 and 60 heads with the Slime Sheds treating the battery tailings.
Ringtail plant	A tailings treatment plant located to treat sluice and Waratah mill tailings.
Catch'em plant	A plant built to retreat the Ringtail tailings. It operated from 1888 until 1895.
North Valley Mill	An entirely separate mill built to handle sulphide ores. It operated with indifferent results between 1892 and 1894.

Mt Bischoff

Crushing

Jaw breakers were installed to handle hard stone that otherwise would have required spalling before it could be fed to the batteries. Portable 14-inch jaw crushers, each driven by its own steam engine were moved around the working faces crushing the oversize from two-inch grizzlies.

The first jaw crusher was installed in 1884. After this crusher was out of operation for a month in 1890 with a broken jaw additional crushers were purchased in 1891, 1894 and 1899, although it appears only two or three were in use at any one time. In 1900 one crusher was moved to the Waratah mill where it was driven by water power.

Waratah Mill

Flowsheet

The flowsheets of both batteries and the Slime Sheds in 1904 are given in Figure 12. Both batteries have similar flowsheets. There were two slime sheds treating the battery tailings.

In essence the method of treatment was:

(1) Crushing in stamp batteries

(2) Classification into coarse and fine jig feed by spitzlutte

60 HEAD MILL:
Items 1-21.

15 HEAD MILL:
Items 22-35.

SLIME SHEDS:
Items 36-59.

FINAL PRODUCTS:

Concentrates:
First Class: Items 3, 4, 13, 21, 26, 35, 53, 59
Second Class: Items 6, 28, 45, 55

Tailings and Overflows:
Items: 3, 24, 36, 47, 48, 49, 54, 55, 56, 57

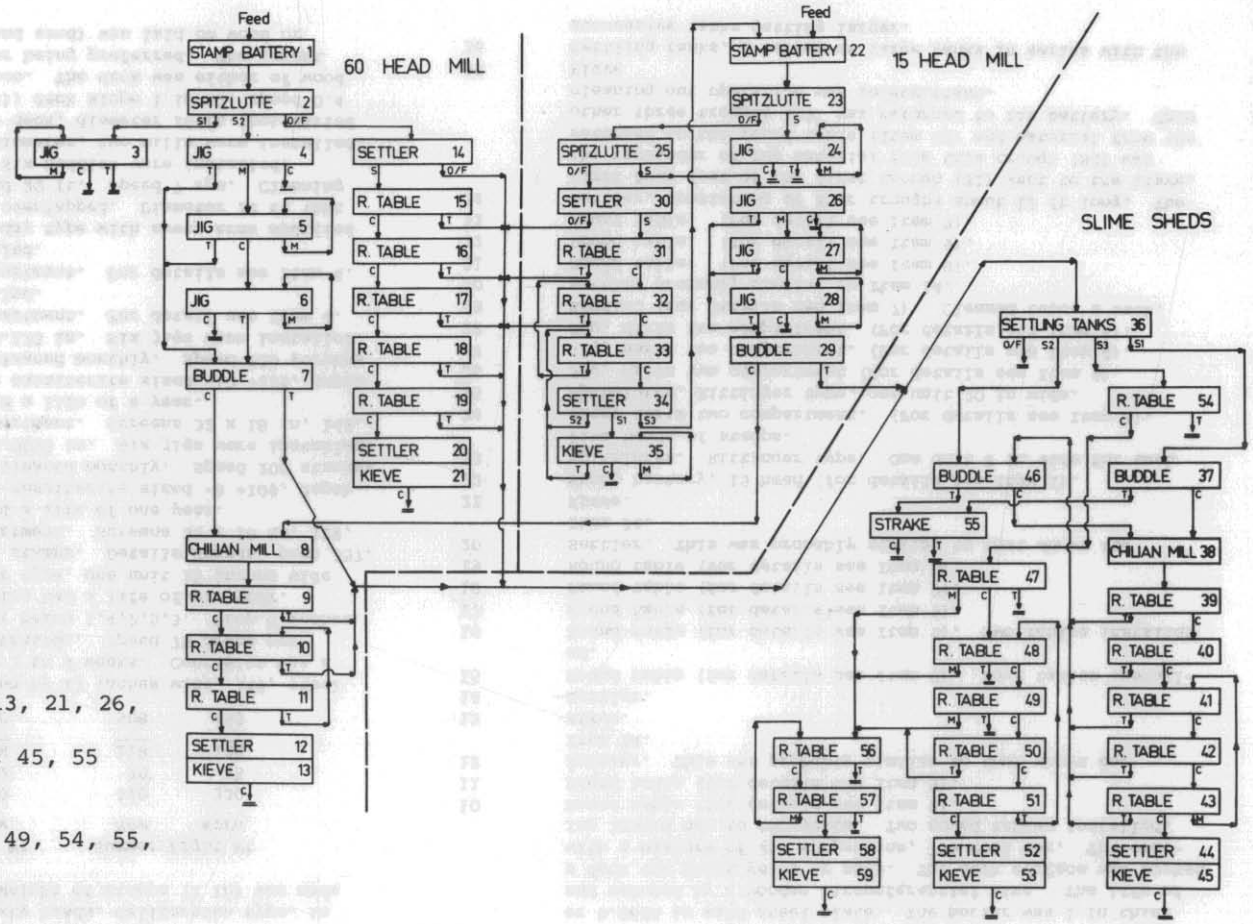


Figure 12. *Mt Bischoff Tin Mining Co. Waratah flowsheet, 1904 [Herman].*

Notes on Figure 12. *Waratah Concentrator Flowsheet, 1904*

Item No.	Description
1	Stamp batteries: sixty heads, Californian type, in batteries of five. Weight of stamps (1 lb) was made up as follows:

Part	Full Wt New	Summer Light Wt New	Summer Light Wt Worn
Shank	320	320	320
Shoe	132	70	35
Head	118	118	100
Total	560	508	455

Screens 19 inches long by 12 inches wide, 14#, steel wire 21 b.w.g. Life 3 to 4 weeks. Corrosion was a problem as well as abrasion. Speed 75 drops per minute the lift order being 1,4,2,5,3. Drop 8 inches. Die weight 70 lb. Dies had a life of one year.

2	Spitzlutte, Rittinger type, one unit 15 inches wide for each ten head of stamps. Details Herman, page 357.
3	Jig, Hartz two compartment. Screens 32 x 18 in, 12#, made of brass and had a life of one year. Ragging: Queen lode cassiterite sized -8 +10#, depth 0.75-1 in. Ragging cleaned monthly. Speed 200 strokes per minute, length 0.0625 in. Six jigs were installed.
4	Jigs, Hartz two compartment. Screens 32 x 18 in, 14#, made of brass and had a life of a year. Ragging: Queen lode cassiterite sized -10 +12#, depth <0.75 in. Ragging cleaned monthly. Speed 220 strokes per minute, length 0.125 in. Six jigs were installed.
5	Jigs, Hartz two compartment. For detail see Item 4. Two jigs were installed.
6	Jigs, Hartz two compartment. For details see Item 4. Two jigs were installed.
7	Buddle, concave, Munday type with sweep arms modified so that areas swept overlapped. Diameter 18 ft (but ranged between 14 and 20 ft. Speed 7 rpm. Cleaning every eight hours. Six buddles were installed.
8	Chilian mill, 5 ft diameter, two mills were installed.
9	Round table: Single deck, diameter 16 ft (but varied between 10 and 16 ft); deck slope 1 in 12; speed 0.4 rpm. Deck composition. The deck was either of wood or cement, the latter being preferred. The cement mortar (1:1 cement and sand) was laid on wood or or 0.0625 in mild steel plate. The mortar was 1 in thick and secured by a wooden circumferential tire. The life of a deck was three years or more. The deck surface was coated with a mixture of 25% turpentine, 75% coal tar. The coating lasted one to two years. Two round tables installed.
10	Round table (for details see Item 9).
11	Round table (for details see Item 9).
12	Settler. This was probably similar to that shown for Item 34.
13	Kieve.
14	Settler.
15	Round table (for details see Item 9). Four tables installed.
16	Round table (for details see Item 9). Two tables installed
17	Round table (for details see Item 9).
18	Round table (for details see Item 9).
19	Round table (for details see Item 9).
20	Settler. This was probably similar to that shown for Item 34.
21	Kieve.
22	Stamp battery, 15 head (for details see Item 1).
23	Spitzlutte. Rittinger type. One unit 8 in wide for each five heads of stamps.
24	Jigs, Hartz two compartment. (For details see Item 3).
25	Spitzlutte, Rittinger type, one unit 20 in wide.
26	Jig, Hartz two compartment (For details see Item 4).
27	Jig, Hartz two compartment. (For details see Item 4).
28	Jig, Hartz two compartment. (For details see Item 4).
29	Buddle, (for details see Item 7). Cleaned twice a week.
30	Settler probably similar to Item 34.
31	Round table. (For detail see Item 9).
32	Round table. (For detail see Item 9).
33	Round table. (For detail see Item 9).
34	Settler, consisting of four troughs about 12 ft long. The first four feet of the first trough (S1) went to the kieve, the remainder of the material from this trough (S2) was returned to the round table (Item S3) and material from the other three troughs (S3) was returned to the battery. This cleaning out operation was intermittant.
35	Kieve
36	Settling tanks, a series of large tanks in series with the successive tanks getting larger.

Notes on Figure 12. *Waratah Concentrator Flowsheet, 1904 - continued*

Item No.	*Description*
37	Buddle. (For details see Item 7). Cleaning every 24 hours. Four buddles were installed.
38	Chilian mill. 5 ft in diameter.
39	Round table (For details see Item 9).
40	Round table. (For details see Item 9).
41	Round table. (For details see Item 9).
42	Round table. (For details see Item 9).
43	Round table. (For details see Item 9).
44	Settler probably similar to Item 34.
45	Kieve.
46	Buddle. (For details see Item 7). Cleaning every 24 hours. Two buddles were installed.
47	Round table. (For detail see Item 9).
48	Round table. (For detail see Item 9).
49	Round table. (For detail see Item 9).
50	Round table. (For detail see Item 9).
51	Round table. (For detail see Item 9).
52	Settler, probably similar to Item 34.
53	Kieve.
54	Round table. (For detail see Item 9). Four tables were installed.
55	Strake 12-15 ft long, 2 ft wide; cleaned occasionally. Three strakes were installed.
56	Round table. (For detail see Item 9).
57	Round table. (For detail see Item 9).
58	Settler, probably similar to Item 34.
59	Kieve.

The equipment in each mill in 1904 is summarised in the following table:

	60 Mill	*15 Mill*	*Slime Sheds*	*Ring-tail*	*Total*
Grinding and Crushing					
Stamps	60	15	-	-	75
Chilian mills	2	-	1	2	5
Concentrating					
Jigs	16	5	-	-	21
Buddles	6	1	6	6	19
Round tables, single	9	3	5	10	27
Round tables, double	2	-	7	-	9

The operations were not quite as continuous as one might infer from the flowsheet. For instance the round tables (Items 56 and 57) were operated by letting the slime from Table 57 accumulate, then at intervals the feed to Table 56 was stopped and this slime fed to that table.

Extensive use was made of a hydraulic elevator for moving pulp from one stage to the next. These elevators were like injectors where a jet of water through the nozzle sucked in pulp and discharged it at a higher elevation to the next treatment stage. This arrangement was used on jigging and round tabling where such units were used in series.

(3) Recovery by jigs of the free cassiterite in the spitzlutte products

(4) Retreatment of the jig tailings on buddles followed by grinding, in Chilian mills, of the buddle concentrate which was then up-graded on round tables

(5) Concentration of the spitzlutte overflow on round tables

(6) Retreatment of the battery tailings in the Slime Sheds which consisted of buddle concentration of settled material; grinding the buddle concentrates in a Chilian mill, followed by concentration on round tables

(7) Up-grading of the round table concentrates in kieves.

Details of the milling equipment are given in the legend to the flowsheet (fig. 12).

Although the flowsheet gives the impression of a continuous flow in all stages this was not the case in most of the round table, settling and kieving operations.

15-Head Battery

The practice of feeding this battery with higher grade ore and sluice box concentrates continued until June 1893 after which both batteries treated a common feed until 1900 when the feed to the 15-head battery again appears to have been richer although only marginally so.

Additions and Modifications

Additional buddles were added to each battery in 1889. Additional Chilian mills were installed in the 60-head battery in 1889 and 1897.

The Slime Sheds which were built between 1887 and 1889 were fitted with double round tables, that is, two tables one above the other on a common vertical shaft. This saved space and power. It was proposed to install triple deck tables in 1890. The first hydraulic elevator, which used a water injection system to elevate a middling to the next treatment stage, was installed in 1893 in the Slime Sheds where it replaced a boy. Later in 1893 forty-three such elevators were installed in the 60-head battery and four in the 15-head battery. In 1896, twenty-five elevators were installed in the Slime Sheds.

In 1893 a round table deck of iron covered by cement was used to replace the former wooden decks.

Brass frames and mesh were fitted to the jigs during 1896 with good results. These gave the bed screen greater rigidity without restricting the open area. Brass also lasted longer than steel.

In 1903 an oscillating table was imported from Europe to replace the buddles.

Operating Notes

In 1888 the round tables were said to save the fine tin but that this was difficult to concentrate. The Chilian mill made a big improvement in slime treatment; it was reported in 1889 that about half the slime tin was recovered on buddles and the other half on the round tables. This presumably is through items 45 and 53 respectively on the flowsheet (fig. 12).

The double round tables gave 'great satisfaction' in 1889.

In 1890 fines were accumulating faster than they could be treated in the slime plant. In 1892 production from the Slime Sheds was down because of material accumulating on the floors and the poor grade of the feed.

In 1892 it was found that when ferruginous material in the battery tailings was crushed it contained a surprising amount of tin.

From 1896 the half-yearly company reports show the average tonnage treated per stamp and the mine operating costs per ton treated.

Battery Operation

Trains delivered the ore to the mill where it was, in Herman's (1914) words '....distributed, with considerable manual labour, by long chutes and partly by re-tramming, to the various battery floors, where it was hand-fed to the stamps'.

Hand feeding was necessary because of the sand and clay content of the ore, some ore merely needing agitation for it to pass through the screen. The rate of crushing was between 4 and 5 tons per stamp per 24 hours through a 14# screen. Twelve men (four per shift) were employed feeding the stamps.

The water added to the battery box was kept to a minimum to prevent sliming of the cassiterite.

In summer, when water was scarce, light weight shoes were fitted to the stamps which then required less water power to drive them. This was said to suit milling operations better than closing down some stamps. This reduction in weight for the summer was only about ten per cent which was the same reduction as that due to wear between a new shoe and a worn shoe due for replacement. Nevertheless Herman reports that between 1896 and 1903 mill stoppages averaged 460 stamp-days per annum due to lack of water.

Jig Operation

The jig feed in 1894 contained 4-9% Sn. The first hutch of the primary jigs (items 3, 4, 24 and 26 in Figure 12) produced 'First Class' concentrates that required no further treatment. About ten tons of 'First Class' concentrate which assayed better than 70% Sn were produced for every ton of 'Second Class' concentrate which assayed 65% Sn.

Buddle Operation

The buddle feed assayed 0.2% Sn in 1894. From such feed a buddle concentrate assaying 5% Sn was made. Re-buddling this concentrate yielded a product assaying 45% Sn.

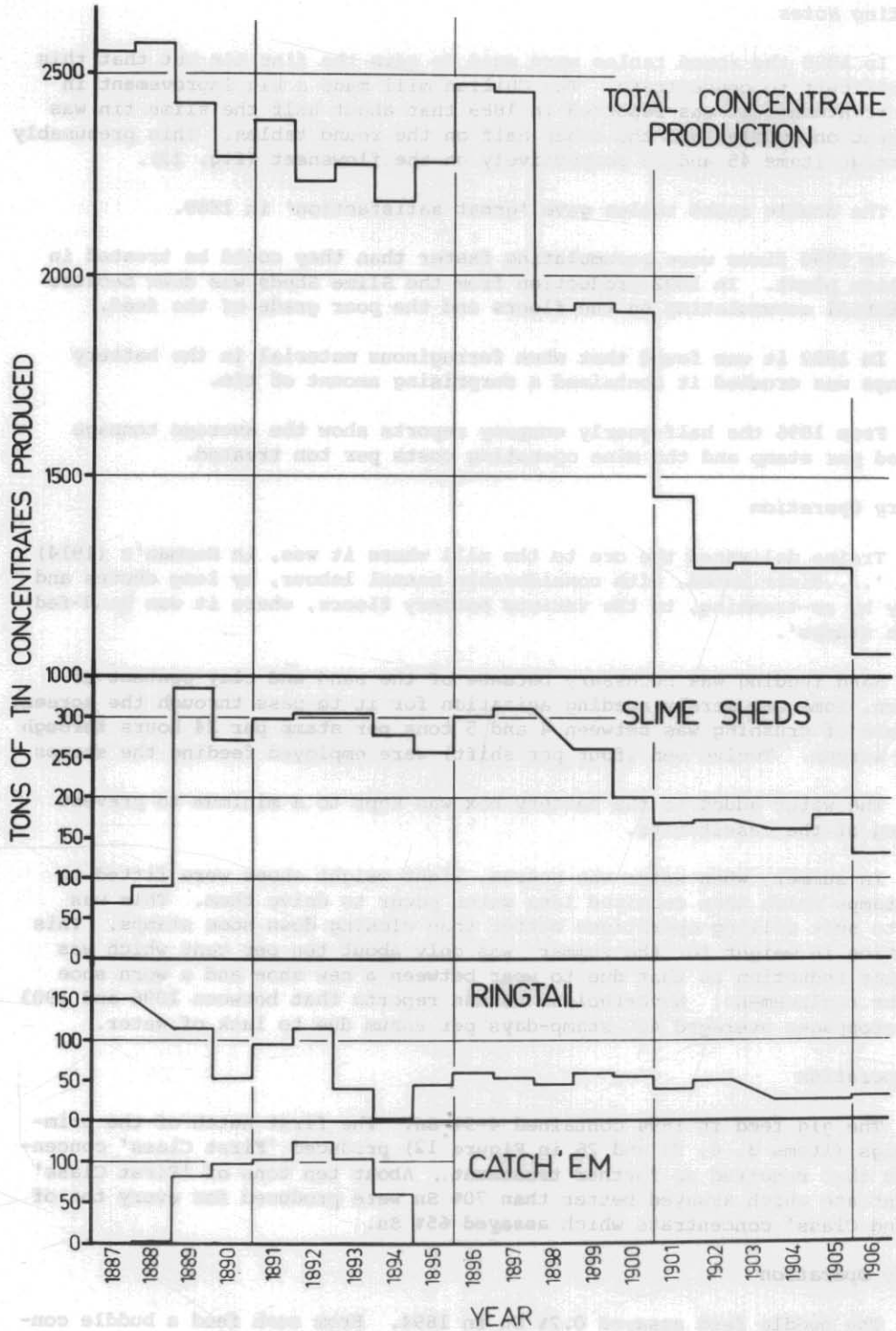


Figure 13. *Mt Bischoff Tin Mining Co. Tin concentrate production, 1871-1970.*

5 cm

Round Table Operation

Round tables were fed at the rate of 800 lb of slime per hour. From a feed assaying between 0.1 and 0.8% Sn a concentrate assaying between 10 and 15% Sn was made. Where double tables were used the concentrate from one deck was further up-graded on the other.

The tables did not operate continuously as implied by the flowsheet. Herman (1914) describes the operation of Tables 56 and 57 (fig. 12) thus: '....in the slime sheds dirty slime from Table 57* was allowed to accumulate, and was then put, at intervals, over Tables 56 and 57 again, the feed of low-grade pulp on Table 56 being stopped in the meantime. This gave a slime product which, after twice kieving, was first-class (over 70%) ore'.

Other Equipment

The hydraulic elevators lifted material from one stage to the next thus ensuring a uniformity of feeding.

The gravity flow between machines was described by Herman thus: 'throughout the mills large and small settlers, labyrinthal troughs, and de-watering tanks were in use at every available opportunity'.

Maintenance

As the 1890's advanced references to the need to maintain buildings and equipment become frequent in the Half-Yearly Reports. Two carpenters were continuously employed on such maintenance although they were apparently unable to keep up with repairs. The following are exerpts from Kayser's Half-Yearly Report of July 1899:

'The remarks I made in my last report with reference to the carpenters and their work I have again to repeat, but on this occasion with more emphasis as it is impossible to keep pace with the work which has to be executed within a given time'.

Of the Ringtail Sheds:

'Nearly all the fluming is either rotten or worn out, and should have been replaced long since but we have neither had the men to do the work nor the material to do it with'.

In the Half-Yearly Report dated 29 June 1901 Kayser reported that: 'in the middle of the present month one of the camshafts broke, but as we had an old one on hand it was quickly substituted and work resumed. This is a warning to renew the camshafts as they have been in work for nearly 20 years, and a breakage may occur at any time.

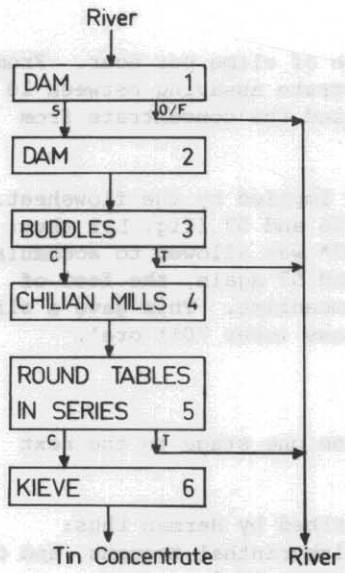
'In the shops, the mechanics have been fully employed and it is hardly possible to keep pace with the requirements'.

Ringtail

Flowsheet

The flowsheet (fig. 14) is quite simple. The coarser material from the river was caught in a dam, then periodically this coarse material was run to a second dam from which the plant was fed.

*numbers given are item numbers from Figure 12 and not those given in Herman's text.



Item No.	*Description*
1	Dam in river impounding tailings. When full sand drained into Dam 2.
2	Dam feeding plant.
3	Buddles, six were installed.
4	Chilian mills; two installed, one for each buddle.
5	Round tables: the concentrate was upgraded over a series of fine tablings. Triple tables were installed.
6	Kieve.

Figure 14. *Ringtail plant flowsheet, 1904.*

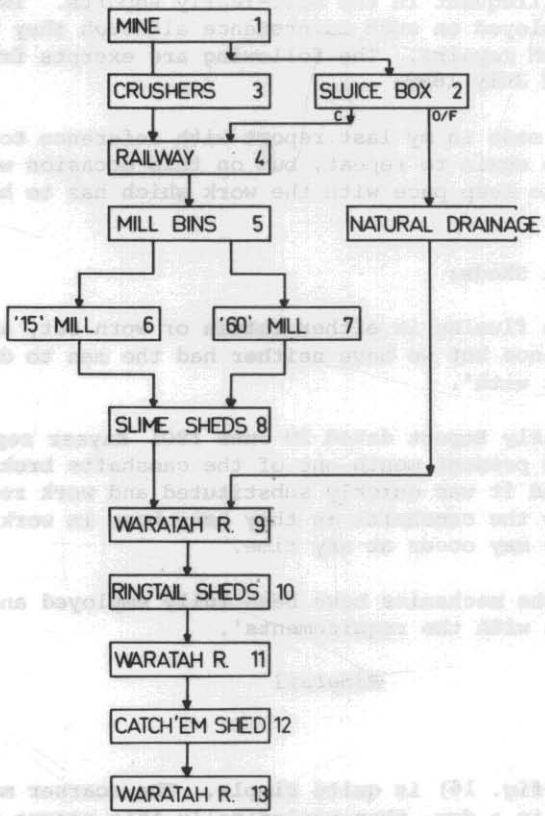


Figure 15. *Material flow diagram, c.1890.*

Operating Notes

Frequently, even before the cessation of sluicing on Mt Bischoff, the Half-Yearly Reports refer to low grade feed, and the need for more grinding capacity.

Late in 1889 a Chilian mill, three buddles and a triple round table were installed. A second triple was installed in 1891 when a hydraulic elevator was installed to feed the Chilian mill. When the Catch'em equipment was added to the Ringtail plant in 1896 there were 21 hydraulic elevators working in this plant.

When the mine or mill were working clayey ore this precluded satisfactory tin recovery in the Ringtail plant. There was practically no production from this plant in 1894, the only year this occurred in the period under review.

A Card table was installed in the Ringtail plant in 1908 not long before it was closed down late in 1909. It was stated that the Waratah mill recovery had so improved, that tailing retreatment was unnecessary.

Catch'em Plant

This plant began operating in December 1888. The flow sheet appears to have been similar to that of Ringtail: dams in the river, initial concentration on buddles followed by the grinding of the buddle concentrate then upgrading on round tables.

The plant appears to have consisted of six buddles, a Chilian mill and three double round tables, all driven by a water wheel.

The initial yield was not up to expectations but hopes were expressed that this plant would pay operating costs and depreciation and yield a small profit. That there was a poor tin recovery meant that the tailings from the other plants were not worth working.

The plant was shut down and the equipment transferred to the Ringtail plant in 1895.

Sources of Concentrate

In discussing the sources of the tin concentrate, items numbered are those given in the flowsheet (fig. 12).

In the Half-Yearly Reports for 1896 the distribution of concentrates was:

	%
Coarse jigs (items 3, 24)	44
Fine jigs (items 4, 6, 26, 28)	29
Slime	23.5
Buddles (presumably item 13)	c. 1.5
Slime Sheds (Ringtail, see below)	2
	100

Kayser and Provis (1895) stated that 65-75% of the tin was produced by jigs, 4.75% came from buddles and 15-20% came from round tables.

The concentrate production in 1896 was:

		Tons	%
Slime Shed	118 tons		
70-foot Shed	185 Total Slime Sheds	303	12
Ringtail		59	2
Jigs (73% of total by calculation)		1,805	73
Battery slime (items 21, 35) by difference		310	13
		2,477	100

From the above it would appear that the source of concentrate other than from the jigs was not well established.

North Valley Mill

Extensive tunnel workings on the northern slopes of Mt Bischoff led to the erection in 1890 of an entirely separate mill which cost £18,000.

The plant erected consisted initially of a jaw crusher, rotary crusher, jigs, buddles, a rotary calciner and round tables. All machinery was driven by a water wheel.

Early in 1892 an attempt was made to run this plant but sticky ore, clay and decomposed pyrite caused so much trouble that operations were suspended until a battery was installed. Washing was considered as an alternative to the battery.

When the battery began operating late in 1892 it was found that steel screens corroded quickly. Copper and brass screens lasted longer. However the rotary calciner would not remove the sulphur and arsenic from the ore which was stated to be 90% pyrite and arsenopyrite. The first shipment of concentrate (about 4 tons) was rejected by the smelter as it required further calcination.

A reverberatory furnace was built during 1893 to replace the calciner. Whilst this furnace produced a calcined product the roasting was slow, the sulphureous and arsenical fumes were a problem and the output was below expectations. Once heated up, the sulphur in the ore was sufficient to maintain combustion.

The grade of ore was lower than had been anticipated and was more difficult to treat hence the whole operation, which was unprofitable, was shut down in 1894. This plant was destroyed in a bush fire in 1906.

Operation in 1942-1943

Although at this time North Valley was just one component of the tributing operations around Mt Bischoff and will be treated as such elsewhere, it is proposed to present some data here which follows on and extends the information on this plant.

In a review of the Mt Bischoff operation (R.73, July 1942) the North Valley plant was using a 10-head stamp battery with a 12# discharge screen. Classification was by spitzlutte. Seven tables were being used to produce a pyritic concentrate which was roasted at Waratah where the calcine was also treated to produce the final tin concentrate.

One hundred and sixty tons of ore were being treated weekly to produce 25 tons of pyrite concentrate assaying 3-4% Sn. In R.74 (December 1942) an analysis of this concentrate is:

	%		%
Sn	3.2	S	39.8
As	5.1	Cu	0.2
Fe	45.0		

The major sulphide minerals were stated to be pyrite, pyrrhotite and arsenopyrite.

Weather Effects

Water shortages affected operations markedly in 1887, 1889, 1890, 1891, 1894 and 1897. The summer of 1894-1895 was the first in which the plant had sufficient water to run throughout the summer.

The plant was damaged in the severe floods which followed the drought of 1894.

Severe frosts in the winter of 1901, with temperatures as low as 16°F (-9°C), caused the water pipes to freeze.

Bad weather in 1904 and 1905 was blamed for lower ore production.

Power

The plants were driven by water wheels. The various sections of the Waratah mill were sited one below the other to utilise the same water many times.

Plant	*Diameter of Overshot Water Wheels (ft)*
15-head battery	28 and 20
Fitting shop	30
60-head battery	40 and 28
Slime Sheds	19.5 and 15

Clark (1904) commented: 'The row of waterwheels is not looked upon with favour by the modern hydraulic engineer, but their existence is justified on the ground of simplicity and suitability, the quantity of water available being large and the fall comparatively small. The waste water from one wheel is made use of wherever possible to drive a wheel below'.

Kayser and Provis (1895) justified the water wheels with: 'Overshot wheels are used in preference to turbines or Pelton wheels, because they require less skilled supervision; further, because the water employed comes from other dressing sheds above and is more or less charged with sand, which would quickly wear out either of the two latter machines.'

Hydro-Electric Power Scheme

The General Mine Manager recommended that a hydro-electric power scheme be installed on 8 October 1904. Herman's (1914) account of the former water wheel plant and the need for its replacement is:

'The effective average head utilised for power was about 100 ft, not allowing for efficiency (or inefficiency) of the water wheels'.

'For a number of reasons not necessary to enter into here it was becoming apparent that power provision would have to be largely increased, and consequently it was decided in 1905 (primarily on the urgent represent-

ations of the writer, who at the time was assistant manager of the mine), to place an electric power station near the Ringtail sheds, utilising the power water under a head of 560 ft instead of 100 ft, and quadrupling the power hitherto obtained from it'.

Construction of the power station began in January 1906 and by December 1907 the mill was being driven by electric motors and the railway from mine to mill had been electrified.

Mill Labour

In 1894 (Kayser and Provis, 1895) the operating crew was 37 men and boys which formed three shift crews of twelve each and on day shift there was one man on general duties who was paid 8 shillings per day.

The shift crew and their daily rates of pay were:

Number	*Description*	*Per day* s	d
1	Attending machinery and supervising work	9	2
3	Feeding 60-head of stamps	8	0
1	Feeding 15-head of stamps	7	6
3	Attending jigs	7	6
2	Attending tables	5	6
1	Attending buddles	5	6
1	Attending tables and buddles in Slime Sheds	5	6

Tin Loss in Mill Tailings

Kayser and Provis (1895) state that the loss of cassiterite varies with the material being treated. The White Face ore was easily treated with an extremely small loss of tin but the Brown and Slaughteryard faces were more difficult to treat on account of the hematite present and gave higher losses.

'Samples taken every day for a fortnight averaged between 0.01 per cent and 0.2 per cent of tin, the higher limit being reached only in one sample'.

Clark (1904) makes the following comments on tin recovery:

'We only know what is won; but what is lost is seldom if ever accurately known - though often put down, knowingly or unknowingly'. He goes on to say: 'It is often stated that since the tailings from any particular mill cannot be profitably worked that it shows there was no loss in the original treatment'. No figures for Mt Bischoff tailing losses are given.

Fawns (1905) merely repeats the information given by Kayser and Provis in 1895.

Herman (1914) states: 'The writer, during the course of some spasmodic testing in 1904, found fine tin, equivalent to 6.25% of the whole output of the mill, going to waste from the overflow of the large settling tanks in the slime sheds alone, or nearly 7 tons of oxide per month. The conditions of running were quite normal, and this loss was evidently continuous. At the time no statements of extraction were issued'.

Millen in the Half-Yearly Report dated 31 December 1907 makes the following comments: 'Our present plant is in a deplorable state, both as

regards efficiency and the condition of the buildings, machinery, and other things incidental to a mill. Extensive work has been accomplished in experiments and tests on our concentrating appliances, with the result that the tables, and buddles especially, are found to have a very low efficiency....

'It will be noticed that very few assays have been given in the body of this report, as the time since my appointment has been so short, and there was no assay plant on the mine.

'It will also be noticed that there is no statement made respecting the assay of the material treated by our mills, as under the conditions prevailing it would, to a very large extent, be mere guess work'.

Capital Expenditure

Capital expenditure of about £6,000 per annum was maintained until 1892 when it was reduced to about £3,000 per annum at which rate it continued until 1907.

In addition to the above there was considerable expenditure on water storage, collection etc., since water was the power source. These outlays are shown in Figure 20.

The total cost of the North Valley Mill was given as £18,000 hence all the expenditure between January 1891 and June 1892 must have been on this project, water projects apparently were stopped to make the money available.

By any standard the money outlayed on the mine was very small compared with the returns being won.

Contemporary Comments on Operating Practice

Clark (1904) reported on Mt Bischoff giving the following criticisms and explanations:

'While there are many points which critics have been severe upon with regard to this plant, such criticisms have no weight when considered apart from local conditions. The weight of the stamps and the fact that there are plants of the same type as the main one, erected to deal with the sand which escapes from the first, do not seem to accord with our modern notions. The explanation given by the manager puts the matter in a very different light. The stampers fully weighted are really 8 cwt, and this was certainly a full weight stamp twenty years ago. Owing to the shortness of water in the summer half shoes only are cast, so that the whole plant can be kept going. A great deal of the material is soft and is discharged by mere agitation. With regard to retreatment, it is explained that if the top sheds were doing all the work there would be no necessity to retreat the tailings, but owing to the water becoming thick, due to the presence of clayey matter and oxide of iron, a portion of the finer tin ore would float away if such treatment were repeated at the upper sheds. The water is used as often as possible, but it was not considered advisable to have finality of treatment here, but deemed a better plan to allow this escaping material to accumulate and retreat it with clean water at the lower sheds. The nature of the ore also has to be carefully considered, probably the bulk of the material does not need much crushing at all, as delivered in the trucks it is a clayey or sandy material.

'While it is an easy matter to criticise severely and adversely, it is doubtful whether any other class of machinery would have dealt so successfully with the large body of ore, over six million tons, treated, and over

sixty-one thousand tons of oxide won. So far as Australasian practice is concerned, there is no doubt that the Bischoff mill was far in advance of anything of its kind at the time of its erection. Since that time minor details have been perfected, and men who have grown up on the works have become so accustomed to each machine that the best possible work is got out of it'.

Herman (1914) comments on the 1904 practice:

'The crushdirt contains so much clay and sand that after passing over 2-inch grizzlies at the mine, the rather unsatisfactory "screened" product of oversize rock and adhering clay was then easily (if unnecessarily expensively) dealt with by two 14-inch jaw crushers worked by separate portable steam engines manned by a driver each.

'Notwithstanding that a large plant (the "slime sheds") had been erected immediately below the 55-head mill to retreat all waste, it was found that the losses therefrom, as well as the escape of fine tin from sluicing at the Mount, warranted the erection of the "Ringtail" mill, on Falls Creek, about one mile below the main Waratah plant.

'Still another mill (the "Catch'em") was erected in 1888 lower down the river to settle and rework the "Ringtail" tailings'.

He comments that the North Valley mill 'apparently had very little ore to treat, and found difficulty in doing that little successfully'.

The Occurrence and Significance of Sulphides in the Ore

In 1879 an adit encountered stanniferous pyrites. In most places once the alluvial or detrital surface was sluiced away pyrite was found.

Pyrite meant difficulty in concentrating the cassiterite. The Mt Bischoff Co. attempted to exploit the pyritic North Valley lode in 1892-1894 but failed.

Yet after this experience Kayser and Provis (1895) can write, in discussing the ore occurrence that veins of tin ore occur but as most of them are highly pyritic and need roasting before they can be successfully dressed, they 'have received comparatively little attention; but in the future, when more readily available sources fail, they will no doubt become of greater importance'.

Clark (1904) states:

'The Queen lode will soon produce pyritic materials with tinstone, while the North Valley lode is unworked because the pyrites are there. From all this it is quite evident that so long as the soft, easily-worked formations last, the yields can be equalised, but when these are done, and that will not be very long, a much more difficult problem will be presented, and one which is of vast importance to the whole of Tasmania and of vital importance to Mount Bischoff. That is the treatment of hard rock carrying only a small percentage of tin and a cheap, effective treatment of tin stone carrying pyrites'.

Production

During the period between the year of maximum production (1885) and 1922, when the mine virtually ceased production for the year, there is an

average decline in tin production of 48 tons per year (fig. 10).

During Phase 3 (1887-1907) the recovered grade of ore treated steadily declined from 2.7% Sn to 0.8% Sn in 1906 and to 0.4% Sn in 1907. The tonnage milled rose during this period until 1900, after which it remained more or less steady until 1907.

The average grade of concentrate produced was near 70% Sn during the earliest years of production but from 1885 it began to decline until it reached a low of 65.8% Sn in 1892. After that the grade gradually rose to a peak of 71.3% in 1901 when it began to fall again reaching 67.2% Sn in 1907 but the declining grade continued beyond that.

In Figure 13 the amount of concentrate from the various slime tin recovery plants is shown together with the total amount of concentrate made. As there is no record of the grades of these concentrates the tin actually recovered is unknown but all show the decline in production characteristic of the whole operation. The proportion of total concentrate made by the three slime plants fell from 21% in 1890 to 16% in 1899 to 15% in 1904.

From 1896 costs of production are given in terms of per ton of ore milled. These costs are the total money spent for the period divided by the tons treated. There were no allowances or distribution of costs over other periods even if they may have been applicable. This results in rather erratic fluctuations but when applied to the tin produced the steady decline in production shows as a steady rise in the cost of producing a ton of tin (see fig. 11).

The Crisis

In the Half-Yearly Report for the period ended 31 December 1906, Kayser, the Mine Manager, begins:

'This is the first time during nearly thirty-two years' service that I cannot lay before you a report pleasing to all concerned and satisfactory to myself. Since the Brown Face has been worked out, and the sudden and unexpected collapse of No. 3 bench, unforeseen difficulties have presented themselves, which will in the near future compel us to face a still further reduced output. Several works are under consideration, and surveys prepared, to start work as soon as the electric scheme is finished and an air-compressor can be supplied, and it is to be hoped that some of the operations will turn out satisfactory and enable us to recover some lost ground; but since the main deposit of the mine has failed and nothing promising has been met with in the deep ground, it is doubtful if the old prosperous times will return'.

The Directors' Report issued at the same time says of the mine and its management:

'The Mine

'Development work at the mine during the past half year has been unsatisfactory, and has given your Board considerable anxiety. Although the reports of the General Mine Manager for some considerable time past have indicated a gradual lowering in value of the ore bearing material, they were not prepared for the sudden falling off in quality, which reduced the output from 100 tons to 60 tons per month, from October last. This was due to a simultaneous change in value in three of the more important faces, and until there is an improvement, for which there is reason to hope, it is advisable for the proper working of the mine while the present quantity of ore material

is treated, to limit the monthly output to the amount now obtained. Meanwhile, further prospecting by the diamond drill, by surface workings on the top of the Mount, and by alluvial work in the North Valley, is being prosecuted vigorously and with fair success. With the near completion of the electric power installation, your Directors expect to be able to reduce the cost of treatment slightly, and with that end in view, it is proposed to get an independent report from the best authority available, to supplement the efforts of your General Mine Manager in effecting an improvement in the economical working of the mine, and in bringing about wherever practicable a saving in the cost of handling and treating the ore material.

'Mine Management

'The Assistant Mine Manager, Mr Herman, retires at the termination of his agreement with the Company. Mr Kayser will reside permanently at the mine, and will be assisted by Mr J.D. Millen, who for some years past has been the Company's Metallurgist. An assay office will be established at the mine, and the closest investigation will be made as to the possibility of economically saving any losses arising during the process of concentration. Further advantage will also be taken of Mr Millen's special knowledge by installing a separate plant for the purpose of treating the pyritic ores, which have not hitherto been treated, and from which an addition to the output may reasonably be expected'.

Kayser resigned during the second half of 1907 and was succeeded by Millen.

Reference to Figure 11 will show that in the second half of 1906 the cost per ton milled reached 92 pence after having risen steeply for two years. While this cost fell dramatically in 1907 the actual cost of producing tin did not, in fact it continued to rise.

The Secretary for Mines in his Annual Report for 1906 states:

'Dividends have been declared by the Mount Bischoff Tin Mining Company, Registered, during the year for £63,000 bringing the total up to £2,088,000. The total quantity of tin ore produced by that mine to the end of December was 65,833 tons.

'Somewhat disappointing results have attended the development of the mine, and towards the end of the year it was found necessary to reduce the output by 40 per cent. Measures are being taken to secure the utmost economy in working, and an exploration policy is being pursued with vigour'.

Discussion

New Equipment and Innovations

During this period the pattern of processing established earlier was continued with the addition of some more items of equipment which were similiar to those already installed.

Multiple deck round tables, cement round table decks and better jig sieve rigidity were introduced. Probably the most radical change was the installation of hydraulic jet elevators which reduced the number of men employed and gave a more uniform feed even if that feed were much diluted with water. There is no reference to any metallurgical innovation until a Wilfley type table was installed in 1903.

Rich Ore

Although the 15-head battery treated a much richer feed early in the period and a somewhat richer feed at the end of the period the flowsheet was almost the same as that for the 60-head battery. The only difference in treatment could possibly be a slower feed rate to the 15-head mill but I can find no reference to this being the case.

Crushing

According to Herman, the screening before crushing was inefficient. The operation of the crushers as single portable units instead of in a fixed central crushing station would be costly in labour and moving time. The permanent location of one crusher at the Waratah mill from 1900 suggests that all crushers could have been located at one place.

Batteries

The distribution of the ore once it was delivered by rail to the mill was laborious and inefficient although by its nature mechanical feeding of the stamps was precluded.

The throughput per stamp was high but as Clark says 'a great deal of the material' required only agitation to separate the cassiterite from the clay and sand. Why then were stamps used on such ore?

In view of the friable nature of cassiterite any machine which allowed the possibility of size reduction should have been used only as a last resort. Log washers and wash trommels were known at this time but apparently were not tried. A reference to washing tests is made regarding North Valley.

If throughput could be maintained using half shoes in summer why were they not used throughout the year? Half shoes would have reduced the crushing of cassiterite.

A minimum of water was added to the battery box. If a large flow of water were used then the reasoning was no doubt that the lower specific gravity minerals would be removed leaving the high specific gravity cassiterite to be slimed. However using a low water flow would give a high viscosity pulp which would impede the settling out of cassiterite but it would also impede its flow out of the box.

When Clark (1904) can write'...probably the bulk of the material does not need much crushing at all...' the conclusion is that for most of the ore crushing in a battery was not only unnecessary but decidedly damaging to recovery.

Concentrating Machines

Most of the tin was recovered in the jigs: the slimes treatment yielded about a fifth or less of the total. Although Millen had only just taken over the management of the mine in 1907 he writes in the Half-Yearly Report dated January 1908: 'Extensive work has been accomplished in experiments and tests on our concentrating appliances, with the result that the tables, and buddles especially, are found to have a very low efficiency...'

For the first time in the mill's history that statement is based on results of tests from which the products were assayed. So much for Clark's comment that the men who have grown up with the machines got the best out of them.

That buddles were inefficient is confirmed by their replacement by Wilfley and Card tables soon after Millen's comments.

In view of criticism from Herman and Millen the performance figures given in the text should be taken as indicative only. The numerous vats and settlers would save some cassiterite but, judging from Herman's description, the periodic treatment of the sediments from these was most haphazard.

Contemporary thoughts on tin recovery are stated by Clark. At Mt Bischoff the coarse free cassiterite was probably efficiently recovered in jigs but the machines employed after them were inefficient and probably the recovery was very low.

Slime Recovery Plants

Clark no doubt quotes Kayser's explanation for the Ringtail plant. The water in the mill circuit had become so viscous with clay that it carried the cassiterite out of the plant. Such pulp was too dense to be retreated so it was run down the creek where it was diluted with the power water from the water wheels. This allowed the tin to settle out in the creek and to be caught later at Ringtail where it was retreated with clean water.

Surely some form of controlled washing of this pulp would have allowed controlled treatment of this pulp and kept the fine cassiterite in a circuit?

The Ringtail and Catch'em plants did best when there were sluice box tailings to treat but once these ceased it was merely the application of the same process over again, there was nothing else done to effect a better recovery hence the recovery at Ringtail is a measure of the main plant's inefficiency in recovering what should have been recoverable tin.

It seems significant that there were severe floods in 1894 and that the Ringtail plant produced very little that year. The supply launder from the mill to the Ringtail was the Waratah River. Did the flood scour out all the Ringtail feed for a year?

Water Wheels

Clark voices the criticism of the mill powered by water wheels but justifies the operation of such water wheels by their simplicity and the abundance of water; despite shutdowns every summer due to lack of water; Clark publishes this comment at the time when the Mine Manager (in Company correspondence regarding the electricity installation) was recommending the acceptance of a hydro-electric generator which would make better use of available water.

Maintenance

The half-yearly reports begin to mention repairs in the early 1890s. However it appears that repairs were never made until there was a breakdown. About the turn of the century the manager reports the necessity to repair an item but it is several reports later before mention is made of such an item being repaired. The worsening maintenance situation can be seen in Kayser's reports and Millen confirms the run-down state of the mill when he took over.

Sulphide Ore

Although sulphide ore was encountered quite early in the exploitation of Mt Bischoff such ore was put aside in some unused part of the mine. At North Valley where the ore body consisted largely of sulphides an attempt

was made in 1892-1894 to treat such ore but without success.

Even with this failure fresh in mind Kayser and Provis could dismiss sulphide ores as becoming of greater importance in the future. Nothing was done about sulphide ores until the soft easily worked material was utterly exhausted, then of sheer necessity the question had to be faced. Clark clearly states the corner into which the Mt Bischoff Company was headed so this situation must have been realised by those at the mine yet nothing was done even two or three years before the crisis.

Mill Efficiency and Tin Losses

When Clark, in 1904, says of current tin ore dressing, that only the tin that is recovered is accurately known, it was certainly applicable to the Mt Bischoff operations.

Herman (1914) gives the loss from the Slime Sheds (presumably fig. 12 , item 36) as 84 tons of cassiterite per year. In view of the inefficiency of the round tables other such losses would have occurred in fact if the Ring-tail at this time was making 25 tons of concentrate per year this would have been part of such losses. However as such losses would have been determined after some form of gravity concentration one could say any -15 µm cassiterite was not considered and the losses in the size range up to 75 µm would be significant.

One would expect that in the battery discharge, the recovery of cassiterite of size suitable for jigging was good but below this recovery was poor. The unknown is how much fine cassiterite was in the battery discharge.

Production

When one looks at the production record of Mt Bischoff (fig. 10) the decline in tin production during this period is most striking. This must have been apparent at the time to anyone who studied tin production figures.

However the Company reports make no reference to this although over the last four years of the period there appears to have been an effort to product a constant weight of concentrate in each half-yearly period.

The smelter metallurgical reports from 1904 mention test work (Half-yearly Report dated July 1905) '...seeing that the probability is that in the near future we may have to deal with foul ores as well as the comparatively clean ones now received.'

This together with the interest in costs from 1896 does indicate some interest in the position but not to the degree one would have expected.

Clark foresaw the crisis that was coming, Herman did also but neither the mine manager, the directors nor the shareholders apparently did, or if they did then they chose to dismiss it.

The following contemporary comment shows how Mt Bischoff's production was accepted by the Mines Department and probably also reflects the government and community attitude as well.

Registrar O'Neill, in the Secretary for Mines Report for 1890-91, states:

'The Mount Bischoff Company is turning out about the usual yield, and the North Valley is pushing on with the erection of machinery'.

Registrar Tegg, in the Secretary for Mines Report for 1893-94, states:

'The Mount Bischoff Tin Mining Company's mine continues its regular output, and from all appearances is likely to do so for years to come'.

The Secretary for Mines, in his report for 1901-1902 states:

'At the Mount Bischoff Mine the batteries have run as usual, producing 1,291 tons of tin ore'.

If one enquires further into the Mt Bischoff Company it will be seen that it was the Company policy to pay as dividends almost all the profits made as they were made. Thus dividends rose and fell with profits (see fig. 19); there was no policy of steady dividends nor was there any accumulation of reserves.

A study of Figure 19 will show that dividends and profits did not decline to the same extent as tin production, in fact throughout the period the half-yearly dividends were around £30,000 except for five years (1894-1898 inclusive) when the dividends were around £20,000 each half year. This dividend is still pretty good when considered in relation to the paid up capital of only £29,600.

Thus one may conclude that the dividend rate was as far as those concerned with the Company delved into its affairs.

The fluctuation in concentrate grade is not indicative of tight mill control. To an extent this decline a grade may have been cuased by a desire to show a larger apparent concentrate production as the company only recorded its mine production in terms of concentrate weight without reference to the grade of such concentrate. However such a practice would be to the detriment of the smelter and no doubt brought comment from there.

As the general level of production fell so did the proportion recovered in slime treatment. This may have been due to better recovery in the initial concentrating plant or more likely in view of its inefficiency to the fact that a constant proportion of the tin in the mill feed was not recovered. The higher apparent earlier recovery of the slime plant could be due to poorer mill operation then and to the treatment of sluice box tailings.

Review of Phase 3

This was a period during which the Mt Bischoff Company earned a vast amount of money, little of which was used to develop the mining and metallurgical operations.

A few mechanical improvements were made to the mill, maintenance was done of necessity and while Clark can say the plant was as good as any in Australasia when installed nothing was done to maintain that position.

Inefficiencies were everywhere, in crushing at many sites, in distributing the ore to the batteries, in the recovery of slime tin, in the use of water for power, and in the use of the Waratah river as a plant launder. The future of the mine after the easily won ore had been exhausted was not considered until such ore had been depleted to the extent that production could not be maintained. Although the Company had known since 1894 that it could not treat the sulphide ore which would comprise its future mill feed nothing was done to solve this at the mine, rather it appears that an attempt was made to pass on the problem to the smelter.

Whenever reference was made to the Mt Bischoff Company throughout most of this period such reference is made against the tons of concentrate produced and the dividends paid. Such a producer and profit maker could not be going far wrong!

The Board of Directors apparently decreed that expenditure should be minimal and with this the mine manager complied, relying on his past record to hold at bay the growing professional criticism in his last years of office.

Phase 4: 1907-1914

During this time experimental work aimed at recovering tin from low grade and pyritic feed was undertaken and greatly increased tonnages were milled.

Meeting the Crisis

The Company published, on 10 January 1907, a report by H. Herman who quit the position of Assistant Mine Manager about this time. In this report Herman reproduces a memorandum he wrote to the General Mine Manager on 19 March 1906.

This memorandum advocates the following changes:

(1) The erection of a battery near the mine on a site able to accommodate up to 100-head of stamps.

(2) The battery to be erected in stages of 20 heads at a time each 20-head unit being independent.

(3) One 20-head unit would be reserved for pyritic ore.

(4) The installation of a calciner to roast the pyritic concentrates.

(5) A single crushing plant.

(6) The erection of an assay office primarily for grade control in mining but also to allow some control of milling as well.

An independent report from J.B. Lewis, General Manager of the Anchor Mine at Lottah, was also obtained by the Board and circulated to shareholders early in 1907.

When J.D. Millen became Mine Manager in the later half of 1907 he wrote in the Half-Yearly Report dated January 1908:

'The question of milling and concentration is one which needs serious attention, and will have to be faced in the near future. Our present plant is in a deplorable state, both as regards efficiency and the condition of the buildings, machinery, and other things incidental to a mill'.

He instituted a regular mill sampling programme, he advocated the installation of the latest equipment after a two year testing programme to ascertain requirements, and a new mill site near the mine. He began building an assay office.

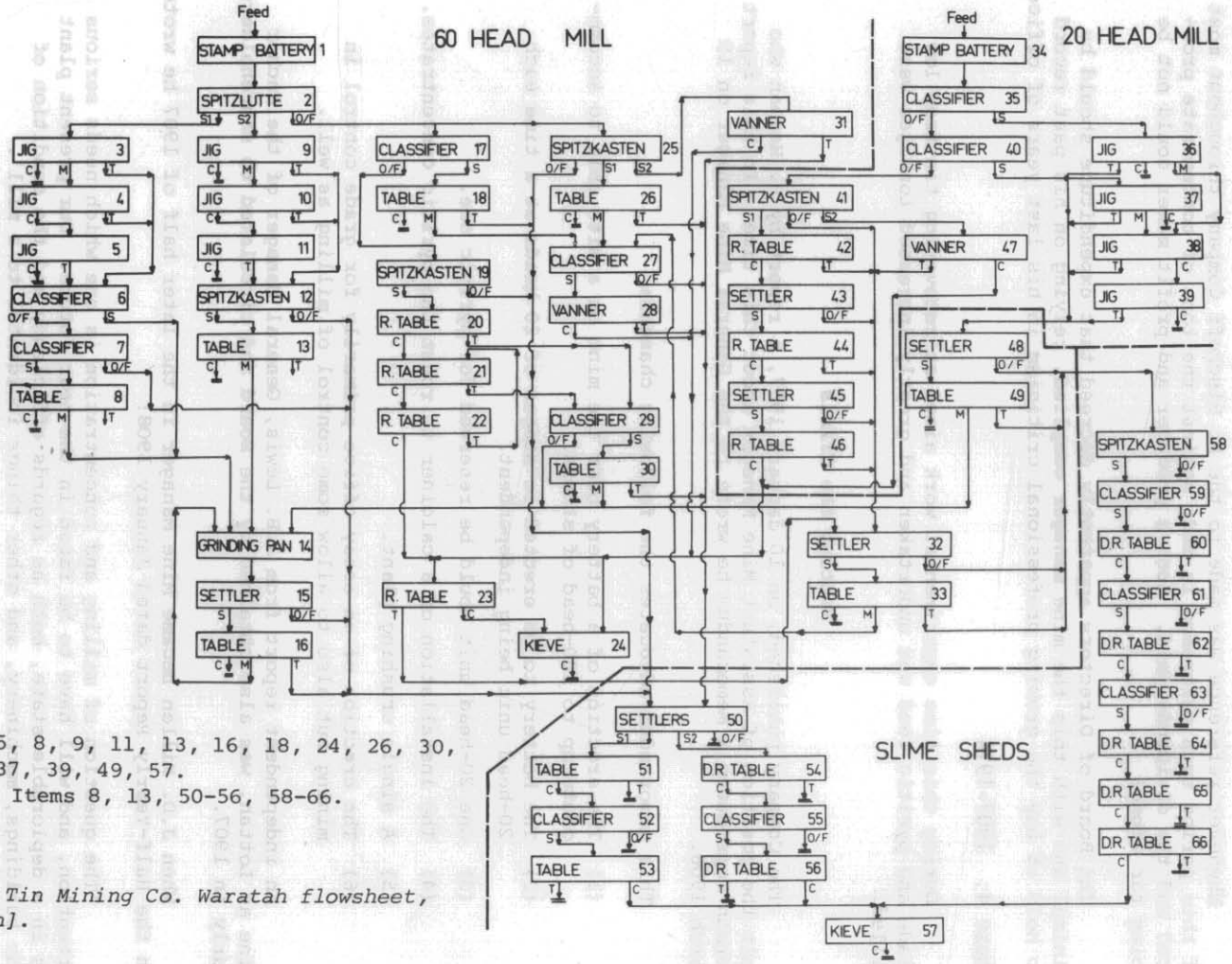
As a result of the sampling campaign and the assaying of such samples alterations were made to the mill.

Accompanying these changes there were considerations of how best capital could be employed to improve earnings.

60 HEAD MILL:
Items 1-33, 48, 49.

20 HEAD MILL:
Items 34-47.

SLIME SHEDS:
Items 48-66.



FINAL PRODUCTS:

Concentrates: Items 3, 5, 8, 9, 11, 13, 16, 18, 24, 26, 30, 32, 36, 37, 39, 49, 57.

Tailings and Overflows: Items 8, 13, 50-56, 58-66.

Figure 16. *Mt Bischoff Tin Mining Co. Waratah flowsheet, 1910 [Millen].*

Notes on Figure 16. *Waratah Concentrator Flowsheet, 1910*

Item No.	*Description*
1	Stamp batteries: 60 heads, wooden frames. Weight of stamps (lb) made up as follows:

Part	*Weight*
Stem	250
Tappet	56
Head	120
Shoe	136
Total	562

Screens discharge area 3 ft^2 with bottom of screen level with top of dies; 12 # woven steel wire 20 s.w.g. Speed 88 drops per minute, drop 8 in; dies were rectangular in plan (7.5 x 8.75 in) and 4 in high. Dies were replaced when height was reduced to 1.5 in.

2	Spitzlutte, two compartment, six were installed.
3	Jig, Hartz type, two compartment: screens 32 in long by 18 in wide; ragging, cassiterite. Speed 200 strokes per minute, length 0.3125 in. Six jigs were installed.
4	Jig (see Item 3) One jig was installed
5	Jig (see Item 3) One jig was installed
6	Classifier (probably a spitzlutte)
7	Classifier (probably a spitzlutte)
8	Table, Card, Model B, 258 strokes per minute, slope 2.75°.
9	Jig, Hartz type, two compartment: screens 32 in long by 18 in wide; ragging, cassiterite. Speed, 250 strokes per minute, length 0.1875 in. Six jigs were installed.
10	Jig (see Item 9) One jig was installed.
11	Jig (see Item 9) One jig was installed.
12	Spitzkasten
13	Table, two tables installed: one Card, Model B, 258 strokes per minute, slope 2.75°; one Wilfley, Model No. 5, 240 strokes per minute, slope 2.6°.
14	Grinding pan, Forward-Down 5 ft diameter, 50 rpm, 18 shoes and 18 dies.
15	Spitzkasten
16	Table, Card, Model B, 258 strokes per minute, slope 2.75°.
17	Spitzlutte, two were installed.
18	Table, two were installed: one Card (see Item 8); one Wilfley, Model No. 5, 243 strokes per minute, slope 2.6°
19	Spitzkasten, two were installed.
20	Round table, convex: diameter 16 ft, slope 5.5°, speed 0.5 rpm; four tables were installed.
21	Round table (see Item 20)
22	Round table (see Item 20)
23	Round table (see Item 20)
24	Kieve
25	Spitzkasten
26	Table, Wilfley Model No. 5, 243 strokes per minute, slope 2.6°.
27	Spitzlutte
28	Vanner, type Sperry slimer: convex vanning table 10 ft in diameter, speed 200 throws per minute of 0.3 in, 0.8 rpm, deck slope 5.5°.
29	Spitzlutte
30	Table, Card (see Item 8)
31	Vanner, Double Luhrig: slope 4.5°; speed - belt 6 ft per minute, 150 bumps per minute.
32	Settler
33	Table, Wilfley (see Item 26)
34	Stamp battery, 20 head (see Item 1)
35	Spitzlutte, 4 were installed.
36	Jig (see Item 3). Three jigs were installed.
37	Jig (see Item 3)
38	Jig (see Item 3)
40	Spitzlutte
41	Spitzkasten
42	Round table (see Item 20)
43	Round table (see Item 20)
44	Round table (see Item 20)
45	Settler
46	Round table (see Item 20)
47	Vanner, type Buss Slimer
48	Settler
49	Table, Wilfley (see Item 26)
50	Settlers
51	Table, Card (see Item 8)
52	Classifier
53	Table Card (see Item 8)
54	Round table, double deck (see Item 20)
55	Classifier
56	Round Table, double deck, (see Item 20)
57	Kieve
58	Spitzkasten
59	Classifier
60	Round table, double deck (see Item 20)
61	Classifier
62	Round table, double deck (see Item 20)
63	Classifier
64	Round table, double deck (see Item 20)
65	Round table, double deck (see Item 20)
66	Round table, double deck (see Item 20)

Plant Changes

During 1908 and 1909 changes were made to overcome the shortcomings revealed by the systematic survey of milling operations.

A single central crushing station equipped with jaw and gyratory crushers was built at the mine.

At the batteries modifications were made to increase the capacity by twenty per cent and to produce less slime.

The Chilian mills were replaced by the latest type of grinding pans.

The inefficient buddles were replaced by Card and Wilfley tables. Sperry and Luhrig vanners were installed and their performance compared with the existing round tables.

The 15-head battery was isolated from the main mill circuit so that it could be used to treat pyritic ores, the pyrite concentrates from which were roasted in a Merton furnace. The calcine was retabled to recover the cassiterite.

A 10-head experimental battery was built so that new methods of treatment could be tested and new ores could be investigated.

Automatic samplers were installed to sample all mill tailings.

Further Proposals

Millen proposed that the mill capacity be enlarged to a rate compatible with the substantial mine ore reserves he hoped to prove. Such enlarged capacity would reduce overhead costs and thus lower the cost of production.

He also proposed the installation of a washing stage ahead of the crushers to separate the clay from the ore.

Interest in a completely new mill site at the mine gave way to consideration of an aerial ropeway instead of the railway to convey ore from the mine to the mill. The ropeway was later installed.

60-Head Mill, 1910

This mill still retained the essentials of the earlier (1904) mill as can be seen in the flowsheet (fig. 16).

The stamp batteries were modified by:

(1) doubling the weight of the shoes which were made from forged steel instead of cast iron,

(2) increasing the drop and speed, and

(3) using coarser battery screens.

Initial concentration was still by coarse and fine jigs but Card and Wilfley tables replaced the buddles for treating jig tailings and the finer classifier products.

Although round tables were retained for slime concentration Sperry, Buss and Luhrig vanners were being investigated as alternatives.

The Luhrig vanners were found superior to round tables but apparently not to the same degree as the tables that displaced the buddles.

In the Slime Sheds where the mill tailings were retreated, oscillating tables handled the coarse material while round tables continued to treat the slime.

Although Millen (1910) gives typical sizing analyses for various mill products it is difficult to identify such on the flowsheet (fig. 16). The sizings plotted in Figure 21 have been identified as closely as possible with items on the flowsheet.

Millen (1910) gives the following capacities and data for the following machines:

Machine	*Capacity tons/day*	*Solids %*	*Extraction %*	*Remarks*
Stamp	7.5	-	-	Through 12# screen
Grinding pan	17	-	-	
Jig, coarse feed	10	8	60	
Jig, fine feed	13	12	60	
Table, Card, coarse feed	15	27	85	
Table, Card, fine feed	25	24	67	
Table, Wilfley, fine feed	23.5	21	65	
Vanner	18	-	-	Feed all -130 µm and assaying 0.56% Sn, tailing 0.03% Sn

10-Head Experimental Mill, 1910

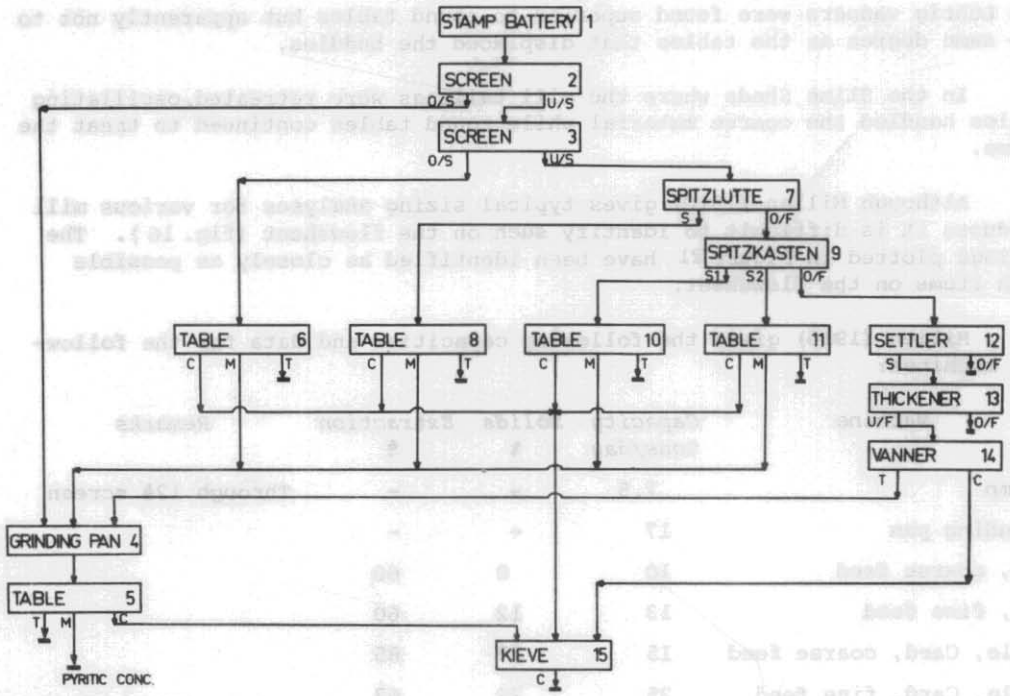
This mill differed considerably from the main mill of the time, (fig. 16), as can be seen in the flowsheet (fig. 17). Although the stamps were initially fed by Challenge feeders the clayey nature of the Mt Bischoff ores precluded satisfactory automatic feeding hence hand feeding had to be used after a few months of operation.

The stamps were heavier, (1,050 lb) and ran faster than those in the main mill. In order to get a larger capacity the discharge screens were increased to 8#.

Screens were used instead of hydraulic classification to size the battery discharge. This alteration was made to conform with the accepted thinking of the day.

The various sizing analyses given by Millen for this battery are plotted in Figure 21. This shows an extremely fine battery discharge for the coarse discharge screen fitted. The screen oversize products, S20/S and T6F/D, were considerably finer than their nominal sizes of 20# and 30# (640 um and 420 µm).

The battery capacity was 100 tons per day. Calculations made from the data given yielded the following flow rates, tin distribution and sizing of spitzlutte spigot product (C7O/F) plotted on Figure 21.



FINAL PRODUCTS:

Concentrate, tin: Item 15. *Concentrate, pyritic:* Item 5.
Residue: Items 5, 6, 8, 10, 11. *Overflows:* Items 12, 13.

Item No. *Description*

1 Stamp battery: 10 heads, wooden frames. Weight of stamps made up as follows:

Part	*Weight (lb)*	*Part*	*Weight (lb)*
Stem	500	Shoe	195
Tappet	125		
Head	230	Total	1,050

Screens discharge area 4 ft^2; woven steel wire 18 s.w.g. Speed 90 drops per minute, drop 8 in.

2 Screens, revolving King type, octagonal cross-section; speed 8 rpm. Size of cloth 13 ft x 3 ft wide, 20 #

3 Screen as for Item 2 but 30 # cloth.

4 Grinding pan, Forward-Down, 5 ft diameter.

5 Table, Card.

6 Table, Card.

7 Spitzlutte, conical.

8 Table, Card.

9 Spitzkasten.

10 Table, Card.

11 Table, Card

12 Settler

13 Thickener, two were installed.

14 Vanner, double Luhrig.

15 Kieve

Figure 17. *Mt Bischoff Tin Mining Co., 10-head experimental mill, flowsheet 1910 [Herman 1914]*

Product	*Flow (solids) tons/day*	*Assay % Sn*	*Tin Distribution %*	*Remarks*
F/D*	100	0.45	100	Mill feed
S2O/S	5	0.26	3	
S3O/S	6	0.59	8	T6F/D
C7S	16	0.65	23	T8F/D
C7O/S	73	0.41	66	

*refer to flowsheet (fig. 17) and graph (fig. 21).

50-Head Mill

Based on test work done by the 10-head experimental mill a new mill was built between 1910 and 1917. Units of ten or twenty heads were commissioned as erected. Heavy stamps (1,050 lb) were used because the lighter (560 lb) stamps of the old mill were unable to cope with the harder ore now being mined.

The flowsheet (fig. 18) shows the differences between this mill and its forebears. The flowsheet shows the simple logical approach of the experimental mill but it also shows the reversion to hydraulic classification and the use of jigs. Card tables and Buss slimers (vanners) were installed to recover the tin after the jigs. Round tables were omitted. Jig tailings were screened prior to grinding.

Herman gives as the reason for the switch back to spitzlutte from screens for primary classification the fact that the spitzlutte gave a richer jig feed. He says that from a feed assaying 0.5% Sn the spitzlutte gave a spigot product sizing the same as the 20# screen but assaying 0.9% Sn whereas the screen oversize assayed 0.3% Sn.

From discussions with Macartney, the mill superintendent, Herman (1914) learnt that Macartney felt there was a need for a small jig to rough out the free cassiterite in the spitzlutte spigot product.

Output and Efficiency

The throughput of all mills was raised from 110,000 tons in 1907 to 200,000 tons by 1909 and remained above this until 1914. The tin produced between 1908 and 1913 averaged 700 tons annually. The recovered tin grade was around 0.3% Sn.

The mill efficiency had been so much increased by the measures taken by Millen that there was no longer any need for the Ringtail plant which was shut down.

From the few figures quoted in references it would appear the mill feed averaged about 0.5% Sn from which the tin recovery was about 50%.

General Mine Operations

A water shortage caused the cessation of milling for one month early in 1910. A new reservoir was built late in 1910. Consideration was given in 1913 to an alternative source of power to augment the hydro-electric supply in times of low rainfall.

Progress on the installation of equipment, particularly the new heavy

stamps was slow throughout the period, due to lack of skilled labour which was apparently being used on the higher priority work (see fig. 20).

The mine operations were closed down from July to September, 1914 due to a heavy fall in the market price of tin due, the Directors said, to the war which began on 4 August 1914. This shutdown caused the miners 'a good deal of distress' (Half-Yearly Report dated January 1915). They approached the Government and after lengthy negotiations with the Company the mine re-opened on a cooperative basis on 15 September 1914. This cooperative basis was terminated on 15 March 1915 after which the men were employed under six monthly agreements which made provision for a bonus payable on the tonnage milled.

Experimental Work

The laboratory note book, which was commenced on 24 October 1913 by A.D. Mackay was given to the Department in 1963. This book covers test work done on the new battery up to possibly 1919.

The batteries were tested with punched plate and woven wire (5, 6, 8 and 10#) discharge screens. The resultant sizing analyses and tin distributions do not follow the logical sequence one would expect but this could be due to the fact that the 6# (and no doubt the 5# also) screen was only fitted to the battery for a few minutes because the coarse discharge caused trouble in the mill circuit.

The Card tables were studied with respect to ribbed glass decks, the height of the riffles above a wooden deck, the angle of the riffles, thickening of the table feed and the feed rate. The glass decks appear to have behaved similarly to the wooden decks but were said to lack stability. On one occasion the glass deck collapsed. Wooden riffles greater than 0.125 in high did not work. Skew riffles allowed the maximum use of the dressing surface. Thickened feeds, 28% solids instead of 14% on fine material were beneficial. The best feed rate for a Card table was found to be 12 tons per day with an extraction rate of 80% and a concentration ratio of 20.

Comparative tests between Buss vanners and double deck round tables showed:

'(1) Double deck rotary tables are better than single deck tables.

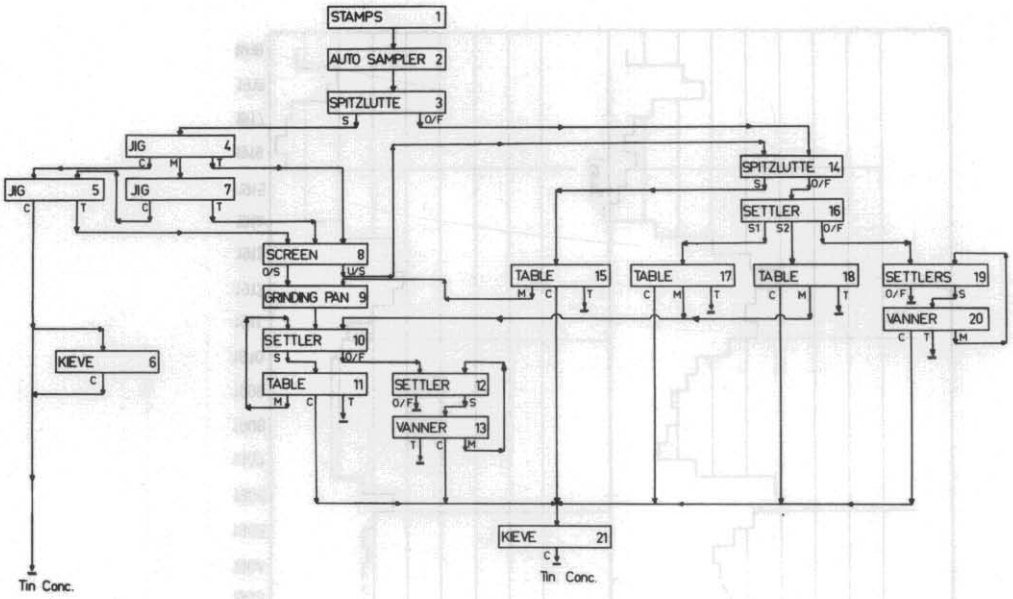
(2) Buss vanners have a much higher concentration ratio than rotary tables with a similar extraction'.

Two sets of figures obtained in these tests were:

	Round Tables	*Buss Vanner*
Feed (tons per day)	5.1	3.7
Feed (% solids)	9	10
Feed assay (% Sn)	0.13	0.39
Tailing assay (% Sn)	0.10	0.15
Concentrate assay (% Sn)	0.30	30.0

Costs

The cost of producing a ton of ore fell sharply in 1908 and remained at below 50 pence per ton until 1911 when it began to rise. This low cost per ton was the result of the larger throughput.



FINAL PRODUCTS:

Concentrates: Items 5 or 6, 21.
Residues: Items 11, 13, 15, 17, 18, 20.
Overflows: Items 12, 19.

No details of the mill machinery are available. The plant consisted of fifty stamps in batteries of twenty, and, presumably, the original ten head experimental mill.

For each unit of twenty stamps the installation was:

Item No.	*Description*
1	Battery, 1,050 lb stamps similar to those in the experimental mill. 20 head were installed.
3,14	Hydraulic classifier or spitzlutte, two were installed.
4,5,7	Jig, two compartment, two installed in Item 4 others one.
8	Screen, neither the type nor mesh stated.
9	Grinding pan, probably Forward-Down type used in experimental mill.
11,15	Table, Card
17,18	Table, Card
13,20	Vanner, type Buss Slimer.
16	Settler, probably a spitzkasten.

Figure 18. *Mt Bischoff Tin Mining Co., New Mill flowsheet, 1913*
[Herman 1914]

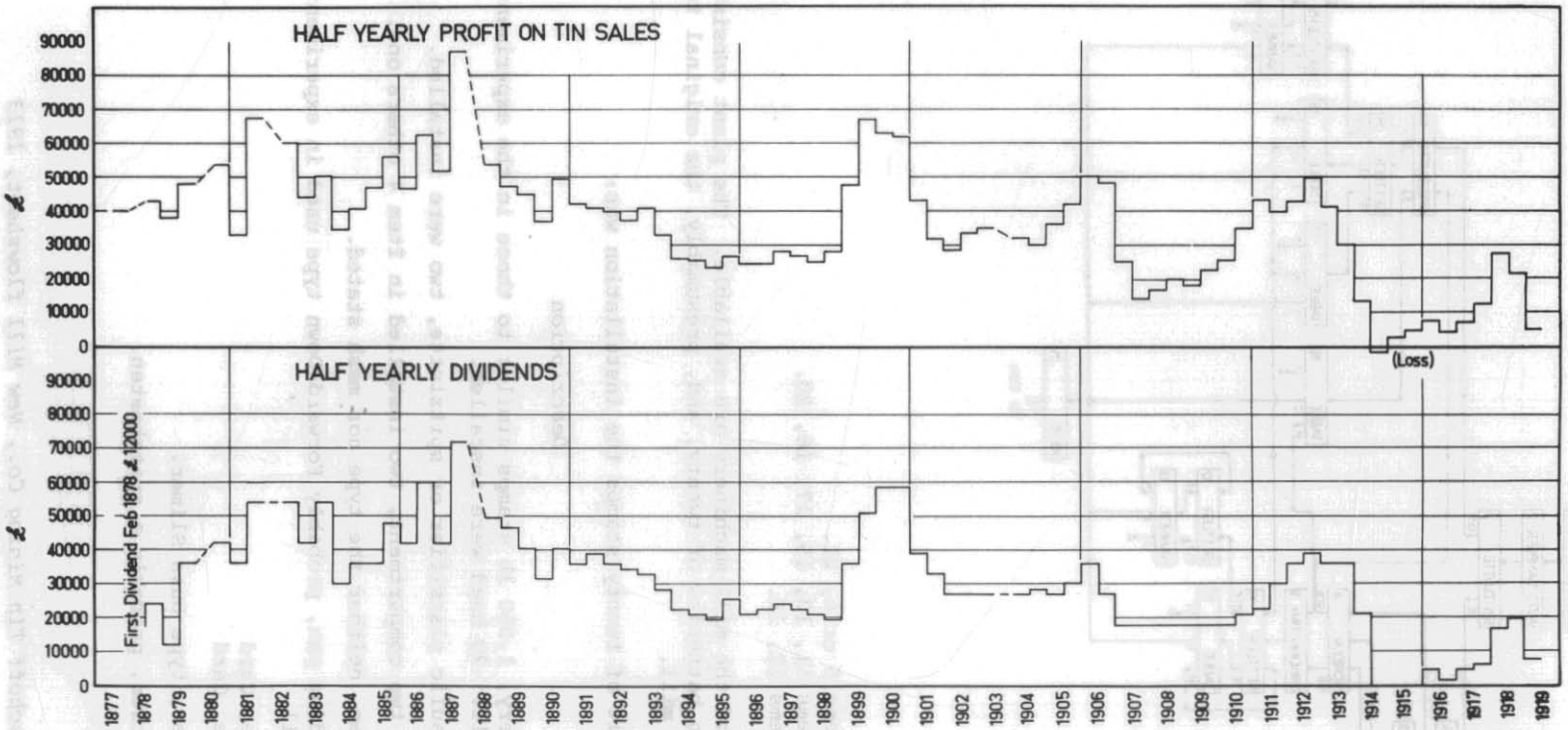


Figure 19. *Half-yearly profits and dividends.*

Figure 20. *Mt Bischoff Tin Mining Co. Capital expenditure at mine and mill.*

However the cost of producing a ton of tin metal after being held steady at about £60 between 1908 and 1911 began to climb after that. In fact mid 1911 marks a turning point in the cost of tin metal produced (see fig. 11). From 1896 until 1911 the cost of producing tin had risen at an average annual rate of £3.2.0. After 1911 that rise increased sharply to an average of £11.10.0 per annum.

Capital

In order to meet the crisis capital expenditure was increased sharply in 1907 although most of the immediate increase was for a tramway to bring ore from the Wheal Bischoff area. Once the Waratah mill had been re-equipped and the experimental battery erected capital expenditure fell sharply only to rise again a year later. It would appear the aerial ropeways were expensive to install. The construction of the additional heavy batteries was suspended when the tin price fell in 1914.

The pattern of expenditure is shown in Figure 11.

Discussion

The crises were met by adopting some of the proposals made by Herman in 1906, namely, to increase efficiency by a logical approach to materials handling.

The application of a systematic sampling programme in the mill revealed the inefficient machines, *e.g.* the buddles, which were quickly replaced. The difficulty in finding a machine that would satisfactorily recover slime tin was no doubt responsible for the delay in replacing the round tables which from Mackay's figures had a very poor performance.

The isolation of part of the mill for the treatment of pyritic ore and the building of a roaster meant that stanniferous material hitherto dumped could be milled. The experimental mill allowed full scale testing of ores and equipment before designing the new plant which had to be built to replace the old, poorly maintained and now inadequate mill.

There is only one mention of washing the ore before processing. In late 1907 and early 1908 about 2,000 tons of ore were sent for hydraulicking which is interpreted as an experiment to remove the clay. As washing seems a logical stage that would have eliminated many milling problems one wonders what reason precluded its adoption. Probably this was lack of water on Mt Bischoff, since one gains the impression that pumping water was not considered practicable.

The light stamps of the old plant could not handle the harder ore: this meant that heavier stamps were needed. In order to maintain production and because the new flowsheet would be quite different from the old, a new mill site was necessary. The experimental mill, which was the prototype, was built near the existing mill. The new mill was not located at the mine and in order to solve the transport problem between mine and mill aerial ropeways were built to replace the railway. This solution would probably have involved a lower capital outlay.

Millen, when he became mine manager, wrote of his ambition to publish assays on mill performance. However no assays were ever published but Millen did report the ore grade in terms of the pounds of concentrate recovered per ton of ore milled. There are no reports of milling operations with assays, sizing etc., published by the Company. Millen's series of

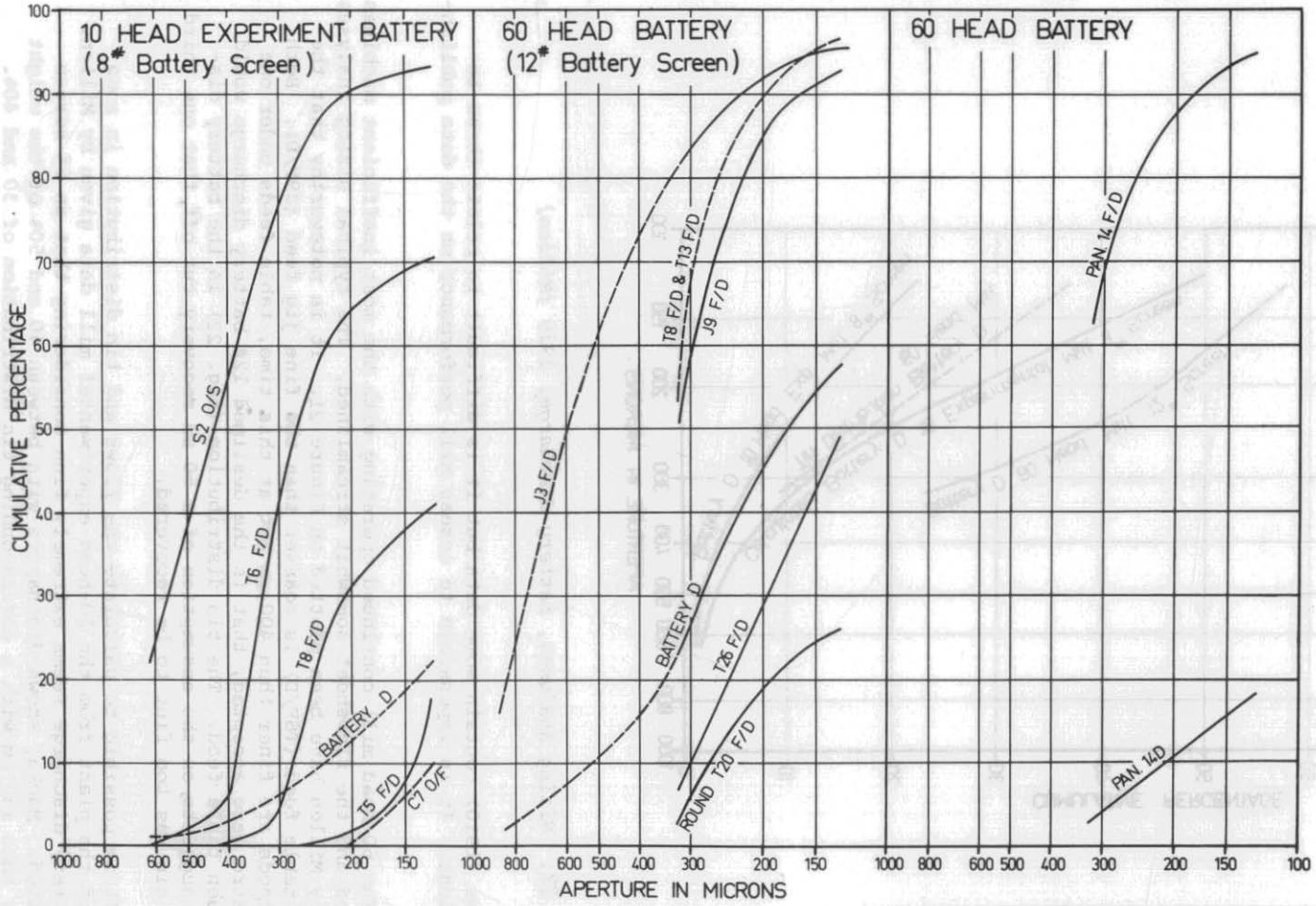


Figure 21. *Mt Bischoff Tin Mining Co. Sizing analyses, 1910.*

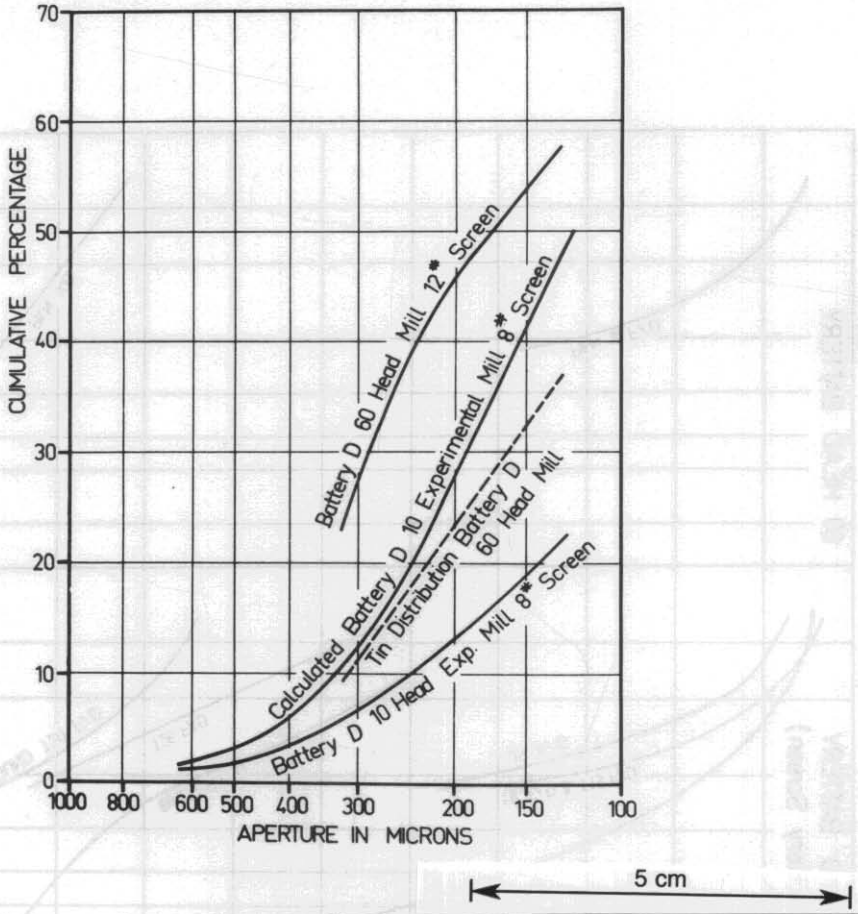


Figure 22. *Sizing analyses, battery discharge, 1910 [Millen]*

articles (1910) contain some data but it is difficult to relate these to operations. It is impossible to assess mill performance on the data published.

The 60-head mill continued operating with the most inefficient machines replaced and the flowsheet somewhat streamlined. The typical sizing analyses cited by Millen have been plotted on Figure 21. It is noteworthy that the coarse table feed (T8F/D) is coarser than the fine jig feed (J9F/D). Both these feeds are finer than 400 µm but at this time, table feeds under one millimetre were accepted, that is the deslimed 12# battery discharge would have been table feed. The tin distribution (fig. 22) in the battery discharge suggests, on the assumption of a 50 µm recovery cut off, that one third of the tin was too fine to be recovered.

It is possible to calculate the flows and tin distribution in some parts of the plant from the 10-head experimental mill data given by Millen. The battery discharge seems extremely fine considering that an 8# screen was fitted. Mackay shows discharges with between 40 and 50% of the weight coarser than 130 µm with a corresponding tin distribution of 30 and 40%. By assuming a recovery cut off at 50 µm it would appear that the recoverable tin was less than in the old mill.

Assuming Millen's feed sizing is wrong; then by making other assumptions and assuming a 130 µm content of 50% it is possible to calculate a

battery discharge sizing analysis (fig. 22) but the conclusion still is that about half the tin would be below 50 µm. However the 10-head mill must have worked better than the 60-head mill but not that much better otherwise one would have expected a much more rapid installation of the 50-head mill than occurred. Therefore one must conclude that although the heavier stamps overground cassiterite the recovery equipment was so much more efficient in the 10-head mill that a better yield of tin was obtained.

Poor screening efficiency in the 10-head mill was probably the cause of the reversion to a spitzlutte in the new mill. What was needed was a more efficient type of screen.

There are practically no available data on the operation of the 50-head mill.

The throughput of ore rose dramatically between 1907 and 1909 but how was this achieved? The alterations to the stamps were stated to have raised the capacity 20% so that the old mill output would have risen from 110,000 tons to 130,000 tons per annum. The new 10-head battery had a capacity of about 30,000 tons per annum. It would thus appear that the remaining 40,000 tons capacity was obtained by the more efficient use of the equipment. The increase in 1912 would be due to another 10-head unit coming on stream in the new mill.

Without information it is difficult to estimate recovery but it cannot have been high. Millen's failure to give assays may have been due to what would have been revealed had they been given. From the information available from the battery discharge sizings about two thirds of the tin appears of recoverable size hence a recovery of half using the equipment in the old mill seems reasonable. In the period considered this plant had a far larger throughput than the newer batteries hence its recovery would be dominant in assessing the mill as a whole.

Water shortage again caused loss of production and the consideration of alternatives to overcome such shortages.

The shutdown in 1914, when operations incurred a loss clearly showed Company policy. The loss was £1,838 for the half-year but at that time Company reserves totalled £81,931. An agreement was made following negotiations with employees and the mine ran profitably again.

An additional reason for the slow progress in building the new mill could be that the money was not provided for it. From Figure 20 it will be seen that the aerial ropeways were being constructed at this time and must have absorbed most of the capital available.

The experimental work during the early part of this phase exposed the inefficiency of the old mill and led to the introduction of new equipment; however the problems confronting any gravity tin plant, namely the concentration of slime tin presented problems that were not solved at Mt Bischoff and have not been solved even today. The building of an experimental mill was a step quite foreign to what had gone before at Mt Bischoff. In a way it was an expression of the realisation that only a greatly improved metallurgical performance could keep the Company going.

The cost of producing tin had been held at about £60 per ton from 1907 until 1911 but then the cost began to rise at a much faster rate than in the past. Such sharply rising costs without increased production adversely affected profitability, with the expectation of an eventual loss. Any down-

ward trend in the price of tin brought that time nearer.

Review of Phase 4

This was the first period in which technology played a part in the Mt Bischoff operations. The climate for such a radical change was created by the crisis of 1906 when the level of profitability could not be maintained.

The first fruits of this change were the changes made in the mill where new machines replaced inefficient ones, a roaster was built so that hitherto untreated ore could be handled, and an experimental mill was built. All these changes occurred in the first two years of the period. They were measures that had to be taken to keep the mine in production. They were measures that had been proposed to the Company before the crisis but ignored until it was imperative to do something. The low efficiency of the old management is well illustrated by the increased mill throughput that was apparently obtained merely by ensuring maximum utilisation of equipment.

However the longer term measures were not implemented. These measures (*e.g.* the location of the new mill at the mine to eliminate ore transportation) would have required a large capital outlay. To justify this expenditure sufficient ore reserves would have had to have been proved. In July 1908 the ore reserves were stated to be 2.5 million tons but no assay was given. Such a tonnage should have been sufficient to justify an outlay on a new mill although with hindsight the grade was probably not high enough as the reserves in 1917 were 1.3 million tons assaying 0.305% Sn.

After the initial expenditure on re-equipment capital expenditure fell sharply but the need to overcome the problem of transporting greatly increased tonnages of ore from mine to mill and the construction of new batteries caused unprecedented expenditure in 1911.

Even after the improvements to the mill and no doubt an improved tin recovery the mill recovery would not have been good. Although the recovered mill head grade was around 0.3% Sn this represents only about half of the tin fed to the mill. Thus one suspects that ore mined during this period was better than ore reserve grade: that is selective mining was practised, so that by 1917 the reserve would have been too low in grade to be mined as a whole.

Although a large tonnage was treated and a profit made with only a 0.3% Sn recovered head grade the production costs were rising with the inevitable result of unprofitable operation.

Phase 5: 1915-1927

Production continued at a decreased rate after 1914 despite difficulties caused by the Great War. However the whole operation was dependent on a high tin price hence a fall in price such as occurred in 1921 caused the virtual cessation of operations. However with better prices operations resumed on a scale commensurate with those prices.

Sources of Ore

The southern slopes of Mt Bischoff continued as the main source of ore but all round the Mount the slopes were scoured for alluvial and detrital material.

However when mining lode material, either from beneath the old surface

workings or from North Valley, sulphide ore was encountered. Finally the position had to be faced that the only ore left was sulphidic. In the Half-Yearly Report dated July 1919 Millen reported '...much laboratory research work is being conducted to obtain data for the improvement of the present methods of treating the admixed sulphides'.

From 1926 onwards the reports spoke optimistically of the North Valley alluvial deposits in the bed of the Waratah River.

Mill Alterations

In the mill Millen was confronted with the complex problem of balancing what was technically possible against what was economically possible. In the Half-Yearly Report dated January 1916 he stated the problem thus: 'The ore being low in grade does not permit of very elaborate appliances or refinements of treatment, and consideration has been given to attaining the maximum output per stamp and the economic limit of concentration'.

Experimental work and improvements covered the inclusion in the flow-sheet of a tube mill eighteen inches long for secondary grinding, tests with glass topped tables which were said to be efficient in separating cassiterite from pyrite, and work on slime treatment.

After several years under construction, a further twenty heads of heavy stamps were commissioned in 1917. When these started up the old light stamps were finally discarded after 35 years of service.

With the greatly increased sulphide content of the ore a large roaster was commissioned in 1921 and another in 1925 when in addition a new treatment plant consisting of a grinding pan and seven tables was built to treat the calcine.

Also in 1925 eight new tables were purchased to replace worn ones in the mill which now centred around a 40-head battery.

Treatment of Sulphide Ore

The experimental work in 1919 led to the new 100-foot long Edwards roaster being started up in 1921, pyritic concentrate having been stockpiled since 1919.

In its first nine months of operation this roaster treated 2,472 tons of pyritic concentrate which assayed 2.5% Sn to yield a calcine which contained about 55 tons of metallic tin. This roaster appears to have had capacity to treat about 3,500 tons of pyritic concentrate per annum.

However this large scale roasting liberated sulphur dioxide in sufficiently large volumes to cause protests from the town of Waratah. In 1922 water sprays were inserted into the roaster flues to reduce the sulphureous gases entering the atmosphere.

The Secretary for Mines, in his annual report for the year 1922 remarked that 'the average grade (recovery) has been 1.88 per cent Sn which makes it the lowest grade pyritic concentrate being profitably treated in the world today', and in 1924, when referring to the installation of the second roaster, 'the methods in use permit of pyrite with one-half of one per cent tin being profitably treated.'

The second roaster was started in 1925. This brought the capacity for

treating pyritic concentrates to about 8,000 tons per annum.

Aerial Ropeways

The aerial ropeway loading station in North Valley was destroyed by a bush fire in 1920. This caused the abandonment of North Valley operations and also the use of the aerial ropeways. Once again the railway became the sole ore carrier between mine and mill.

Some idea of the wartime quality of goods can be gained from this reported experience in 1917 when a new rope lasted only six months whereas the one it replaced had been in service six years.

Production

In the six years 1915 to 1920 the ore milled averaged about 100,000 tons per year with an average production of about 300 tons of metallic tin per annum. There was little ore produced in 1921 and even less in 1922. During these two years the roasting of stockpile pyritic concentrate yielded 300 tons of metallic tin. During the next five years ore production averaged about 70,000 tons per annum with an average yield of about 250 tons of metallic tin. The average grade of concentrate in the period 1917 to 1919 was very low, no doubt reflecting the difficulties being experienced with pyritic ore.

Once roasting began a grade of 66% Sn was maintained in the concentrates.

Recovered Grade

The recovered head grade showed some improvement during this phase of operations (see fig. 10). However the mill feed assay was never stated.

However after 1917 assays were given for positive ore reserves which should have been indicative of the mill feed assay. In June 1917 ore reserves were given as 1.3 million tons assaying 0.306% Sn. After the milling of 195,000 tons of ore the reserves were stated to be 1.1 million tons assaying 0.28% Sn in June 1919. This would indicate a mill feed of about 0.45% Sn over the two year period. With stated recovered head grades of around 0.25% Sn for 1919 and 1920 the overall recovery of tin would have been about half of what was fed to the mill.

These figures also indicate that the company was not mining reserve grade but was selectively mining the richer ore.

The reserve stated in December 1920 was 253,000 tons assaying 0.353% Sn: a revised estimate based on a more realistic grade. Substantially the same reserve figures were given in December 1922 but recovered grades given for the years 1925, 1926 and 1927 were 0.36, 0.38 and 0.37% Sn respectively which indicate much higher grade feeds than the 1922 reserve grade.

Financial Aspects

Only a few figures are available after June 1919 regarding expenditure, costs etc.

Costs continued to rise rapidly after 1914 (fig. 11). Millen had made the point that unit costs would be reduced when he advocated a greatly increased treatment rate. Now with decreased mill throughput the converse applied. Capital expenditure was mainly directed at completing the heavy

batteries (fig. 20). Profits were made (fig. 19) but as in earlier times these were immediately paid out as dividends.

In the 1920's there would have been considerable outlay on the roasters (the 1921 roaster cost £4,600), calcine treatment plant and plant renewals. From the few figures available profits and losses alternated during this time.

Discussion

The Company, after scouring the area for easily treated stanniferous detrital material, finally had to face the fact that unless it could handle a largely pyritic ore there was nothing else to mine.

Experimental work led to the installation of roasters to treat this ore. No mention has been found of flotation to remove the sulphides. Playford (1928) says that it had not been used on the Mt Bischoff field up to 1926. Perhaps in 1919 flotation would have still been too new even to have been included in the laboratory investigations at Mt Bischoff at that time.

The roasting of a pyritic concentrate had been practised at Mt Bischoff Extended for a number of years.

It is apparent that pyrite was causing great trouble in the mill, witness the average concentrate grades (fig. 10) which fell to 60% Sn in 1918. While the war may have been responsible for the delay in building a plant to cope with it one must also suspect that the Board withheld approval for such expenditure until the last minute.

The roasters, once installed, apparently ran efficiently and at low cost. The stockpile of pyritic concentrate allowed production of tin to continue even though mining virtually ceased during 1921 and 1922.

The overall mill recovery up to 1921 appears to have been about 50% which was in line with previous practice.

As this phase progressed the recovered grade improved. Whilst this may have been due to higher recoveries the fact that the Company policy was to mine the richest ore, not an average reserve grade, strongly suggests that to meet rising costs the head grade rose from between 0.4 to 0.5% Sn to something around 0.7% Sn. Such a policy would rapidly hasten the day when, despite a reasonable tin price, ore of sufficient grade for profitable operation could not be mined.

The three aerial ropeways were abandoned after only from six to eight years of operation. Although lower tonnages would be a factor their abandonment after a fire at a loading station there is a strong suggestion that they were not a success.

During this phase the Company's fortunes and operations were closely tied to the price of tin; a low price meant the cessation of mining, a high price could mean the return of a sizeable dividend, for example dividends in 1918 totalled £36,000. This dividend, which was the result of an isolated highly profitable year (see fig. 19), reveals the Board's attitude, for at that time the mine was beset by problems which were to require a large capital outlay to solve. Towards the end of this phase the Company was running a bank overdraft.

From 1926 onwards hopes were being pinned on the successful exploitation of the North Valley alluvial deposit which had been sampled in 1915.

The mine was now a marginal operation, very much at the mercy of the price of tin.

Capital had to be employed to provide facilities to treat pyritic ore. While such facilities appear to have been technically successful other factors like the selective mining of the richest ore foretold the ultimate end.

The Board policy of paying out in dividends the profits as made, continued despite the changed circumstances so that under conditions of alternating profit and then loss finance was provided by overdraft.

Phase 6: 1928-1942

This phase saw the commencement of work by the Company on the alluvial material in the Waratah River at the North Valley. The world wide economic depression with the fall in the tin price caused the cessation of all operations on 22 October 1929.

From then on all mining was tributors. The Company erected a number of batteries around the Mount which were rented to tributors. The Company maintained and operated the calciners and calcine concentrating plant on a custom basis.

This situation continued until the Japanese occupied Malaya, Sumatra and Thailand in World War II. The shortage of tin thus created caused the Federal Government to examine all sources of tin and led to its acquisition of Mt Bischoff.

North Valley Alluvial Sluicing

In 1915 fifty test shafts were sunk into this alluvial ground. Levings (Half-Yearly Report dated January 1926) stated 'the result was given as 1,005,000 tons with a value of 0.27 per cent tin...'. Subsequent working of this material and extensions, he said, had raised the tonnage to two million tons but with a much reduced tin content namely 0.15% Sn. Ample water was said to be available for sluicing.

When sluicing from a pontoon in a pond began in 1928 the unusual composition of the wash which contained huge boulders led to breakages of the equipment, hence delays and a more costly operation than envisaged.

In 1929 to overcome the difficulties a big grizzly and an electric winch were installed. However the tin yielded was below expectations so along with all other Company work this operation closed in October 1929.

Tributors were then let work the alluvial. Water shortages several times caused the suspension of these operations. In 1934 a water shortage precluded work then a severe flood damaged the sluicing equipment. In 1937 the Company built a race to assist the tributors.

Although in Figure 10 there is quite an extended period over which 'alluvial' was treated, except for the first two years of this phase most of this is considered to be from areas other than the North Valley deposit. It is difficult to differentiate from the reports between alluvial and old tailings as most of these had ended up in the Waratah River.

Waratah Mill

In 1928 it was reported that, as the 40-head mill was very worn, a new 10-head battery was being installed. The mill building was also repaired.

The recovered grade for 1928 was 0.53% Sn and for 1929 0.67% Sn these being the last two years of Company operation of both mine and mill.

In 1933 the mill ran on a two-shift basis for the whole year using ten heads. All the ore treated was supplied by tributors. A spitzlutte, jigs and four Card tables were in use. During the year the jigs were reconditioned and a further ten head of stamps were made ready for service.

The following indicates how the mill operated in succeeding years:

Year	*Shifts Worked*	*Average No. of Stamps used*
1934	422	14
1935	450	14
1936	406	15
1937	425	10

With the building of other batteries the main mill became just another battery with the trucks of ore being drawn by horses from the mine to the mill.

Roasting Plant

During 1928, 8,322 tons were roasted and before the closure in 1929 a further 2,961 tons were calcined. As alluvial was the main ore for the next two years the roasters were probably not used.

In 1933 a fire damaged the furnace building. Four Card tables were made operational in the calcine plant and the furnaces were repaired.

From then on the calciners appear to have worked for about three quarters of the available time.

Small Treatment Plants

In 1930 assaying and ore testing for tributors was done free.

Although it was reported that tributors were erecting their own treatment plants in 1933 as no details were given one assumes that these plants were quite primitive.

In 1936 a tributing party set up a 4-head battery at Happy Valley. A year later the Company erected 5-head batteries at the Slaughteryard Face and at the North Alluvial Face. These batteries were rented by the tributors.

In 1938 there were two batteries operating in the open cut area on Mt Bischoff and a battery on the North Alluvial Face. Two tables were installed at the Happy Valley battery.

A 5-head battery was installed in North Valley in 1939 and another five head in 1941 together with a sluice box.

Four Card tables were installed in a dressing shed on the North East Lode in 1939.

The Happy Valley battery was moved to the White Face in 1940. Small parties also worked areas where the Mt Bischoff Extended Company had operated.

Production

Tin production (fig. 10) declined to a low of 70 tons in 1930 then recovered to over 300 tons in 1932 after which it declined steadily to about 100 tons per annum in 1942.

The main source of tin in the early thirties was described as 'alluvial' but this was probably tailings deposited downstream from the Waratah Mill. At a later date tailings from dumps were treated. The high tin yields shown from the tonnages shows that cleaning up by the tributors was taking place.

The recovered mill head grades for 1928 and 1929 show that the Company was continuing selective mining of the richest ore apparent in previous phases.

The tributors' ore, as one would expect, was richer than that of the last year of Company operation. From the recovered head grade the ore must have assayed between 1.5 and 2% Sn during 1934 and 1935. However then it fell to between 0.5 and 1% Sn until 1942 when wartime incentives no doubt encouraged the tributors to gouge out some richer ore.

In 1940 a witness in the Arbitration Court stated the Company's treatment methods were obsolete. The Company sent a sample of ore to the Council for Scientific and Industrial Research in Melbourne for flotation tests to remove the sulphides but these tests, the Company stated, showed that the method employed, namely gravity pyrite concentration then the roasting of that concentrate and the gravity separation of the cassiterite from the calcine was the more economical.

Financial Aspects

The Company incurred losses in the early 1930s during the height of the Depression but in the later 1930s profits were made which were immediately distributed as dividends. In the early 1940s losses were again incurred.

The Situation in 1942

On the 23 March 1942 the principal tributors and the Company's Superintendent, Mr Porte, met at Waratah in order to prepare a case which was presented to the Metals Advisory Council, of the Commonwealth Department of Supply and Development. They advocated a price of £400 per ton for tin metal and a capital expenditure of £7,920 to re-equip the mine for larger scale production. Included in this sum were £1,600 for batteries and £3,000 for the re-location of the calciners.

Officers of the Tasmania Department of Mines (Henderson, Vaudeau and Manson) prepared a report (R.73) on the situation at Mt Bischoff. This report which contained geological, mining and metallurgical sections, was issued on 22 July 1942. It contained a review of tributing activity plus recommendations. A summary of the metallurgical part follows:

Tributing Parties Working Alluvial Material (R.73)

The tributing parties can be split into two groups, namely those working alluvial deposits and those working lode material.

Those parties working alluvial deposits were:

Party	*Location*	*Remarks*
Stevens	North Valley	The deposit the Company began working in 1928
Packett, Campton, Housego	Waratah River	Retreating mill tailings deposited along the river

The tributors working the alluvial material and tailings all had similar plants which were located near their workings. The material was fed to a coarse screen to remove large stones. The screen undersize was fed to a stamp battery the discharge from which was concentrated on a table from which a pyritic concentrate was taken for roasting in a single rabble, 36 ft^2 hearth roaster. The calcine was then tabled to get a sale tin concentrate.

Tributing Parties Working Lode Material (R.73)

The parties were engaged in the mining and milling of lode material, each rented a battery from the Company for the production of a pyritic concentrate which was roasted at the Company calciners and then tabled to produce a final concentrate in the adjoining plant which contained five tables on which 91% of the tin was recovered. The calciners treated 10 tons per 24 hours.

The parties and the lode material each treated were as under:

Party	*Lode*	*Remarks*
Crawford	North Valley	Hard pyrrhotite and pyrite ore
Clarke	Slaughteryard Face	Quartz and pyrite ore
Fraser	White Face	Soft pyritic ore
Allen	Gossan Face	Soft ore which was transported to the Waratah mill for concentration

The following are outline details of the various batteries:

Description	*Crawford*	*Clarke*	*Fraser*	*Allen*
Battery Location	North Valley	Slaughteryard	Happy Valley	Waratah
Lode	North Valley	Slaughteryard	White Face	Gossan Face
Plant				
Stamps	10	5	4	6-9
Tables	7	4	4	5
Operation				
Running (hours/week)	120	40	88	32
Treatment rate (tons/week)	160	40	90	120
Pyritic concentrate (tons/week)	25	8	6-8	6
Pyritic concentrate assay (% Sn)	3-4	5-7	4.5-7	4-6
Recovered head grade (% Sn)	0.56	0.24	0.17	0.34
Mill tin recovery (%)	62.7	64.7	56.7	66.3

All the batteries used 12# battery discharge screens with spitzlutte for classification. Allen also used a spitzkasten. The stamps were all 900 to 1,000 lb in weight except Fraser's which were about 500 lb.

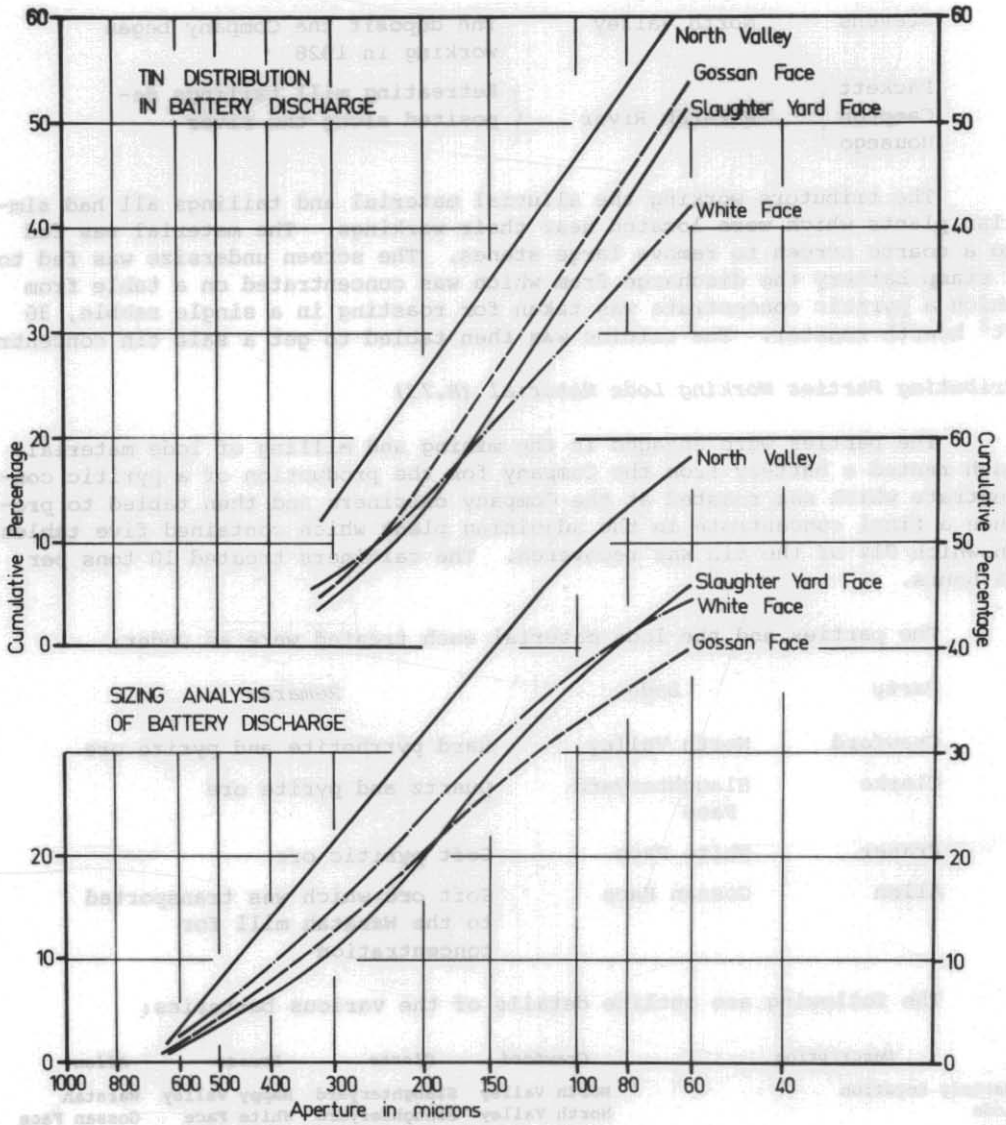


Figure 23. *Mt Bischoff. Sizing analyses and tin distribution in battery discharge [R 73].*

5 cm

The weekly rentals paid to the Company for the batteries were: Crawford £9, Clarke £2.10.0 and Fraser £1.16.0. Allen paid no rent for his battery. In addition there were charges for using the calciner and a royalty payment of 24% of the value of the tin contained in the final tin concentrate providing always that sufficient money was left to pay the tributors the Basic Wage.

Other Considerations and Recommendations (R.73)

The calciners would need relocation if they were to continue to be used. The larger calciner needed extensive repairs but the other calciner and the concentrating plant were in good order.

However as flotation was being used to eliminate the sulphides at Renison Bell hopes were entertained that flotation would eliminate the sulphides at Mt Bischoff and thus render a roasting plant unnecessary.

Most of the batteries were in poor repair. A crusher was needed ahead of the stamps at North Valley which was the only plant besides the Waratah mill of sufficient size to warrant alterations.

The Tasmanian officers recommended an expenditure of £10,045 to rehabilitate the field and favoured the North Valley for development to increase tin production.

The End of the Company

In the Director's Report dated 18 February 1942, J.L. Craw, the chairman, attributed the low production of tin in 1941 to a falling off in the grade of ore and a shortage of labour.

He wrote that '...it is important that every effort should be made, in the National interests, to step up production in all tin mines during the present year, and with additional labour your mine should be able to help in this direction.

'Your Directors regret that the past year resulted in a loss of £191.0.11 but an increase in the price of tin, would enable us to work much of our low grade deposits, and with additional labour should go a long way to solve the Company's difficulties'.

In the Director's Report dated 15 April 1943, Craw stated: '...your Directors regret that owing to operations at the mine being continued under instructions from the Commonwealth Government, a loss of £2,365.2.0, was incurred due to mining operations, which was reduced to £1,534.15.4 by a subsidy of £833.6.8 received from the Commonwealth Government Tin Pool'.

The Commonwealth Government acquired the mine and plant at Waratah from the Company on the 28 November 1942 with a view to increasing the production of tin from it.

The Mt Bischoff Tin Mining Company Registered, after receiving from the Commonwealth in respect of settlement for its assets £10,358 was finally wound up in August 1947.

Discussion

The world wide Depression, which began in 1929, certainly affected the Company, but it should not mask the fact that after it there was no return to organised tin production.

The North Valley Alluvial deposit, upon which the Company based its hopes at the end of Phase 5, was worked for only two years. In that time the tin yield had not been up to expectations and the plant as originally installed had been unable to handle the huge boulders in the wash. Although the plant was modified in 1929 the fact that a large scale operation never resumed suggests that it was not an economical proposition even with better tin prices. It should be noted that the original, 1915, ore reserve stated by Levings would have to have been diluted with nearly barren material, to get the 1925 reserve. However, as Levings says the area covered by the later reserve was much more extensive it is clear the original area must have been written down in grade. Some tributing was done in the late thirties and this was still going on in 1942.

The Waratah mill ran in 1928 and 1929 with increasing recovered head grades thus extending the previous pattern of mining richer ore to meet higher operating costs. The mill had been allowed to deteriorate but in 1928 some repairs were made. These allowed ten stamps to be used in 1933 when there were further repairs and more stamps were made serviceable.

As the Company never resumed mining after 1929 its mill became a custom plant for the tributors. Some tributors began processing their own ore in 1933 probably using dishes, cradles and sluices. From such beginnings the erection of a battery in 1936 was a logical expansion.

The erection of a battery by a tributing party would eliminate the overheads incurred by technical control, assaying and the bookwork a company operation required.

In 1937, when the Company began erecting batteries, which were let to tributors, centralised ore dressing ceased and with it any hope of metallurgical progress.

With the mining by tributors the practice followed by the Company of mining the richest ore was accentuated. Production rose as the tributors cleaned up the pickings left by the Company but by the mid 1930s these were gone, hence the low grade ore and low production.

In 1940 the Company sent a sample of ore to C.S.I.R. for flotation tests. However, with the operations at Mt Bischoff then dispersed among tributors it is difficult to see how flotation could have been implemented if it had been shown to have been successful.

From the 1942 survey the alluvial operations received little consideration hence the inference that they showed little promise. The lode milling operations look surprisingly efficient but when the losses after roasting are taken into account the recoveries were between 50 and 60%.

The battery operating data show quite wide differences for instance the Waratah mill was treating 4 tons per hour compared with one ton in the others. Some tables were treating ten tons per week while others handled more than twenty tons.

The wartime emergency caused the Commonwealth Government to look at Mt Bischoff. The Company approach was that if the government made it worth while they would produce the tin but they would not and, having no reserves, could not operate at a loss. The small profits of the late 1930s in accordance with the long followed practice of the Company were paid out in dividends as made.

While the Tasmanian Government report stated no ore reserve figures it did refer to operations on the North Valley lode in terms of 23,000 tons per annum. It envisaged flotation under 'effective metallurgical direction and control' for the removal of the sulphides.

As the Company could not meet the emergency and as there were indications of sufficient tin to make mining worthwhile the Commonwealth Government, acting to meet wartime needs, acquired the mine. The Company thus shorn of its sole asset had no longer any cause for existence once settlement with the Government was reached.

Review of Phase 6

Company organised mining and milling occupied only the first two years of this phase. The hopes entertained for the North Valley alluvial deposit were not realised. With this failure the Company had no more ore to exploit. This coincided with the Depression but would have occurred without it.

The long established practice of mining the richest ore left large tonnages of low grade material from which tributors gouged the best ore in the early 1930s.

The Company, unable to carry on mining itself, let out the mining to tributors to whom it acted as a custom ore dresser. When the tributors preferred their own ore dressing plants the Company assisted the dismemberment of its milling facilities by erecting and letting batteries around the mine area. Finally after several years that incurred losses the Company was unable to carry on. Faced with a national emergency the Commonwealth Government acquired the mine and the Company was wound up.

Although the Company made a move toward flotation in 1940 the circumstances suggest it was rather forced to it. Apart from that the Company's actions were such as to progressively destroy it.

With the Japanese occupation of the world's major tin fields government attention was turned to Mt Bischoff. After seventy years of being exploited with profit the only objective no easily won ore was at sight in 1942. Flotation might be the answer, under the emergency conditions the government would investigate. Thus this phase concluded with government hopes that at Mt Bischoff it could get tin where the old Company could not.

Phase 7: 1943-1947

Following acquisition of the mining assets of the Mt Bischoff Company the first year was spent evaluating the position while continuing production with assistance in the form of better equipment.

During 1944 improvements allowed greatly increased tonnages to be mined and processed. At the same time metallurgical testing was done with a view to building into the mill sufficient modern equipment to achieve a reasonable tin recovery. However, towards the end of that year an appreciation of the position showed Mt Bischoff would not meet the immediate tin deficiency caused by the war hence Commonwealth interest waned.

The first half of 1945 was spent with the Tasmanian Government endeavouring to keep Waratah alive by interesting the Commonwealth Government in continuing its exploration and metallurgical improvements at Waratah.

Once the Governments agreed more test work was done then a Board of

Management took over the operation in November 1945.

The installation of new equipment proceeded through 1946 until April 1947 when the commissioning of the new circuit began. While the alterations were being made milling was carried on using the batteries, tables and calciner.

The new circuit which incorporated a ball mill, screen, flotation and tilting decks dispensed with the calciner. This circuit did not work efficiently, tin recovery was only about half what had been predicted, the grade of ore was lower than had been estimated, the whole operation was uneconomic and hence was closed down in August 1947.

Commonwealth Action

Upon assuming control the Commonwealth sent H.H. Dunkin, Lecturer in Mining Engineering at the University of Melbourne to report on the mine.

Dunkin's review of the situation was largely the same as that given by the Tasmanian officers the year before. Dunkin believed the tin production could be increased by appointing a mining engineer to manage the mine. He recommended that the North Valley lode be investigated but also observed that the Pig Flat lode, although low grade was cheap to mine and crush. He sampled the North Valley and Waratah mills to assess performance.

He suggested methods of treatment for investigation including the removal of sulphides by flotation. He also took samples for metallurgical investigation.

The Commonwealth objectives were outlined by Lindesay Clark as:

'(1) keep the mine operating and the township in being;

(2) put in such modern metallurgical plant as would enable the recovery with high class equipment to be established; and

(3) give time for a drilling and development campaign to test the remaining prospects of further ore.'

A mining engineer, A.A. McLeod, was appointed manager. Tilting decks were recommended to him by M.A. Mawby who had seen them being used for the recovery of fine tin in Canada. Ore dressing investigations on North Valley ore began at the C.S.I.R. Ore-Dressing Laboratory in Melbourne and in the Department of Mines Laboratory in Launceston. At Mt Bischoff a hydrosizer was designed to produce better table feed in the North Valley mill.

Ore Dressing Research

C.S.I.R. Investigations on North Valley Ore

The C.S.I.R. mineragraphic investigations showed that the quartz and cassiterite were intimately associated but these two minerals were not closely associated with the sulphides. Stannite and some chalcopyrite were present.

Tabling of the battery discharge after hydraulically sizing into three spigot products plus an overflow gave a 61.4% recovery compared with a plant recovery of 56.4% which was obtained by tabling two products from a spitzlutte (fig. 24). By roll crushing the ore in the laboratory and hydraulically sizing it into five products a recovery of 66.9% was obtained (fig. 24).

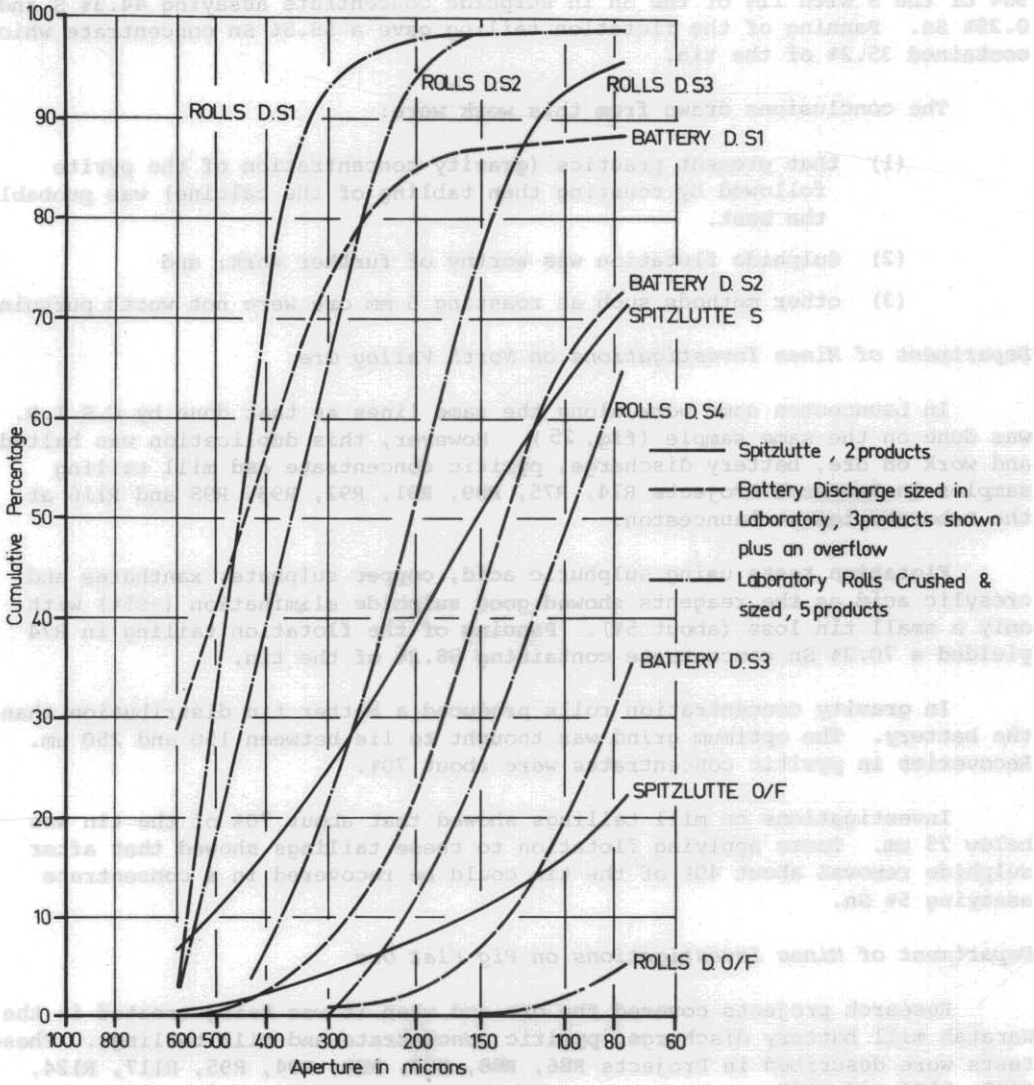
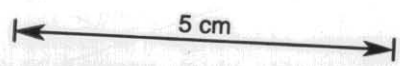


Figure 24. *North Valley plant and laboratory table feeds, 1943.*



All these recoveries were obtained in pyritic table concentrates assaying between 3.8 and 5.0% Sn.

Sulphide flotation after grinding to pass a 210 µm screen eliminated 95% of the S with 11% of the Sn in sulphide concentrate assaying 44.3% S and 0.25% Sn. Panning of the flotation tailing gave a 58.5% Sn concentrate which contained 35.2% of the tin.

The conclusions drawn from this work were:

(1) that present practice (gravity concentration of the pyrite followed by roasting then tabling of the calcine) was probably the best.

(2) Sulphide flotation was worthy of further work; and

(3) other methods such as roasting 6 mm ore were not worth pursuing.

Department of Mines Investigations on North Valley Ore

In Launceston some work along the same lines as that done by C.S.I.R. was done on the same sample (fig. 25). However, this duplication was halted and work on ore, battery discharge, pyritic concentrate and mill tailing samples in Research Projects R74, R75, R89, R91, R92, R93, R98 and R116 at the Laboratories at Launceston.

Flotation tests using sulphuric acid, copper sulphate, xanthates and cresylic acid as the reagents showed good sulphide elimination (>95%) with only a small tin loss (about 5%). Panning of the flotation tailing in R74 yielded a 70.3% Sn concentrate containing 88.2% of the tin.

In gravity concentration rolls produced a better tin distribution than the battery. The optimum grind was thought to lie between 150 and 250 µm. Recoveries in pyritic concentrates were about 70%.

Investigations on mill tailings showed that about 70% of the tin was below 75 µm. Tests applying flotation to these tailings showed that after sulphide removal about 40% of the tin could be recovered in a concentrate assaying 5% Sn.

Department of Mines Investigations on Pig Flat Ore

Research projects covered the ore and when it was being treated in the Waratah mill battery discharge, pyritic concentrate and mill tailings. These tests were described in Projects R86, R88, R89, R90, R94, R95, R117, R124, R141, R144 and R145.

Head grades of samples of this ore were quite variable, the assays being either around 0.2% or 1% Sn.

Investigations were made to try and disperse the clayey material in this ore using a concrete mixer and a pebble mill.

Flotation after grinding to pass a 180 µm screen using copper sulphate, lime, xanthate and pine oil was not a success. Sulphide elimination was only 63% and the tin concentrate which assayed 22.8% Sn contained only 30.5% of the tin (R141). This low recovery was confirmed by repeating the test procedure on another sample and by a run of this ore through the Waratah mill, (fig. 26). By regrinding the mill tailings from this test to pass a 105 µm screen, then retabling, the recovery was raised to 47.8%. Further tests on

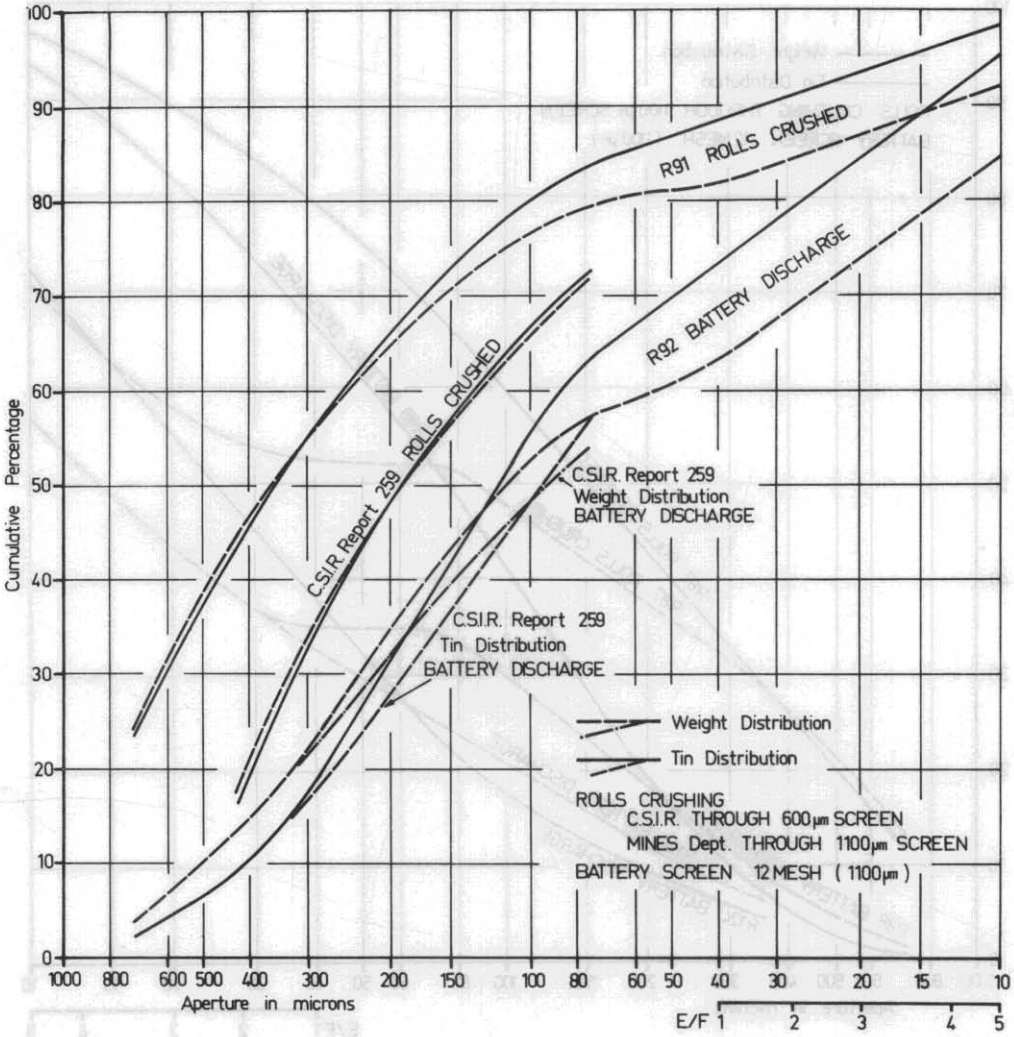


Figure 25. *North Valley mill, battery and rolls crushing, 1942.*

5 cm

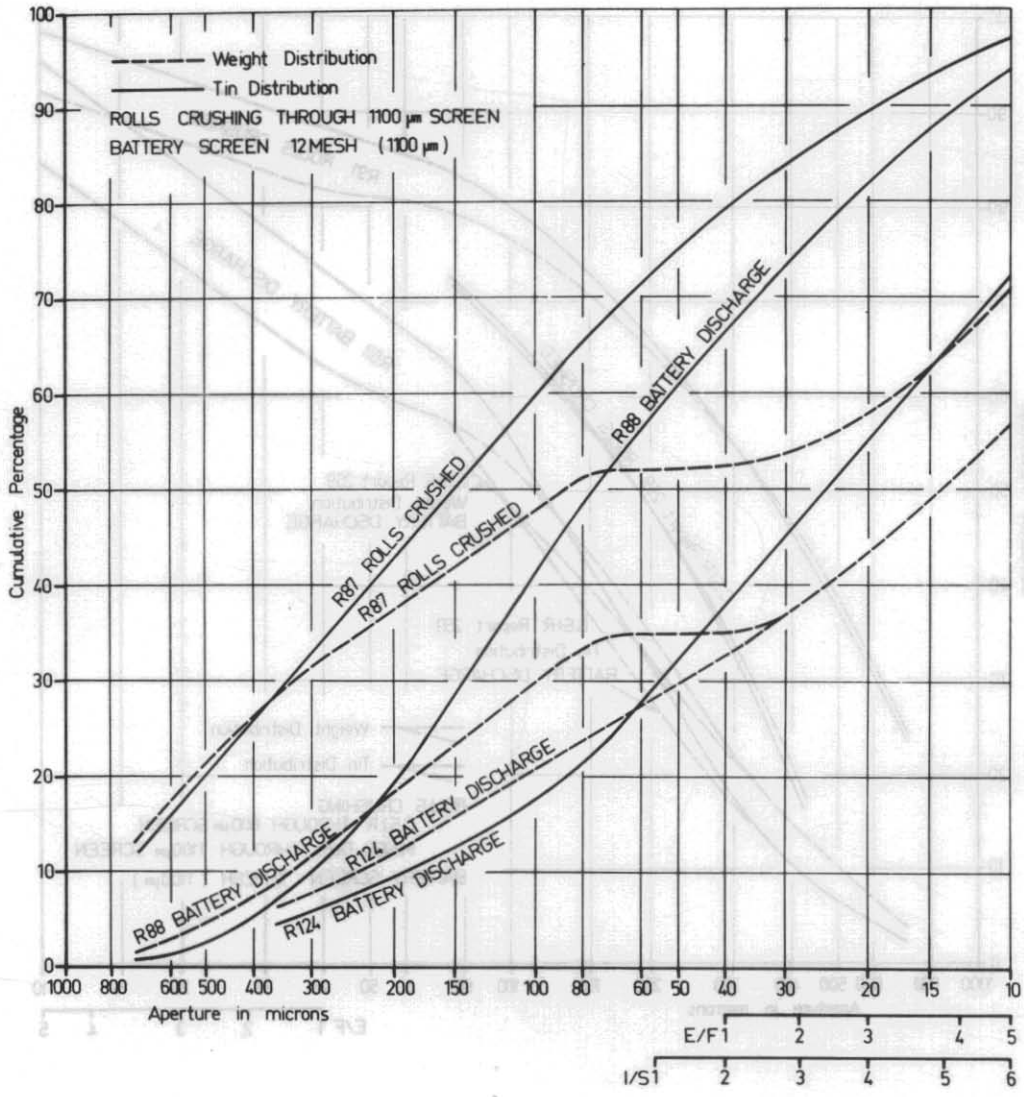
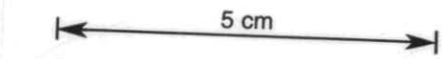


Figure 26. *Pig Flat ore, battery and rolls crushing, 1943-1944.*



different Pig Flat ore yielded around 40% recoveries in low grade (5%) tin concentrates.

The tin distribution in the mill tailings was similar to that of North Valley. Efforts to apply flotation to this tailing were unsuccessful as a non-sulphide gangue mineral floated readily with the sulphides.

Department of Mines Investigations on Greisen Ore

As the geological appraisal progressed the emphasis passed from North Valley to the Greisen Orebody. Work on this ore falls into two sections, namely that done in the last half of 1944 and that done after August 1945 when the Commonwealth had agreed to continue the operations which were to mine and mill Greisen and some Pig Flat ore.

The 1944 tests were described in Projects R118, R121, R122, R123, R125, and R127.

Gravity tests showed that to achieve a recovery of tin greater than 70% in a low grade (4%) tin concentrate grinding to pass a 105 µm screen was necessary.

Flotation tests were made on ore ground to pass a 160 µm screen. Talc and dolomite were found to be the non-sulphide minerals that floated readily with the sulphides in fact talc would float with the addition of frother only but the talc concentrate so made contained an excessive amount (about half) of the tin.

These tests showed recoveries around 60% but the grade of tin concentrate was low (15 to 38% Sn) except in Test R118, N8 where a grade of 57% Sn was achieved.

Tests to depress the talc showed this could be done by floating at pH 11 which required a lime addition of 3 kg per metric ton. Using this and a cleaner stage of flotation a tin concentrate assaying 67.4% Sn and containing 72.8% of the tin was produced (R118, N22).

Further gravity testing confirmed the results achieved earlier.

Tests to determine the recovery of 'slime tin' in R122 used a spigot product and a classifier overflow: the former assayed 1.05% Sn and sized 70% between 28 and 105 µm. The latter assayed 0.51% Sn and sized 100% -40 µm and 33% -14 µm. These tests showed about half the tin could be recovered in concentrates assaying approximately 50% Sn. The Waratah battery discharge sizing and tin distribution is shown in Figure 27.

The renewed investigation of the Greisen ore in 1945 is described in Projects R140, R141, R143, R152, R180 and R181.

The grindability of the ore was established by comparison with King Island scheelite ore.

Tests on three samples of Greisen ore (R141) showed over 70% recoveries in two of these samples but only 30% in the third in concentrates assaying 20 to 38% Sn.

The low recovery was attributed to the oxidation of the sulphides over a four month period but a composite of all three samples floated some time later yielded a recovery close to the average of the three recoveries

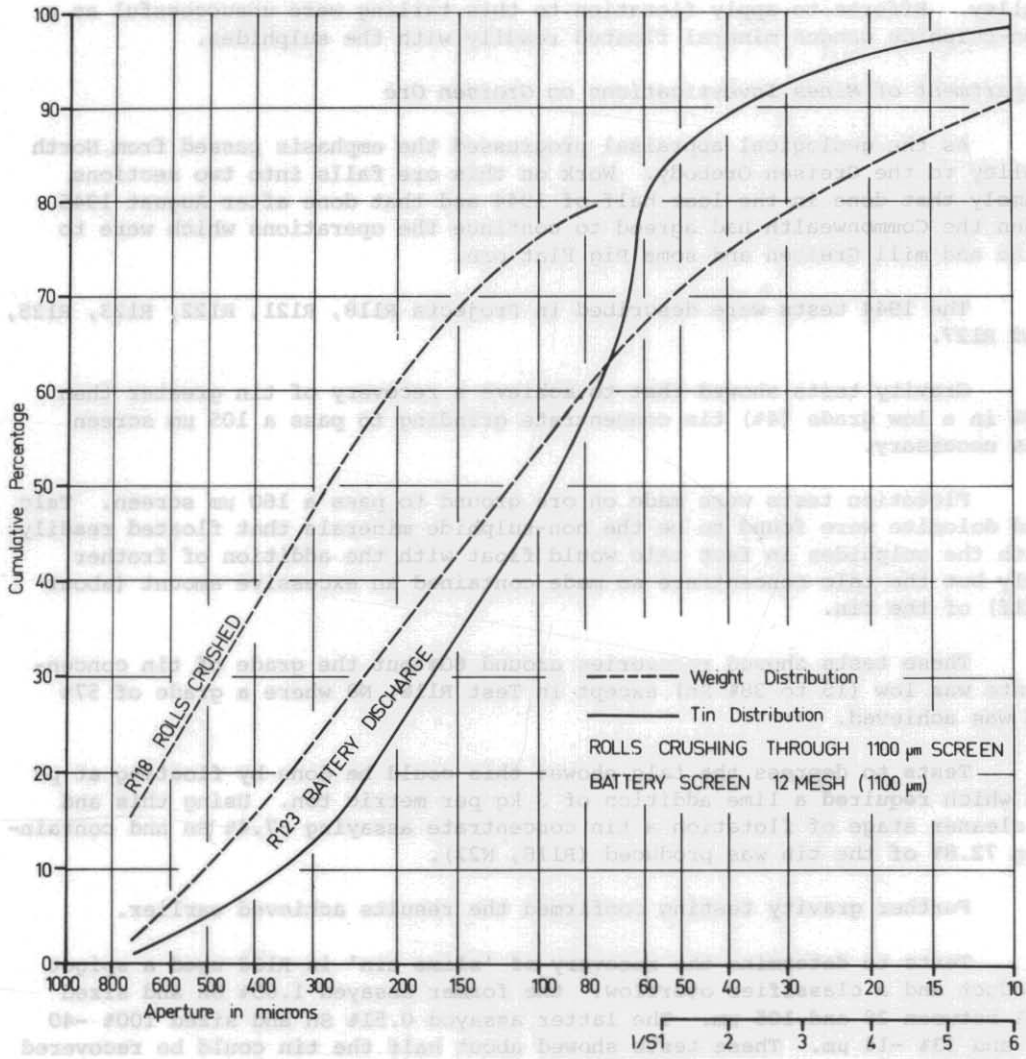
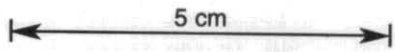


Figure 27. *Greisen ore, battery and rolls crushing, 1943-1944.*



obtained earlier.

At this time a large parcel of ore was put through the Renison Bell mill.

A sample of mixed Greisen and hard Pig Flat ore was ground to pass a 180 µm screen and the sulphides were floated at natural pH using copper sulphate, xanthates and pine oil. The rougher concentrate so made was cleaned at pH 11.2 using the same reagents plus lime to correct the pH. The result was a tin concentrate assaying 12.1% Sn and containing 43.1% of the tin (R152). A sample of this ore was sent to C.S.I.R. for cassiterite flotation tests.

Tests on a sample of Waratah mill tailings (R180; fig. 29) showed that the plant tabling was inefficient.

Other tests on Glory Hole samples which were tabled to yield a pyrite concentrate which was subjected to flotation gave tin concentrates of grades 10.9 and 19.8% with respective recoveries of 62.3 and 71.4%.

Tests on the calciner discharge showed 84% of the tin was recovered in a 57.8% tin concentrate.

The Test at Renison Bell on Greisen Ore, October 1945

In order to assess the recovery of tin under mill conditions a shipment of 276 tons of Greisen ore was railed to the Renison Bell mill in October 1945. This was reported in R143.

The circuit used in shown in the flowsheet (fig. 28). The rougher concentrates were worked up to sale grade in a kieve with the retreatment of kieve tailings etc., by batch flotation and tabling.

The classifier overflow which assayed as under, was sized and the tin distribution determined (fig. 29).

	%			%
Sn	0.68	Acid soluble	Fe	35.1
S	20.9	Acid soluble	MnO	0.81
Cu	0.08	Acid soluble	P_2O_5	0.14
Bi	Trace	Acid soluble	CaO	0.60
As	0.15	Acid soluble	MgO	5.4

The acid insoluble portion comprised 25.2% of the classifier overflow and assayed:

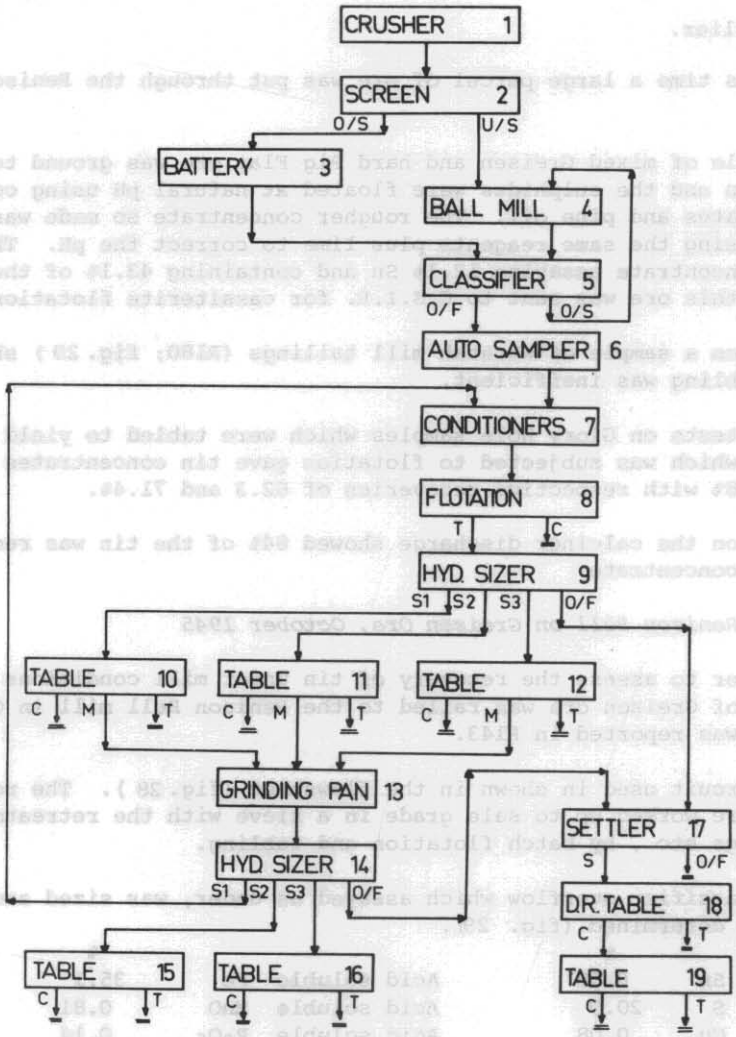
	%		%		%
SiO_2	63.2	Fe_2O_3	1.7	MgO	26.7
Al_2O_3	1.1	CaO	Nil	SnO_2	3.3

Sulphide flotation was run at pH 11.5, this pH being controlled by lime addition. Reagent consumption was:

	g/m.ton	*lb/long ton*
Copper sulphate	450	1.0
Limil	1.4 kg	3.14
Mixed xanthate*	450	1.0
Eucalyptus phellandrene	53	0.12

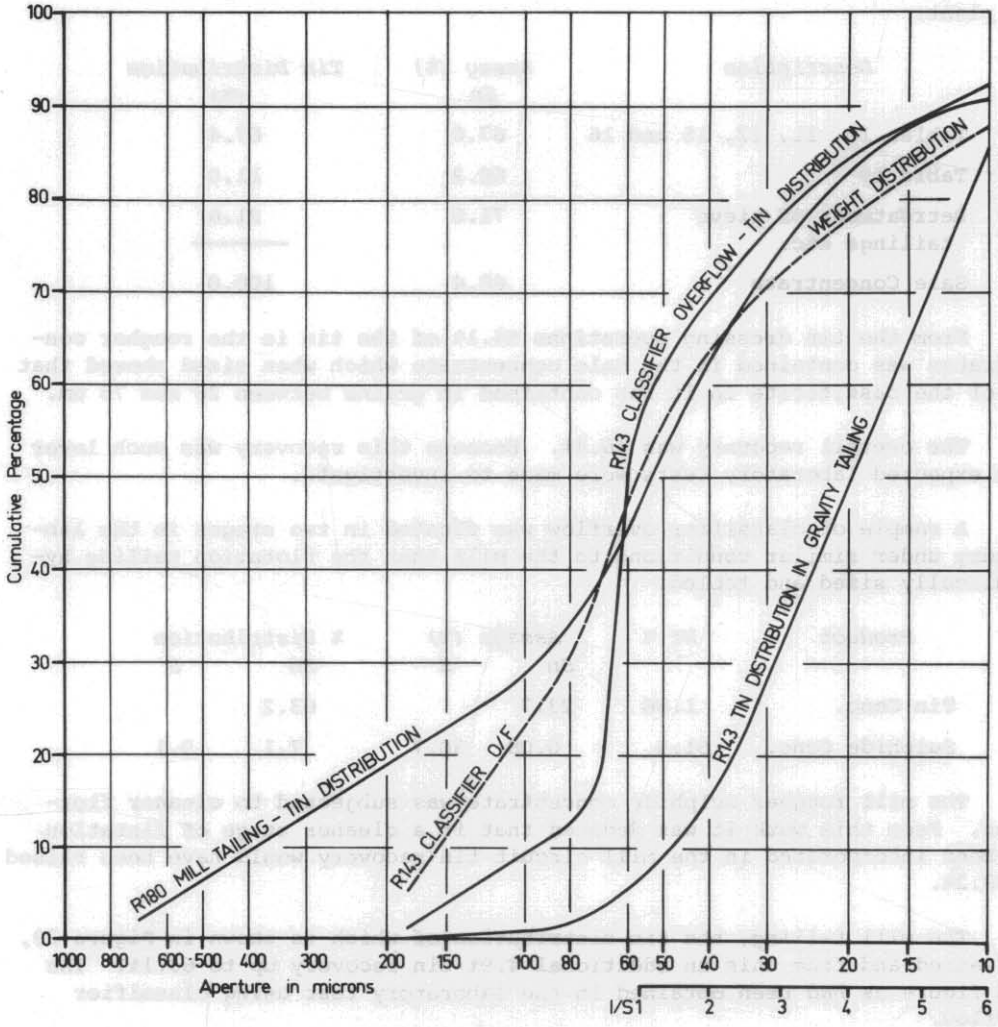
The sulphide flotation concentrate F8C (fig. 28) assayed 0.3% Sn and contained 63% of the feed weight, 24% of the tin and 82% of the sulphur.

* 2 parts sodium ethyl xanthate + 1 part T43.



Item No.	*Description*	*Item No.*	*Description*
1	Crusher, jaw	10,11,12	Tables, Bannister
2	Screen, 0.75 in aperture	13	Grinding pan.
3	Battery, stamp, 0.25 in aperture screen. Discharge averaged 35% solids.	15,15,16	Tables, Wilfley.
		17	Settler
		18	Double round table
4	Ball mill, 6 ft long x 2.5 ft diameter.	19	Table, Diester
5	Classifier, drag, O/F density 27% solids, 2.7 tons of O/F per hour.		
6	Automatic sampler		
7	Batch conditioner		
8	Flotation cells. 8 M.S. type, 26 in x 60 in deep.		
9	Hydraulic sizer.		

Figure 28. *Renison Bell test on Greisen ore, October 1945, flowsheet.*



R 180: Waratah mill tailing, 1946.

Figure 29. *Renison Bell test on Greisen ore, October 1945. Sizing curves for mill products [R 143].*

5 cm

The rougher table concentrates (T10C, T11C, T12C, T15C and T16C) averaged 10.5% Sn (Range 7-13% Sn) while the slime concentrate (T19C) assayed 30% Sn. The table tailing assayed 0.72% Sn.

The sale tin concentrate was derived from the following sections of the plant:

Description	*Assay (%) Sn*	*Tin Distribution (%)*
Tables 10, 11, 12, 15 and 16	67.8	67.4
Table 19	66.2	11.0
Retreatment of kieve tailings etc.	71.8	21.6
Sale Concentrate	68.4	100.0

From the tin dressing operations 94.1% of the tin in the rougher concentrates was contained in the sale concentrate which when sized showed that 90% of the cassiterite in it was contained in grains between 28 and 75 µm.

The overall recovery was 48.4%. Because this recovery was much lower than expected laboratory tests were made to investigate.

A sample of classifier overflow was floated in two stages in the laboratory under similar conditions to the mill then the flotation tailing hydraulically sized and tabled.

Product	*Wt %*	*Assays (%)*		*% Distribution*	
		Sn	*S*	*Sn*	*S*
Tin Conc.	1.86	23.7		63.2	
Sulphide Conc.	61.9	0.08	30.7	7.1	9.1

The mill rougher sulphide concentrate was subjected to cleaner flotation. From this work it was deduced that if a cleaner stage of flotation had been incorporated in the mill circuit tin recovery would have been raised to 58.2%.

The mill tailing, the tin distribution of which is shown in Figure 29, was vanned and from this an additional 4.9% tin recovery up to 63.1%. The same figure as had been obtained in the laboratory test using classifier overflow.

As a result of this mill test and subsequent laboratory investigations the following conclusions were made in R143:

(1) The ball mill-classifier circuit over-ground cassiterite hence screens should be used in the grinding circuit as were used in laboratory tests.

(2) The higher recovery in the laboratory was attributed to:

(*a*) the use of a cleaner flotation stage, and
(*b*) more efficient tabling.

(3) The ore dolomite content, which was higher than expected, could have affected the concentration of the finest cassiterite.

(4) The flotation machines 'sanded up' periodically: this caused tin losses.

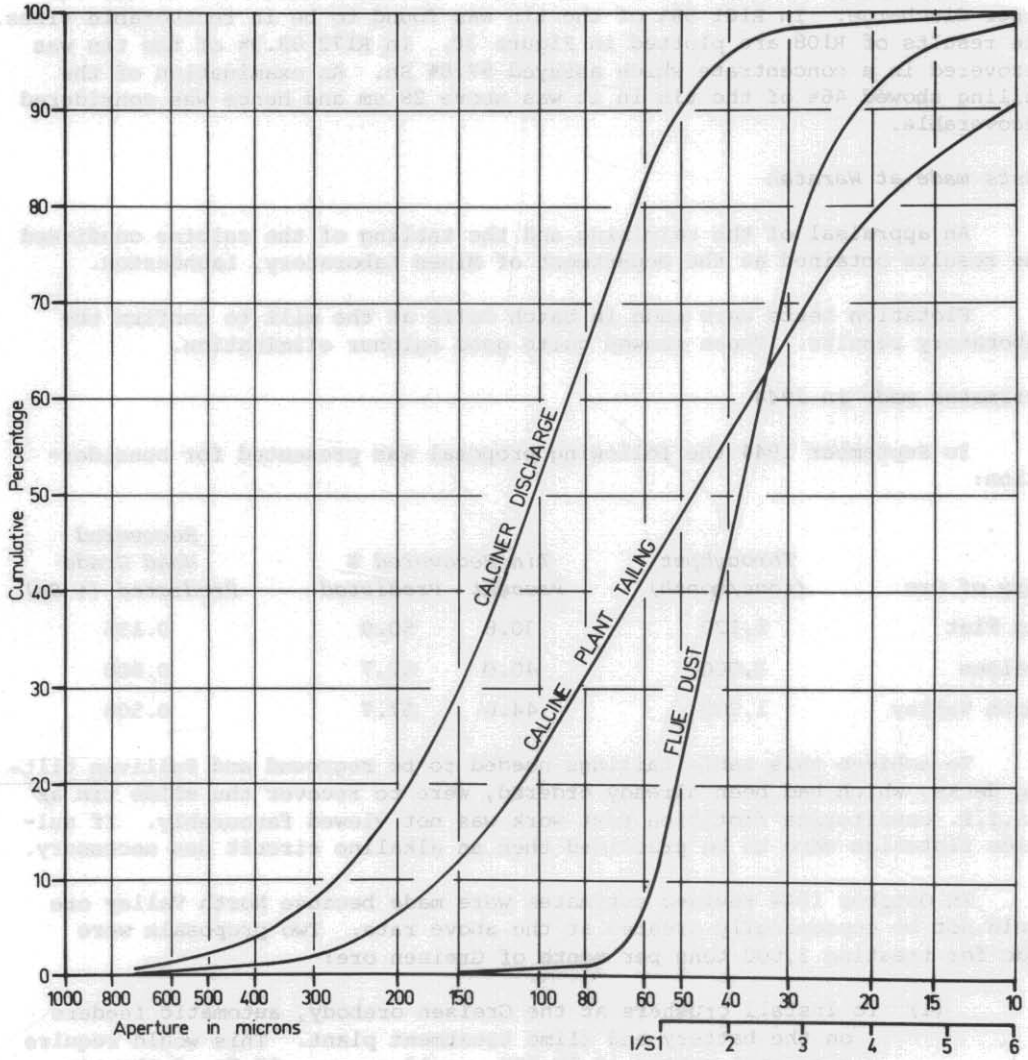
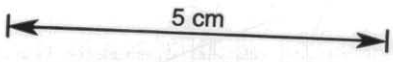


Figure 30. *Mt Bischoff. Tin distribution in calciner products, 1944.*



Another point made in R143 was that the expected head grade from mine sampling was 0.9% Sn not the 0.78% Sn obtained.

Department of Mines Investigations on the Calciner Discharge

Three projects were done to determine the recoverable tin in the calciner discharge. In R101 99% of the tin was found to be in recoverable sizes. The results of R108 are plotted in Figure 30. In R172 83.9% of the tin was recovered in a concentrate which assayed 57.8% Sn. An examination of the tailing showed 46% of the tin in it was above 28 μm and hence was considered recoverable.

Tests made at Waratah

An appraisal of the calcining and the tabling of the calcine confirmed the results obtained at the Department of Mines Laboratory, Launceston.

Flotation tests were made in batch cells at the mill to confirm the laboratory results. These showed quite good sulphur elimination.

Estimates made in 1944

In September 1944 the following proposal was presented for consideration:

Type of Ore	*Throughput (tons/month)*	*Tin Recovered %*		*Recovered Head Grade Predicted (% Sn)*
		Present	*Predicted*	
Pig Flat	1,120	30.8	50.0	0.135
Greisen	3,000	40.0	57.7	0.508
North Valley	1,200	44.6	57.7	0.508

To achieve this table tailings needed to be reground and Sullivan tilting decks, which had been already ordered, were to recover the slime tin as C.S.I.R. cassiterite flotation test work was not viewed favourably. If sulphide flotation were to be practised then an alkaline circuit was necessary.

In October 1944 revised estimates were made because North Valley ore could not be economically treated at the above rate. Two proposals were made for treating 3,000 tons per month of Greisen ore:

(1) To install crushers at the Greisen orebody, automatic feeders on the battery and slime treatment plant. This would require a capital outlay of £8,800, would recover 57.7% of the tin and show a monthly profit of £1,937, and

(2) to install a ball mill/classifier circuit to grind the battery discharge to -180 μm followed by flotation to remove the sulphides. This required an outlay of £14,350 to yield a 60 to 65% tin recovery and £2,700 profit per month.

Production, 1944-1947

The Waratah and North Valley mills operated throughout 1944 and 1945 while metallurgical investigations were made. The North Valley mill was closed on 30 August 1946. The milling of Pig Flat ore in the Waratah mill ceased on 25 June 1946. Production at Waratah was hampered during 1946 and early in 1947 due to construction and circuit alterations. Mill head grades

and recoveries for 1946 are shown in Figure 32.

The following production figures have been taken from the Director of Mines' annual reports, except for 1945 where I have used those given by the Mine Manager in his annual report.

	1944	*1945*	*1946*	*1947*	
Ore milled:					
North Valley	12,979	13,638	7,192	-	tons
Greisen	4,880	1,255	13,650*	21,788	tons
Pig Flat	28,172	29,978	5,400*	-	tons
White Face	458	-	-	-	tons
TOTAL	46,489	44,871	26,242	21,788	tons
Pyritic conc. calcined	4,084	2,755	2,272	879†	tons
Metallic tin produced	90.7	65.3	44	31.5	tons
Grade of sale conc.	66.4	67.7	66.8	59.2	% Sn
Recovered head grade	0.20	0.14	0.17	0.14	% Sn

*Estimated from mine production. †From routine mill reports.

The figures given in the Director of Mines' reports for 1945 were:

milled	45,316	tons
metallic tin produced	61.862	tons
grade of sale conc.	65.2	% Sn

Initiation of the Joint Venture

In late 1944 it was apparent that Mt Bischoff would not be able to quickly supply tin for the war effort hence Commonwealth interest waned but the Tasmanian Government wishing to maintain the town of Waratah began enquiries aimed at inducing the Federal Government to continue its support for exploration and metallurgical investigation.

In August 1945 the two governments agreed upon a joint venture believing there were sufficient ore reserves assaying about 0.85% Sn and that a recovery of not less than 60% would be achieved from Greisen ore in the revamped mill. For Pig Flat ore the recovery was not expected to exceed 50%. It was proposed to mill 3,000 tons of Greisen and 1,500 tons of Pig Flat ore per month.

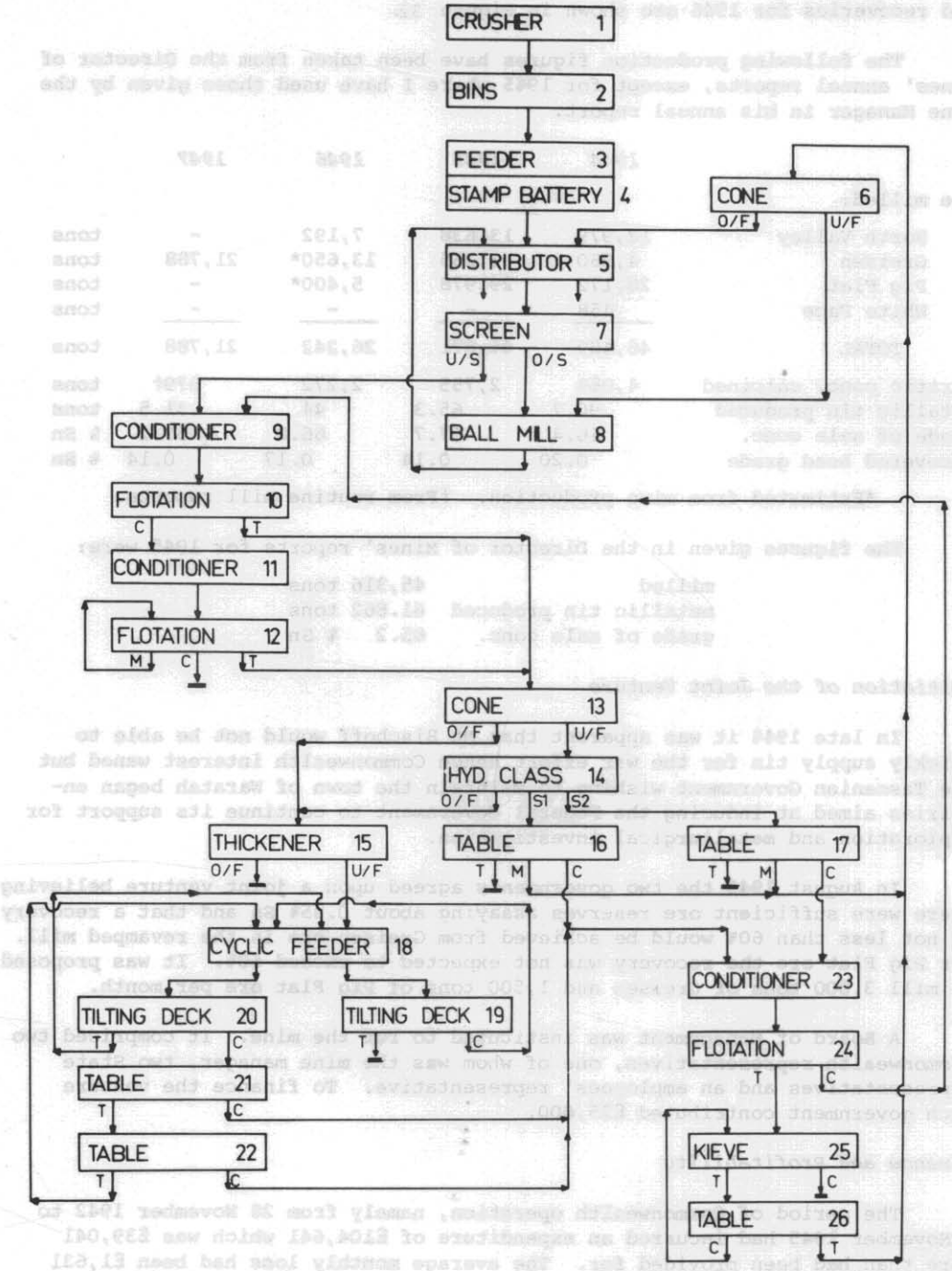
A Board of Management was instituted to run the mine. It comprised two Commonwealth representatives, one of whom was the mine manager, two State representatives and an employees' representative. To finance the venture each government contributed £25,000.

Finance and Profitability

The period of Commonwealth operation, namely from 28 November 1942 to 6 November 1945 had incurred an expenditure of £104,641 which was £39,041 more than had been provided for. The average monthly loss had been £1,631 but over the last six months (June-November 1945) it had averaged £2,069.

When the Board of Management took over on 6 November 1945 it had been voted £50,000 to establish Mt Bischoff as a viable operation.

A loss was incurred during every period of the Board's control. By September 1946 the Board was seeking a further £20,000 from the Commonwealth which granted it. By May 1947 the Board had overspent its vote by £5,000



FINAL PRODUCTS:

Concentrate: Item 25.
Waste Residues:
Sulphide: Item 12.
Tailing: Items 16, 17, 19.
Overflow: Item 15.

Figure 31. *Waratah flowsheet, 1947*

Notes on Figure 31. *Waratah Concentrator Flowsheet, 1947*

Item No.	*Description*
1	Crusher, jaw 20 in x 16 in set to crush to 3 in.
2	Mill bins
3	Challenge feeders, one for every five stamps.
4	Stamp battery, 15 head, 1,200 lb each; discharge screen aperture, 0.5 in.
5	Distributor, three way, for distribution of screen feed.
6	Cone, de-watering 6 ft diameter, with Richards type hydraulic vortex fitting.
7	Screen, No. 38 Hummer: 4 ft wide, 5 ft long; mesh: Toncap No. 1 2475 (aperture 200 µm). Three screens were installed.
8	Ball mill, 5 ft diameter, 6 ft long; grate discharge.
9	Conditioner, 6 ft diameter, 6 ft deep, Denver type.
10	Flotation cell bank, 6 Denver No. 21 Sub A cells.
11	Conditioner, 10 ft diameter, 6 ft deep, Denver type.
12	Flotation cell bank, 6 Denver No. 21 Sub A cells.
13	Cone, desliming, 6 ft diameter.
14	Hydraulic classifier, two spigot discharge.
15	Thickener, 20 ft diameter, 7 ft deep.
16	Table, Wilfley No. 11, four were installed.
17	Table, Wilfley No. 11, four were installed.
18	Cyclic feeder. Thickener underflow handled by one side of feeder and cleaning deck feed on the other. Cyclic feeder in closed circuit with surge tanks in each circuit to handle build up during period when decks were not receiving feed.
19	Tilting deck, Denver-Buckman type, 5 decks each 6 ft x 6 ft.
20	Tilting deck, see Item 19.
21	Table, Wilfley No. 11.
22	Table, Wilfley No. 11.
23	Conditioner, 3.5 ft diameter, 4.5 ft deep.
24	Flotation cell bank, 2 Mineral Separation cells each 28 x 28 x 54 in deep.
25	Kieve, 34 in diameter, mechanically tapped. Two were installed.
26	Table Wilfley No. 6, two were installed.

The capacity of the mill was 8.6 tons per hour.

The object of mill was to reduce the feed to -210 µm, remove the sulphides by flotation and then concentrate the tin on coarse rougher tables (Items 16, 17) to 10-30% Sn and on slime tables, (Items 21, 22), to 20-40% Sn.

and was seeking a further £20,000. The Prime Minister agreed to grant a further £10,000 provided the State did likewise but this would be the last money to be made available for this project.

The Board incurred expenditure of £86,900 or an average monthly loss in excess of £4,000.

Reconstruction and Testing, 1946

The post-war shortages and inflation made it difficult to procure machinery and keep within the estimates during 1946.

The tilting decks were delivered in February and given a trial run in March but it was reported in July that the decks were not yet being properly fed. At the end of 1946 it was reported that there was only sufficient deck area for a roughing stage.

In March priority was given to completion of the Greisen crushing station for the automatic battery feeders could not be used until crushed feed was available. However, in September when such feed was available the ore trucks needed modification to feed the battery bins.

The new Wilfley tables were poorly made and were made of poor quality materials hence they needed continual attention.

Some flotation cells were commissioned in April. Some experimental table flotation was practised.

Absenteeism and the failure of the mine to produce sufficient ore seriously affected mill production.

In November it was reported that metallurgical results to date were disappointing, the main cause being that pyrrhotite carried tin with it off the tables. However, when the ball mill and new flotation cells were operating this would be overcome.

Reconstruction and Testing, 1947

When the plant metallurgist quit early in January, Department of Mines personnel became actively engaged in the plant operation and continued thus after the arrival of the new plant metallurgist in April.

The ball mill/screen circuit commenced on 11 April but the stamp batteries could crush only 5 tons per hour through the 6 mm battery screens. This was investigated (R195). The required 8 tons per hour was achieved by fitting 12 mm screens to the battery on 10 May.

The flotation section was commissioned on 1 May and the tilting decks on 5 May. With these sections operating the flowsheet became that shown in Figure 31. Flotation was poor from the start. The flowsheet was changed so that battery discharge was fed to the ball mill rather than the screen in the hope that grinding would produce fresh sulphide surfaces and hence better flotation. A proposal to use sulphuric acid was vetoed in early May because of the cost of acid.

Two plant recovery tests were run during May one of 60 hours the other of 127 hours duration. The tin recoveries obtained were stated to be 47 and 58% respectively. During these tests various plant machines were sampled to assess performance. A mill tailings sample so obtained assayed 0.28% Sn.

Retabling of it in the laboratory (R195) recovered 30.8% of the tin in it in a 19.2% Sn concentrate. Examination of this same tailing showed 40% of the tin was coarser than 75 µm, (fig. 34).

At the end of June the milling position was reviewed. The following production figures were given:

4 weeks Period Ended	*Ore Milled (tons)*	*Recovered Head Grade (% Sn)*
22 April 1947	2,195	0.099
20 May 1947	2,576	0.166
17 June 1947	3,491	0.113

The poor metallurgical results were attributed to low grade mill feed. No mill recovery figures could be given because most of the assaying capacity was being used on experimental work.

Elimination of the sulphides by flotation was more difficult than expected. This was attributed to some oxidation of the sulphides. The sulphide content of the ore varied widely and averaged more than was anticipated. In the dressing shed coarse sulphide grains could not be removed from the tin concentrates.

There were also mechanical troubles with the secondhand equipment. Mill spillages were hosed into the tailings launders. The mill operation was placed under Department of Mines metallurgical control at the end of June.

The tin content of the ore mined increased after June but the sulphide content also increased. The former poor flotation elimination of sulphides was now accentuated, sulphide minerals overloaded the tables reducing tin recovery to 25-30%. Increasing the amount of collector did not improve flotation. It was found that if the flotation cell impeller speed was increased flotation was better. If sulphuric acid were added to the slower speed cells flotation improved.

On 16 July a test run lasting 3.7 hours was made using sulphuric acid in the rougher flotation. During this test tabling was said to have been 'impressive' and the mill tin recovery 'of the order of 60 per cent'. The sample of mill tailing taken during this test assayed 0.3% Sn. In this sample 42% of the tin was in the +75 µm fraction and 20% of this cassiterite was in liberated grains.

The following figures were given for July 1947:

Period	*Ore Milled (tons)*	*Metallic tin in Concentrates (tons)*	*Recovery (%)*
2-15	1,915	3.721	46.3
16-22	788	1.533	51.2
23-29	799	1.456	50.6

After this test with sulphuric acid there was a delay of two weeks while more acid was obtained. In the two weeks from 30 July until 13 August sulphuric acid was added to the rougher flotation stage but cleaner flotation was done at a high pH.

Production figures for these two weeks were:

Ore milled	1,984 tons
Tin in concentrates produced	4.387 tons

Flotation reagents used were:

Sulphuric acid	2 kg/metric ton
Copper sulphate	430 g/metric ton
Sodium amyl xanthate	54 g/metric ton
Sodium ethyl xanthate	22 g/metric ton
Potassium amyl xanthate	31 g/metric ton
Eucalyptus oil	71 g/metric ton
Lime	788 g/metric ton

The mill metallurgist averaged the results of two calculations (73.5% and 59.2%) to get a recovery of 66.35% for these two weeks. Some time later at the Department of Mines a figure of 66.9% was obtained but this was qualified for it did not balance with other data.

Final Report

When the Prime Minister closed down operations because the low grade of ore and poor tin recovery were resulting in heavy financial losses, the Board of Management presented a report to the responsible ministers. In this report the Board stated that it had processed 52,000 tons of ore to yield 78 tons of metallic tin (recovered head grade 0.15% Sn). It had spent £47,558 on capital works and incurred operating losses of £39,308.

It attributed failure to the inaccurate estimates of ore reserves but claimed 'approximately 60 per cent' recoveries for the July test run and the last two weeks of operation.

Comment on Data Reported

Up to this point production figures, recoveries etc., have been quoted verbatim.

However, as there appear to be conflicting figures given and as some of the calculations appear to have been based on erroneous presumptions it is necessary to attempt to establish from primary sources what actually happened by:

(1) establishing the grade of each ore type milled, then

(2) establishing the grade of mill feed for each year or period of significance, then

(3) establishing the recovery obtained knowing the head assay and the recovered head grade which was determined from the tin produced and the tonnage milled.

To do this certain assumptions have had to be made for most of the time being considered. The confusion caused by the use of 'van tin' and 'chemical tin' assays permeates all phases of operations making the task of resolving this tedious and long. Fortunately sizing analyses and tin determinations were done after operations at Mt Bischoff ceased. These results, which were derived from chemical tin assays, have been used and from them and their corresponding 'van tin' assays a relationship has been established to enable an estimation of the chemical result where no such result existed.

Grade of the Mill Feed and Recovery

The grade of the Greisen ore has been established for each year of operation by reference to the grade of samples submitted for research, the head grade of parcels milled during tests, fortnightly metallurgical reports (fig. 32) and from estimates of grade. The same has been done for North Valley. Pig Flat ore varied greatly in grade making it impossible to calculate a grade for any year hence 0.3% Sn has been adopted for the whole period.

The following head grades have been obtained for the Greisen ore:

Period	*Head Assay (% Sn)*
1944	0.8
1945	0.7
1946	0.4
1947 1 January to 22 April	0.47
23 April to 2 June	0.48
3 June to 30 July	0.55
31 July to 13 August	0.66

From these head grades for the various ores the mill head grades for each year have been calculated and hence with the known recovered head grade the recovery.

Year	*Ore Type*	*Milled (tons)*	*Head Assay (% Sn)*	*Recovered Head Assay (% Sn)*	*Recovery* (%)*
1944	North Valley	12,980	0.94		
	Greisen	4,880	0.8		
	Pig Flat	28,630	0.3		
	Total	46,490	0.53	0.20	40
1945	North Valley	12,640	0.8		
	Greisen	1,250	0.7		
	Pig Flat	29,980	0.3		
	Total	44,870	0.49	0.14	30
1946	North Valley	7,200	0.65		
	Greisen	13,600	0.4		
	Pig Flat	5,400	0.3		
	Total	26,200	0.45	0.17	40
1947	Greisen	21,790	0.51	0.14	30

*to the nearest one significant figure

During 1947 a split up of the overall figure is necessary in order to assess the new circuit which was stated to have only operated satisfactorily during the last two weeks although it had operated since early in May.

Period (1947)	*Ore Milled*			*Tin in Conc.*	*Recovered Head Grade*	*Recovery*
	Total (tons)	*Rate (t/wk)*	*Assay (% Sn)*	*(tons Sn)*	*(% Sn)*	*(%)*
1 Jan-22 Apr.	7,886	492	0.47	11.77	0.15	32
23 Apr-17 Jun.	6,067	758	0.48	7.03	0.12	25*
18 Jun-30 Jul.	5,851	977	0.55	8.36	0.14	25*
31 Jul-13 Aug.	1,984	992	0.66	4.39	0.22	33
1 Jan-13 Aug.	21,788	-	0.51	31.55	0.14	28

*to nearest 5%

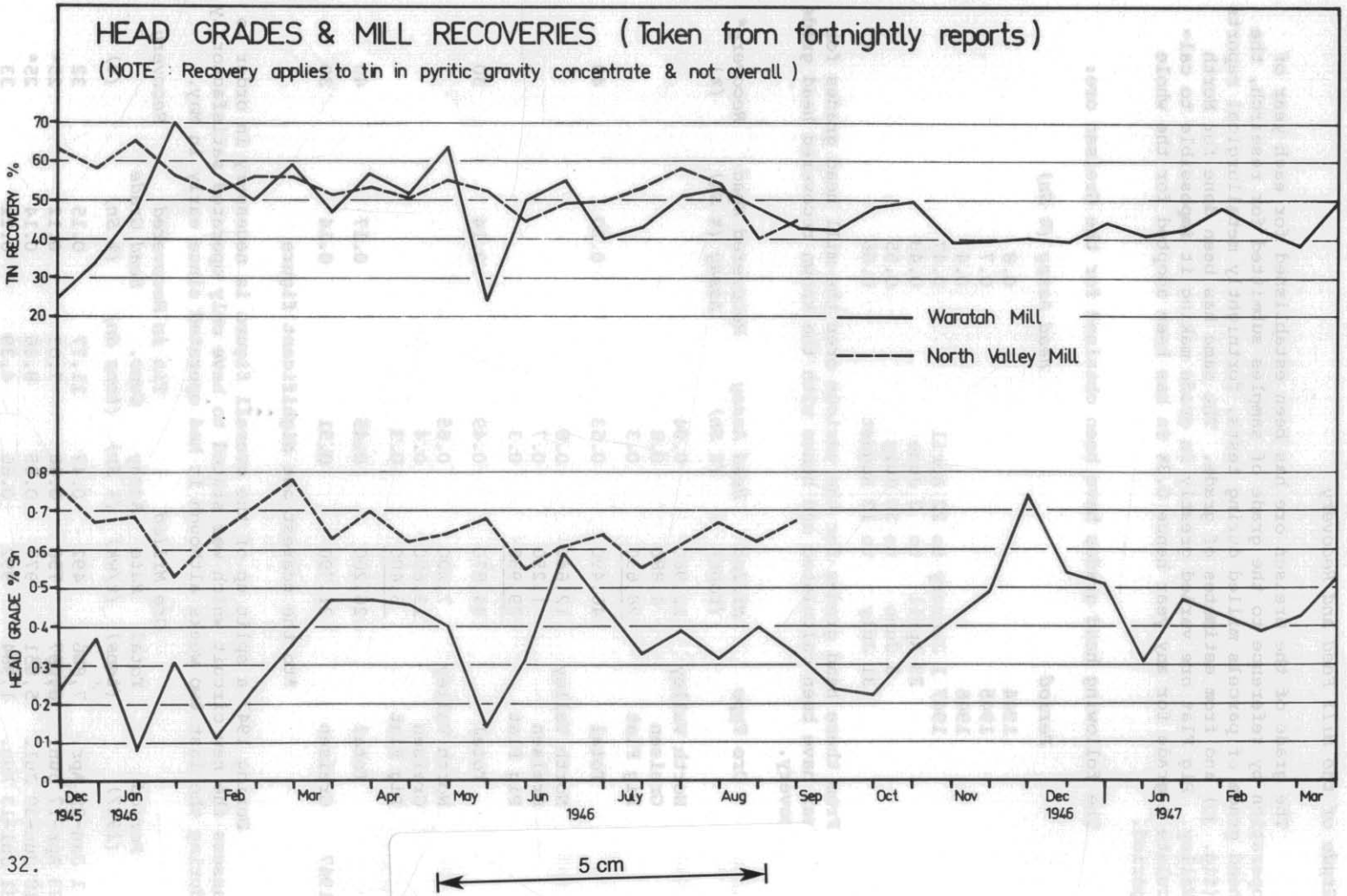

HEAD GRADES & MILL RECOVERIES (Taken from fortnightly reports)
(NOTE : Recovery applies to tin in pyritic gravity concentrate & not overall)
TIN RECOVERY %
70
60
50
40
30
20
Waratah Mill
North Valley Mill
HEAD GRADE % Sn
0·8
0·7
0·6
0·5
0·4
0·3
0·2
0·1
0
Dec 1945
Jan 1946
Feb
Mar
Apr
May
Jun
July
Aug
Sep
Oct
Nov
Dec 1946
Jan 1947
Feb
Mar
1946
5 cm

Figure 32.

Calculation of Recovery

It will be seen that the recovery figures derived above differ greatly from those published at the time. The greatest cause for difference is the 'van tin' assay which was used. This gives an indication of the recoverable tin and recoveries derived from its use will only give a measure of the recovery of the so called 'recoverable' tin. However the 'van tin' assay must be questioned as to its worth at all.

For the May tests 'van tin' and 'chemical tin' results exist for the products used to derive the recovery figures. By using the same formula but substituting chemical figures for the van figures used at the time the respective recoveries of 47 and 58% become 42 and 36%.

The recovery figures given at the time were for short duration trial runs with no account taken of inventory factors. In the early operations with simple stamp battery/table flowsheets inventory considerations would not have been as important as in the enlarged Waratah mill.

Van Tin Assays

Herman (1914) in his survey of the Australian tin mining industry devoted several pages to the problem of fine cassiterite in sulphide lode material. The problem of assaying was important in this context so he circulated to various assayers the same sample and received in return a widely ranging set of results. Herman presents these results but does not comment on the methods himself. He does say however that 'P.J. Thibault is of opinion that the valuation of crude tin ores by the vanning assay is too crude and barbarous to be worthy of detailed consideration'.

Playford (1928) in his 1926 report on Mt Bischoff Extended wrote: 'The average grade of ore is 0.75% Sn, but this may be after vanning the weighed assay sample, and is called the saveable tin. The reason for reporting mine samples in this way is plain to all metallurgists, but to report assays as such is unscientific and misleading.'

The first manager appointed by the Commonwealth, McLeod, had doubts about the use of van tin assays in determining ore reserves. Van tin assays were not used from early 1944 until 2 June 1947 when they were adopted for mill control assaying although they had been used in the determination of recovery in earlier Department of Mines test runs.

In their reports for the last period of mill operation both Pearson, the manager, and Quarmby, the metallurgist, expressed misgivings about the use of van tin assays in the calculation of metallurgical balances and Manson, Chief Chemist & Metallurgist, Department of Mines at that time, who advocated the use of van tin assays, could not define what they meant.

The temptation to substitute a simple assay for an ore dressing test is great. Renison Limited were attempting to run their new mill on 'van tin' assays as late as 1966. Knowing daily the recoverable tin in a gravity mill feed is desirable but is it desirable to use a method that can be so misleading?

Discussion

General

In test work done along comparable lines to mill-practice laboratory

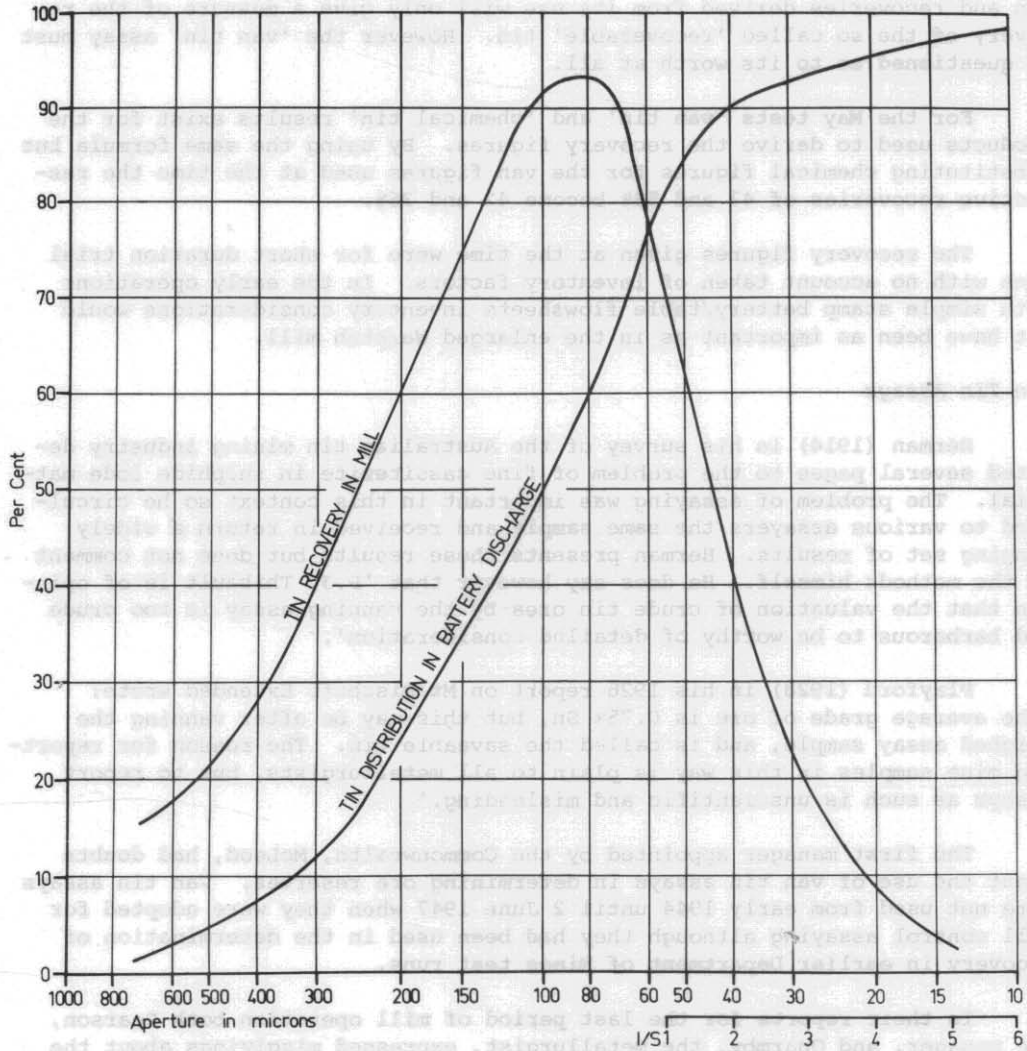
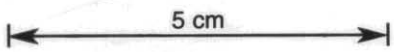


Figure 33. *Tin recovery, Greisen ore, 1944.*



results were consistently better than what was achieved in the mill. From a study of recovered cassiterite in Mt Bischoff mill products and from the test run at Renison Bell a guide to the recoverable tin in a product can be obtained by determining the tin in the product between 45 and 250 µm (quartz size (fig. 33). By regrinding any +250 µm material further recovery is possible.

In the earlier work recovery from both laboratory and mill tests meant tin recovery in pyritic concentrate hence a factor of 0.85, the roaster loss, is necessary to convert this to a figure for comparison with those obtained when the sulphides were eliminated by flotation.

North Valley Mill, 1943-1946

Operation of this mill is best summarised as follows:

Year	*Ore Treated*		*Tin Recovery %*	
	Rate tons/year	*Head Assay % Sn*	*In Pyritic Conc.*	*Overall*
1943	8,000	1.0	60	50
1946	16,000	0.65	50	40

See also Figure 32 for mill feed grades and recoveries.

Waratah Mill, 1943-1944

Mill recovery tests showed a low recovery, 20-30%, from Pig Flat ore while laboratory tests showed a range of recoveries from 30 to 60% for this ore. It was difficult to treat because of the fineness of the cassiterite and the clayey nature of some of the ore.

Work on Greisen ore in 1944 showed recoveries of between 48 and 73%. A sample upon which slime recovery estimates were based contained up to 105 µm material and hence was not true slime.

The Use of Rolls

Although laboratory work on ore samples showed rolls produced less slime than a battery their use in any reconstruction of the Waratah mill was precluded because of the structural alterations required.

Estimates Made in 1944

The figures given in late 1944 for tin recovery from Pig Flat ore was 30% but with the installation of tilting decks 50% was expected. However from the Sullivan practice of the time and the expected sizing of deck feed only a 35% recovery should have been expected.

Mill tests on Greisen ore during 1944 showed a recovery of 40% which was much below what laboratory tests showed. From sizing analyses the improvement tilting decks would make on recovery would be small. The estimation of the recovery using flotation to remove the sulphides was based on a single sample that was known to yield quite a high recovery in a pyritic concentrate.

These estimates, upon which the Commonwealth and State governments joint venture was based:

(1) over-estimated, the recovery improvement likely to be achieved

by the tilting decks,

(2) based the Greisen recovery on a sample that yielded a much higher recovery than the mill was currently achieving on this ore, and

(3) did not clearly set out what ores were to be treated and at what rates.

Waratah Mill, 1945-1946

After the joint venture had been agreed upon test work upon the ore to be milled showed only a 30% recovery for Pig Flat which merely confirmed what had been shown previously. The Renison test on Greisen ore gave a 48% recovery which was below expectations but laboratory work on the products raised this figure to 63%. The grade of ore treated at Renison was below expectations.

During 1946 reconstruction and supply difficulties hampered operations. Laboratory tests on the ore to be milled showed a 43% recovery in one test but 62 and 71% in others. Mill tests showed tabling to be inefficient. This was attributed to pyrrhotite on the tables. The tilting decks had insufficient capacity.

Milling : January-June 1947

When the metallurgist quit in January Department of Mines metallurgical staff were used to continue the testing and commissioning of the new circuit. The new metallurgist arrived during the commissioning period and had little time to get acquainted with the situation.

Fortunately the ball mill circuit had sufficient capacity to handle the coarser battery discharge which was beneficial in that it contained less slime cassiterite. However, the running of the battery discharge into the ball mill rather than the screen only ground the cassiterite further.

The table tailing contained 60% of the tin in the range 45 to 250 µm hence tabling was most inefficient.

Apart from the Renison test and some tabling tests at Waratah little had been done to upgrade the rougher table concentrates hence difficulties were experienced here.

Fortnightly metallurgical balances were suspended when commissioning started hence operations proceeded without the essential technical control data that a metallurgist needs. To this was added the use of the unreliable van tin assays to assess test runs.

Flotation was not going as well as expected but it was considered acceptable as was the table concentration.

Makeshift equipment and over-estimated capacities of some machines increased the commissioning difficulties.

Flotation

Although the earliest flotation testing in the Department of Mines Laboratories of North Valley ore had employed sulphuric acid this reagent was excluded from consideration when it was found that a high pH was necessary to depress the talc, which readily floated with the sulphides, and, which floated with it a considerable amount of tin.

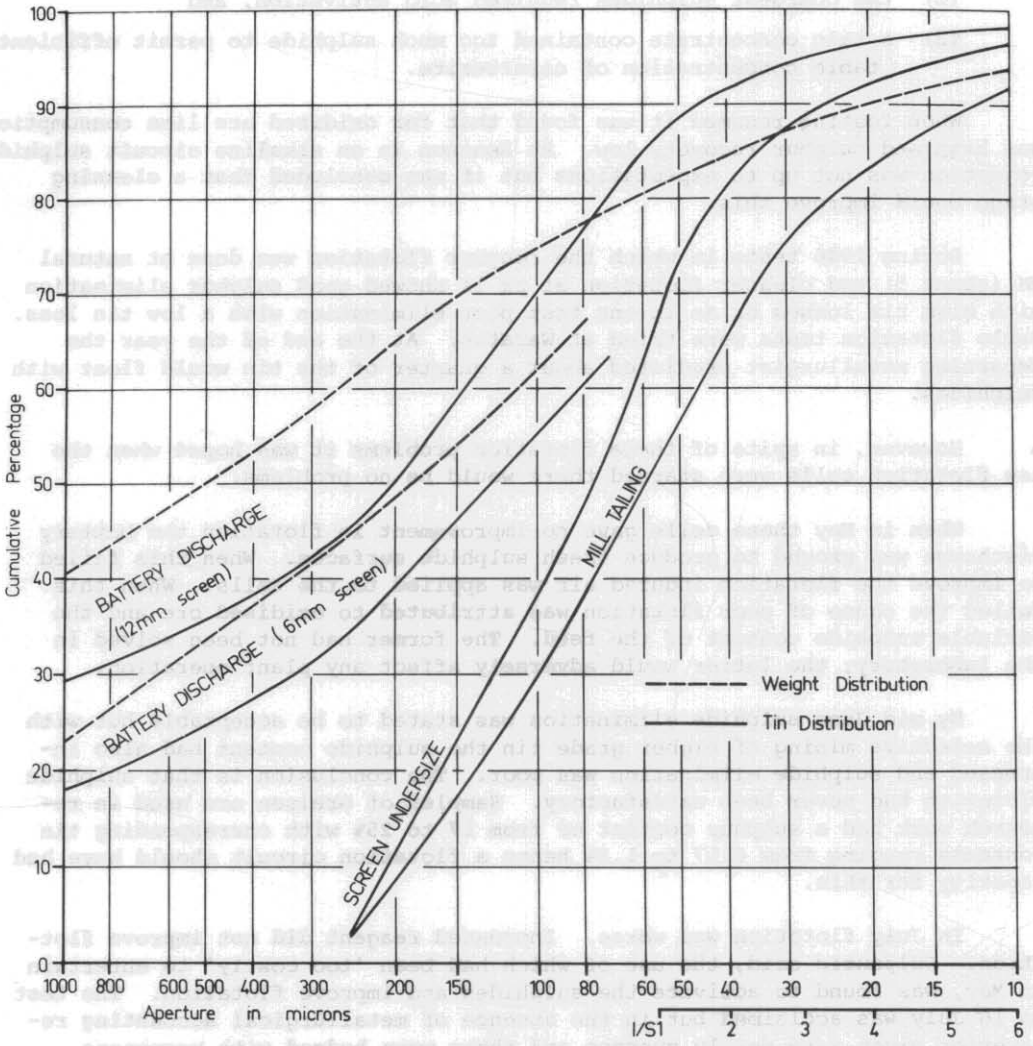


Figure 34. *Sampling tests, Greisen ore, May 1947.*

5 cm

Late in 1944 when flotation testing stopped for almost a year the position was

(1) talc could be depressed at pH 11,

(2) the coarsest sulphides required acid activation, and

(3) a talc concentrate contained too much sulphide to permit efficient table concentration of cassiterite.

When testing resumed it was found that for oxidised ore lime consumption was high and sulphur recovery low. At Renison in an alkaline circuit sulphide rejection was not up to expectations but it was concluded that a cleaning stage would improve this.

During 1946 tests in which the rougher flotation was done at natural pH (about 5) and cleaner flotation at pH 11 showed good sulphur elimination with high tin losses or as in one test poor elimination with a low tin loss. Table flotation tests were tried at Waratah. At the end of the year the departing metallurgist predicted about a quarter of the tin would float with sulphides.

However, in spite of these flotation problems it was hoped when the new flotation cells were started there would be no problems.

When in May these cells gave no improvement in flotation the battery discharge was ground to produce fresh sulphide surfaces. When this failed to improve the flotation induced air was applied to the cells. When this failed the cause of poor flotation was attributed to oxidised ore and the variable sulphide content of the feed. The former had not been solved in the laboratory; the latter would adversely affect any plant operation.

By mid June sulphide elimination was stated to be acceptable but with the selective mining of higher grade tin the sulphide content had also increased and sulphide elimination was poor. The conclusion is that sulphide flotation had never been satisfactory. Samples of Greisen ore used in research work had a sulphur content of from 17 to 25% with corresponding tin contents ranging from 0.57 to 1.2% hence a flotation circuit should have had capacity for this.

In July flotation was worse. Increased reagent did not improve flotation. Sulphuric acid, the use of which had been 'too costly' to entertain in May, was found to activate the sulphides and improve flotation. The test on 16 July was acclaimed but in the absence of metallurgical accounting recoveries given were merely guesses and these were hedged with vagueness. The use of sulphuric acid may have improved flotation and thereby tin recovery but the improvement only brought the tin recovery up 33%.

At the conclusion of operations when the Board claimed all major milling problems had been overcome, flotation cells were being installed in the tilting deck circuit to remove sulphides.

Milling, July-August 1947

Gravity concentration efficiency reached its lowest ebb in July. However, when the tin distributions in the mill tailings are studied those in August were only slightly worse than they had been in May and both were very poor when compared with the tailing from the Renison mill test (fig. 29, 34, 35).

The reason given at the time was the presence of sulphides on the

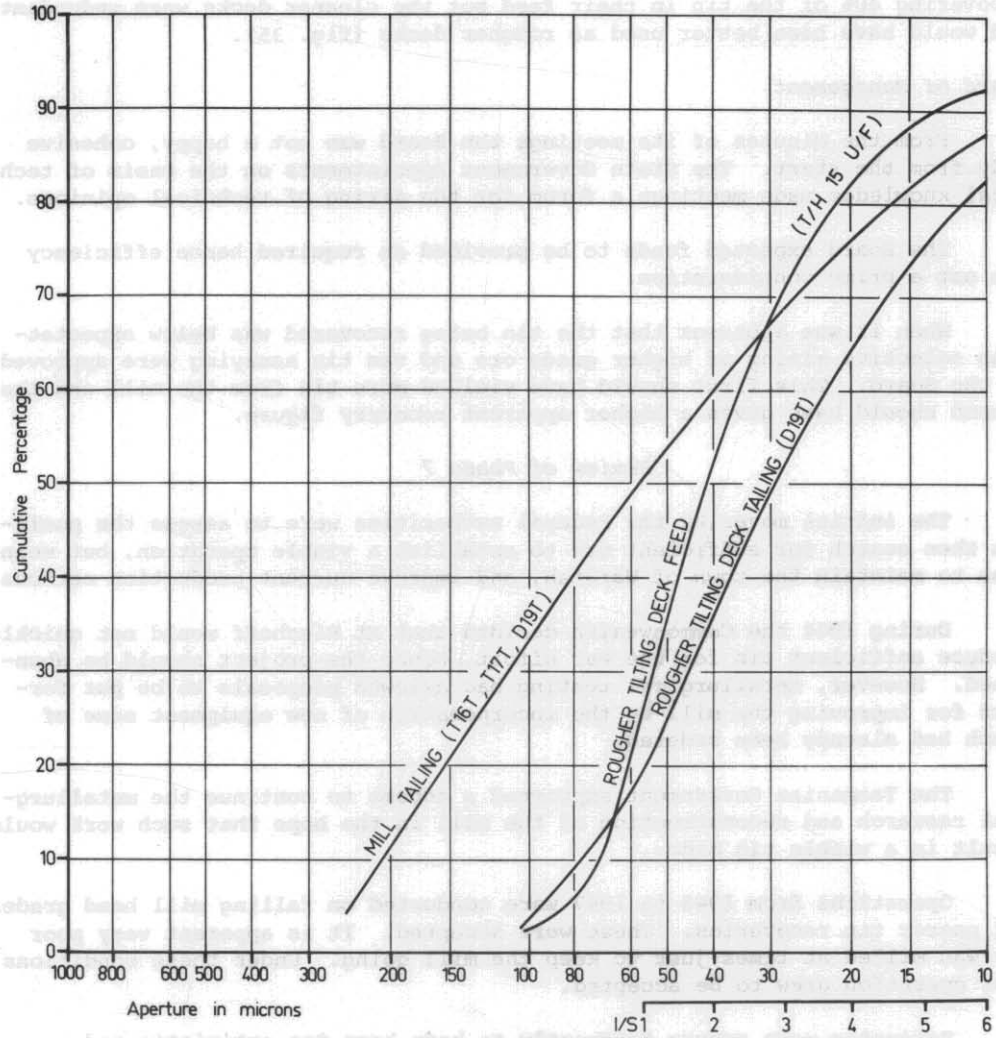
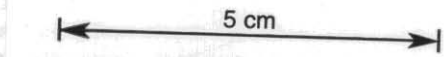


Figure 35. *Mt Bischoff. Sampling tests, Greisen ore, August 1947.*



tables. Flotation at Renison was not good hence there must have been interfering sulphides there too. It would appear that tabling practice was bad at Waratah. This would merely be a continuation of the poor tabling that test results had shown from 1945 onwards, (fig. 29 , 35).

Samples from the tilting deck circuit showed the rougher decks were recovering 40% of the tin in their feed but the cleaner decks were redundant and would have been better used as rougher decks (fig. 35).

Board of Management

From the Minutes of its meetings the Board was not a happy, cohesive body from the start. The State Government appointments on the basis of technical knowledge made meetings a forum for the airing of technical opinions.

The Board expected funds to be provided as required hence efficiency was not a prime consideration.

When it was apparent that the tin being recovered was below expectations selective mining of higher grade ore and van tin assaying were approved by the Board. This first should have yielded more tin from the mill and the second should have given a higher apparent recovery figure.

Review of Phase 7

The initial moves by the Federal authorities were to assess the position then search for sufficient ore to establish a viable operation, but meantime to maintain the town of Waratah, and improve current production methods.

During 1944 the Commonwealth decided that Mt Bischoff would not quickly produce sufficient tin for the war effort, hence the project should be abandoned. However, metallurgical testing had allowed proposals to be put forward for improving the mill by the incorporation of new equipment some of which had already been ordered.

The Tasmanian Government supported a scheme to continue the metallurgical research and reconstruction of the mill in the hope that such work would result in a viable tin mine.

Operations from 1943 to 1947 were conducted on falling mill head grades and poorer tin recoveries. These were accepted. It is apparent very poor ore was milled at times just to keep the mill going. Under these conditions poor operation grew to be accepted.

Estimates made appear frequently to have been too optimistic and unquestioned. Problems like the Pig Flat ore were shelved rather than solved. Results that should have caused further investigation were explained away.

The opportunity to test equipment in the circuit, especially continuous flotation before the revamped circuit was completed was only partially exploited.

When the new circuit commenced the essential data for metallurgical control, routine mill samples, were so much reduced that a metallurgical balance was impossible. Even these samples were assayed by vanning so that these results were quite unreliable. The new circuit was being run blind.

There were periodic results to cause concern in the two years before the new circuit ran but these were unheeded. The flotation problem was never solved. If sulphuric acid had been too expensive in May then the alleged

solution in August would have been uneconomic. The new circuit which recovered only about a third of the tin, did not perform as well as the circuit it superseded.

Apart from flotation there were other shortcomings, tabling efficiency was poor and the tilting decks were not properly used.

While the period of Commonwealth management can be considered one of investigation and maintenance and hence not expected to be profitable the joint operation was expected to become an economic success but in fact its losses were at twice the rate of those in the earlier period.

Phase 8: 1948-1971

After the government operations ceased there was little activity until production from small scale operations gradually rose from 1950 to a peak in 1953 after which it declined to about ten tons of tin per year by 1957, since when it has remained fairly constant.

In the early 1960s Mt Costigan Mines Limited, of Canada and Broken Hill South Limited became interested in Mt Bischoff. They initiated further metallurgical investigation which included chloridisation for the extraction of tin. This test work was done by Geo-Met Reactors Limited of Ottawa, Canada.

Ore Dressing Investigational Work

Mt Costigan Mines requested that gravity concentration tests be done in Launceston on a sample of Greisen ore assaying 0.5% Sn. Their objective was to produce a 10% Sn concentrate with less than 10% Fe for a hydrogen reduction process but if the iron were greater than 10% then for a chloridisation process. In either case a 90% recovery in the gravity concentrate was the objective.

In Project R.431 a gravity pyritic concentrate was produced by grinding to pass a 75 µm screen then tabling. The concentrate produced assayed 5.2% Sn and contained 66% of the tin. The tailings assayed 0.2% Sn.

In Canada Geo-Met tried flotation, magnetic separation and gravity tests to concentrate the tin ahead of the chemical attack. Magnetic separation removed 22% of the weight with only 2% of the tin. Flotation tests removed a talc concentrate containing 25% of the weight and 12% of the tin. A pyrrhotite concentrate was made containing 38% of the weight and 13% of the tin. The flotation tailing then contained 63% of the weight and 75% of the tin and assayed 2-3% Sn. It was noted that at pH 5.2 the talc contained 30% of the tin but at a higher (unstated) pH only 12%.

In view of the pyrometallurgical operations to follow, elimination of the talc was considered most desirable as this operation would require the presence of pyrrhotite but an excessive amount could be removed by magnetic or flotation treatment.

Chloride Volatilisation

The advantages of this process relative to the Mt Bischoff ore were stated to be:

(1) The selective volatilisation of tin from large masses of gangue and iron,

(2) the almost complete extraction of the tin, and

(3) the use of low grade ore or concentrate, thus eliminating losses in beneficiating to high grade concentrates.

The process comprises three stages:

(1) chloridising during which the tin is volatilised as a chloride,

(2) collection of the tin chloride with purification if necessary, and

(3) recovery of the tin from the chloride may be done either by:

(a) electrolysing the fused anhydrous chloride, or
(b) reducing the chloride with hydrogen, or
(c) precipitating chemically.

Tests were made feeding the reactor with material which was 30% -150 µm and assayed 1.5% Sn and 5% moisture, presumably a gravity concentrate. After chlorination the reactor residue assayed 0.061% Sn the tin volatilised being 95% of the feed. Pyrrhotite consumed about 5% of the total chloride but talc also used chloride. The reactor ran at between 450° and 600°C.

The tin chloride was pure. It contained negligible iron and the elemental sulphur could be removed. The recovery of tin from the tin chloride was 99% using hydrogen. The reaction was carried out in a molten tin bath at 360°C. The steam hydrolysis of the tin chloride to tin oxide was considered 'a good possibility'.

The Situation in 1971

The investigation by the large companies continues through Comstock Pty Ltd in the form of geological work. Progress with cassiterite flotation at other West Coast lode mines suggests that it could be applied to Mt Bischoff.

In the North Valley, Ringarooma Mining Pty Ltd contemplates working the alluvial that the old Company began to work. Meanwhile this company works a sluice box to recover tin from residual material on the mountain side. Quite large cassiterite nuggets have been recovered here, shades of the earliest phases of the Mt Bischoff story.

Nearby in North Valley a terraced slope littered with bleached and charred timber and rusting iron marks the site of the North Valley battery.

Further upstream is another small part-time plant. A trommel removes the coarsest stones. The tin is recovered first in a sluice box then in a jig.

The Waratah river here flows between well timbered slopes, but the valley itself with its banks of stones and debris well shows its mining history. Almost concealed by bush on its bank is an abandoned plant consisting of a water wheel driving a 3-head stamp battery and a table. Alongside these is a single rabble roaster, arch fallen, ironwork rusting, and its sheltering shed disintegrating.

Back along the old North Valley road a rusty steel tank marks the head of the penstocks that once supplied the turbines in the power station, now only a rusting shed in the trees. Water still follows the old power station

race by pipeline to this point then runs in the roadside drain to the sluice box in the valley below.

The Mount itself presents a colourful scar from which the enormous wealth was won. Beside the North Valley road amid the old workings stands a neatly built shed of variegated colour, zinc coated and red rusty iron. Inside a 2-head battery and table recover tin for part-time fossickers.

To the west, on the bush-covered slopes of Mount Bischoff are the scars of mullock rills and the site of the Mt Bischoff Extended mill. Overlooking the mill site a dozen bleached and charred timber columns still stand, but on the terraced slope below the odd blackened timber post or rusting battery shoes are all that remain. Across the small tributary gully a brick chimney rises from the hillside, below it a rubble of brick from which project three rusting roaster shafts each crowned with a gear wheel, the remains of the first large scale attempt to handle pyritic concentrates at Bischoff. On the opposite side of Tinstone Creek the pile of rusting galvanised iron sheeting, assorted steel shapes and some charred wood, marks the site of a small plant destroyed by fire although at one end a single rabble roaster still stands remarkably intact, the rabble shaft projects above the arch at an angle, the stack a few bricks short.

At Waratah in the valley beneath the town, if the widely spread buildings can be so described, the Forty Mill, a mass of weathering timber scantily clad with rusting iron, decays. Inside the stamps are seized with rust, save five which are driven by an electric motor for infrequent crushings. A plastic pipe conveys the discharge to the calcine plant across the valley. A leaking pipeline cuts across the mill conveying water to the North Valley sluice.

MOUNT BISCHOFF COMPANY TIN SMELTER

In 1874 the Mt Bischoff Company erected a tin smelter in Launceston where it occupied about half a city block with frontages on the Esplanade, Tamar and William Streets.

Production and Sources of Tin

From the beginning the smelter purchased concentrates in addition to the Company's own Mt Bischoff concentrates. However, as the Mt Bischoff production fell the purchased materials rose as the following figures illustrates:

Period	*Percentage of Mt Bischoff Tin Smelted*	*Annual Tin Production (tons)*
1877-1887	75-95	2,000-3,000
1888-1902	45-65	
1903-1914	20-35	
1915-1919	13	
1924-1927	-	c. 1,000
1928	-	960
1929	-	460

In 1888 the Company made an agreement with T.H. Kelly (Penman, 1970) who had until then operated a smelter in Launceston, to smelt the concentrate purchased by him. This agreement was metallurgically good since the ferruginous Mt Bischoff concentrates were self fluxing with the largely alluvial purchased concentrates which were siliceous. This agreement was also opportune, since it came at a time when the Mt Bischoff production began to decline (fig. 10).

For some years the Mt Bischoff Company smelted concentrates on behalf of O.T. Lempriere of Sydney (Penman, 1970) but this agreement terminated with the commissioning of Lempriere's smelter in Sydney in 1923.

The main source of alluvial tin was north-eastern Tasmania. Anything that affected production there was bound to be reflected in the production of the smelter, so that the destructive flood of 1929 and the onset of the economic depression soon after, adversely affected the smelter as the production figures show. This led to its closure in 1930.

The Early Plant

The original smelter consisted of two furnaces although at first only one furnace was used. The first charge of the Mt Bischoff Company's ore was made on 4 January 1875; three days later ore from Walker and Beecraft's claim was smelted. The second furnace was started on 25 January. A more detailed account of the early operations is given on pages 32-33.

An enforced suspension (Half-Yearly Report dated January 1878) of smelting in the middle of 1877 due to the Waratah-Emu Bay road being impassable provided the opportunity to enlarge the furnaces to allow 25 tons of concentrate per week to be smelted in each, and thereby to effect a saving on coal of 20%. A year later the average capacity of these enlarged furnaces was reported (Half-Yearly Report dated January 1879) as 25.6 tons per week with a coal consumption for all purposes of 0.95 tons per ton of concentrate smelted.

In late 1878 a well was sunk to provide water for a jig that was being installed. In 1879 the smelter capacity was increased by the erection of two more furnaces and a chimney to serve both. To improve tin recovery a jaw crusher was installed to crush slag. The Tamar and William Street wooden fences were replaces by brick walls and a blacksmith's shop was erected.

Smelting and its Problems

When Mt Bischoff was the main source of concentrate its grade evoked favourable comment from the Smelting Manager when it rose as in 1880 (Half-Yearly Report dated July 1879) but the reverse when it fell as in 1881, (Half-Yearly Report dated July 1881) because the smelter aimed to recover in ingots within 2% the tin in all concentrates assaying over 70% Sn.

The Mt Bischoff Company had earned a reputation for its tin quality. In 1881 samples were sent to Johnson, Mathey and Company in England for check assays (Half-Yearly Report dated January 1882). The Company's assays, which averaged 99.89% Sn, were verified. However, the Company was encountering smelting problems. Concentrates from North Valley contained arsenopyrite which required special attention. The iron content of the Mt Bischoff ore made it harder and therefore more costly to smelt.

Progress in the 1880s

In 1880 a clean up of accumulations under the furnaces from the previous six years yielded 20.8 tons of tin. In 1882 a new furnace for the treatment of 'impure ore' was built. The boiler needed extensive repairs in 1884 when all machinery connected with the Chilian mill was said to be in good order.

In 1887 alterations began. The older furnaces were not maintained and accumulations of impure tin and stanniferous material rose. In 1889 four small chimneys were replaced by two larger ones. Presumably the original furnaces had individual stacks but the improvements resulted in arrangements similar to the installation of 1879.

I have drawn on Clark (1904) for a description of this plant. The furnaces were of Cornish type with an oval hearth of interior dimensions 13 ft 3 in x 9 ft 8 in with an adjoining fire box of inside dimensions 4 ft 6 in wide x 4 ft long. The external dimensions were 12 ft 9 in wide x 15 ft long plus an additional 6 ft 6 in for the fire box. The arch was 18 in above the floor. There were two charging and rabbling doors on one side and a tap hole on the other. Beneath the hearth, which needed to be well made to prevent loss of tin, was a vault to catch any leaking tin. A hearth lasted three months. The reverberatory furnace charge was 2.5 tons of concentrate plus 0.5 tons of coal. When the charge melted, the tin was tapped into a float. The reverberatory furnace was then strongly heated for an hour and the slag tapped and set aside for retreatment. A fresh charge was then put in the furnace.

After standing in the float for an hour the dross was skimmed then the molten tin was ladled to a refining kettle where it was poled (a green pole being plunged into the molten metal). The dross so formed was skimmed and the tin cast into ingots which weighed 75 lb for export and less for the Australian market. The purity was checked by casting a bar 6 in long with a square section 2 in x 2 in. This was cut half way through at the middle then the two ends were bent back on each other. A fine silky fracture indicated pure metal.

The tin in the slag was present as a silicate. These slags were reduced by charging 1.3 tons of 2-inch slag, 0.3 tons of coal, 0.05 tons of lime and some scrap iron into a furnace which was then kept at high temperature for 10 hours after which it was tapped. The slag, which assayed 5% Sn, was discarded and the metal, an alloy of iron and tin known as hard-head, was returned to the original concentrate smelting charge.

Research in 1904 and 1905

In 1904 George Latta the Smelting Manager was replaced by F.B. Jackson and in addition a metallurgist, John Millen was employed. Millen (Half-Yearly Reports dated July 1904, January 1905), who later became mine manager, was engaged on research into the metallurgy of tin as well as being responsible for routine assaying.

This research work continued in 1905 (Half-Yearly Report dated July 1905). At the smelter it was foreseen that the Mt Bischoff concentrates would in future contain impurities that would have to be eliminated either from the concentrates before smelting or after smelting in the refining of the metal.

Test work aimed at removing from the concentrates sulphur, arsenic,

molybdenum and tungsten. Electrolytic and chemical methods of refining the crude metal were investigated. In addition to this, research had led to the use of a different variety of coal that had reduced the tin tenor of the slag. Effort was also directed to the consideration of labour saving appliances that would cut costs.

Alterations in 1908 and 1909

During this two year period (Half-Yearly Reports dated July 1908, January 1909) the six old reverberatory furnaces were replaced by four larger furnaces. The stack built in 1879 was replaced. A leaching plant (Half-Yearly Report dated July 1909) was installed to remove copper from the ores.

The Plant After 1909

For information on smelting practice in the last phase of operations I am indebted to Claude Penman (1970) the last Smelter Superintendent and Metallurgist.

The four enlarged *reverberatory furnaces* were coal fired. The fire box and furnace roof were of silica brick and the furnace hearth and walls of fireclay refractory bricks. The *hearth* was 20.5 ft long and 10 ft wide giving an area of 190 ft^2. The *firebox* was 4 ft long and 6 ft wide with a grate area of 24 ft^2. The overall dimensions, including the firebox, were 28 ft x 13 ft. There were three charging doors on one side with the tap hole and a rabbling door on the other. The bridge between firebox and hearth was air cooled.

Beneath the hearth was a *vault* in which leaking tin was collected and baled for refining. Beneath the firebox an *ash pit* allowed access to the grate bars.

Metal from concentrate smelting, reduction of first slags and liquation was tapped into the *float*, cooled and skimmed before transfer to the refining kettles. The floats consisted of a mild steel shell, 5.25 ft x 3 ft, with rounded ends. This was lined with firebrick. Its capacity was 3-4 tons of tin.

Slag from the furnace was tapped into a *settler* or *dummy*. This was a mild steel shell 4 ft x 2.25 ft at the top, with rounded ends and tapering to the bottom. It was lined with firebrick. There was an overflow spout. Cast into the slag were eye bolts. The solid slag was raised by these to recover the hard-head that had settled from the slag.

The *refining kettles* were of cast iron with a hemispherical base and slightly conical sides and had a capacity of 5 tons. The kettle was coal fired. Above the kettle was a cage with which the green poles were plunged into the molten tin.

The *refined metal* was cast into export ingots, 75-80 lb, or medium (28 lb), or small (20 lb), ingots for the Australian trade. As each ingot was poured the surface was skimmed to obtain a good appearance.

An Edwards Roaster was used for roasting pyritic concentrates, and those containing arsenopyrite in which case sodium chloride was added to assist arsenic elimination. The roaster was fed by a screw feeder, the calcine was elevated by a bucket elevator. The furnace gases passed through a large number of settling chambers before being discharged up a 60 ft stack.

The *copper leaching* plant consisted of four square wooden vats heated by lead steam coils. Roasted concentrate was fed to this plant.

When *furnace linings* etc. were replaced the old linings were put through a Blake crusher then further reduced by Cornish rolls in closed circuit with a screen to jig feed. The jig recovered the metallics which were returned to the circuit.

The coal used in smelting was slack from New South Wales. Between 600 and 1,000 tons could be held in the open storage area. This was sufficient to tide the smelter over the New South Wales coal strikes in 1910 and 1917.

Limestone, which came from Flowery Gully, West Tamar, was jaw crushed at the smelter. *Iron scrap* was mainly local tin plate scrap pressed into hundredweight bundles. *Fireclay* was obtained from the Company's pit near the Launceston aerodrome.

Metallurgy in the 1920s

The smelter received the following concentrates:

(1) High grade alluvial
(2) Alluvial seconds in minor amounts
(3) Pyritic concentrates from Mt Bischoff and elsewhere
(4) Roasted concentrates from Mt Bischoff and Mt Bischoff Extended

The alluvial impurities were mainly oxide minerals whereas the concentrates from lode mining contained sulphides, tungsten minerals, quartz and siderite. A 70% Sn concentrate contains 11% of impurity.

The sulphur, arsenic and antimony were removed by roasting, sodium chloride being used to assist the arsenic elimination. Hydrochloric acid was used to leach the *copper* and rarely *bismuth*, from the calcine. Bismuth was recovered.

The charge was made up by the metallurgist whose experience of the various concentrates allowed him to successfully blend them without chemical analysis of the impurities. Only a little limestone was used for fluxing during slag smelting. The reducing iron for concentrate smelting came from returning refining kettle skimmings, hard-head from second slag smelting and liquation dross.

The *concentrate smelting* charge was:

	lb
Blended concentrate	8,960
Coal (variable)	1,440
Return ferruginous material about	2,000

The firing cycle was completed in eight hours. After seven hours the crude tin was tapped into the float, allowed to cool then transferred to the refining kettle. The slag remained in the furnace for a short time during which its temperature was raised. It was then tapped via the settler into slag beds. The dummy was then removed and any tin in the settler transferred to the refining kettle. The slag from this charge, the First Slag would weigh 2,300 lb and assay 21 to 27% Sn.

The *First Slag* was then smelted with the following:

	lb
First Slag	6,720
Coal	800
Limestone	448
Scrap iron (tin plate)	112

The firing cycle was completed in eight hours. The tin silicate was reduced by the iron as only small amounts of limestone were employed. There was no data on whether higher lime slags would have been more economical. The crude tin was cast into ingots as it was too high in iron for refining. The slag as in the concentrate smelting was reheated then tapped via the settler to slagbeds. From the original concentrate smelting charge about 2,000 lb of Second Slag assaying 8% Sn was produced. Any metal in the settler was cast into ingots.

The *Second Slag* was then smelted with the following:

	lb
Second Slag	4,928
Coal	640
Limestone	448
Scrap iron (tin plate)	112

This too was run on an eight hour cycle. The metal cast (*'hard-head'*) was high in iron and often returned to the concentrate charge. The slag after further heating was tapped via the settler to slag beds. Any settler metal was returned to the concentrate charge. The slag produced from this operation was discarded. It assayed 2.5-3.0% Sn and 9% CaO. The amount of *Discard Slag* from an original concentrate charge (8,960 lb) was 2,100 lb.

An analysis of a discard slag in early 1930, when a higher than usual lime addition was being used, was:

	%	*Remarks*
SnO	2.93	(Sn 2.31%)
SiO_2	37.80	
TiO_2	4.01	
FeO	26.60	
MnO	0.46	
Cr_2O_3	0.32	
Al_2O_3	10.77	
CaO	12.64	
MgO	0.57	Moisture, alkalies, fluorine etc.,
P_2O_5	0.46	were not determined
S	0.20	
Total	96.4	

The hard-head assayed 70% Sn, 30% Fe.

If the metal from concentrate or First Slag smelting contained more than 1% Fe (*i.e.* 5.2% $FeSn_2$) it was kept for *liquation*, which was carried out in the smelting furnaces. The furnaces were cooled to 300-400°C then charged with ingots. The temperature was then raised to above the melting point of tin (232°C) when the uncombined tin ran out of the tap hole into the float. Then the temperature was raised to 500°C when the $FeSn_2$ broke down to FeSn thus releasing more tin. The tin from the tap hole was transferred to the refining kettle, the dross remaining in the furnace was removed through the stirring door with rabbles. It was then returned to the concentrate charge.

Fire refining was done to remove iron. Dilution was an aid to refining. Cooling the metal in the float to 400°C allowed a dross to form. In poling the boiling action and release of sap and moisture as steam reduced the metal temperature to a point of very low iron solubility. The dross was skimmed off. Some dross would sink in the kettle. After casting the top part of the kettle, a further skimming took place before the lower part was cast. Casting purity was controlled by casting and bending the small 2 in x 2 in x 6 in ingots.

Discussion

The production records are incomplete. Nothing is known of the earliest production. However, the State tin production (Secretary for Mines Report for 1930) during the twenties should indicate how the smelter fared. Production figures were:

Year	*Tons of Tin**	*Remarks*
1919	1,580	*metallic tin
1920	1,310	
1921	790	
1922	679	
1923-1928	1,100	steady production during these years
1929	640	
1930	512	

From these figures it appears the Mt Bischoff smelter ceased to buy appreciable concentrates from outside Tasmania after the Great War. The falling production then must have reduced smelting activity and no doubt there was competitive buying indicated by Lempriere's Sydney smelter being commissioned. Therefore it seems likely that production was about 1,000 tons in 1920 falling to about half that in 1922 then rising to about 1,000 tons again until 1928.

In 1929 there were great floods. 'In the North-Eastern District few mines escaped serious damage and the Briseis, one of the biggest tin mines of the State, suffered enormous damage' (Director of Mines Report for 1929). The Briseis sent its tin to Launceston. At the end of 1929 the Great Depression, had begun. The smelter was closed.

The smelter record contrasts with that of the mine. In order to make up for the declining production from the mine it must have pursued an active campaign to secure concentrates.

The early problems experienced when mainly Mt Bischoff ores were smelted, were largely overcome when large scale purchasing of alluvial concentrates began in 1888. The smelter capacity was increased to cope with the increased Bischoff output in 1879. Alterations were again made in 1887 but from then until 1908 when in line with the improvements at Waratah the six reverberatory furnaces were replaced by four larger ones, few changes were made. After 1909 there were no further changes.

Improvements were made in the early days (jig and jaw crusher), but I have not found when the roaster was installed. Some roasting facility should have been available in 1881 to cope with the arsenopyrite mentioned, but North Valley arsenical concentrate was rejected by the smelter in 1892 indicating no roasting facilities.

In 1904 research began with a view to extending the ability of the

smelter to handle concentrates with impurities. No doubt this was prompted both by the declining grade of the Waratah concentrates (fig. 10) and to give the smelter ore buyers a wider buying scope. It is apparent that the approaching crisis at Mt Bischoff was foreseen by the smelter staff. It is also apparent that they were aware of the necessity for the smelter to be kept up to date in order that it could remain economically viable.

Possibly ore buying outside the State was suspended by the Mt Bischoff Board as an economy measure when profits fell in 1919. On their performance elsewhere it is unlikely that capital was forthcoming in the twenties hence the smelter languished and the double disasters of 1929 finished it.

Review of Smelting Operations

Although the smelter was built to service the Mt Bischoff mine it soon became a custom smelter because this was necessary to improve metallurgy and maintain the production level.

One is led to believe the smelter was a profitable business but not a bonanza. Concentrates were brought into Tasmania for smelting thus this smelter set the pattern to be followed later by zinc and aluminium smelters. However, it appears the Mt Bischoff Board were not interested in conducting a smelting business. With reduced profits in 1920 it appears they restricted purchasing of concentrates to Tasmania thereby strengthening competing smelters on the mainland.

REVIEW OF MOUNT BISCHOFF METALLURGY

Evolution of a Flowsheet

Sluice boxes on Mt Bischoff were used to recover the tin initially. The cassiterite from these boxes, after some hand dressing at Waratah, was sent to the smelter. Any composite grains and agglomerated lumps were stockpiled until a battery was erected in 1876.

With the batteries came jigs and with the expansion in 1879 the first buddle to treat jig tailings. Both the sluice and mill tailings must have contained a lot of tin as efforts to recover this began in the early 1880s.

Slime Sheds were erected to treat the battery tailings at Waratah and further down the Waratah River below where the natural drainage from the Mount joined it, the Ringtail plant was built. When tin was found below this the Catchem plant was erected close to the lease boundary. Thus evolved in 1889 the general flowsheet (fig. 15).

Waratah Mill Operating Prior to 1907

The percentage distribution of the plant tin production has been calculated for the 1904 plant (fig. 12) from 1894 and 1896 data.

Primary jigs	(Items 3, 4, 24 and 26)	68%
Other jigs	(Items 6 and 28)	7%
Slime plants		25%

The primary jig concentrate required no further upgrading. The slime tin production was not well documented; the impression gained is that it was rather a haphazard business that made money. The main production from the jigs required little equipment and probably not a great deal of labour either.

Regrinding of the buddle concentrate began with the installation of a Chilian mill in 1885. The slime tin recovery was made by passing the pulp through numerous settlers the sediments from which were periodically fed to round tables from which a concentrate was taken for kieving to a sale grade product.

The plant as a whole was inefficiently run: witness the lost time due to lack of water, half shoes in summer on the batteries, absence of maintenance and the tin losses. In 1894 Kayser and Provis refer to the higher losses from the Brown Face, which was the principal source of tin, due to the presence of hematite.

Of the plant equipment the jigs were probably reasonably efficient in collecting the coarser tin but they would not recover much below 300 µm in size. The buddles were most inefficient and were replaced immediately their performance was assessed by Millen. The round tables could not have been very efficient either although finding another machine to treat slime tin was a problem. Mackay's notes show a 35% recovery on a round table treating low grade feed.

The Down-River Plants

Both the Ringtail and Catch'em plants recovered quite a lot of tin in their first years of operation. Doubtless this was mainly from sluice box tailings. However after sluicing stopped in 1893 and the scouring of the river bed by the 1894 flood the Ringtail production was cut in half and the Catch'em ceased operation. After the flood the Ringtail production can be attributed to tin in the Waratah mill tailings.

Mill Recovery to 1907

Before 1907 there was no control of the mill by sampling and assaying, hence no metal balances were made.

From the comments made about operating practice losses of fine tin occurred continually, with periods when a high clay content of the ore made mill pulps viscous and greater losses inevitable. Even when the machines were later assessed for efficiency, figures were not given. Under such circumstances any recovery must be a guess, but it is unlikely that more than a third of the -300 µm tin was recovered. Using this the following tin loss has been calculated for the period before 1907.

		tons Sn	
	Total production	44,000	
	Jig concentrate	33,000	(assumed 75% of production)
Therefore	Slime concentrate	11,000	(by difference)
Therefore	Total slime tin in ore	33,000	
Hence	Total tin in ore	66,000	

From the above the overall recovery was 66% and the loss of -300 µm tin 22,000 tons.

Reorganisation of the Mill

While the Waratah mill was being reorganised a new mill was built at Mt Bischoff Extended in 1908. This mill used screening instead of classification for the preparation of gravity feed and made some pyritic concentrate for roasting. The 1910 experimental mill at Waratah followed this practice.

In the alterations at Waratah buddles were replaced by Wilfley tables, Chilian mills by grinding pans and vanners were installed but in the rest of the circuit, concentrates were made on tables at stages throughout the circuit, *e.g.* Tables T8, T13, T18 and T26 in Figure 16. In the Slime Sheds the rejection of a waste product at each processing stage indicates stage, and probably better, recovery of the tin.

The equipment was better employed hence throughput was increased without additional plant.

In 1910 the Mt Bischoff Extended reverted to classification of table feed which was followed by the new 1913 Waratah mill but in it, screening was retained for the jig tailing retreatment.

About this time automatic samplers were put into the circuits. This indicates that metallurgical balances were being calculated, but no results were published.

The use of vanners in the 1913 mill suggests that they were now considered better than round tables but it should be noted that round tables had replaced vanners at Mt Bischoff Extended in 1926. Playford was unable to compare the efficiencies of round and Wilfley tables, but he appears to have had doubts about the round tables.

The reorganisation raised the milling rate from 100,000 tons per annum to 200,000 tons but only for six years, then it fell to the former level for a further six years when milling ceased during a period of low tin prices. After that milling for a further five years at 70,000 tons per annum completed active milling by the Company.

Treatment of Sulphide Ores

The Mount Bischoff Company built a plant to treat North Valley sulphide ores, but efforts to successfully work this plant between 1892 and 1894 were unsuccessful. Nothing was done about treating sulphides until 1910, although Clark (1904) mentions the necessity of doing so if the mine were to endure for the oxidised ore was being exhausted.

The Mt Bischoff Extended began producing a gravity pyritic concentrate as the first stage of concentration in 1910. This was roasted then the calcine gravity concentrated. This method was used there for the following twenty years.

However, the Mt Bischoff Company did not begin stock-piling pyritic concentrate until 1919. Roasting began in 1921. This method remained in use until 1947.

Playford mentions flotation in 1926, the Mt Bischoff Company sent a sample for testing in 1940 but said on receiving the result that, its roasting process was more economical, a conclusion also reached by Dunkin in 1943.

Flotation tests were done on various ores from 1943 to 1947. A test parcel of ore, put through the Renison mill in October 1945, probably gives the best idea of flotation performance for the tests done in the Waratah Mill between May and August 1947 were poorly documented, and the operation at that mill was inefficient.

Laboratory tests showed flotation to effectively eliminate sulphides from the ore but talc which was closely associated with cassiterite tended

to float with the sulphides. Depression of the talc by using high pH was effected in the laboratory but in plant tests the sulphides carried excessive tin or they were not completely eliminated from the gravity feed. Canadian tests in 1963 also encountered this flotation problem.

The Government Operation

From 1921 the Company operated the roasting and calcine concentrating plant. When its milling operations ceased in 1929 it roasted pyritic concentrates on a custom basis for tributors. The Waratah mill was reduced to a simple battery crushing and concentrating circuit used by a tributor. The Government installed new equipment and a complex circuit.

The parcel of ore processed at Renison Bell (see fig. 28) was subjected to a ball mill/drag classifier grinding circuit, flotation to remove the sulphides and hydrosizing to prepare the table feeds. The foregoing were new to Bischoff ore dressing, but the double round table in that circuit was an echo from the past.

In the reconstructed Waratah mill the coarseness of the battery discharge (fig. 34) virtually eliminated slime tin at this point. Grinding was in a ball mill/screen closed circuit, the product from which (fig. 34) was better as regards tin distribution than that of the Renison test classifier overflow (fig. 29).

Flotation incorporated a cleaner stage, which was absent at Renison. Tilting decks were installed to recover fine tin. The flotation circuit never operated successfully hence sulphides interfered with all subsequent gravity operations. The tin distributions in the screen undersize and mill tailing (fig. 34) confirm this poor gravity concentration. Thus the tilting decks which were stated by Wilkins to have insufficient capacity for a cleaning stage were not given a fair test.

Testing in 1963

A new approach was brought to Mt Bischoff by Geo-Met in 1963. This was the application of a pyrometallurgical treatment with a high recovery of tin to a gravity sulphide concentrate but recovery of tin in this concentrate was low.

Tin Losses into the Waratah River

It has been shown that up until 1907 an estimated 22,000 tons of metallic tin was lost in tailings discharged into the Waratah River. The only significant production since then was from 1908 to 1928 during which 8,000 tons of tin were produced with a mill recovery of about 50%. Thus the total metallic tin discharge into the river was 30,000 tons which was contained in about eight million tons of gangue. During the latter period the tin would have been finer than the earlier period but more than half of this tin should be in grains sufficiently coarse for gravity concentration.

The Effect of Board Policy on Metallurgy

In the initial sluicing operations the Board policy was to extract what tin it could by using the sluice box. The tailings were run to waste. Thus the operation was one in which the most easily won tin was recovered with the minimum capital expenditure. This policy was maintained throughout the life of the Company.

In those early days of poor communications, harsh working and living conditions it was justified but could only have been sustained by the unique wealth of Mt Bischoff.

However, when the mine became very profitable only about a tenth of the profits were reinvested in capital equipment during the period to 1894. Quite a large part of this capital was used to ensure a water supply, the motive power of the mill. From 1895 to the crisis of 1906 only minimal amounts were spent on the mine and mill. This tight rein on capital caused frequent comments from the mine manager about the turn of the century regarding the need for maintenance of buildings and equipment but short of breakdowns little money was forthcoming. Such mill operation would neither enhance recovery nor personal efficiency.

This was the situation Herman, a young mining engineer, found in 1904. By then Kayser, the mine manager, appears to have grown complacent. Herman's suggestions got nowhere until production could not be maintained, then the board wanted instant answers. When Clark visited the mine in 1904, he wrote that it was easy to criticise but with a record such as Mt Bischoff's, could their methods be far wrong?

Millen was given the task of meeting the crisis, he roundly criticised the previous milling practice and certainly improved efficiency. He was given far more money to do it with than in the past, but still it was only a small part of the profits. Then adverse profitability cut sharply the money for capital works. Millen expected to do better than he did in respect of mill recovery. Soon he faced the problem of rising costs, low grade ore and a falling tin price. Depending on current earnings to finance capital works he too was restricted.

Mt Bischoff Extended, which was never a profitable operation, was technically more advanced than Waratah. It would appear the Board would not allow expenditure on a project until it was a demonstrated success. Hence Waratah technology followed what was done at the Extended.

Last century concepts of ore reserves to justify capital outlay, a stable mill head grade and production were not applied as they are today. But by the first decade of this century Mt Bischoff employed young engineers who had been trained in these terms. It was not then too late to devote a substantial amount of earnings to exploration and research with a chance of success. However, the money continued to flow out as dividends while the population of Waratah suffered the full force of changes in the price of tin. Under these conditions staff would not be easily held, good staff would not be recruited and a mine in need of technical assistance suffered.

The smelter appears to have been a successful custom operation until about 1920. If the Board had been interested in running a smelting business they would have been, one would think, in a commanding position to focus tin smelting in Australia in Launceston rather than allow smelters to be established in Sydney. Australian tin production (Kalix *et al*, 1966) would have allowed tin smelting to have continued at 2,000-3,000 tons per annum up until the present time with about half of the tin coming from Tasmania.

Subsequent Operations

In contrast to the money made available by the Mt Bischoff Board, was the wartime and subsequent government expenditure. If money was all that was needed to rehabilitate Mt Bischoff then success should have attended this operation.

The current exploration programme has brought with it the application of new metallurgical techniques to improve tin recovery, but problems remain. In the North Valley alluvial mining, using modern earth-moving equipment may present the metallurgist with the problems that confronted the early Mt Bischoff workers, namely the recovery of tin ranging in size from grains microns in diameter to nuggets of one hundred pounds weight, but now with much more gangue between the tin grains.

Conclusion

The name of Mt Bischoff is known throughout the mining world for its fabulous riches. Today Mt Bischoff is a colourful scar on a hill, while at Waratah beneath the falls the terraced slope is all that marks the old mill site where the tin was separated from the ore. A reminder of this old mill can be found in the Zeehan Museum where there is a model showing the machines in use at the turn of the century. At the touch of a switch the water wheel, stamp batteries, jigs, a buddle and a round table spring to life for a minute or two. Between these moving items the circuit is completed with spitzlutte, settlers, pipes and launders.

The Mt Bischoff Tin Mining Company Registered has been long defunct, scarcely remembered now. This contrasts with another mining and smelting company that began working a bonanza ten years after the Bischoff Company, and whose name is now a household word throughout Australia, The Broken Hill Proprietary Company Limited.

MT BISCHOFF TIN MINING COMPANY REPORTS CONSULTED

Half Yearly Reports

No.	*Date*	*No.*	*Date*
9	January 1878	40	July 1893
11	January 1879	41	January 1894
12	July 1879	42	July 1894
13	January 1880	43	January 1895
15	January 1881	44	July 1895
16	July 1881	45	January 1896
17	January 1882	46	July 1896
19	January 1883	47	January 1897
20	July 1883	48	July 1897
21	January 1884	49	January 1898
22	July 1884	50	July 1898
23	January 1885	51	January 1899
24	July 1885	52	July 1899
25	January 1886	53	January 1900
26	July 1886	54	July 1900
27	January 1887	55	January 1901
28	July 1887	56	July 1901
29	January 1888	57	January 1902
31	January 1889	58	July 1902
32	July 1889	59	January 1903
33	January 1890	60	July 1903
34	July 1890	62	July 1904
35	January 1891	63	January 1905
36	July 1891	64	July 1905
37	January 1892	65	January 1906
38	July 1892	66	July 1906
39	January 1893	67	January 1907

No.	Date	No.	Date
68	July 1907	83	January 1915
69	January 1908	84	July 1915
70	July 1908	85	January 1916
71	January 1909	86	July 1916
72	July 1909	87	January 1917
73	January 1910	88	July 1917
74	June 1910	89	January 1918
75	January 1911	90	June 1918
76	July 1911	91	January 1919
77	January 1912	92	July 1919
78	July 1912	103	January 1925
79	January 1913	105	January 1926
80	July 1913	107	January 1927
81	January 1914	109	December 1927
82	July 1914		

Annual Reports

Year Ended	Year Ended
31 December 1928	31 December 1936
31 December 1929	31 December 1937
31 December 1930	31 December 1938
31 December 1931	31 December 1939
31 December 1932	31 December 1940
31 December 1933	31 December 1941
31 December 1934	31 December 1942
31 December 1935	

SECTION 4:
Geology

D.I. GROVES

ABSTRACT

The cassiterite-sulphide deposits at Mt Bischoff represent one of a series of cassiterite-sulphide deposits (*e.g.* Renison Bell, Cleveland, Razorback-Grand Prize, Mt Lindsay) that occur in western Tasmania and have many geologic similarities.

The deposits of Mt Bischoff fall into three main categories: stratiform deposits that represent the replacement of a dolomite horizon and consist essentially of pyrrhotite, pyrite, arsenopyrite, chalcopyrite, sphalerite and cassiterite; quartz fissure lodes containing cassiterite and varied sulphides; and topazised quartz-feldspar porphyry dykes that contain cassiterite and sulphides. The ore deposits occur in a folded sequence of Upper Proterozoic-Lower Cambrian sedimentary rocks that form a faulted, complex anticlinal structure. This structure has been the locus for intrusion of dyke swarms of late Devonian-early Carboniferous quartz-feldspar porphyries that have been extensively topazised and tourmalinised. The major and trace element chemistry of these granitic rocks, combined with isotopic dating, indicates that they probably represent the most highly fractionated portions of penecontemporaneous, high-level, post-kinematic granitic batholiths occurring throughout western Tasmania. The spatial association of the polyascendant tin mineralisation with the granitic rocks, and their associated pre-joint tin mineralisation, suggests a genetic relationship.

Around the Mt Bischoff mine area is a halo of dispersed Pb-Zn-Ag mineralisation, which on the basis of field relationships is considered to be related to the same granitic activity. A uniform distribution of Co and Ni in Fe-sulphides, Cd in sphalerites, Se in sulphides, and Mn in both the cassiterite-sulphide and Pb-Zn-Ag deposits throughout western Tasmania provides empirical support for a single metallogenic province covering western Tasmania in the late Devonian.

In the cassiterite-sulphide deposits at Mt Bischoff cassiterite has generally crystallised before the earliest formed sulphide, which is arsenopyrite, followed by pyrrhotite. Chalcopyrite, sphalerite and stannite were formed penecontemporaneously and were followed by tetrahedrite, jamesonite, and finally galena with associated boulangerite. Most pyrite appears to have formed by hypogene replacement of pyrrhotite. At Mt Bischoff pre-sulphide metasomatism of dolomite at temperatures in excess of 400°C resulted in formation of tourmaline, quartz, wollastonite, chondrodite and garnet. These minerals were subsequently replaced by sulphides and talc, fluorphlogopite, serpentine, chlorite, Fe-Mn-Mg carbonates, fluorite and sellaite.

A mineralogical and isotopic lateral zonation can be demonstrated at Mt Bischoff. The zonal and paragenetic sequences are essentially parallel. Fluid inclusion studies demonstrate that the zoned sequence is partly a function of declining temperature, with initial deposition of fluorite up to 580°, decreasing to 200°C in the marginal zone. It is probable that the temperature, salinity and composition of the ore-forming fluids were constantly changing during mineralisation as an initially hot, saline fluid was mixed with cooler, less saline meteoric and connate waters, in conjunction with heat loss to the wall rocks. Ore deposition was probably initiated by increasing alkalinity of acid ore-forming fluids caused by reaction with dolomitic host rocks.

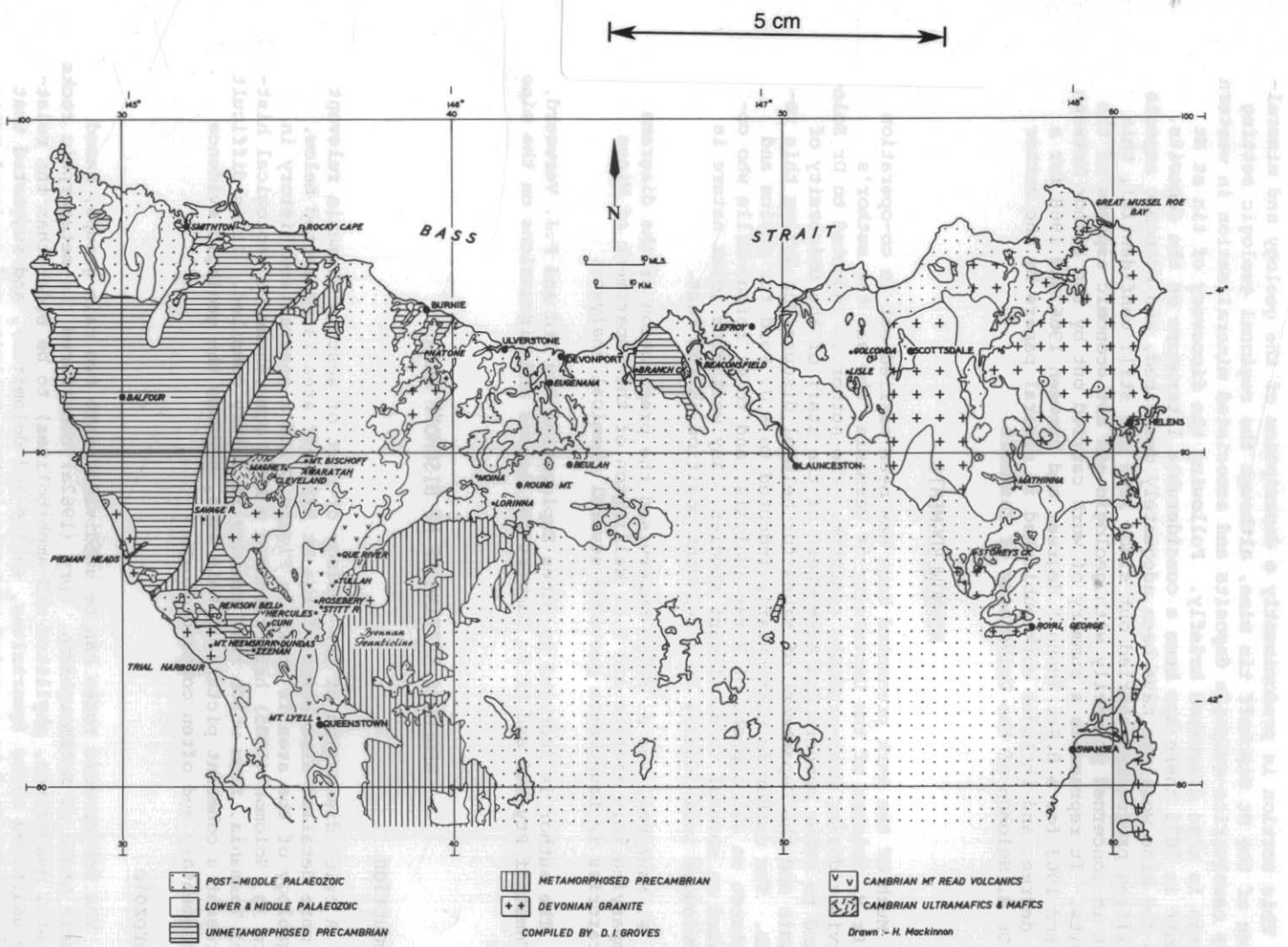


Figure 36. *Geological sketch map and locality map of northern Tasmania [Adapted from Geological Map of Tasmania, 1961].*

INTRODUCTION

This section is predominantly a description of the geology and mineralisation of the Mt Bischoff tin mine, although the regional geologic setting of the cassiterite-sulphide deposits and associated mineralisation in western Tasmania is also discussed briefly. Following the discovery of tin at Mt Bischoff in 1871 there has been a considerable literature on the deposits. Since the mine workings have been adequately described, and economic aspects of drilling programmes carried out since 1963 are still confidential, this report is concerned primarily with geologic and not economic aspects of the deposits. It represents a summary of work carried out by the author between 1962 and 1963 for a B.Sc. (Hons.) degree, and between 1966 and 1968 for a Ph.D. degree, and includes work published in several papers by the author with Dr M. Solomon of the University of Tasmania.

ACKNOWLEDGMENTS

Much of the work presented here has been carried out in co-operation with Dr M. Solomon of the University of Tasmania, who was the author's supervisor during post-graduate research. The author is indebted to Dr Solomon and to other members and ex-members of the staff of the University of Tasmania and the Department of Mines for helpful discussions during this research. The author is particularly indebted to R.J. Ford for advice and assistance on all aspects of X-ray analysis, and to G. Loftus-Hills who co-operated on several trace element studies. Any work of a joint nature is acknowledged individually in a relevant position in the text.

P. Nankivell and J. Walsh supervised the preparation of the diagrams accompanying this section and H.K. Wellington of the Department of Mines Laboratories in Launceston provided several chemical analyses.

The author is indebted to Mines Exploration Pty Ltd and P.J. Verwoerd, and Comstaff Pty Ltd and Dr J. Lambert for access and discussions on the mine area.

GEOLOGIC SETTING OF MT BISCHOFF AREA

Introduction

A brief discussion of the geology of part of western Tasmania relevant to a more detailed discussion of the Mt Bischoff area is presented below. The geology of the area is shown in Figure 37 and a generalised history in Figure 38. Solomon (1965) has presented a more comprehensive geological history of Tasmania, from which this summary is partly compiled. It is difficult to present a coherent picture of this area, because the geologic evidence is fragmentary and often contradictory.

Proterozoic

The Proterozoic rocks can be subdivided into two main groups, based on their grade of metamorphism. Spry (1962a) considered the metamorphic rocks (schists, quartzites, phyllites and amphibolites) to be older than the relatively unaltered rocks (quartzites, shales and dolomites), and suggested that they were separated by a period of basic igneous activity and regional metamorphism defined as the Frenchman Orogeny.

A belt of metamorphic rocks (Whyte Schist) extends from the Pieman River,

GEOLOGY
WARATAH – MT. MEREDITH – RENISON BELL AREA

MILES 0 1 2 3
KILOMETRES 0 1 2 3 4

GEOLOGY BY
D.I. GROVES

AMG 364383E, 5409852N

AMG 355592E, 5384345N

WARATAH
MOUNT BISCHOFF
MAGNET
CLEVELAND
MT. CLEVELAND
MOUNT WRIGHT
HEAZLEWOOD
OLD JASPER
BOXING DAY
BELLS REWARD
DISCOVERER
NEW JASPER
GODKIN EXTENDED
GODKIN
WASHINGTON
WASHINGTON HWY
CONFIDENCE
MT. STEWART
MOUNT STEWART
MT. MEREDITH
WOMBAT FLAT
SOUTH BISCHOFF
MT. RAMSAY
TRIG 455
2804
MT. LIVINGSTON
MT. LINDSAY
MT. LINDSAY MINE
PARSONS HOOD
RILEY KNOB
MT. KERSHAW
CHESTER
FARRELL JUNC.
BURNS PEAK
2166
THE PINNACLES
ROSEBERY
ROSEBERY MINE
RENISON BELL
COMMONWEALTH HILL
PENZANCE
LEAD BLOCKS
SERPENTINE HILL
DREADNOUGHT
MONTANA
BOULDER
POSEIDON
SUCCESS EXT
BON ACCORD
OWEN MEREDITH
SUCCESS
PROPRIETARY
COLEBROOK
OLYMPIC
ATHENIC
FENTON
NORTHRIDGE FAULT
PIEMAN RIVER FAULT
DUNKLEY FAULT

REFERENCE

QUATERNARY
- Qra — Alluvium and swamp deposits. Qra(Sn) – tin bearing.
- Qt — Talus deposits – (Qtu) of ultrabasics, (Qtp) of porphyry.
- Qf — Fluvioglacial deposits.
- Qm — Moraine.
- Qg — Gravel, grit, sand and clay.

TERTIARY
- Tg — UNCONFORMITY

DEVONIAN
- Db — Bell Shale.
- Df — Florence Quartzite.

SILURIAN (ELDON GROUP)
- Sac — Austral Creek Siltstone.
- Sk — Keel Quartzite.
- Sa — Amber Slate.
- Sc — Crotty Quartzite.

SILURIAN – DEVONIAN
- DS — Undifferentiated quartzite, siltstone and shale.

ORDOVICIAN
- Og — Gordon Limestone.
- Oz — Mt Zeehan Conglomerate.

UNCONFORMITY

CAMBRIAN
- Cd — Dundas Group and correlates.
- Cgc — Greywacke conglomerate – sandstone and mudstone sequence.
- Cwg — Greywacke sandstone and mudstone sequence of the Waratah River.
- Ccm — Mudstone and greywacke sequence of the Coldstream River.
- Cam — Red mudstone and sandstone sequence of the Arthur River.
- Cc — Crimson Creek Formation.
- C — Probable correlates of Crimson Creek Formation.
- Cu — Undifferentiated sedimentary sequence.
- Cr — "Rosebery Group" minus Stitt Quartzite.

UNCONFORMITY ?

PRECAMBRIAN
- PCb — Quartzite mudstone and dolomite of Mt Bischoff.
- PCo — Oonah Quartzite and Slate.
- PCs — Stitt Quartzite.
- PCw — Whyte Schist.

IGNEOUS ROCKS

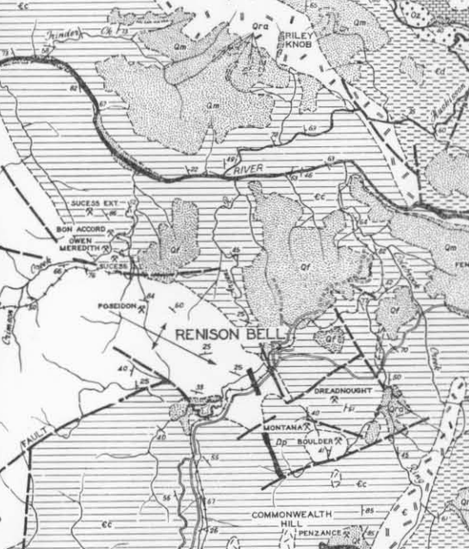
TERTIARY
- V Tb V — Basalt.

JURASSIC
- Jd — Dolerite.

DEVONIAN
- Dp — Quartz porphyry.
- Dgt — Tourmalinized granite.
- Dmg — Microgranite dyke with tourmaline nodules.
- Dg — Meredith Granite.

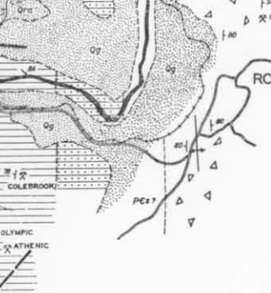
CAMBRIAN
- Cp — Predominantly quartz and feldspar porphyries (Mt Read Volcanics).
- Ci — Intermediate igneous suite (albite syenite and granophyre).
- Cb — Basic igneous suite (gabbro, Cbg dolerite Cbd spilite, Cbs and agglomerate Cba).
- Cu — Ultrabasic igneous suite (pyroxenite and peridotite).
- Cs — Serpentinite.
- C — Undifferentiated predominantly igneous rocks.

- Geological boundary – approximate.
- Geological boundary – inferred.
- Strike and dip of beds.
- Strike and dip of beds – overturned.
- Strike and dip of cleavage.
- Fold axis.
- Fault – approximate position (downthrown side indicated).
- Major jointing in granite.
- Mine or prospect.
- Trend line (Photo-interpretation).
- Trig station.
- Road.
- Vehicular or pack track.
- Railway.
- Tramway.
- Lake or dam.



SOURCE OF INFORMATION FOR GEOLOGIC MAP

1. Detailed and regional mapping D.I. Groves (1962–1967)
2. Regional and reconnaissance mapping D.I. Groves and R. Jack (1964)
3. Photo-interpretation with scattered information from L.L. Waterhouse (1914) and G. Urquhart (1966)
4. Photo-interpretation D.I. Groves (1967)
5. Compilation of mapping by Rio Tinto Australian Exploration Pty. Ltd.
6. Mackintosh 1 mile to 1 inch map sheet, Tasmanian Dept. of Mines (1966) with some modifications.
7. Compilation of traverses by B.L. Taylor (1954)
8. Zeehan 1 mile to 1 inch map sheet, Tasmanian Dept of Mines (A.H. Blissett and A.B. Dulline 1962)
9. Sketch map from information by G. Loftus-Hills and R.L. Braithwaite (1967)



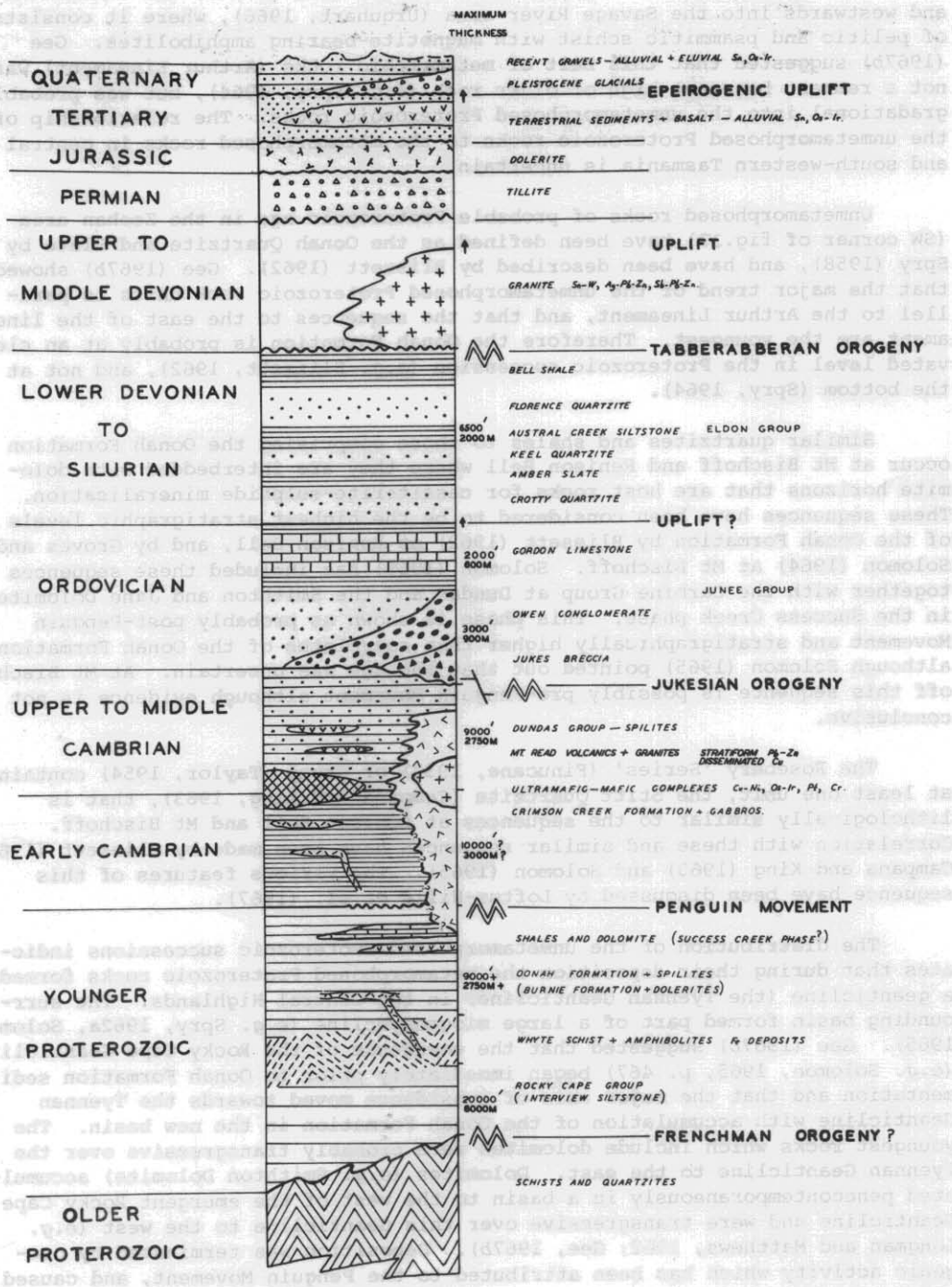
5 cm

AMG REFERENCE POINTS ADDED

Drawn:– P. B. Nankivell

GS854

Figure 37



GEOLOGICAL HISTORY OF WEST TASMANIA

5 cm

Figure 38.

along the western flank of the Meredith Granite (fig.36,37), and northwards and westwards into the Savage River area (Urquhart, 1966), where it consists of pelitic and psammitic schist with magnetite-bearing amphibolites. Gee (1967*b*) suggested that this belt of metamorphic rocks (Arthur Lineament) was not a remnant basement high of older rock (*e.g.* Spry, 1964), but was probably gradational into the unmetamorphosed Proterozoic rocks. The relationship of the unmetamorphosed Proterozoic rocks to the metamorphosed rocks in central and south-western Tasmania is uncertain.

Unmetamorphosed rocks of probable Proterozoic age in the Zeehan area (SW corner of fig.37) have been defined as the Oonah Quartzite and Slate by Spry (1958), and have been described by Blissett (1962). Gee (1967*b*) showed that the major trend of the unmetamorphosed Proterozoic rock units is parallel to the Arthur Lineament, and that the sequences to the east of the lineament are the youngest. Therefore the Oonah Formation is probably at an elevated level in the Proterozoic succession (*e.g.* Blissett, 1962), and not at the bottom (Spry, 1964).

Similar quartzites and shales to those comprising the Oonah Formation occur at Mt Bischoff and Renison Bell where they are interbedded with dolomite horizons that are host rocks for cassiterite-sulphide mineralisation. These sequences have been considered to be the highest stratigraphic levels of the Oonah Formation by Blissett (1962) at Renison Bell, and by Groves and Solomon (1964) at Mt Bischoff. Solomon (1965) has included these sequences together with the Carbine Group at Dundas and the Smithton and Jane Dolomites in the Success Creek phase. This phase is shown as probably post-Penguin Movement and stratigraphically higher than correlates of the Oonah Formation, although Solomon (1965) pointed out that the age was uncertain. At Mt Bischoff this sequence is possibly pre-Penguin Movement although evidence is not conclusive.

The Rosebery 'Series' (Finucane, 1932) or Group (Taylor, 1954) contains at least one unit, the Stitt Quartzite (Campana and King, 1963), that is lithologically similar to the sequences at Renison Bell and Mt Bischoff. Correlation with these and similar sequences have been made by Blissett (1962), Campana and King (1963) and Solomon (1965). The various features of this sequence have been discussed by Loftus-Hills *et al.* (1967).

The distribution of the unmetamorphosed Proterozoic successions indicates that during their deposition the metamorphosed Proterozoic rocks formed a geanticline (the Tyennan Geanticline) in the Central Highlands. The surrounding basin formed part of a large miogeosyncline (*e.g.* Spry, 1962*a*, Solomon, 1965). Gee (1967*b*) suggested that the emergence of the Rocky Cape Geanticline (*e.g.* Solomon, 1965, p. 467) began immediately prior to Oonah Formation sedimentation and that the major axis of subsidence moved towards the Tyennan Geanticline with accumulation of the Oonah Formation in the new basin. The youngest rocks which include dolomites were probably transgressive over the Tyennan Geanticline to the east. Dolomites (*e.g.* Smithton Dolomite) accumulated penecontemporaneously in a basin to the west of the emergent Rocky Cape Geanticline and were transgressive over this Geanticline to the west (*e.g.* Longman and Matthews, 1962; Gee, 1967*b*). Deposition was terminated by orogenic activity which has been attributed to the Penguin Movement, and caused folding of the sedimentary piles, which were transported to the south-east towards the Tyennan Geanticline (Gee, 1967*b*). The effect, intensity and exact position of this Movement in the area investigated is difficult to determine.

Albite dolerites were intruded into unmetamorphosed Proterozoic rocks at Burnie in the early stages of folding and have been dated as 700 million

years (Spry, 1962*b*). Dolerites also occur in unmetamorphosed Proterozoic rocks in the Interview River area (Spry and Ford, 1957; Gee, 1967*b*), and the amphibolites in the Whyte Schist are probably the metamorphic equivalents of these rocks (Gee, 1967*b*). No dolerites occur in the Upper Proterozoic rocks at Mt Bischoff or Renison Bell, although volcanic rocks occur towards the top of the Oonah Formation in the Zeehan District (*e.g.* Twelvetrees and Ward, 1919; Blissett, 1962), and some fragmental rocks occur towards the top of the sequence at Mt Bischoff.

Cambrian

The Oonah Formation in the Zeehan-Renison Bell area is overlain, apparently conformably, by a maximum of 3000 m of unfossiliferous purple and green mudstone, greywacke and slate of the Crimson Creek Formation (Blissett and Gulline, 1961*a*, after Taylor, 1954). This passes upwards into the fossiliferous Dundas Group, which ranges in age from lower Middle Cambrian to Franconian (Banks, 1962*a*). The Dundas Group consists of alternations of greywacke, siltstone, mudstone and shale with conglomerate and grit and is a maximum of 2700 m thick (Blissett, 1962; Elliston, 1954). It includes the Huskisson Group of Taylor (1954).

Correlation with these major subdivisions of the Cambrian succession elsewhere in the area is made difficult by the lack of fossils and coherent structural interpretation in critical areas.

The sequence along the Wilson River extends north-west for 11 km from the type section of the Crimson Creek Formation and contains the Mt Lindsay tin deposit (fig.37). It is lithologically similar to the Crimson Creek Formation and is probably an extension along strike, as suggested by Taylor (1954). The sequence extending northwards from the Pieman River just west of Rosebery to the headwaters of the Huskisson River is probably co-extensive with the mudstone and greywacke sequence of the Coldstream River shown on the Mackintosh 1 inch to 1 mile map sheet (Department of Mines, Tasmania). These sequences are also lithologically similar to the Crimson Creek Formation and are unfossiliferous. Loftus-Hills *et al.* (1967) recorded possible inter-bedding of sedimentary rocks of this sequence with the Rosebery 'Series' near Rosebery. The mudstone and sandstone sequence of the Arthur River (fig.37), which contains the Cleveland cassiterite-sulphide deposit, may represent the northern extension of the Crimson Creek Formation. It appears to be structurally equivalent to Crimson Creek Formation correlates south of the Meredith Granite. Sequences of greywacke-conglomerate, greywacke-sandstone and mudstone occur to the east of the predominantly mudstone sequences and possibly overlie them. They exhibit similarities to the Dundas Group, but are apparently unfossiliferous.

Spilites and associated pyroclastics are common in Cambrian sequences throughout the area. Numerous spilites occur with Cambrian sedimentary rocks in the Waratah district (*e.g.* Scott, 1954; Groves and Solomon, 1964; Solomon, 1964; Cox and Glasson, 1967). Large areas along the northern end of Bett's Track and the Heazlewood area (fig.37) which were considered to be predominantly ultramafic rocks (*e.g.* Nye, 1923; Reid, 1923) have been shown to contain a significant proportion of extrusive rocks including altered porphyritic and spherulitic basalts and agglomerates (Jack and Groves, 1965; Groves, 1966).

Large ultramafic-mafic igneous complexes occur in the Heazlewood-Bald Hill area north of the Meredith Granite, in the narrow, elongate strip from Colebrook Hill via Rileys Knob to the Harman River south of the Meredith Granite, and at Serpentine Hill near Renison Bell (fig.37). These complexes

contain layered pyroxenites, peridotites and hartzburgites with serpentinites, gabbros, norites and dolerites in association with basalts and agglomerates (*e.g.* Jack and Groves, 1965; Groves, 1966; Rubenach, 1967). Small gabbro and dolerite bodies which are commonly approximately concordant with the regional structure are abundant in the Arthur River sequence (*e.g.* Groves and Solomon, 1964; Cox and Glasson, 1967). Rubenach (1967) recorded that Dundas Group conglomerates overlying rocks of the Serpentine Hill Complex contain abundant detrital pyroxenes and gabbro pebbles, which suggests that the ultramafic and mafic group of this complex crystallised prior to the Middle Cambrian, and were exposed to erosion at the beginning of Dundas Group sedimentation.

Another important feature of Cambrian igneous activity was the development of a thick volcanic pile, which is now represented by an arcuate zone of acid volcanic rocks known as the Mt Read Volcanics. These volcanics are intruded by granitic bodies of similar composition. The volcanics appear to be interbedded with slates of late Middle Cambrian age in the Que River area (Gee *et al.*, 1970) although it is possible that they are in part older than this (Loftus-Hills *et al.*, 1967).

The stratiform Pb-Zn-pyrite and disseminated Cu ore bodies at Rosebery-Hercules and Mt Lyell respectively occur within this arcuate zone of volcanic rocks, and a genetic relationship between ore deposition and vulcanism has been suggested (Campana and King, 1963; Solomon, 1964). Several recent studies (Brathwaite, 1967; Groves and Loftus-Hills, 1968; Loftus-Hills *et al.*, 1969; Solomon *et al.*, 1969) have supported this relationship.

The Cambrian sedimentation included abundant vulcanism and involved a deepening of the sedimentary basin (the Dundas Trough) towards the Tyennan Geanticline and a change to sedimentation typical of eugeosynclinal development (see Solomon, 1965, p. 468). Solomon (1965) suggested that the development and destruction of local ridges and troughs was a feature of Cambrian development and that this resulted in variable, impersistent accumulation of sediment.

Sedimentation was terminated over most of western Tasmania in the Upper Cambrian by the Jukesian Orogeny, which produced folding of the Cambrian and older rocks on trends subparallel to the margin of the Tyennan Geanticline. Solomon (1965) suggested that the major feature was faulting on a similar trend which uplifted the Tyennan and Rocky Cape Geanticlines, and produced an intervening basin divided by an axial ridge of Cambrian rocks (the Porphyroid Anticlinorium of Carey (1953). The Owen Conglomerate and correlates was deposited in these basins, with lateral transgression of the younger sandstones and limestones over the initial highland areas.

Ordovician

The Mt Zeehan Conglomerate is the oldest formation of the Junee Group exposed in this area and was deposited in the western basin at the same time as filling of the Owen Basin to the east. The Zeehan Conglomerate correlate is 120 m thick in the Huskisson River area. Its relationship to the Dundas Group is obscure at this locality, but Blissett (1962) recorded that they were essentially conformable elsewhere in the Zeehan Quadrangle. The Mt Zeehan Conglomerate is apparently thinning rapidly to the north-east, and is absent north of the Meredith Granite. The Gordon Limestone is also thinning to the north, and is absent beneath Silurian sandstones at Heazlewood on the Corinna Road (fig. 37). The limestone is apparently discontinuous around the base of the Eldon Group (fig. 37). This may be an original depositional or erosional feature, although similar discontinuity of limestone in the Godkin area, north

of the Meredith Granite, is probably the result of intersection of minor NNW-trending faults with a major NW-trending fault zone. An Ordovician age for this limestone is indicated by the occurrence of *Lichenaria ramosa* (M. Clarke, pers. comm.).

Silurian-Devonian

Minor uplift of the source area at the end of Gordon Limestone sedimentation is suggested by the occurrence of coarse sandstones (Crotty Quartzite) which conformably overlie the Gordon Limestone. The whole Silurian-Devonian sedimentary sequence (the Eldon Group) represents a tectonically quiescent period.

An extensive area between the Wilson and Huskisson Rivers is covered by sedimentary rocks of the Eldon Group, which is a maximum of 2000 m thick in this area, compared with about 1800 m in the Zeehan area (Blissett, 1962). The Eldon Group is incomplete in the Heazlewood-Godkin area, north of the Meredith Granite, where it consists of thickly bedded, white, saccharoidal sandstones lithologically similar to the Crotty Quartzite, interbedded with quartzite, and conglomerate (Groves, 1966). The sandstones unconformably overlie Cambrian igneous rocks to the west and are faulted against similar rocks to the east.

Devonian

The Tabberabberan Orogeny followed the close of sedimentation in the Middle Devonian (Banks, 1962*b*). Undeformed late Middle Devonian cave deposits in deformed Gordon Limestone at Eugenana (Banks and Burns in Banks, 1962*b*, p. 185) indicate an upper limit for the age of the major tectonic phase.

Solomon (1962, 1965) suggested that Tabberabberan deformation took place in two stages. The earliest deformation was largely controlled by the Tyennan and Rocky Cape Geanticlines and consisted of differential vertical movement which produced long wavelength, arcuate synclinoria and anticlinoria. The following deformation was not influenced by the orientation of the geanticlines and resulted in structures of smaller wavelength of approximately north-west trend. Marked changes of plunge are produced by the resultant interference of these two generations of folds. Large structures of the second generation include the Huskisson and Zeehan Synclines with cores of Eldon Group rocks (see Solomon, 1965, p. 471). A pronounced axial surface cleavage is common in most rock types, except the most competent Owen Conglomerate and Proterozoic quartzites (Solomon, 1965). In the area investigated, however, cleavage is even rare in the Cambrian mudstone sequences.

Several large granite-adamellite stocks were intruded in the late Devonian, and were essentially post-folding although they have been faulted in places (*e.g.* Heemskirk and Pieman Heads). Carey (1953) has suggested that they were intruded along large scale anticlinal structures which were the first phase folds of the Tabberabberan Orogeny.

Deposits of cassiterite and wolframite with minor molybdenite and bismuthinite occur near the granite margins in fissures and stockworks (*e.g.* Heemskirk). Small lead-zinc deposits at Mt Stewart (Groves, 1966) are also adjacent to granite intrusions. Pyrrhotite-cassiterite replacements of carbonate rocks occur at Mt Bischoff and Renison Bell, where they are spatially associated with altered granitic rocks, and also at Cleveland and Razorback where no major occurrences of granitic rocks are recorded. Haloes of lead-zinc mineralisation occur around many of these tin deposits, and zoning has been postulated around the Heemskirk Granite (*e.g.* Twelvetrees and Ward, 1910;

Both and Williams, 1968), and around Mt Bischoff (*e.g.* Groves and Solomon, 1964). The position is complicated along the Zeehan - Dundas - Farell line of mineralisation where tin mineralisation occurs in several areas including Renison Bell, Razorback and Queen Hill. Solomon (1965) suggested that the lead-zinc deposits possibly formed complicated, overlapping haloes around the tin deposits in this area.

Post Late-Devonian

The post late-Devonian history of the area is represented by periods of igneous activity in the Jurassic and Tertiary and by shallow terrestrial deposition in the Tertiary and Quaternary. Some faulting is considered to have occurred in the Tertiary.

Jurassic dolerite similar to that blanketing much of central and eastern Tasmania occurs in the Pieman River some 6 km west of Renison Bell where it forms a large cone sheet in contact with Oonah Quartzite and Slate (Spry, 1958). Other sill-like bodies occur at Mt Dundas and Firewood Siding (Blissett 1962) and north of the Magnet mine (Reid, 1923), and small dykes of probable Jurassic age occur at Renison Bell and in Yellowband Creek.

Sub-basalt sediments of Tertiary age occur in the Waratah area (Reid, 1923; Groves, 1963) and contain minor accumulations of cassiterite. A maximum thickness of 60 m is recorded for Tertiary sediments in the Zeehan Quadrangle (Blissett, 1962), and similar sediments have been described from the Meredith Granite area by Jack and Groves (1965). It is probable that Tertiary sedimentation was more widespread, particularly in the Meredith Granite area, but that the sediments have been subsequently reworked or removed during the Quaternary. An extensive plateau area south and east of Waratah is covered with a series of basalt flows which have a cumulative thickness of about 15-60 m, the base being at an elevation of 550-610 m. Thin dykes of basalt have also been recorded from this area by Groves (1963).

Pleistocene glacial activity was restricted to an area north of Renison Bell, where moraine and fluvioglacial deposits occur on both sides of the Pieman River.

Recent deposits of gravel and sand occur in the present streams and along higher alluvial terraces. They consist largely of reworked Tertiary or Pleistocene deposits and contain some cassiterite (*e.g.* Stanley River, Wombat Flat, Pine Creek), osmiridium (*e.g.* Wilson River, Loughnan Creek) and monazite (*e.g.* Yellowband Creek).

GEOLOGY OF THE MT BISCHOFF AREA

Introduction

Mt Bischoff, situated about 1 km north of Waratah, is a small monadnock rising some 180 m above an extensive, deeply dissected plateau area which is largely covered by Tertiary gravel and basalt flows (fig. 37). The oldest rocks exposed are the quartzite, shale and dolomite sequence of Mt Bischoff (the Mt Bischoff sequence) which was originally called the Mt Bischoff Series by Reid (1923) who considered it to be of Ordovician age. Subsequent authors (Carey, 1953; Knight, 1953; Groves and Solomon, 1964) have suggested they are Upper Proterozoic, although Solomon (1965) suggested that they may be Lower Cambrian in part, and may be included within the Success Creek phase. Overlying this sequence are Cambrian sequences of indeterminate thickness, which consist of greywacke, mudstone, shale, chert and spilite. There is at least

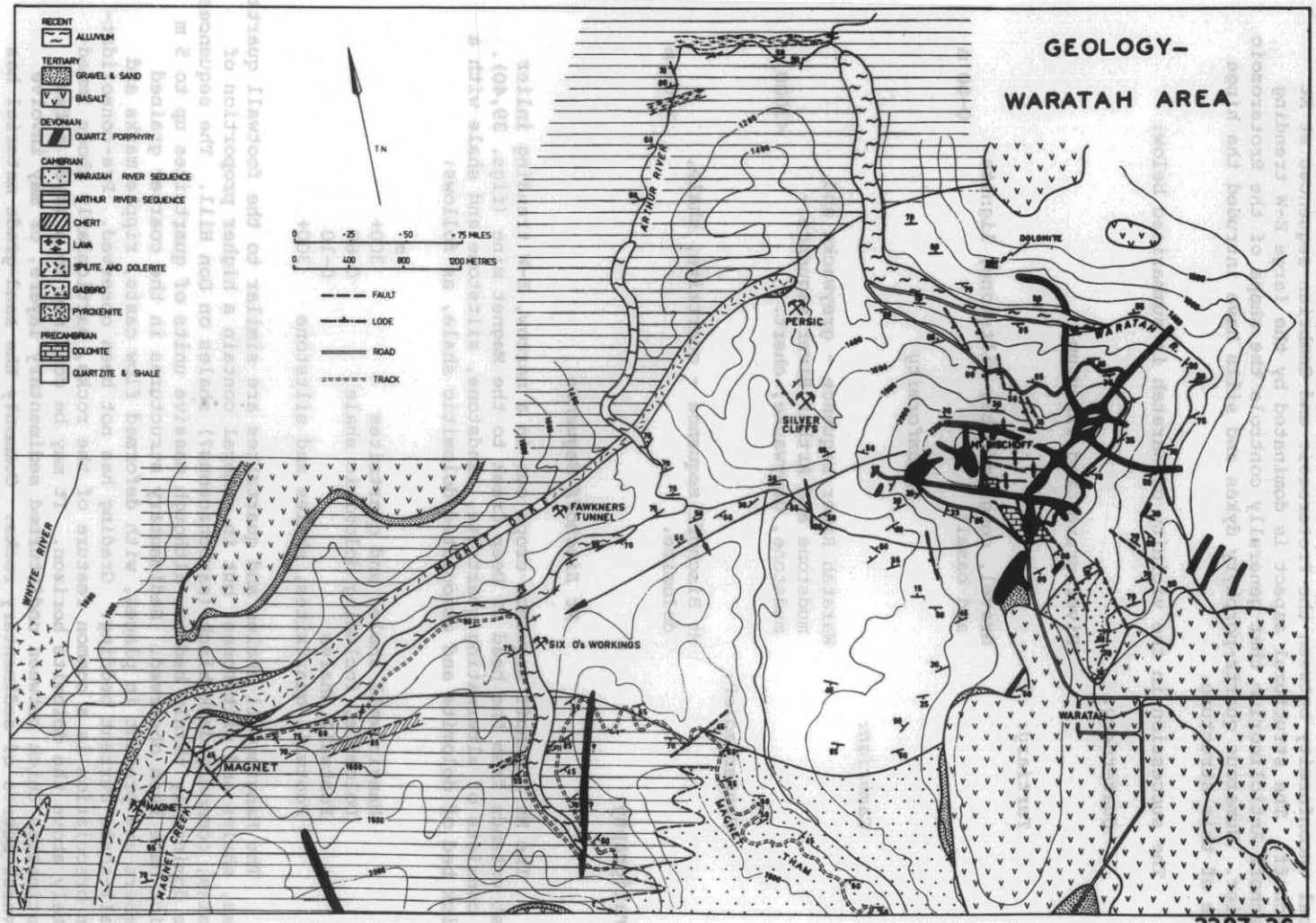

5 cm
GEOLOGY-
WARATAH AREA
RECENT
ALLUVIUM
TERTIARY
GRAVEL & SAND
BASALT
DEVONIAN
QUARTZ PORPHYRY
CAMBRIAN
WARATAH RIVER SEQUENCE
ARTHUR RIVER SEQUENCE
CHERT
LAVA
SPILITE AND DOLERITE
GABBRO
PYROXENITE
PRECAMBRIAN
DOLOMITE
QUARTZITE & SHALE
TN
0 .25 .50 .75 MILES
0 400 800 1200 METRES
FAULT
LODE
ROAD
TRACK
ARTHUR RIVER
DOLOMITE
PERSIC
SILVER CLIFFS
WARATAH
FAWKNERS TUNNEL
SIX O's WORKINGS
WHITE RIVER
MAGNET
MAGNET CREEK
MAGNET TRAM
3387-36

Figure 39.

local unconformity between the Proterozoic and Cambrian sequences at Mt Bischoff. The structural aspect is dominated by the large E-W trending Bischoff Anticlinorium that generally controls the shape of the Proterozoic inlier. Numerous quartz-porphyry dykes and sills have intruded the hinge zone of this structure.

The succession in the vicinity of Waratah is summarised below:

Quaternary:

River gravel and alluvium.

Tertiary:

Gravel, conglomerate, siltstone, lignite and basalt 0-60 m

Unconformity

Cambrian:

Waratah River sequence - greywacke and mudstone and Arthur River sequence - mudstone, greywacke, chert. <3000 m

Proterozoic?:

Mt Bischoff sequence - quartzite, shale, dolomite. >600 m

Stratigraphy

Mt Bischoff Sequence

The Mt Bischoff sequence crops out in a narrow, E-W trending inlier that extends from the head of Deep Creek to the Magnet mine (figs. 39,40). It consists of alternating quartzite, sandstone, siltstone and shale with a thick bed of dolomite and associated dolomitic shale, as follows:

	m
Hangingwall shales and quartzites	300+
Dolomite, including dolomitic shale	0-60
Footwall shales	0-10
Footwall quartzites, shales and siltstone	300+

The Hangingwall shales and quartzites are similar to the footwall quartzites, shales and siltstones, but in general contain a higher proportion of shales, commonly dark black (carbonaceous?) shales on Don Hill. The sequences are dominantly thinly bedded, although massive units of quartzites up to 5 m thick have been observed. Sedimentary structures in the coarser grained members are present in places, with deformed flow casts, ripple marks and small-scale current bedding. Grading has not been observed. Pre-consolidation brecciation is a common feature of the rocks, particularly those immediately above the dolomite horizon. It may be confined to a layer a few centimetres thick between undisturbed sedimentary layers, or may involve several metres of sedimentary rocks. Commonly the sand-grade material has liquefied and has incorporated fragments of adjacent mud layers, which retained a shearing strength during deformation. However discontinuous sand layers occur in shale matrix in places. Lateral discontinuity of beds and abrupt facies changes are typical of the sequence, and prohibit any confident prediction of stratigraphic units.

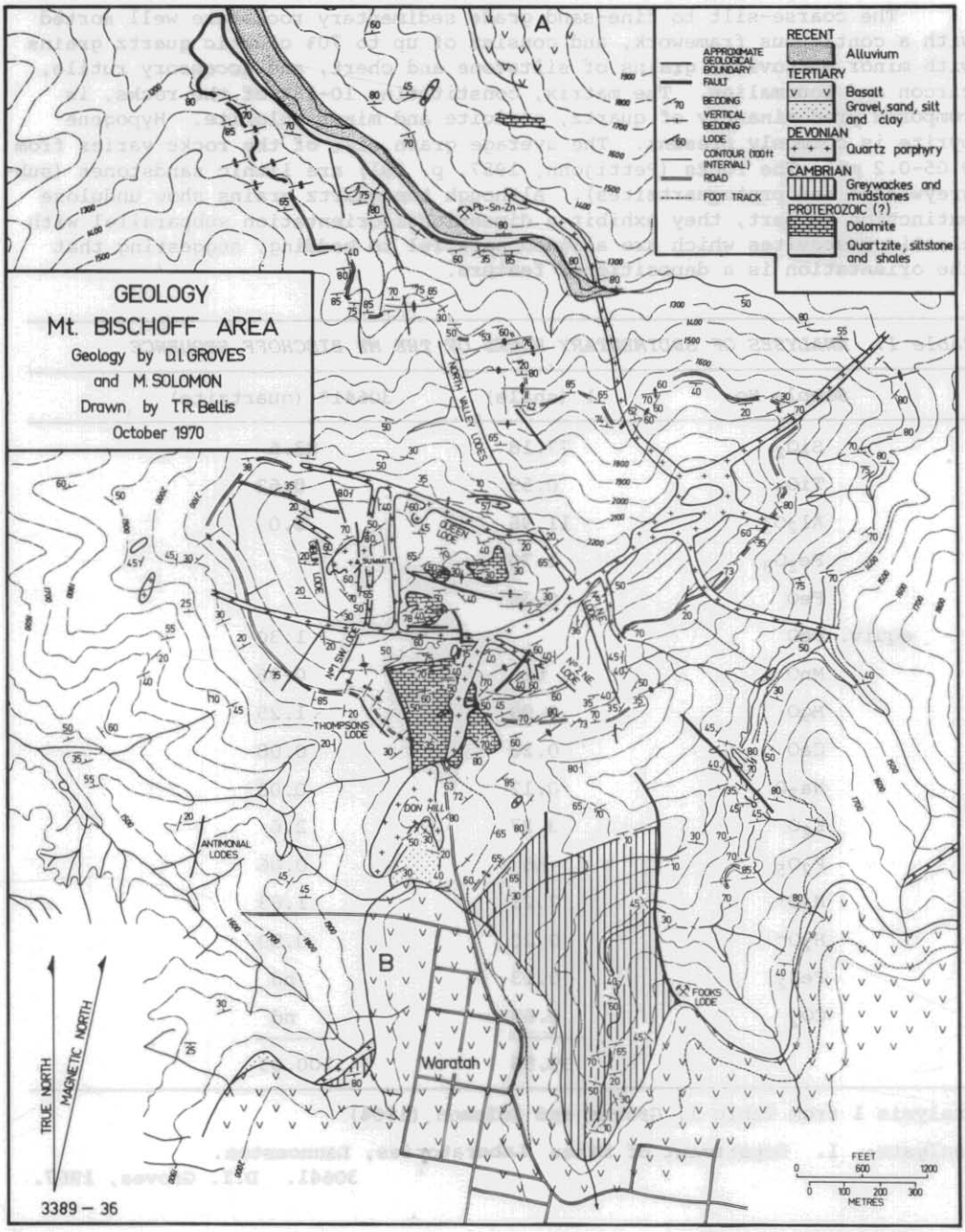


Figure 40.

5 cm

The coarse-silt to fine-sand grade sedimentary rocks are well sorted with a continuous framework, and consist of up to 70% clastic quartz grains with minor muscovite, grains of siltstone and chert, and accessory rutile, zircon and tourmaline. The matrix, constituting 10-15% of the rocks, is composed predominantly of quartz, sericite and minor chlorite. Hypogene pyrite is commonly present. The average grain size of the rocks varies from 0.05-0.2 mm. The rocks (Pettijohn, 1957, p. 291) are lithic sandstones (sub-greywackes and protoquartzites). Although the quartz grains show undulose extinction in part, they exhibit a dimensional orientation subparallel with clastic muscovites which are aligned parallel to bedding, suggesting that the orientation is a depositional feature.

Table 1. ANALYSES OF SEDIMENTARY ROCKS OF THE MT BISCHOFF SEQUENCE

Sample No:	1 (shale)	30641* (quartzite)
SiO_2	77.18	83.6
TiO_2	0.59	0.62
Al_2O_3	11.96	8.0
Fe_2O_3	1.70	
FeO	0.38	
equiv. FeO		1.30
MnO	tr	0.06
MgO	0.99	1.25
CaO	0.20	0.08
Na_2O	0.12	0.03
K_2O	3.27	2.6
P_2O_5	nd	0.06
H_2O^+	2.21	1.63
H_2O^-	0.20	0.80
FeS_2	0.13	nd
CO_2	0.63	nd
	99.56	100.03

Analysis 1 from Table 1, Groves and Solomon (1964)

Analysts: 1. Department of Mines Laboratories, Launceston.
30641. D.I. Groves, 1967.

The fine-silt grade horizons consist of fine-grained clastic quartz and chert grains with a high proportion of clastic muscovite, in a finer grained matrix of quartz and sericite. The grain size of the rocks is generally 0.015 mm and grains of this diameter comprise approximately 10% of the clay-grade rocks. Lamination appears due to variations in the proportions of clastic muscovite. The extremely siliceous nature of the rocks and their high K_2O/Na_2O ratios are shown by analyses of both quartzite and shale Table 1).

* All specimen numbers refer to a collection housed in the Department of Geology, University of Tasmania unless otherwise stated.

The dolomite shows a local transition to dolomitic shales. It is a cream to pale grey, fine-grained rock consisting almost exclusively of crystalline dolomite with minor interstitial quartz grains. Irregular patches of coarsely crystalline carbonate were developed during recrystallisation related to porphyry intrusion and/or mineralisation. The dolomite also exhibits a fine macroscopic banding which almost certainly is bedding lamination. Analyses of the dolomite (Table 2) indicate that it is almost pure dolomite mineral, with only minor SiO_2, $FeCO_3$ and $MnCO_3$.

Table 2. PARTIAL ANALYSES OF DOLOMITES (WEIGHT PER CENT) FROM MT BISCHOFF

Sample No:	1	2	100207	100208
SiO_2			2.7	2.7
equiv. FeO			0.49	0.77
MnO			0.35	0.43
MgO	20.8	21.1	19.7	19.2
CaO	30.6	30.2	31.1	30.7
approx. CO_2	46.7	46.7	45.6	45.4
H_2O^-			0.04	0.04
Total	98.10	98.00	99.94	99.24
Sr (ppm)			43	71
Rb (ppm)			<3	16

Sample Localities: 1. Hangingwall, Greisen Face. 2, 100207, 100208. South end of open cut, Mt Bischoff.

Analysts: 1, 2. Department of Mines Laboratories, Launceston (from Groves, 1963). 100207, 100208. D.I. Groves, 1967.

The thin banding (bedding?) and fine-grained texture of the dolomite and its regional concordant relations to continguous rocks, indicate a sedimentary origin (*e.g.* Knight, 1953; Groves and Solomon, 1964). There is considerable doubt that dolomites may form by direct precipitation (*e.g.* Degens and Epstein, 1964), although it is evident that dolomites may form early in diagenesis by replacement of calcite adjacent to the surface of deposition, and that they may be presently forming in this way in shallow water environments (*e.g.* Wells, 1962; Alderman and Von der Borsch, 1963; Curtis *et al.*, 1963; Von der Borsch *et al.*, 1964).

Waratah and Arthur River Sequence

Detailed descriptions of the Cambrian sequences, particularly the volcanic activity associated with sedimentation, are given by Groves (1963), Groves and Solomon (1964) and Groves (1966).

Unfossiliferous sedimentary rocks of probable Cambrian age occupy the major portion of the Waratah area (figs. 37,39). They can be subdivided into the greywacke-sandstone and mudstone sequence of the Waratah River (Waratah River sequence) and the mudstone and sandstone sequence of the Arthur River (Arthur River sequence). The sand-grade rocks of both sequences are usually greywacke (Pettijohn, 1957, p. 291) in contrast to the lithic sandstones of the Mt Bischoff sequence.

The Waratah River sequence, which is restricted to an area near Waratah, consists of greywacke-sandstones and siltstones with grey to yellow-brown laminated mudstones and rare cherts and beccias. Lavas are generally absent. The Arthur River sequence consists of interbedded red-brown mudstones, greywackes, massive chert-breccias and abundant volcanic rocks. It contains numerous, generally concordant bodies of dolerite and gabbro (fig. 37, 39). Limestone occurs in this sequence in the Arthur River to the north of Waratah. It is dark grey in colour, fine-grained and contains small bands of coarsely crystalline calcite.

The sand-grade rocks of both sequences are similar and are both compositionally and texturally immature. They are greywackes, showing varying degrees of grading, are poorly sorted with a disrupted framework, and consist of large, sub-angular to sub-rounded clastic grains in a finer grained matrix that forms up to 50% of the rock. The clastic grains are largely quartz with albite, muscovite, hornblende, augite, chlorite, magnetite and rare rock fragments including altered lava, siltstone, mudstone and quartzite. Some microcline is also present (*e.g.* 30659), the grains showing a fine, spindle-like normal albite twinning and some pericline cross-twinning; they are optically negative and $2V\alpha$ measurements range from 79° to 87°.

The clay-grade rocks are extremely fine-grained, and the composition cannot be determined microscopically although some small, angular fragments of quartz, plagioclase and chlorite are recognisable. Small carbonate veinlets are commonly present. The red colouration of mudstones in the Arthur River sequence is due to limonite and possibly hematite.

Individual lava flows occur within the Arthur River sequence and are largely spilitic (*e.g.* 636, 642, 30658a, 30662a and 35G5, 35G9 of the Department of Mines). They are described by Groves (1963) and Groves and Solomon (1964). Many are porphyritic with phenocrysts of albite, augite and chlorite in a felted groundmass of albite, chlorite, calcite, epidote, magnetite and ilmenite. Some have intersertal texture, and consist of interlocking aggregates of albite, augite and chlorite with interstitial chlorite, calcite etc. The albite as in the Cambrian spilites elsewhere in Tasmania, is partially altered to sericite, displays low temperature optics, and varies in composition from $Ab_{90}An_{10}$ to $Ab_{95}An_5$. Chlorite-quartz spherulite rocks and amygdaloidal rocks with vesicles filled with chlorite and quartz spherulites are common in places. Pillow-structure occurs in spilites on the Corinna Road some 13 km from Waratah and confirms the submarine nature of the volcanic rocks.

Keratophyric lavas occur rarely. The quartz phenocrysts in these lavas are rounded, euhedral or shard-like in form, and are commonly embayed in identical fashion to crystals in the keratophyres of the Mt Read Volcanics (*e.g.* Solomon, 1964). Volcanic rocks with fragmental texture are not common. An interesting example (646) contains a pale reddish-brown isotropic mineral that is probably hydrogrossular, similar to that described by Scott (1951) in spilites from King Island.

The occurrence of microcline in the greywackes, which are largely locally derived, is unusual in that K-feldspar is absent in the spilites and keratophyres. Groves and Solomon (1964) suggested that it is possibly derived from potassic rhyolite or quartz keratophyre flows and necks that have been largely disintegrated by explosive activity during eruption.

Massive sequences of Cambrian ultramafic and mafic rocks associated generally with spilites and agglomerates and in places with syenitic and granophyric rocks occur to the west and south of Waratah. They have been

described by Nye (1923), Scott (1954), Groves (1963), Groves and Solomon (1964), Solomon (1964), Jack and Groves (1965), and Groves (1966). They are being studied in detail by M. Rubenach at the University of Tasmania.

An important sequence of these rocks in the Waratah area is the 'Magnet Dyke' which has localised ore deposition at Magnet, Fawkners Tunnel and Persic (fig.39, 41). It is an elongate strip of igneous rocks which occurs along the northern contact of the Mt Bischoff sequence and the Arthur River sequence, and is roughly concordant with both sequences. It reaches a maximum thickness of about 300 m at the Magnet Mine. It consists predominantly of porphyritic to amygdaloidal spilites or albite dolerites which have been deuterically altered locally to spherulitic quartz-chlorite rocks. Marginal lenses of ultramafic rocks (websterites and orbicular websterites, which have been locally serpentinised) appear concordant with the probable extrusive rocks, but contain blocks of both Proterozoic and Cambrian sedimentary rocks. The 'dyke' has been described in detail by several authors (*e.g.* Scott, 1954; Groves, 1963; Groves and Solomon, 1964; Solomon, 1964) and it is generally considered that it represents a sequence of extrusive or shallowly intrusive rocks.

Proterozoic - Cambrian Contact

Reid (1923) considered that the contact was a fault, but Groves and Solomon (1964) showed that the boundary was regionally conformable. Recent sluicing at Don Hill has revealed the contact over a strike length of a few hundred metres (fig. 43). This contact has been described by Groves (1971). It is a few centimetres wide, is locally subparallel to the strike of both sequences (*i.e.* NNE to NE) and dips at a variable high angle to the southwest. The contact zone contains numerous slickensides. Small fractures subparallel to the contact intersect both sequences imparting a shredded appearance, particularly in the greywacke-sandstone of the Waratah River sequence. It is evident that the contact has been a surface of movement. The exact relationship of the bedding in both sequences to the orientation of the contact zone is difficult to determine. In general, the siltstone and shale layers of the Mt Bischoff sequence dip steeply away from the contact while the greywacke-sandstone beds dip towards the contact at variable angles.

The greywacke-sandstone of the Waratah River sequence locally encloses large blocks of laminated mudstone, which are irregular in detail but show an overall subparallelism to bedding. These blocks are apparently confined to within 100 m of the contact. In general it is impossible to determine whether these blocks consist of mudstones of the Mt Bischoff sequence. At least one large block, approximately 12 m in length, in the north-east corner of the mapped area is of grey shale identical to those of the Mt Bischoff sequence. These blocks indicate extensive slumping penecontemporaneous with deposition of the Waratah River sequence. The irregular to wispy contacts of some blocks suggest that they were partly unconsolidated at the time of slumping; these blocks possibly represent penecontemporaneously deposited mudstones that have been transported from another part of the depositional basin. It is probable that the surface of deposition was a zone of movement represented by the present contact zone, movement on this zone possibly occurring at several times.

Structure of Proterozoic and Cambrian Sequences

Introduction

There have been distinct differences in interpretation of the general and detailed structure of Mt Bischoff in recent years. This has been due to

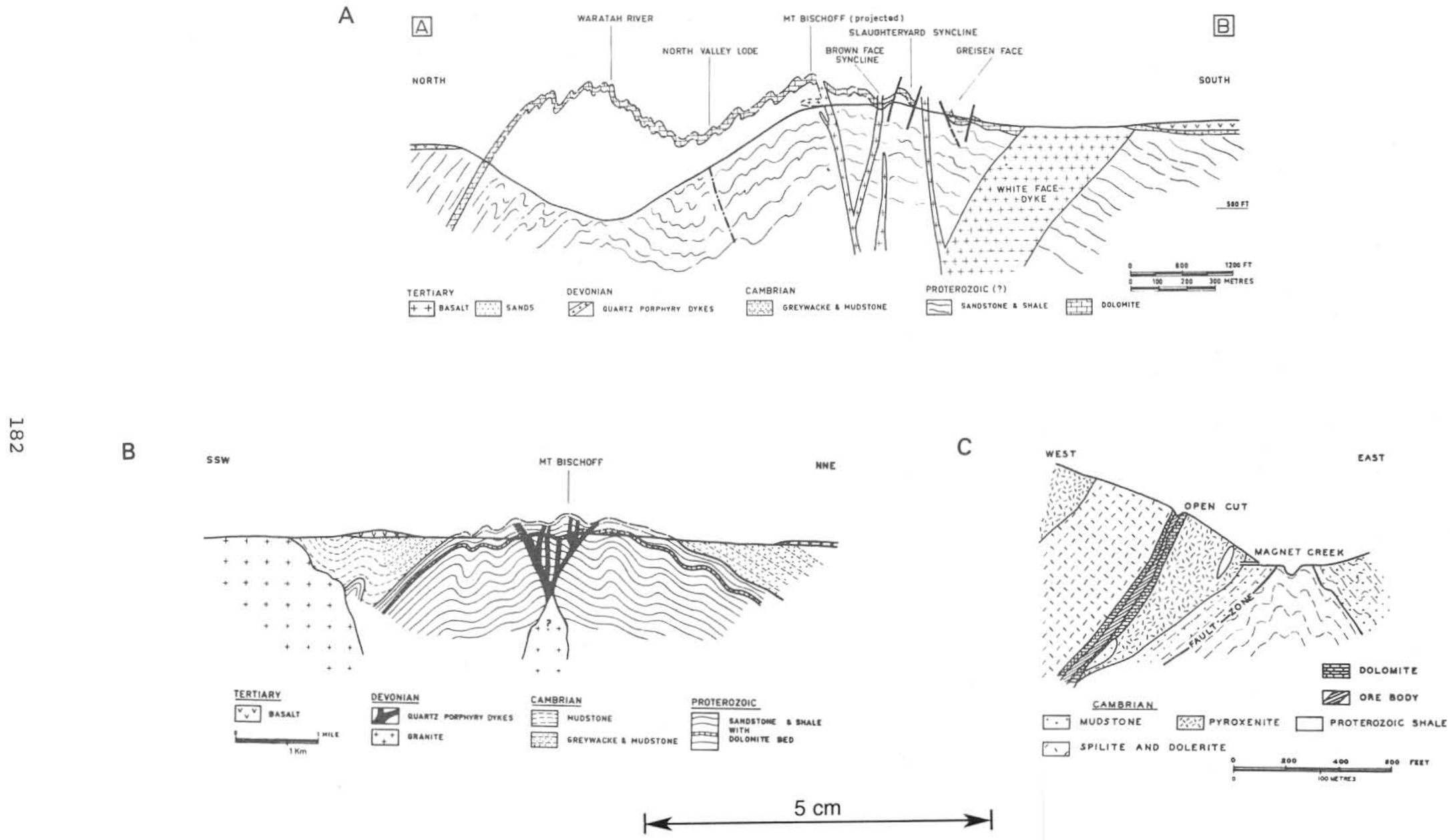


Figure 41. *Cross sections through the Bischoff anticlinorium in the Waratah district.*

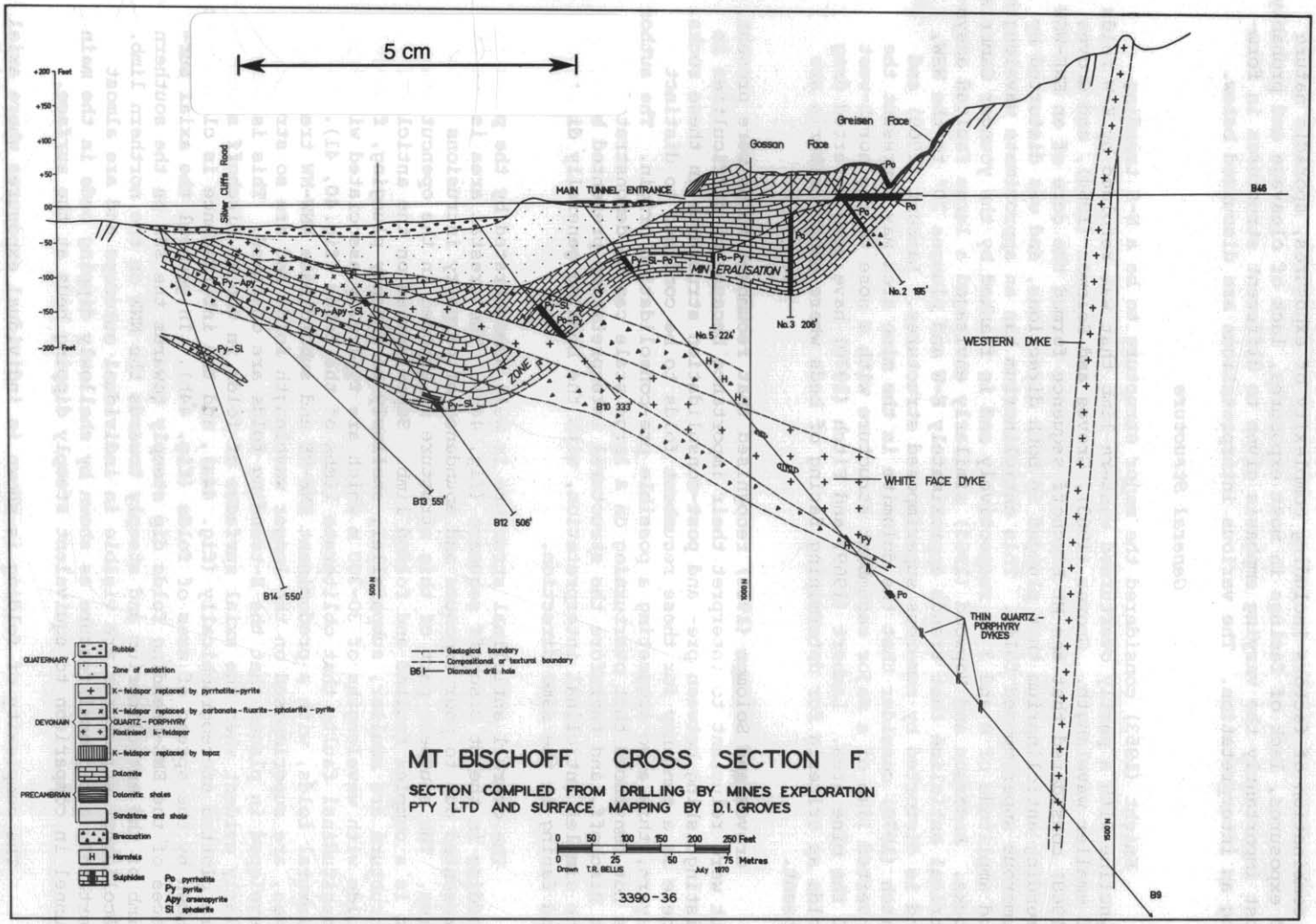

5 cm
Silver Cliffs Road
MAIN TUNNEL ENTRANCE
Greisen Face
Gossan Face
B46
Po
Py
Po-Py
Py-Sl-Po
MINERALISATION
Py-Apy
Py-Apy-Sl
Py-Sl
No.2 185'
No.3 206'
No.5 224'
B10 333'
B12 506'
B13 551'
B14 550'
WESTERN DYKE
WHITE FACE DYKE
THIN QUARTZ-PORPHYRY DYKES
B9
Geological boundary
Compositional or textural boundary
Diamond drill hole
QUATERNARY
Rubble
Zone of oxidation
DEVONIAN
K-feldspar replaced by pyrrhotite-pyrite
K-feldspar replaced by carbonate-fluorite-sphalerite-pyrite
QUARTZ-PORPHYRY
Kaolinised K-feldspar
K-feldspar replaced by topaz
PRECAMBRIAN
Dolomite
Dolomitic shales
Sandstone and shale
Brecciation
Hornfels
Sulphides
Po pyrrhotite
Py pyrite
Apy arsenopyrite
Sl sphalerite
MT BISCHOFF CROSS SECTION F
SECTION COMPILED FROM DRILLING BY MINES EXPLORATION PTY LTD AND SURFACE MAPPING BY D.I. GROVES
0 50 100 150 200 250 Feet
0 25 50 75 Metres
Drawn T.R. BELLIS
July 1970
3390-36

Figure 42.

a combination of factors including complexity of exposures, sporadic nature of exposures, lack of facings in most exposures, lack of cleavage and probably most importantly the varying emphasis given to different structures in forming an interpretation. The various interpretations are discussed below.

General Structure

Knight (1953) considered the major structure to be a N-S trending syncline with a partly overturned western limb that was refolded by E-W folds of smaller wavelength. Groves (1963), Groves and Solomon (1964), and Groves (1968) considered that the Mt Bischoff sequence formed the core of an ENE-WSW trending anticlinorium that plunged in both directions, and was distorted by numerous superimposed folds. This anticlinorium has an approximate wavelength and amplitude of 8 and 3 km respectively and is flanked by the younger Cambrian rocks. Anderson and Hopwood (1963) similarly envisaged a large faulted assymetrical anticline that trends approximately E-W and plunges 20° to the WSW, and is distorted by numerous superimposed structures. Lambert (1969) and Fitch (1970) consider that the dolomite in the mine area may represent the inverted limb of a major recumbent structure with a nose to the north-west of the mine area. Lambert (1969) and Fitch (1970) have used inverted drag folds as criteria for determining facing of beds where other criteria are absent.

Groves and Solomon (1964) recognised that recumbent folds were present, but were reluctant to interpret their importance because of difficulties in distinguishing between pre- and post-consolidation structures in these rocks. There is a tendency for these recumbent folds to be confined to distinct layers, this also suggesting a possible pre-consolidation origin. The author is not convinced that overturning on a large scale can be demonstrated at Mt Bischoff, and therefore the structural interpretation presented below is the simpler anticlinal interpretation, with the bedding generally dipping and facing in the same direction.

The overall anticlinal structure is well demonstrated by the poles to bedding in the Mt Bischoff sequence (fig. 44c). The crestal area is not represented due to poor exposure and abundance of porphyry intrusions in this area. The southern part of this structure is exposed in the opencut where it is a complex faulted and folded limb. Superimposed on the anticlinal structure are smaller, subparallel, shallowly to steeply plunging, flexural folds with wavelengths of 30-300 m which are typically associated with sublongitudinal faults that obliterate limbs of the folds (fig. 40, 41). Smaller flexural folds, with a predominant ENE-NE and subordinate NNW-NW trend of axes, are superimposed on the longer wavelength folds and are so strongly developed in places that the E-trending folds are obscured. This is particularly evident when the axial surfaces of folds in the Mt Bischoff sequence are plotted stereographically (fig. 44a), and the interference is clearly sho shown by the spread of axes of folds (fig. 44b). In general the axial surfaces of the ENE-trending folds dip steeply towards the SSE on the southern limb of the anticlinorium and steeply towards the NNW on the northern limb. Décollement structures are visible in individual outcrops and are almost certainly a regional feature as shown by shallowly dipping beds in the main tunnel in comparison to equivalent steeply dipping beds at the surface.

The complexity of folding is shown in individual exposures where axial surface and plunge variations are ubiquitous, and bent hinges and refolded fold axes are common.

A regional comparison of fold geometry in the Mt Bischoff sequence and Cambrian sequences is virtually impossible due to the lack of suitable folds

Strong fracturing

GEOLOGICAL SKETCH MAP
WARATAH DISTRICT
(AFTER GROVES & SOLOMON, 1964)

0 0.5 1
SCALE OF MILES

TERTIARY — Basalt and underlying sediments
DEVONIAN — Quartz-feldspar porphyry
CAMBRIAN — Waratah River sequence
Arthur River sequence
PRECAMBRIAN — Mt. Bischoff sequence
Approx. contact
Fault

Waratah
Arthur River
MT BISCHOFF
AREA COVERED BY MAIN PLAN
TO MURCHISON HIGHWAY
WARATAH
MAGNET
TO CORINNA

LEGEND

RECENT — Soil and talus
CAMBRIAN ? — Laminated mudstone } WARATAH RIVER SEQUENCE
Greywacke sandstone
PRECAMBRIAN ? — Sub-greywacke siltstone } MT. BISCHOFF SEQUENCE
Shale and fine siltstone
Trace of bedding with dip
Rock boundary
Fault
Contour (interval 5ft)
Drainage channel
Small escarpment

DEPARTMENT OF MINES — TASMANIA

DETAILED GEOLOGY
SLUICED AREA — DON HILL
MT BISCHOFF

DATE May 1967	SCALE OF FEET 0 10 20 30
GEOLOGIST D. Groves	
DRAUGHTSMAN P. Rankcell	MAP SHEET & N° Val Peak 36
REVISIONS	FILE N° 3041

GSB54

Figure 43

5 cm

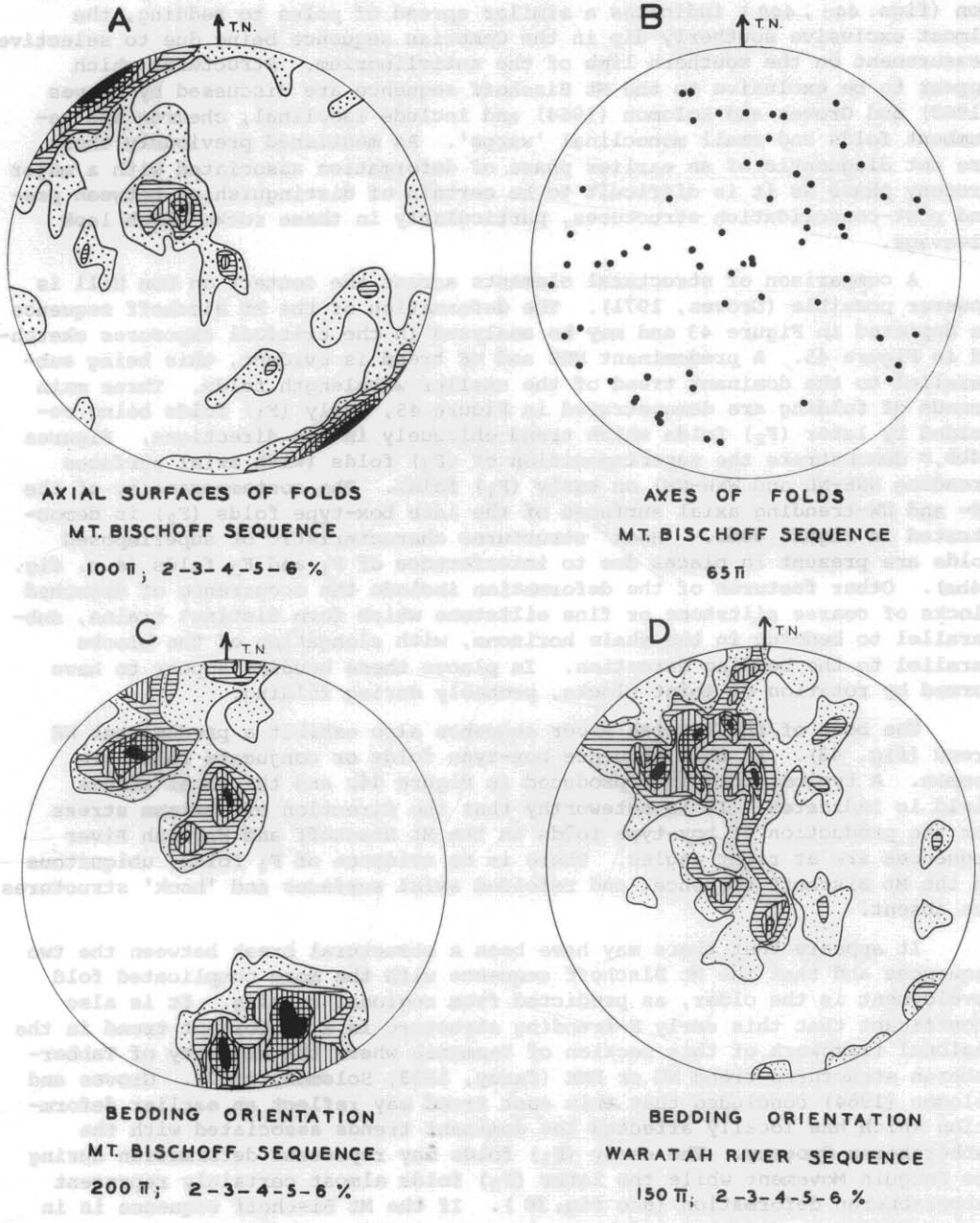


Figure 44. *Stereogram of structural elements, Mt Bischoff. Equal angle, lower hemisphere projection.*

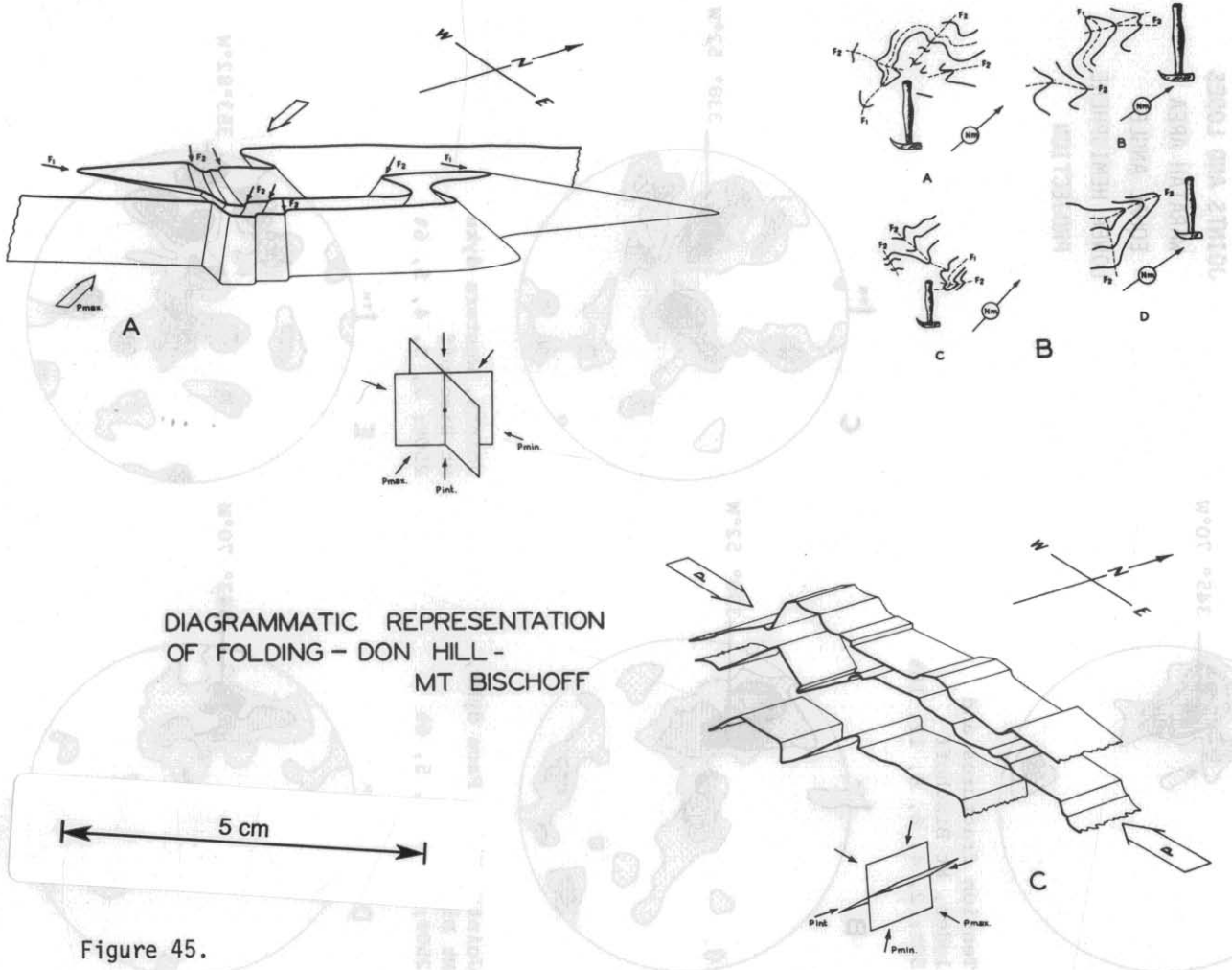
for measurement in the Cambrian sequences. A comparison of bedding orientation (figs. 44c , 44d) indicates a similar spread of poles to bedding, the almost exclusive southerly dip in the Cambrian sequence being due to selective measurement on the southern limb of the anticlinorium. Structures which appear to be exclusive to the Mt Bischoff sequence are discussed by Groves (1963) and Groves and Solomon (1964) and include isoclinal, chevron and recumbent folds and small monoclinal 'warps'. As mentioned previously these are not diagnostic of an earlier phase of deformation associated with a major orogeny phase as it is difficult to be certain of distinguishing between pre- and post-consolidation structures, particularly in these rocks which lack cleavage.

A comparison of structural elements across the contact on Don Hill is however possible (Groves, 1971). The deformation of the Mt Bischoff sequence is depicted in Figure 43 and may be analysed in the critical exposures sketched in Figure 45. A predominant NNE and NE trend is evident, this being subparallel to the dominant trend of the smaller wavelength folds. Three main trends of folding are demonstrated in Figure 45, early (F_1) folds being refolded by later (F_2) folds which trend obliquely in two directions. Figures 44b*b*,*c* demonstrate the superimposition of (F_2) folds (with axial surfaces trending NNE-NE and WNW-NW) on early (F_1) folds. The contemporaneity of the NE- and NW-trending axial surfaces of the late box-type folds (F_2) is demonstrated in Figure 44b*d*. 'Hook' structures characteristic of superimposed folds are present in places due to interference of F_1 and F_2 folds (*e.g.* fig. 44b*a*). Other features of the deformation include the occurrence of detached blocks of coarse siltstone or fine siltstone which form distinct trains, subparallel to bedding in the shale horizons, with elongation of the blocks parallel to the bedding direction. In places these boudins appear to have formed by rotation of joint blocks, probably during folding.

The beds of the Waratah River sequence also exhibit a predominant NE trend (fig. 43). In this sequence box-type folds or conjugate folds are common. A typical fold is reproduced in Figure 44c and the local stress field is indicated. It is noteworthy that the direction of maximum stress for the production of box-type folds in the Mt Bischoff and Waratah River sequences are at right angles. There is no evidence of F_1 folds, ubiquitous in the Mt Bischoff sequence, and refolded axial surfaces and 'hook' structures are absent.

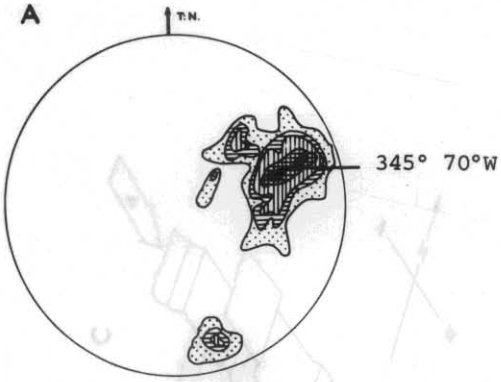
It appears that there may have been a structural break between the two sequences and that the Mt Bischoff sequence with the more complicated fold development is the older, as predicted from regional mapping. It is also significant that this early E-trending structure is a discordant trend in the regional framework of this section of Tasmania where the majority of Tabberabberan structures trend NW or NNE (Carey, 1953; Solomon, 1962). Groves and Solomon (1964) concluded that this east trend may reflect an earlier deformation which has locally affected the dominant trends associated with the Tabberaberan Orogeny. The early (F_1) folds may represent deformation during the Penguin Movement while the later (F_2) folds almost certainly represent Tabberabberan deformation (see fig. 38). If the Mt Bischoff sequence is in fact pre-Penguin Movement and hence in an equivalent position to the Oonah Formation it does not occupy the position of the Success Creek phase as envisaged by Solomon (1965).

The faulting and jointing has been particularly important in localising ore deposition. Stereographic plots of joints in Proterozoic and Cambrian rocks together with the porphyry dykes (fig. 46) show common maxima which indicate a strong post-folding and intrusion joint set which trends approximately NNW and dips steeply to the WNW. A stereographic plot of tension fractures and lodes (fig. 46a) indicates a similar orientation, which is almost normal to the axis of the anticlinorium. The post-intrusion age of the fractures is demonstrated by displacement of porphyry dykes, the fractures

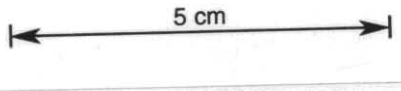


DIAGRAMMATIC REPRESENTATION
OF FOLDING – DON HILL –
MT BISCHOFF

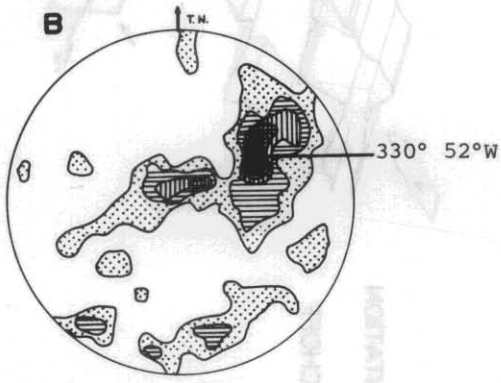
Figure 45.



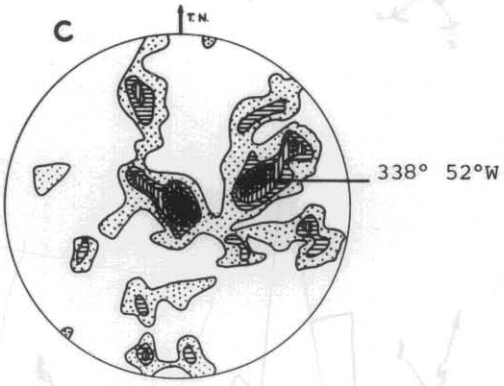
Tension fractures and
lodes, Mt Bischoff
50π; 2, 4, 6, 8, 12, 16%



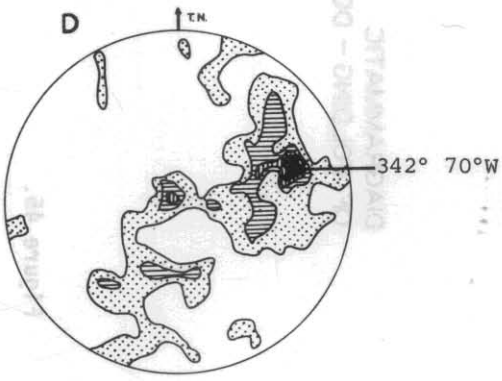
JOINTS AND LODES
WARATAH AREA
EQUAL ANGLE
LOWER HEMISPHERE
PROJECTION



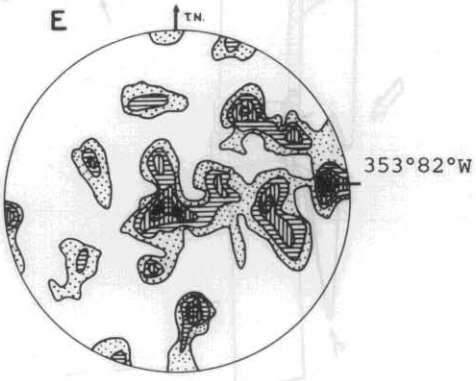
Joints, White Face dyke,
Mt Bischoff
250π; 2, 3, 4, 5, 6%



Joints, Western dyke
Mt Bischoff
250π; 2, 3, 4, 5, 6%



Joints, Mt Bischoff
sequence, Waratah
200π; 2, 3, 4, 5%



Joints, Cambrian rocks,
Waratah-Magnet
200π; 2, 3, 4, 5, 6%

Figure 46.

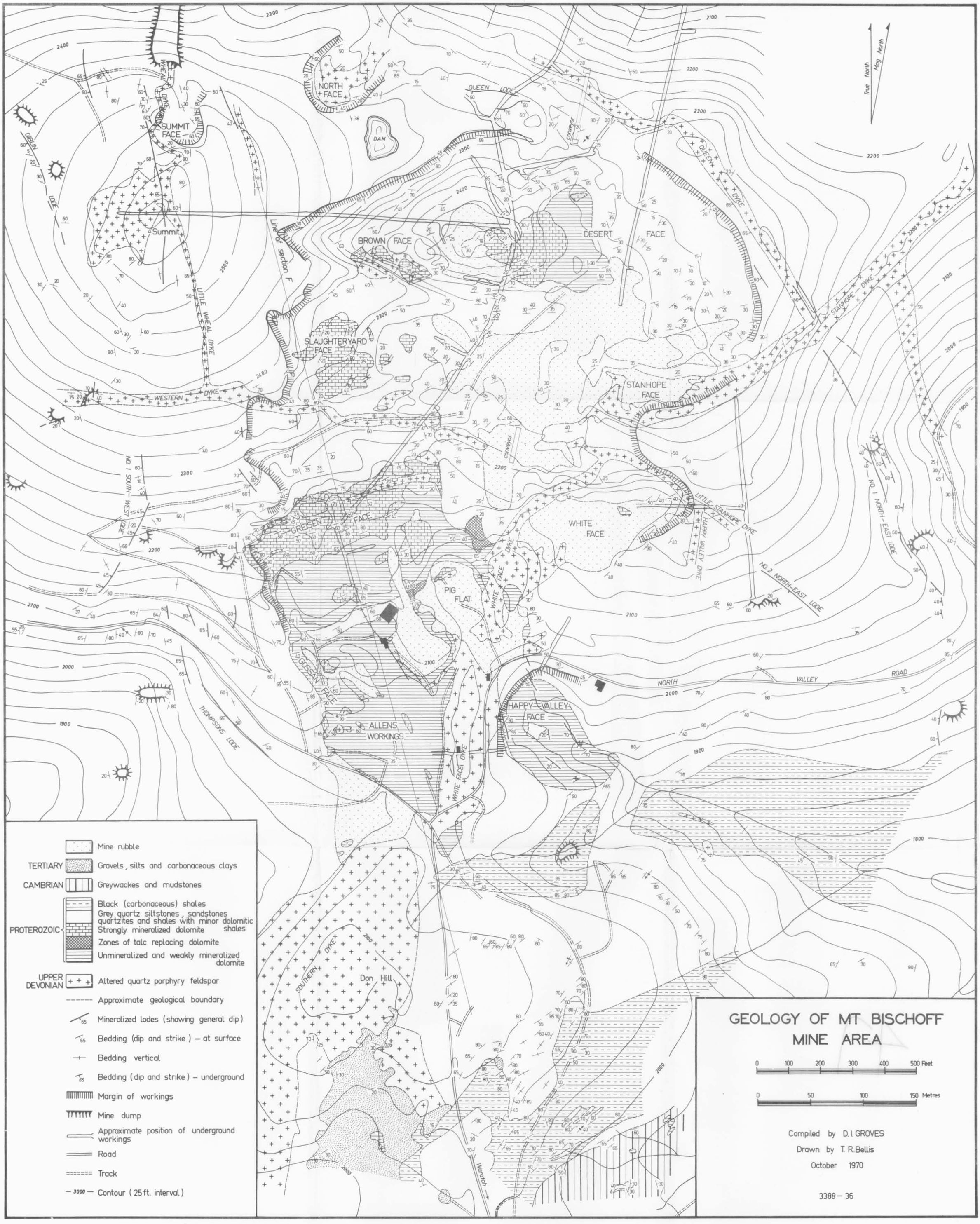


Figure 47.

5 cm

being filled with quartz and cassiterite and forming small lodes. The displacements (*e.g.* Western Dyke, fig. 47) can all be explained in terms of normal movement on these faults.

Structures Within the Open Cut

The factual information available for publication at the present time is shown in Figure 47. The structure of several of the main faces within the cut is discussed below.

Brown Face and Desert Face

Brown Face is part of a doubly plunging asymmetrical synclinal fold with a faulted southern limb (fig. 40, 47). A remnant of altered dolomite forms the core of this fold, and the sedimentary rocks beneath the dolomite are shallowly dipping. The large synclinal fold appears to plunge over Desert Face, and changes in form to a series of smaller folds with shallowly dipping limbs.

Slaughteryard Face

Slaughteryard Face contains part of the crest of the faulted anticlinal fold immediately south of Brown Face, and the southern dipping limb of this fold. In detail the structure is exremely complicated with numerous small faults and superimposed folds. The western face may be a fault although this is uncertain.

Greisen Face - Gossan Face

Greisen Face and Gossan Face represent the gently south-dipping flexured limb of a fold which steepens along the northern extremity of Greisen Face, with the dolomite horizon rising above the present surface (fig. 42). Fitch (1970) considers that this steepening represents part of a recumbent structure with an approximately E-W axis. The western side of the Greisen Face - Gossan Face structure appears to be partly faulted, possibly with a downthrown side to the west, although Fitch (1970) suggests that the dolomite may form a steep NW-dipping limb along the west side of Greisen Face.

White Face

The structure of White Face appears to be an extension of the Greisen Face structure with dolomite forming the south-dipping limb of the regional fold. There appears to be an E-W trending fault between this face and Happy Valley Face (fig. 40).

Happy Valley Face and Southern Area

South of the opencut the dolomite horizon dips beneath overlying sedimentary rocks, and at Happy Valley Face also appears to dip beneath sedimentary rocks at its southernmost extension. However it has not been encountered in exploratory drilling south of this area, and may be thinning rapidly to the south. The eastern contact of dolomite and sedimentary rocks in the Happy Valley Face is probably a fault contact with the fault downthrowing the dolomite horizon to the east.

To the south-west of the main dolomite mass three drill holes have intersected dolomite, indicating a possible SW-dipping extension of the southern limb of the major anticlinal fold. The dolomite horizon has also been found by drilling beneath the flat-lying porphyry mass on Don Hill.

Summit Area

A sequence of pale grey shales, siltstones and minor quartzites crop out in the Summit area. They are intensely brecciated and folded with common recumbent structures, and Fitch (1970) has called them 'rheomorphic rocks'. The type of structures are superficially similar to pre-consolidation deformation elsewhere, but the rocks have not been studied in enough detail by the author to strongly support a particular origin for them.

Devonian Igneous Rocks

Anastomosing quartz-porphyry dykes and sills intrude the Mt Bischoff sequence near the crest of the anticlinorium. They have been dated at 349 ±4 million years, a similar age to granitic rocks of the Meredith Granite (353±7 m.y.) which crop out some 6.5 km south-west of Waratah (Brooks, 1966). Other discordant granite stocks in western Tasmania have a similar isotopic age (table 3).

Table 3. SUMMARY OF ISOTOPIC DATING OF GRANITES, WESTERN TASMANIA

Granite Mass	Rock Type	Age m.y. (initial Sr^{87}/Sr^{86})	Method	Reference
Heemskirk	Granite?	338	K-Ar	Evernden and Richards (1962)
Heemskirk	Red and White Adamellites	340±5	K-Ar	McDougall and Leggo (1965)
Heemskirk	White Granite A Series	353±3 (0.734±0.002)	Rb-Sr	Brooks and Compston (1965)
Heemskirk	White Granite B Series	357±7	Rb-Sr	Brooks and Compston (1965)
Heemskirk	Red Granite	354± (0.705±0.720)	Rb-Sr	Brooks and Compston (1965)
Pieman	Granite?	356±9 (0.7354±0.0018)	Rb-Sr	Brooks (1966)
Meredith	Micro-adamellite	350	K-Ar	McDougall and Leggo (1965)
Meredith	Porphyritic	353±7 (0.7155±0.0015)	Rb-Sr	Brooks (1966)
Renison Bell	Greisenised	354±4 (0.700)	Rb-Sr	Brooks (1966)
Mt Bischoff	Quartz-Orthoclase Porphyry	349±4 (0.700)	Rb-Sr	Brooks (1966)
Granite Tor	Granite	355±6	K-Ar	McDougall and Leggo (1965)
Housetop	Adamellite	362	K-Ar	McDougall and Leggo (1965)
Housetop	Adamellite	375±10 (0.710±0.002)	Rb-Sr	McDougall and Leggo (1965)

Meredith Granite

The Meredith Granite is a large stock that crops out over an area of about 300 km^2. It has extremely irregular contacts against the country rocks locally (*e.g.* Groves, 1966) and is discordant to the regional structure (fig. 37). The stratigraphic position and structure of the country rocks to the north and south of the intrusion are similar and suggest possible continuity prior to intrusion. Regional jointing consists of approximately N-S and E-W sets of near vertical joints, the N-trending set swinging in strike from NNE in the northern part of the granite to NNW in the southern area (fig.37). Major dykes within the granite mass subparallel this trend and Waterhouse (1914) recorded that tourmaline-quartz veins commonly filled NW-trending fissures in the Stanley River area.

Table 4. MODAL ANALYSES OF EVEN-GRAINED ROCKS, MEREDITH GRANITE (VOLUME %)

	Microgranite			
Mineral	*64-33*	*Maximum variance*	*64-39*	*Maximum variance*
Quartz	37.1	5.1	39.2	5.3
K-feldspar perthite	43.0	5.9	40.8	5.6
Plagioclase	14.8	2.0	14.2	1.9
Biotite	3.9	0.5	3.7	0.5
Muscovite	1.1		1.9	
Accessories	0.3		0.3	
Total	100.2		100.1	

	Biotite Granite	
Mineral	*1404*	*Maximum variance*
Quartz	34.9	4.8
K-feldspar perthite	41.5	5.7
Plagioclase	16.3	2.4
Biotite	6.9	0.9
Muscovite	0.0	
Accessories	0.3	
Total	99.9	

Variance calculated from Hasofer (1963) and Solomon (1963)

The Meredith Granite is composed predominantly of equigranular biotite/adamellite (table 4) and porphyritic biotite granite/adamellite. They are mineralogically similar (see Groves, 1968, appendix A1) and consist of quartz, acid andesine ($An_{33}Ab_{67}$-$An_{38}Ab_{62}$), and orthoclase-microperthite with subordinate biotite and minor muscovite, hornblende, apatite, zircon and sphene.

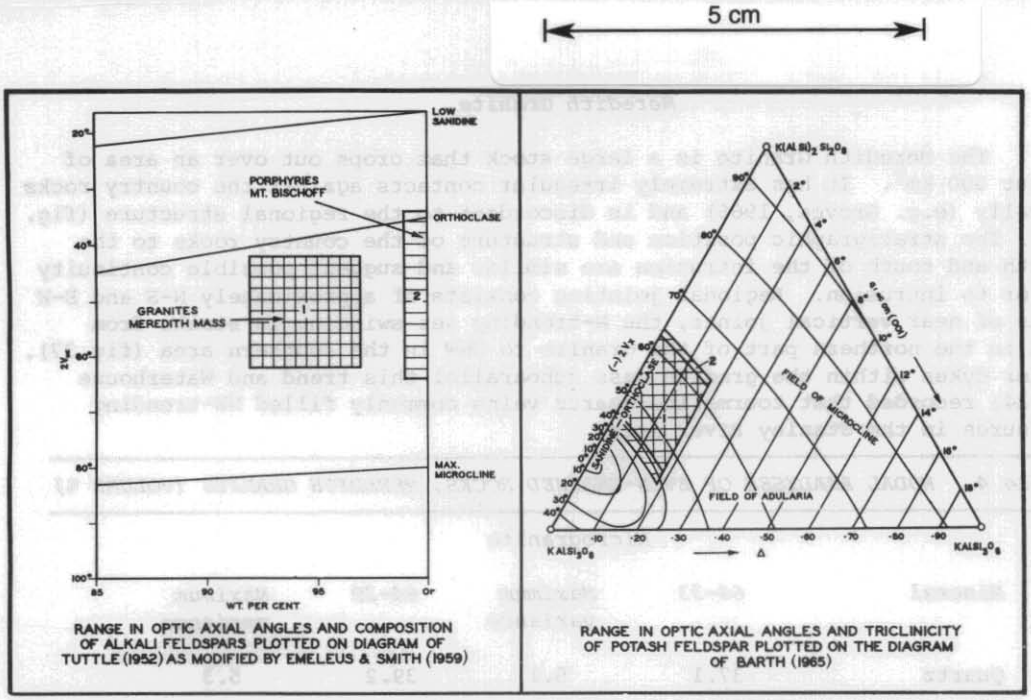


Figure 48. *Properties of potash feldspar in granitic rocks, Meredith granite and Mt Bischoff.*

Orthoclase phenocrysts up to 6 cm in length are common in the porphyritic variety. The relationship between the texturally different rocks is not clear (*e.g.* Waterhouse, 1914; Reid, 1923), the only exposed contact examined containing 'rafts' of hornfelsed sedimentary rocks. The granite/adamellites are intruded by aplites and microgranites (table 4) which consist of granular intergrowths of quartz, orthoclase and albite (An_6Ab_{94}-An_9Ab_{91}) and in places contain abundant tourmaline nodules. The range of optic axial angles, compositions and obliquities of potash feldspars in rocks of the Meredith Granite is summarised in Figure 48, and it is evident that they plot below the low albite-orthoclase join but well above the low albite-maximum microcline join. A plot of optic axial angle against composition for the plagioclases (fig. 49) indicates that those from the granite/adamellites generally lie between the determined curves for plutonic (low temperature) and heated plutonic (high temperature) plagioclases, but it is difficult to draw conclusions from the results as the two curves approach closely at these compositions. The plagioclases from the microgranites cluster around and generally slightly above the low temperature curve. The coefficient of distribution of albite between potash feldspar and plagioclase (K_T) is approximately 0.41 for the granite/adamellites.

The contact aureole of the Meredith Granite is up to 2.5 km wide in plan and the hornfelses are predominantly of the albite-epidote hornfels and hornblende hornfels facies, and locally of the pyroxene hornfels facies. Estimates of lithostatic load at the time of granite emplacement vary from 1,000-2,500 atmospheres. The occurrence of sillimanite in the immediate contact zone in the Stanley River area (Waterhouse, 1914) indicates a minimum temperature of approximately 650°C at 2,500 atmospheres, this temperature increasing with decreasing pressure (*e.g.* Heitanen, 1967). Using the temperature of 650°C as a minimum initial contact temperature, a minimum initial temperature of the magma of 850° is indicated if appropriate values of the geothermal gradient (*e.g.* Jaeger and Sass, 1963) and diffusivities and thermal conductivities of the rocks involved (*e.g.* Lovering, 1936) are substituted in the equations of Jaeger (1959). Xenoliths within the granite/adamellites

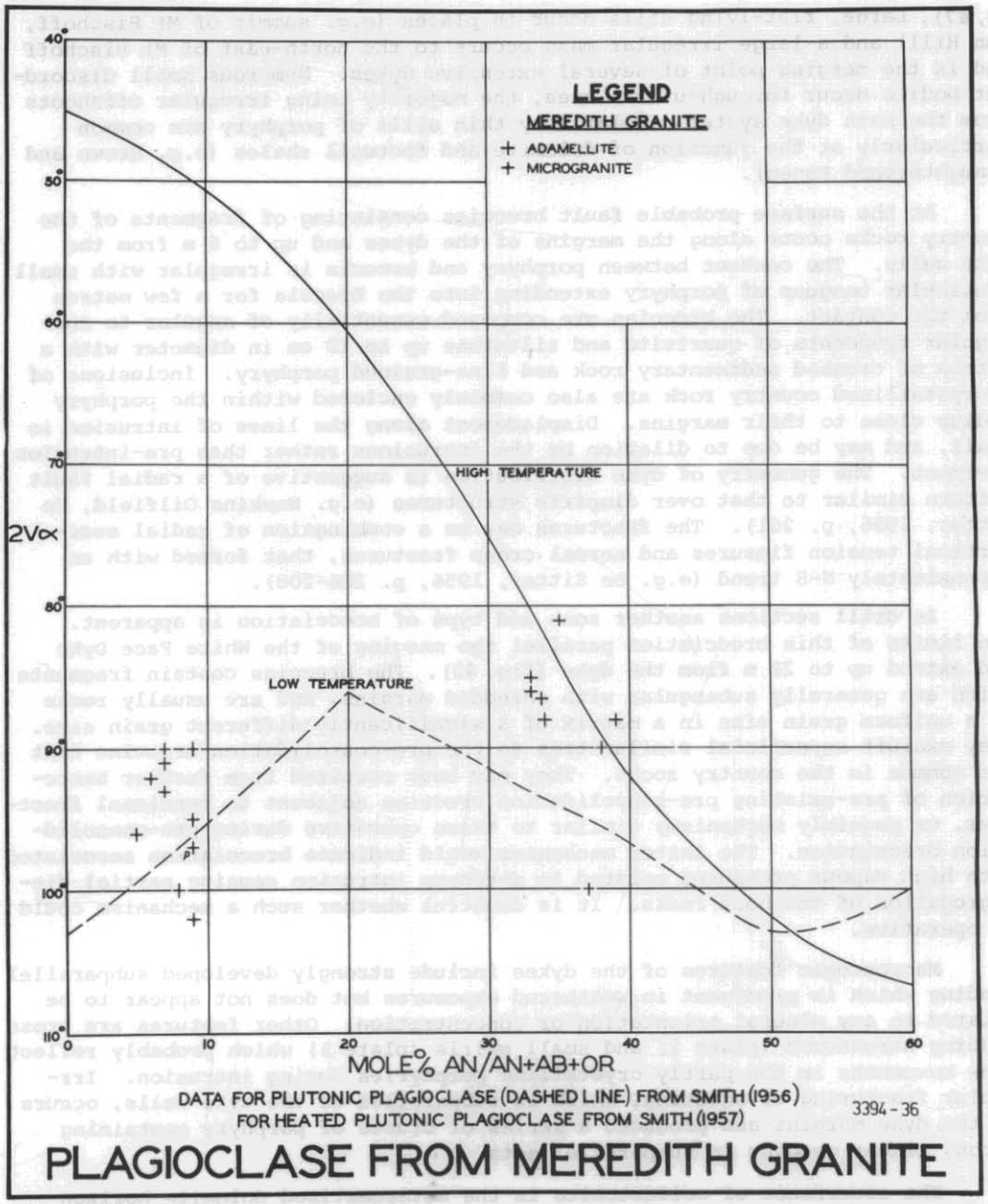


Figure 49.

5 cm

along the northern contact consist of a granoblastic mosaic of quartz, biotite, acid andesine and potash feldspar. The metamorphic facies of these rocks is not demonstrable.

Mt Bischoff Porphyries

Macroscopic Structures

The majority of the quartz porphyry intrusions are dyke-like bodies, of variable width, which form a general radial pattern with dominant orientations E-W and N-S, and steep dips to the north and west respectively (fig.

40,47). Large, flat-lying sills occur in places (*e.g.* summit of Mt Bischoff, Don Hill) and a large irregular mass occurs to the north-east of Mt Bischoff and is the merging point of several extensive dykes. Numerous small discordant bodies occur throughout the area, the majority being irregular offshoots from the main dyke system. Relatively thin sills of porphyry are common particularly at the junction of dolomite and footwall shales (*e.g.* Brown and Slaughteryard Faces).

At the surface probable fault breccias consisting of fragments of the country rocks occur along the margins of the dykes and up to 6 m from the dyke walls. The contact between porphyry and breccia is irregular with small lenticular tongues of porphyry extending into the breccia for a few metres from the contact. The breccias are composed essentially of angular to subangular fragments of quartzite and siltstone up to 10 cm in diameter with a matrix of crushed sedimentary rock and fine-grained porphyry. Inclusions of recrystallised country rock are also commonly enclosed within the porphyry bodies close to their margins. Displacement along the lines of intrusion is small, and may be due to dilation by the intrusions rather than pre-intrusion movement. The geometry of dyke distribution is suggestive of a radial fault pattern similar to that over diapiric structures (*e.g.* Hawkins Oilfield, De Sitter, 1956, p. 261). The fractures may be a combination of radial near-vertical tension fissures and normal cross fractures, that formed with an approximately N-S trend (*e.g.* De Sitter, 1956, p. 206-208).

In drill sections another zone and type of brecciation is apparent. The limits of this brecciation parallel the margins of the White Face Dyke and extend up to 22 m from the dyke (fig. 42). The breccias contain fragments which are generally subangular with shredded margins, and are usually rocks of a uniform grain size in a matrix of a significantly different grain size. They exhibit superficial similarities to the pre-consolidation breccias that are common in the country rocks. They may have resulted from further brecciation of pre-existing pre-consolidation breccias adjacent to tensional fractures, or possibly mechanisms similar to those operative during pre-consolidation brecciation. The latter mechanism would indicate brecciation associated with high vapour pressures related to porphyry intrusion causing partial disaggregation of the host rocks. It is doubtful whether such a mechanism could be operative.

Macroscopic features of the dykes include strongly developed subparallel banding which is prominent in weathered exposures but does not appear to be related to any mineral orientation or concentration. Other features are cross bedding structures (plate 1) and small swirls (plate 3) which probably reflect flow movements in the partly crystalline porphyries during intrusion. Irregular fracturing or sheeting, which is subparallel to the dyke walls, occurs on the dyke margins and produced a series of crusts of porphyry containing strong linear grooves in subparallel sets.

The occurrence of wollastonite in the metasomatised dolomite horizon indicates attainment of at least the hornblende hornfels facies of metamorphism. The muscovite and quartz in the quartzites and shales have been recrystallised with introduction of topaz, carbonate, fluorite and tourmaline.

Petrography of Porphyries

The porphyries at Mt Bischoff exhibit a wide range of composition and texture resulting from the subsequent alteration of quartz-feldspar porphyries as described below.

The unaltered quartz-feldspar porphyries (*e.g.* 1388-1389, 1391, 1393, 30617) consist of bipyramidal cracked quartz phenocrysts up to 5 mm in diameter, with rare braxilian twinning, and lath-shaped phenocrysts of K-feldspar up to 5 mm in length, in a fine-grained groundmass.

In thin section the quartz phenocrysts are abundant, generally ranging in size from 0.5-2.0 mm with a maximum diameter of 5.0 mm and containing small inclusions of K-feldspar. The K-feldspar phenocrysts are partially altered and a D.T.A. curve and X-ray diffraction analysis confirmed the presence of kaolinite. The groundmass consists of a fine intergrowth of quartz and sericitised feldspar. Muscovite occurs sporadically as flakes up to 0.5 mm in length. No plagioclase is present.

Obliquity measurements on two specimens of K-feldspar, comprising several phenocrysts, give Δ values less than 0.18, indicating largely monoclinic symmetry. Optic axial ($2V\alpha$) measurements on several phenocrysts in sections 1388a and 30617 indicate a variation from 36° to 63°, the majority being between 44° and 52°. Measurement of the composition of the K-feldspar indicates that it is almost pure orthoclase ($Or_{100}Ab_0$-$Or_{99}Ab_1$), and that the composition of the homogenised feldspar is $Or_{98}Ab_2$. These results confirm the observation that the K-feldspar is non-perthitic. The equant pyramidal habit of the quartz phenocrysts with small or non-existent prism faces and the strong cracking suggest that the quartz may have reverted from β-quartz which originally crystallised above 573°C (Wright and Larsen, 1909). These features however are not conclusive as the so-called β-quartz habit has been found in the α variety (Frondel, 1945).

Alteration of Porphyries

The greater proportion of the dykes at Mt Bischoff are altered porphyries containing abundant topaz and sulphides and insignificant proportions of original feldspar. They are largely topazites (Johannsen, 1932). They have been described previously by Twelvetrees and Petterd (1897), Weston-Dunn (1922), Reid (1923), Groves (1963) and Groves and Solomon (1964), although the recent availability of drill core has provided fresher material for examination.

The porphyries consist essentially of quartz phenocrysts up to 6 mm in diameter and lath-shaped masses of topaz, quartz, fluorite, carbonate or sulphides, with sporadic muscovite flakes in a fine groundmass of quartz and topaz. The quartz phenocrysts are generally euhedral adjacent to the margins of the dykes but within the dykes are corroded or possess a narrow rim of quartz in optical continuity with the phenocrysts. In some sections (*e.g.* 1433) rows of muscovite inclusions occur parallel to the crystal outlines of the quartz and may represent evidence of overgrowths. Rows of fluid inclusions are present in some crystals that also contain minute inclusions of zircon and topaz. Topaz also occurs as partial rims around the phenocrysts. Lath-shaped aggregates up to 5.0 mm in length are common and are generally multiple bodies of topaz, quartz, fluorite, carbonate, muscovite, cassiterite and sulphides (plate 11).

The fine prismatic topaz which has been called pycnite (*e.g.* Petterd 1910) has refractive indices $\alpha = 1.620$ and $\gamma = 1.610$ and $2V = 62°$ (+)ve indicating that it has low molecular percentage OH/(OH+F) (*e.g.* Deer, Howie and Zussman, 1962, vol. 1, p. 148). A recent analysis of the topaz (table 5) supports the low molecular percentage OH/(OH+F) ≃ 6.5.

Lath-shaped areas consisting of fine sericite and talc are less common (*e.g.* 1468) and large discrete muscovite flakes and rosettes of muscovite are rare (*e.g.* 1439, 30630). Cassiterite is present in most sections as granular, subhedral cracked crystals up to 1 mm in diameter associated with lath-shaped areas of topaz and as smaller crystals, 0.1 mm in diameter, associated with the groundmass. Tourmaline occurs rarely as small euhedral crystals up to 2 mm in length. Sulphides are abundant as anhedral grains in

lath-shaped forms. Pyrrhotite and pyrite are predominant and arsenopyrite, sphalerite and wolframite occur sporadically. An interesting feature of the distribution of sulphides is the strong dominance of either pyrrhotite or pyrite in specific parts of the porphyries; they seldom coexist in equal proportions.

Table 5. PARTIAL ANALYSIS OF TOPAZ FROM MT BISCHOFF

SiO_2	35.72
Al_2O_3	52.33
Fe_2O_3	0.11
TiO_2	nil
CaO	0.26
MgO	0.28
F	17.40
H_2O^-	0.22
H_2O^+	1.20
less O ≡ F	7.33
	100.19

$OH/(OH + F) \simeq 6.5$

Analyst. C.J. Penman, Department of Mines Laboratories, Launceston.

The North-eastern Dyke (fig. 3) is unusual in that it contains large euhedral cubic pyrite crystals, with a side length up to 1 cm, together with subhedral quartz phenocrysts and minor topaz. The pyrite generally occurs as large crystals in the groundmass but in places encloses and replaces quartz phenorcrysts. A feature of the pyrite cubes is the occurrence of zones within the crystals that are oblique to the crystal margins.

A further variety of quartz porphyry occurs about 0.8 km west of Waratah (fig. 40). This porphyry (30633) consists of deformed quartz phenocrysts with elongate aggregates of sericite in a fine quartz-sericite groundmass. It is the only porphyry examined to show deformation textures.

The occurrence of quartz-feldspar porphyries towards the limit of mineralisation, and the retention of relic lath-shaped forms in topazites in which no feldspar is present, suggests that the topaz, sulphides and associated minerals have selectively replaced feldspar in original quartz-feldspar porphyry dykes. This is confirmed by excellent topaz pseudomorphs after orthoclase, which have been extracted from weathered porphyries in the White Face. At White Face the alteration has been so extensive that the porphyritic texture and original composition are obliterated. In Sections 694, 498, 1469 and 2171 topaz occurs as radial aggregates of columnar crystals in association with euhedral zoned crystals of cassiterite up to 3 mm in diameter (plates 12, 13).

In parts of the White Face and the south end of the open cut the porphyries that are in contact with dolomite have been totally replaced by dark green tourmaline, fluorite, carbonate and muscovite (*e.g.* 30634 a-f). In thin section the rocks consist of large crystals of fluorite which contain inclusions of small aligned, acicular crystals of tourmaline and minor sericite and carbonate and are commonly rimmed by sheaf-like masses of muscovite (plate 7). The tourmaline is intermediate between a schorlite and dravite in composition (table 6).

Table 6. ANALYSES OF TOURMALINES FROM MT BISCHOFF AND THE MEREDITH GRANITE

	1477	1402	1478	1*
SiO_2	35.7	36.3	34.8	36.86
TiO_2	0.79	0.00	0.00	nd
Al_2O_3	26.0	32.4	32.8	36.72
FeO (equiv.)	18.1	9.9	12.95	5.66
MnO	0.12	0.24	0.41	0.66
MgO	3.9	5.6	2.9	3.92
CaO	0.86	0.42	0.35	0.34
Na_2O	nd	nd	nd	3.57
K_2O	0.04	0.19	0.05	1.11
P_2O_5	0.01	0.03	0.01	nd
H_2O^+	2.48	2.58	3.20	nd
H_2O^-	0.04	0.04	0.08	nd
B_2O_3	7.72	8.05	9.98	10.56
Total	95.76	95.75	97.53	99.40

Occurrence and Locality: 1477 Vein in Meredith Granite, Corinna Road, 6 km from Waratah; 1402 Replacement of porphyry, S end of Mt Bischoff; 1478 Replacement of dolomite, Power House, Mt Bischoff.

Analyst: D.I. Groves. X-ray fluorescence spectroscopy. 1* Replacement of dolomite, Mt Bischoff, Read (1923, p. 55). Analyst A.D. Mackay.

The White Face Dyke exhibits a compositional gradation from a quartz-feldspar porphyry to a topazite (fig. 42). The quartz-feldspar porphyry at the southern end of the dyke grades in a distance of about 1 metre into porphyries in which the feldspar has been largely replaced by carbonate and sericite with inclusions of quartz, muscovite, topaz and fluorite (*e.g.* 1433, 1468). The sulphides present are predominantly sphalerite and pyrite. Small relic feldspar phenocrysts remain in places (*e.g.* 30618) and the groundmass consists largely of quartz, muscovite and talc. This rock then grades rapidly into an altered porphyry in which topaz, pyrrhotite and pyrite are predominant. The intermediate zone between quartz-feldspar porphyry and topazite is a maximum of 30 m in width. It is interesting that the zonation shown by the White Face Dyke is similar to the zonal sequence of minerals in the replacement deposit. A further feature of the alteration is the occurrence of several small patches of quartz-feldspar porphyry within the topazised dyke. The heterogeneous nature of the alteration is also shown by the occurrence of patches of topazite containing no appreciable sulphides (*e.g.* 1437, 1438) in zones of generally intense sulphide enrichment.

Mineralisation

Tin Mineralisation

At Mt Bischoff the tin mineralisation is mainly restricted to an approximately circular area of 1 km radius from Mt Bischoff summit. The mine workings and individual deposits have been described by several authors (*e.g.* Reid, 1923; Keid, 1943) and are not repeated here.

The main ore bodies have resulted from replacement of the dolomite horizon of the Mt Bischoff sequence by pyrrhotite with associated cassiterite,

pyrite, arsenopyrite, chalcopyrite, sphalerite and stannite accompanying talc, phlogopite, quartz, fluorite and Fe-Mn-Mg carbonates. Pyrite, sphalerite, galena and jamesonite increase, and pyrrhotite decreases, towards the southern margin of the mine area. The mineragraphy and alteration are described in detail in a later section. Folding, faulting and erosion of the southern limb of the Bischoff Anticlinorium has resulted in a series of spatially separated ore bodies which are, from north to south, Brown Face, Slaughteryard Face, Greisen Face, Pig Flat and Happy Valley Face (fig. 3, 40, 47). Sulphide mineralisation is irregular although it is generally more intensive towards the base of the dolomite, particularly in the southern part of the mine (fig. 42, 47). Cassiterite is microscopically associated with non-sulphides and zones of tin concentration are commonly slightly oblique to massive sulphide zones.

Replacement of the dolomite becomes sporadic towards the south of the open cut, and unmineralised dolomite is common in the Happy Valley Face and in drilling intersections south of Pig Flat (fig. 42). The dolomite horizon is presumed to be above the present surface to the north of Mt Bischoff peak (fig. 41) but at its reappearance north of the Waratah River (fig. 40) it is unaltered and unmineralised. The replacement of the dolomite horizon is apparently limited to the area of most intense development and alteration of porphyry dykes in the hinge zone of the anticlinorium. Numerous small veins of sulphides and cassiterite occur below the replacement deposit in this area.

The search for further ore is obviously tightly controlled by prediction of the occurrence of the dolomite horizon outside the area of the open cut, and much of the recent exploration has been concerned with locating this horizon. The dolomite horizon has never been found at any distance from the open cut except on the north side of the Waratah River (fig. 40), although an extensive search has been made for it. The lack of stratigraphic marker horizons other than the dolomite, the structural complexity of the area, the possible lack of lateral extension of the dolomite, and the relative lack of mineralisation away from the area of the open cuts are all serious impediments to this search. As the author is aware of the latest drilling results but cannot use them from an economic standpoint, it is impossible to give any unbiased recommendations for further exploration.

Fissure vein deposits carrying cassiterite occur throughout the area within a 1 km radius of Mt Bischoff peak. The lodes occupy late fractures which displace porphyry dykes and cut the replacement ore body. They have strike and dip lengths up to 750 m and 300 m respectively and pinch and swell along both dip and strike. The lodes commonly branch and converge forming a complex system of subsidiary ore bodies. They consist essentially of quartz and/or carbonate with fluorite and tourmaline carrying cassiterite, wolframite, pyrite, pyrrhotite, arsenopyrite, sphalerite, chalcopyrite, galena, jamesonite, bismuthinite and stannite. The relative proportions of these constituents vary considerably from one vein to another (*e.g.* Stillwell, 1943).

The major deposits include the North Valley Lode, Giblin Lode and Thompson's Lode (fig. 3). The North Valley lode consists essentially of quartz and pyrite carrying cassiterite in association with carbonate, muscovite, fluorite, sphalerite, chalcopyrite, galena etc. (Stillwell, 1943). The Giblin Lode also has a predominantly quartz gangue with pyrite, arsenopyrite, sphalerite and cassiterite. Zoning has been recorded from this lode (Weston-Dunn, 1922), cassiterite and wolframite being abundant and galena rare near Mt Bischoff peak while sphalerite, galena and jamesonite are predominant and cassiterite and wolframite are rare towards its south-west extremity. Thompson's Lode is unusual in that the gangue consists largely of fluorite and Fe-rich carbonates with only minor quartz and contains abundant cassiterite with rare sulphides (sphalerite and pyrite).

Fooks Lode (fig. 40) occurs outside the general area of tin mineralisation but contains cassiterite in association with abundant fluorite, sphalerite, pyrite and jamesonite. Recent drilling by the Department of Mines has proved the extension of Fooks Lode for 300 m in an E-ESE direction from the old workings. The lode which is dipping at a variable angle to the south has been drilled to 75 m down dip and is approximately 3 m wide and averages 1% tin. The drilling programme is incomplete and will be reported in detail at a later date.

Apart from these major veins, which generally have an average width of about 0.5-1 m, there are subparallel sets of minor veins, some 2-5 cm in width and a few metres long, which commonly consist solely of quartz and cassiterite. Both major and minor veins generally trend from NW to NNW and dip steeply west (fig. 46a).

The quartz porphyry intrusives have been extensively mineralised by topaz, tourmaline, fluorite, Fe-sulphides and cassiterite. These minerals generally form pseudorphs after K-feldspar. In places the proportion of disseminated cassiterite was high enough for the porphyry to consitute an orebody (*e.g.* White Face dyke). In addition cassiterite occurs with quartz and tourmaline on joint faces intersecting the porphyry intrusives. Where the joint frequency is high, the porphyries have been mined by quarrying (*e.g.* Stanhope Face and eastern end of Western Dyke).

Lead-Zinc-Silver-Antimony Mineralisation

Isolated, small Pb-Zn-Ag veins occur throughout the Waratah district (fig. 39), and very small occurrences are relatively abundant around the area of tin mineralisation at Mt Bischoff. They generally occur within the Mt Bischoff sequence or associated with Cambrian volcanic rocks (fig. 39).

Several veins occur close to the margin of the Magnet Dyke. The largest deposit occurs at the Magnet Mine where a vein some 3-5 m wide occurs on the contact of spilite (and dolerite) and websterite (fig. 41). The lode is composed essentially of carbonates with bunches and veinlets of galena and sphalerite with minor arsenopyrite, pyrite, boulangerite, pyrargyrite, tetrahedrite and chalcopyrite (Edwards, 1960). Crustification and cockade textures are abundant in the ore and there is strong evidence of post-ore deformation. The ore body has been described by Nye (1923), Cottle (1953) and Groves (1965). A small Pb deposit occurs in a similar structural position at the Persic Section (fig. 39) where irregular uneconomic splashes of galena and carbonate are present (Nye, 1923). Small veins of quartz and carbonate carrying galena and chalcopyrite occur in a similar structural position at Fawkner's Show (fig. 39).

Small, uneconomic Pb-Zn-Ag veins occur within the Mt Bischoff sequence around Mt Bischoff, the largest being at the Silver Cliffs Mine where a well banded lode consists of galena, jamesonite, sphalerite, pyrite and minor boulangerite in a gangue of carbonates and quartz. A similar lode occurs north of the Waratah River (fig. 40) and is subparallel to the North Valley Lode. Small banded lodes consisting of jamesonite, stibnite, galena, tetrahedrite, boulangerite, pyrite, arsenopyrite, pyrrhotite, marcasite and pyrargyrite occur in Tinstone Creek (fig. 40) and appear to be related to the Pb-Zn-Ag deposits which commonly contain abundant jamesonite.

Oxidation of the Orebodies

Gossans probably developed in the Tertiary over several outcropping sulphide orebodies. The gossan over the Pb-Zn-Ag orebodies (*e.g.* Magnet) are largely indigenous and consist essentially of limonite with bands of secondary

minerals such as cerussite, anglesite, pyromorphite and crocoite. Gossans derived from the pyrrhotite-rich ores at Mt Bischoff formed extensively and were apparently mixtures of indigenous and exotic types. Pyrrhotite is very unstable in temperate conditions and breakdown resulted either in development of friable limonitic crust with cassiterite concentrated at the base or in complete removal of Fe and S, leaving cassiterite-quartz sand on the surface. Where the orebodies cropped out on hillsides the cassiterite-quartz sand travelled down slope to form extensive eluvial and alluvial deposits, which are still being worked at present.

Post-Mineralisation Activity

Dolerites of probable Jurassic age intrude rocks of the Arthur River sequence in the Magnet area (Groves and Solomon, 1964). These rocks (Department of Mines, 715 and 35-G-2) consist of augite and labradorite forming an opitic to doleritic texture with interstitial quartz, feldspar and magnetite.

Irregularly distributed deposits of Tertiary sediments (probably lacustrine), which include conglomerate, gravel, sand and lignite occur up to a maximum thickness of 30 m in the Waratah area (Groves and Solomon, 1964), and have been dated palynologically as Mid-Tertiary (Harris, 1968). At Don Hill these sands have been deposited against low bedrock cliffs, and slumping and differential compaction has produced folds and steep dips near these cliffs (plate 2). Some of the basal conglomerates contain small quantities of cassiterite and gold.

Basalts occur extensively over the Waratah area, to a maximum thickness of 60 m. Recent drilling by the Department of Mines near Fook's Lode has shown that there are at least two flows of basalt separated by terrestrial gravels in the lower part of the sequence which is exposed in cliffs on either side of the Waratah River (fig.40). The following descriptions have been provided by F.L. Sutherland of the Tasmanian Museum.

The lower basalt flows contain olivine phenocrysts, up to 3.2 mm in diameter, in an intergranular groundmass of zoned labradorite, colourless augite, dark glassy mesostasis charged with iron oxides, and amygdaloidal chalcedony, opal and carbonate (Tasmanian Museum T.S. 114-117). Compositionally similar but texturally dissimilar basalts occur also in the small quarry near Mackay's dam. The lower flows are transitional tholeiitic olivine-basalts, varying in type from the Jordan Type to the Midlands Type of McDougall (1959).

The lower flows are apparently overlain by a porphyritic basalt exposed in the quarry on the eastern side of Waratah. This type contains glomerophenocrysts of olivine and zoned labradorite up to 4.5 mm in diameter in an intergranular groundmass of labradorite, slightly titaniferous augite, olivine, iron oxide and minor mesostasis (T.S. 1445). The rock is a near-saturated alkali olivine-basalt and represents the Waratah Type of Edwards (1950).

The topographically highest basalt cropping out on the hill behind Waratah contains olivine glomerocrysts up to 2.2 mm in diameter in a subophitic to ophitic groundmass of zoned labradorite, titanaugite, iron oxide and minor carbonate and clay (T.S. 1544). This rock is an undersaturated alkali olivine-basalt resembling the Deloraine Type of Edwards (1950).

F.L. Sutherland (pers. comm.) considers that the Waratah lavas represent a tholeiitic association passing transitionally into more undersaturated alkaline basalts in the upper part of the succession. Similar sequences are known in the tholeiitic basalt associations at Great Lake in the Central Highlands (Sutherland and Hale, 1970).

Quaternary gravels and sands occur in the present streams and in places contain considerable cassiterite (*e.g.* Reid, 1925). Thin veneers of gravels and cassiterite-bearing soils occur over much of the slopes of Mt Bischoff, and have been successfully sluiced on a small scale in recent years.

MINERAGRAPHY OF THE MT BISCHOFF ORES

Introduction

The stratiform replacement cassiterite-sulphide deposits at Mt Bischoff have a generally simple mineralogy with a limited number of metal sulphide

Table 7. COMPOSITION OF MORE IMPORTANT MINERAL OCCURRENCES AND ORE DEPOSITS OF THE WARATAH AREA INCLUDING INFORMATION FROM STILLWELL (1943, 1945, 1947) AND EDWARDS (1960)

Mineral	Localities								
	1	2	3	4	5	6	7	8	9
Arsenopyrite	xx	xxxx	xxxx	xx	xx	xx			xxx
Bismuthinite	x	x							
Boulangerite	x					x	x		xx
Cassiterite	xx	xx	xx	xx	x				
Chalcopyrite	xx		xx				xx		x
Galena	x		xx	xx			xxx	xxxx	xxxx
Hematite	x								
Jamesonite	x	x	x		xx	xxxx	xxx		
Marcasite	x		xx						
Molybdenite		x							
Native Arsenic		x	x						
Native Bismuth		x	x						
Native Copper		x							
Pyrargyrite						x			xx
Pyrite	xx	xxxx	xxxx	xx	xx	xx	xxx	xx	xxx
Pyrrhotite	xxxx		xx			x			
Sphalerite	xx	xxx	xx	xx	xx	xx	xxx		xxxx
Stannite	x	x	xx						
Stibnite		x	x			xx			
Tetrahedrite	x		x				x		xx
Wolframite	x	xx	x						

x Rare xx Common xxx Abundant xxxx Dominant

Localities: 1. Replacement Deposit, Mt Bischoff; 2. Giblin Lode, Mt Bischoff; 3. North Valley Lode, Mt Bischoff; 4. Thompson Lode, Mt Bischoff; 5. Fooks Lode, Waratah; 6. Antimonial Lodes, Tinstone Creek, Waratah; 7. Silver Cliff Lodes; 8. Persic Lodes; 9. Magnet Lodes.

and oxide components. Macroscopically the ore consists predominantly of pyrrhotite with minor pyrite, arsenopyrite, chalcopyrite and sphalerite. Cassiterite is generally too fine-grained and indistinct to be visible macroscopically although clusters of cassiterite crystals are rarely visible. The sulphide mineralogy of the fissure veins is generally more varied and the cassiterite is much coarser grained. The sulphides and oxides recognised microscopically in this investigation and previous mineragraphic examinations by Stillwell (1943, 1945), are listed in Table 7, and their relative abundance in the respective ore deposits and mineral occurrences is shown. The ore at Mt Bischoff has averaged about 1% tin throughout the life of the mine.

Modal analyses of typical ore specimens were carried out (see Groves, 1968, appendix D2) and the elemental composition of the sulphide phase calculated (table 8). It is evident that almost 98 wt% of the sulphide components are Fe and S and that the bulk Fe/S ratio is 1.38/1 compared to 0.87/1 for pyrite and 1.74/1 for stoichiometric FeS.

The ore minerals occurring in the deposits at Mt Bischoff are described in the following section. The descriptions are based primarily on the replacement deposit, although similar textures are evident in most of the cassiterite-bearing fissure veins. The mineragraphy of the Magnet Deposit has been described by Edwards (1960, and the Antimonial Lodes and North Valley Lode by Stillwell (1947) and Stillwell (1943) respectively. The mineralogy of several of the small lodes is summarised by Groves (1963).

Although interpretation of textural relationships can only be tentative it is apparent that cassiterite and wolframite have crystallised before the sulphides.

Table 8. MODAL ANALYSES OF ORE FROM REPLACEMENT DEPOSIT, MT BISCHOFF

Mineral	Volume %	Weight %	Metal	Weight %
Arsenopyrite	1.0	1.5	As	1.0
Chalcopyrite	0.7	0.8	Cu	0.5
Galena	0.1	<0.1	Fe	56.7
Jamesonite	0.2	0.3	Pb	0.25
Pyrite	4.3	5.5	Sb	0.13
Pyrrhotite	49.2	57.7	Sn	0.12
Sphalerite	0.3	0.3	Zn	0.25
Stannite	0.3	0.3	S	41.0
Supergene Pyrite	2.2	2.8		99.93
Tetrahedrite	0.1	<0.1		
			Fe : S	1.38 : 1
Non-sulphides	41.7	33.6		
	100.0	100.0		

Weight percentage calculated from modal analyses (volume per cent) assuming a specific gravity of 2.9 for the non-sulphide phase. Elemental composition calculated from mean compositions of components (weight per cent).

The general sequence of deposition of the ore minerals appears to be:

Cassiterite, wolframite
Arsenopyrite (?)
Pyrrhotite, pyrite (?), chalcopyrite
Pyrite, marcasite
Chalcopyrite, sphalerite, stannite
Jamesonite, galena, boulangerite, tetrahedrite.

Detailed descriptions of the minerals in their approximate sequence of formation follow.

Systematic Mineral Descriptions

Wolframite

Wolframite is present as rare bladed crystals, up to 0.5 mm in length, associated with quartz in most deposits. It is most commonly associated with topaz in the porphyry dykes (*e.g.* Tasmanian Museum 2231), or as encrustations with quartz on joints in these dykes. A chemical analysis (TM 2231) indicates 5.1 wt% MnO and 16.7 wt% FeO.

Cassiterite

Cassiterite occurs throughout the ores as granular cracked crystals which average between 0.05 and 0.1 mm in diameter. The maximum diameter of cassiterite grains in the sections examined is approximately 1 mm, and these larger cassiterite crystals are commonly zoned. The cassiterite grains generally occur in clusters in non-sulphide gangue, commonly quartz, carbonate or micaceous minerals. They are relatively rarely enclosed in sulphides where they have been corroded and invaded along irregular cracks by the enclosing sulphides. Partial rims of stannite, similar to those described by Novák *et al.* (1962) are rarely developed at cassiterite=pyrrhotite interfaces.

Arsenopyrite

Arsenopyrite occurs generally as small euhedral crystals forming a minor component of the ore. It encloses small quartz and cassiterite grains, and rarely contains inclusions of pyrrhotite. Etching with 1:1 HNO_3 reveals well developed growth zoning, particularly in the larger, discrete crystals. Euhedral crystals of arsenopyrite are commonly partially replaced by pyrrhotite with convex embayments of pyrrhotite against arsenopyrite. Commonly the arsenopyrite is fractured by veinlets of pyrrhotite, chalcopyrite and stannite and more rarely by sphalerite. Matched irregularities in vein walls and offsets of zonal textures proportional to the width of the dyke and the angle of intersection demonstrate a dilational emplacement. This may indicate that the arsenopyrite is pre-pyrrhotite in the deposition sequence, or may be due to slight post-ore deformation with cracking of the more brittle arsenopyrite.

Pyrrhotite

Pyrrhotite is the predominant sulphide in the replacement ores, but in the fissure lodes at Mt Bischoff it is subordinate to pyrite. It is almost free of other sulphides in the Brown and Slaughteryard Faces, but south of the Greisen Face the ore becomes richer in pyrite, sphalerite and other sulphides. The pyrrhotite occurs as anhedral grains which form large irregular interlocking masses and small disseminations. It replaces and invades fractures in cassiterite and arsenopyrite, and is itself replaced and invaded

by second generation pyrite and marcasite, chalcopyrite, sphalerite and non-sulphide gangue minerals. Growth twinning and/or transformation twinning is rarely present in the pyrrhotite. A minor second generation or pyrrhotite occurs as minute exsolution bodies in sphalerite.

Etching with a saturated solution of chromic acid produces different etch reactions in the two structural types of pyrrhotite (*e.g.* Arnold, 1966). Mixtures of hexagonal and monoclinic pyrrhotite (determined from X-ray diffraction) from the replacement deposits were etched and found to consist predominantly of hexagonal pyrrhotite. Elongate blebs of monoclinic pyrrhotite, recognised by its more intense etching, occur in the host of hexagonal pyrrhotite. The lamellae are apparently crystallographically orientated and have the appearance of an exsolved phase; the lamellae generally terminate at grain boundaries in the host.

Pyrite

Pyrite is ubiquitous in the deposits. It occurs in a few sections as coarse-grained, euhedral crystals that possibly formed penecontemporaneously with pyrrhotite. This pyrite contains only rare inclusions of arsenopyrite and cassiterite, and encloses small grains of chalcopyrite in places.

A second generation of pyrite, probably resulting from the hypogene alteration of pyrrhotite, occurs typically in the ores. The distribution of this pyrite is not controlled by the present land surface and in some sections it is apparently replaced by jamesonite (*e.g.* Tasmanian Museum, 447) or chalcopyrite (15-86). The early stage in alteration appears to be the formation of aggregates of granular pyrite containing minute patches and fine specks of marcasite with fine-grained carbonate or quartz. These masses appear to have recrystallised in places to form areas of bladed or lamellar pyrite, containing marcasite, with interstitial carbonate or quartz. In places euhedral to subhedral pyrite crystals occur within the lamellar pyrite-marcasite zones. In many examples (*e.g.* plate 4) the pyrite crystals are completely surrounded by quartz or carbonate and it appears probable that pyrite and quartz-carbonate have recrystallised and segregated from the lamellar pyrite-marcasite and quartz-carbonate areas. Fryklund (1964) observed similar alteration from the ore deposits of the Coeur d'Alene District, Idaho and also considered the alteration to be hypogene and in the direction suggested here. Kullerud (1967) has shown that pyrite and marcasite are stable together up to 432±30°C, this temperature being independent of confining pressure.

A further generation of pyrite is present in some specimens which occur adjacent to the present surface at Mt Bischoff. The pyrite is finely granular or framboidal and occurs in colloform masses, up to 5 mm in diameter, in pyrrhotite. These masses commonly have a rim of uniformly orientated marcasite and contain small irregular grains of hematite with rare chalcocite. In places botryoidal forms have developed around small veins of quartz and carbonate in the pyrrhotite (plate 5). In these examples rhythmically banded granular pyrite, commonly containing small specks of marcasite, alternates with earthy layers of unknown composition. The concentric forms commonly have cores of non-banded, framboidal pyrite or are centred on coarsely crystalline irregular pyrite masses (plate 5). This generation of pyrite (and marcasite) is considered to be supergene.

Marcasite

Early reports on the outcropping cassiterite-sulphide ores at Mt Bischoff (*e.g.* Reid, 1923) stressed the abundance of supergene marcasite. At present, with no gossans being stripped, it is extremely rare at Mt Bischoff, where

it is only present in association with hypogene and supergene pyrite as described above.

Chalcopyrite

Chalcopyrite is present as small disseminated anhedral grains in pyrrhotite, or small veinlets at the boundaries of pyrrhotite with non-sulphide gangue minerals. The chalcopyrite also commonly occurs as veins in pyrrhotite and pyrite and is partially replaced by sphalerite, stannite, quartz and carbonate.

The chalcopyrite contains ubiquitous exsolution bodies of sphalerite which are present as minute, star-shaped skeletal crystals, rarely greater than 0.01 mm in length. Larger blebs of sphalerite occur adjacent to the grain boundaries of the chalcopyrite and probably represent migration of the sphalerite towards the grain boundaries. Brett (1964) found that this was a common effect during exsolution in experimental studies in the Cu-Fe-S system. In several sections (*e.g.* 62-425, 100-135) small irregular included patches of sphalerite occur in chalcopyrite grain boundaries and contain minute exsolution lamellae of chalcopyrite. This appears to be an example of progressive unmixing, with exsolved sphalerite subsequently exsolving a further generation of chalcopyrite. It may be significant that Edwards (1954) recorded a higher temperature for exsolution of sphalerite from chalcopyrite than chalcopyrite from sphalerite as this would be in agreement with the sequence described.

Microscopic blebs of pyrrhotite also occur within the chalcopyrite and may represent exsolution bodies although they do not exhibit crystallographic orientation within the chalcopyrite. Rare native copper and cuprite have probably been derived from oxidation of chalcopyrite.

Sphalerite

Sphalerite occurs sporadically throughout the ore and appears to have formed penecontemporaneously with chalcopyrite. It occurs as granular interlocking aggregates which generally form irregular areas up to 15 mm in length and 1-2 mm in width. The sphalerite replaces cassiterite, arsenopyrite, pyrite and pyrrhotite and is replaced by galena, jamesonite and carbonates. Etching with HI reveals coarsely crystalline aggregates which have well developed growth and/or translational twinning. Further minor generations of sphalerite are represented by exsolution bodies in chalcopyrite and stannite.

The sphalerite in most sections contains minute blebs of chalcopyrite, generally less than 0.01 mm in diameter, which form 'emulsion' or 'mottled' texture. The chalcopyrite bodies vary widely in size and are heterogeneously distributed throughout the sphalerite. Larger bodies of chalcopyrite, up to 0.1 mm in diameter, occur preferentially along grain boundaries in the sphalerite and are commonly surrounded by haloes of minute chalcopyrite blebs while more distant areas of sphalerite may be free of such accumulations. This is probably due to arrested diffusion of exsolution chalcopyrite into discrete bodies at sphalerite grain boundaries. The more common phenomenon of 'drained' areas adjacent to coarse chalcopyrite bodies (*e.g.* Edwards, 1954) is rare in these ores.

More rarely small blebs of pyrrhotite and stannite occur within the sphalerite and are probably exsolution bodies. In some sections the sphalerite contains spherical exsolution bodies of chalcopyrite and pyrrhotite and some compound bodies of both sulphides. Lawrence and Chand (1962) recorded similar exsolution textures in the Rockvale Mine, N.S.W. and termed

the process of formation 'exsolution pseudomorphism'. They considered that the spheroids formed by pseudomorphism of exsolved chalcopyrite by penecontemporaneously exsolved pyrrhotite that was drawn out of solid solution in the wake of diffusing chalcopyrite.

Compound exsolution bodies of chalcopyrite and stannite also occur in sphalerite. The exsolution bodies are irregular in shape and some show marked discontinuity at the junction of chalcopyrite and stannite indicating accretion, while some appear to have resulted from replacement of chalcopyrite by stannite.

Stannite

Small grains of stannite (average diameter 0.01 mm) are present in small proportions in most specimens that contain sphalerite and/or chalcopyrite. The stannite has been confirmed by X-ray diffraction and is generally olive-green in colour and isotropic. This makes the optical distinction between stannite and tetrahedrite difficult. Minor buff-coloured stannite from Mt Bischoff (*e.g.* PS 8-44) is anisotropic.

The stannite occurs as small areas which enclose or are enclosed by sphalerite and chalcopyrite with the development of 'mutual boundaries' in places. These minerals appear to have formed penecontemporaneously. Stannite which is associated with sphalerite generally occurs as small inclusions within the sphalerite, as elongate masses along sphalerite grain boundaries (plate 6), and as narrow rims between sphalerite and non-sulphide gangue minerals. Small areas of stannite also rarely occur within pyrrhotite and as partial rims around cassiterite crystals enclosed by pyrrhotite, similar to those described by Novák *et al.* (1962) from Kutná Hora. Further minor generations of stannite are represented by exsolution bodies in chalcopyrite and sphalerite.

The stannite almost universally contains minute exsolution bodies of chalcopyrite which are only observed under high magnification (x 1250). In some sections star-shaped skeletal crystals of sphalerite, similar to those exsolved from chalcopyrite, occur in the stannite. In several instances small discrete granular bodies of chalcopyrite occur on the margins of the exsolved sphalerite, and in a few cases they occur between the lobes of the star-shaped bodies. The textures suggest that sphalerite exsolution bodies formed initially, with subsequent accretion of chalcopyrite.

Tetrahedrite

It is not known whether the mineral belonging to the tetrahedrite group in these ores is tetrahedrite, tennantite or an intermediate type, as insufficient is present for X-ray diffraction or micro-chemical analysis. It is generally olive to buff-grey in colour suggesting that it may be tennantite. In the specimens examined it is found exclusively as small inclusions in arsenopyrite and galena and is an almost insignificant component of the ores.

Jamesonite

Jamesonite occurs as a minor constituent of the ores at Mt Bischoff. Numerous specimens from Mt Bischoff which have been referred to as berthierite and boulangerite by Petterd (1910) and Reid (1923) have been shown to be jamesonite by X-ray diffraction. The jamesonite is particularly abundant in the fissure lodes.

The jamesonite at Mt Bischoff occurs as large curved fibrous masses

replacing sphalerite or as large acicular crystals, up to 1 cm in length, replacing pyrrhotite, arsenopyrite, sphalerite and second generation pyrite.

Galena

Galena is generally a rare constituent of the cassiterite-sulphide ores. It occurs as fine threads and anhedral grains replacing pyrrhotite and sphalerite. Rare veins of coarsely crystalline galena occur (*e.g.* Tasmanian Museum, 2217) and contain irregular patches and anastomosing veinlets of boulangerite. Small inclusions of probable native bismuth also occur in galena. The rare sulphosalts canfieldite and franckeite which have been recorded from Renison Bell (Stillwell and Edwards, 1943) have not been seen in this study.

Other minerals that are generally confined to rare occurrences in the fissure lodes include native arsenic, molybdenite, bismuthinite, stibnite and pyrargyrite (table 7, p. 201). Secondary carbonates (azurite and malachite) and Fe, Cu and Zn sulphates occur in minor amounts throughout the mine.

Solid Solution

An interesting feature of the ores is the range of solid solution shown by sphalerite, chalcopyrite, stannite and pyrrhotite. The resultant exsolution pairs are summarised below:

Solvent	*Solute*
Sphalerite	Pyrrhotite
Sphalerite	Pyrrhotite + chalcopyrite (pseudomorphism)
Sphalerite	Chalcopyrite
Sphalerite	Chalcopyrite + stannite (accretion)
Sphalerite	Stannite
Chalcopyrite	Sphalerite (sphalerite 2nd generation chalcopyrite)
Chalcopyrite	Pyrrhotite
Chalcopyrite	Stannite
Stannite	Sphalerite
Stannite	Sphalerite + chalcopyrite (accretion)
Stannite	Chalcopyrite

The formation of the compound exsolution bodies has not been satisfactorily explained by Lawrence and Chand (1962). Guy (1959), Brett (1964) and others have suggested that nucleation of atoms of the exsolved phase begins when they collect in sufficient concentrations to form nucleii at points throughout the lattice of the solvent, the particles growing spontaneously when they have achieved a certain critical size. The nucleations are facilitated by imperfections in the lattice of the solvent. A concentration gradient is produced due to removal of the exsolved phase in the vicinity of the growing bodies with acceleration of diffusion and subsequent growth of the exsolution bodies. As the exsolution bodies grow they cause an increase in strain energy due to volume differences with the host and a series of dislocations form around each body. If a further phase exsolves from the same solvent it will again nucleate at points of high strain energy in the lattice. The previously formed exsolution body interfaces probably represent the areas of highest strain energy excepting grain boundaries in the matrix and thus will be favourable loci for nucleation of the second exsolved phase. The

driving force for the replacement of chalcopyrite by pyrrhotite ('exsolution pseudormorphism') is probably the disequilibrium of chalcopyrite in a host saturated with FeS.

Most of the exsolution bodies tend towards a spheroidal shape which results in a minimum surface area with a reduction of the surface free energy of the exsolved body. An exception is the star-like forms of sphalerite in chalcopyrite. These forms are generally elongated in the (111) plane of the chalcopyrite as there is a mutual S-plane parallel to this crystallographic direction (Gruner, 1929). It is interesting that the relative shapes involved in the sphalerite-chalcopyrite exsolution pair are the reverse of those indicated from measurements of interfacial free energy ratios from dihedral angles at two phase triple junctions by Stanton (1964). Stanton (1964) interpreted the occurrence of sphalerite as spheroids in galena, and galena as cuspate forms in sphalerite in terms of a higher surface free energy of the sphalerite determined by the interfacial free energy ratio, galena : galena/sphalerite : sphalerite = 0.61. The determined free energy ratio chalcopyrite : chalcopyrite/sphalerite : sphalerite is 0.75 suggesting that the sphalerite should occur as spheroids in chalcopyrite and chalcopyrite as cuspate forms in sphalerite, whereas the reverse geometric relationship is observed.

NON-SULPHIDE COMPONENTS OF MT BISCHOFF DEPOSITS

Replacement Deposit

A wide variety of silicates, oxides, fluorides and carbonates have replaced dolomite at Mt Bischoff. Pre-sulphide minerals occur rarely and include cassiterite, tourmaline, quartz, wollastonite and chondrodite. Garnet and corundum are also probably pre-sulphide formation. These minerals are extensively replaced by hydrous and fluorine-bearing minerals including fluor-phlogopite, talc, chlorite, muscovite, serpentine, fluorite and sellaite, and by several generations of carbonates and quartz. Minor amounts of sphene, apatite, topaz and zeolite are also present.

A macroscopic banding is evident in the majority of specimens examined. Adjacent bands are rich in sulphides, tourmaline, carbonates, fluorite, sellaite or micaceous minerals. The micaceous bands are commonly separated into talc- and phlogopite-rich layers. Cassiterite also occurs in clusters in distinct, generally sulphide-deficient layers.

In thin section tourmaline occurs as small, idioblastic, zoned crystals which are commonly included in fluorite, sulphides and carbonate (plate 7). The crystals are commonly 0.05 mm in diameter in cross-section and 0.5 mm along their long axis. The tourmaline has refractive indices $\omega = 1.6580$, $\varepsilon = 1.6325$, $\delta = 0.0255$ and chemical analyses (table 6) indicate a variable composition between schorlite and dravite, in agreement with the optical properties. It is essentially a magnesian schorlite.

Quartz occurs as rare idioblastic crystals which are enclosed by later sulphides and micaceous minerals.

Garnet has only been identified in one section from the Greisen Face (1532) where it occurs as idioblastic crystals, approximately 1 mm in diameter, which have been replaced by talc and quartz. Insufficient garnet is present for identification of its composition.

Corundum has also been recognised in one section from Greisen Face (1535) where it occurs as short, rectangular, prismatic, weakly birefringent

crystals, generally 0.06 x 0.04 mm, which are included in pyrrhotite, carbonates and serpentine. The occurrence of corundum is unusual as the assemblage is generally Al-deficient, although topaz is also present in minor proportions.

Chondrodite is abundant in specimens from the eastern end of Brown Face (*e.g.* 33453) and occurs rarely in other sections from Greisen Face (*e.g.* 1517). In section 33453 it has been extensively replaced by serpentine (plate 8) and to a lesser extent by pyrrhotite. It is colourless and non-pleochroic, has $2V = 72° - 82°$ (+)ve (6 determinations), $\alpha = 1.615$ and $\beta = 1.635$ and $\alpha : z = 30° - 32°$ (5 determinations). It has complex twinning (plate 9): it is twinned on (001) and other composition planes which have not been determined. The identification has been confirmed by X-ray diffraction. Its typical occurrence is with grossular and wollastonite in metasomatised limestones (Deer *et al.* 1962, vol. 1, p. 48).

Wollastonite occurs in some sections from the Greisen Face (*e.g.* 1502*b*, 1535) as columnar aggregates with a maximum diameter of 2 mm, which have been replaced by sulphides, micaceous minerals and carbonates (plate 10). The presence of wollastonite could not be confirmed by X-ray diffraction because of its rare occurrence. It is colourless and weakly birefringent, $2V = 50°$ (-)ve with the optic axis almost normal to the length of the crystal. Longitudinal sections have a maximum extinction of 12° and the refractive index is within the range for wollastonite as it lies between $\pi\varepsilon$ and $\pi\omega$ for the carbonate. The presence of wollastonite has thermometric significance as Harker and Tuttle (1956) have determined the univariant P_{CO_2} - T curve for the reaction $CaCO_3 + SiO_2 \rightleftarrows CaSiO_3 + CO_2$. The curve indicates that the minimum temperature of formation is approximately 400°C at atmospheric pressure, and about 650°C at a P_{CO_2} of 1000 atmospheres. Fluid inclusions data (see later section) indicates a confining pressure of about 750 atmospheres, and as it is doubtful if P_{CO_2} equalled the total pressure it is probable that the formation temperature lies between 650°C and 400°C.

Fluorite, which varies from pink, purple and green in colour to colourless, is generally post-sulphide although some large pods which occur in the Slaughteryard and Greisen Faces appear to be penecontemporaneous with pyrrhotite. In thin section it contains inclusions of quartz and zoned tourmaline (plate 7) and is partly replaced by micaceous minerals. It also occurs as a late stage mineral in vughs in carbonate. The cell size of the fluorites (5.4622 to 5.4630 ± 0.0005 Å) is independent of their colour, and is probably due to trace impurities rather than high strontium contents (Steyn, 1954).

Sellaite occurs abundantly in some sections from Greisen Face. It is weakly birefringent, uniaxial (+)ve and has an extremely low refractive index ($\omega = 1.378$). Its identification has been confirmed by X-ray diffraction. It appears to have formed penecontemporaneously with fluorite (*e.g.* 1514) and is typically replaced by fluor-phlogopite and talc. Sahama (1945) recorded that sellaite was extremely rare relative to fluorite in natural occurrences, but that sellaite had been recorded from metasomatised dolomites. It occurs much more commonly in evaporite sequences (*e.g.* Kuhn, 1952; Herrmann and Hoffman, 1961). At Mt Bischoff it has probably formed from the reaction of excess F with $MgCO_3$ when available $CaCO_3$ has been replaced. The almost complete removal of $CaCO_3$ is indicated by compositions of carbonates in the replacement deposit (fig. 48). Sellaite is not a stable phase during experimental formation of fluorite from dolomite by alkaline fluoride-bearing solutions in the kinetic system dolomite - NaF, and magnesite is the stable form in the magnesite - NaF system (Ames, 1961). However the maximum temperature and pressure of the experiments was only 60°C and 1 atmosphere respectively.

Table 9. PARTIAL ANALYSIS OF CARBONATE ROCKS (WT%) FROM REPLACEMENT ORE-BODY AND LODE DEPOSITS, WARATAH

	3	4	5	100209	100210	100211	100219	100217	100218
SiO_2	6.00	1.00	9.54	1.9	6.6	28.7	3.8	0.0	0.0
equiv. FeO	2.56	12.72	21.37	28.8	28.7	19.3	29.0	42.7	27.7
MnO	0.92	13.26	13.20	20.3	16.5	4.8	5.1	8.1	27.5
MgO	11.75	14.46	16.57	4.8	7.4	17.0	16.9	5.0	2.5
CaO	21.08	3.20	0.92	1.3	0.5	2.1	0.2	0.5	0.5
approx. CO_2	49.6	47.4	37.6	34.1	36.6	32.0	41.5	38.4	36.7
H_2O-	7.00(±)	6.40(±)		0.06	0.05	0.02	0.01	0.02	0.03
Total	98.91	98.44	99.20	91.26	96.35	103.92	96.51	94.72	94.93
Sr (ppm)				<3	14	9	9	<3	<3
Rb (ppm)				28	12	11	12	<3	<3

Sample Localities: 3, 4. Happy Valley Face, Mt Bischoff, 5. Greisen Face, Mt Bischoff, 100209-100211, 100219. South end of open cut, Mt Bischoff, 100217. Thompson's Lode, Waratah, 100218. Silver Cliffs Lode, Waratah.

Analysts: 3, 4. from Reid (1923, p. 86), 5. Department of Mines Laboratories, Launceston, 1962, from Groves (1963), 100209-100211, 100217-100218. D.I. Groves, 1967 - X-ray fluorescence.

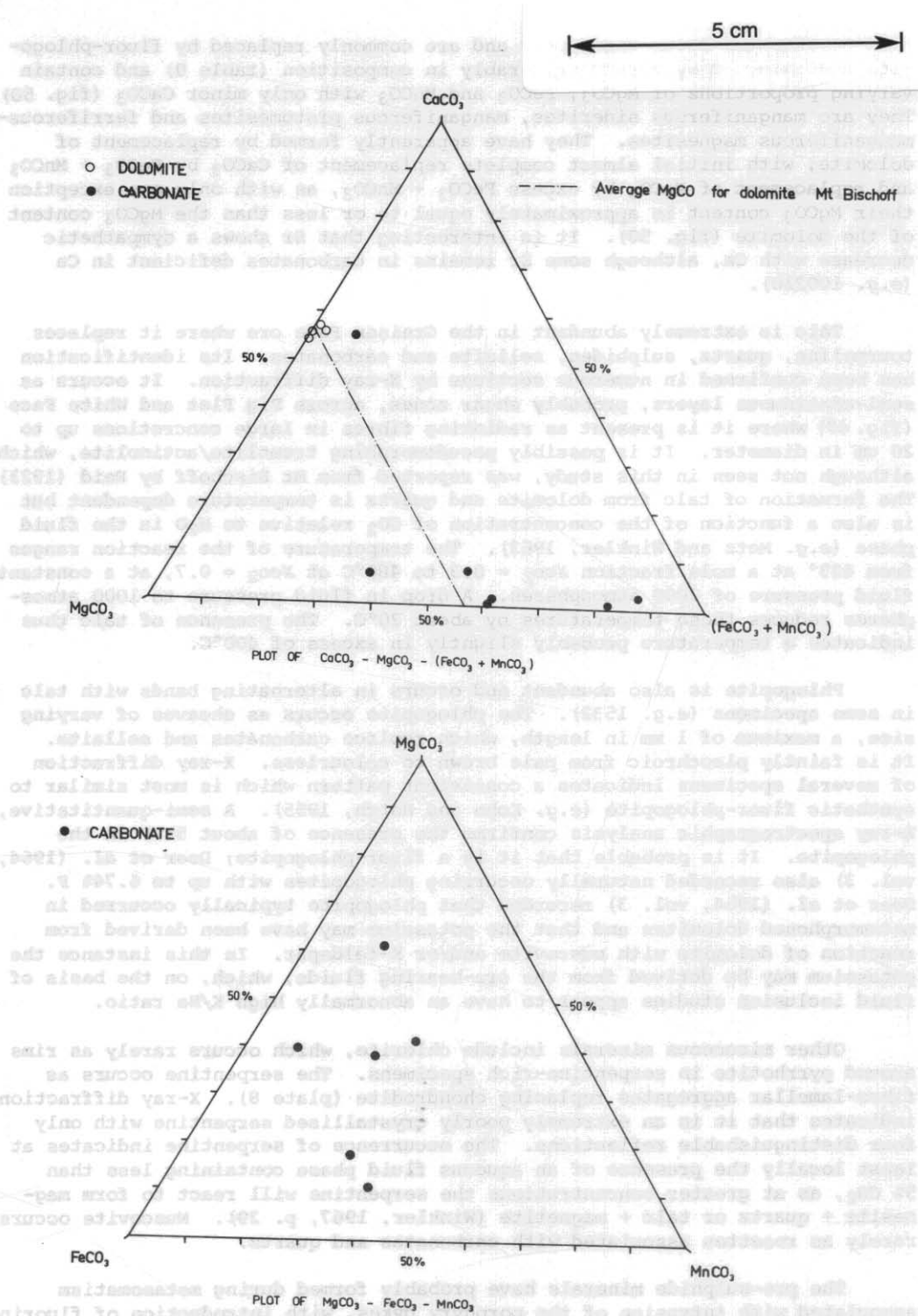
Carbonates occur abundantly and are commonly replaced by fluor-phlogopite and talc. They vary considerably in composition (table 9) and contain varying proportions of $MgCO_3$, $FeCO_3$ and $MnCO_3$ with only minor $CaCO_3$ (fig. 50). They are manganiferous siderites, manganiferous pistomesites and ferriferous-manganiferous magnesites. They have apparently formed by replacement of dolomite, with initial almost complete replacement of $CaCO_3$ by $FeCO_3$ + $MnCO_3$ and replacement of $MgCO_3$ by excess $FeCO_3$ + $MnCO_3$, as with only one exception their $MgCO_3$ content is approximately equal to or less than the $MgCO_3$ content of the dolomite (fig. 50). It is interesting that Sr shows a sympathetic decrease with Ca, although some Sr remains in carbonates deficient in Ca (*e.g.* 100210).

Talc is extremely abundant in the Greisen Face ore where it replaces tourmaline, quartz, sulphides, sellaite and carbonates. Its identification has been confirmed in numerous sections by X-ray diffraction. It occurs as semi-continuous layers, probably shear zones, across Pig Flat and White Face (fig. 47) where it is present as radiating fibres in large concretions up to 20 cm in diameter. It is possibly pseudomorphing tremolite/actinolite, which although not seen in this study, was reported from Mt Bischoff by Reid (1923). The formation of talc from dolomite and quartz is temperature dependent but is also a function of the concentration of CO_2 relative to H_2O in the fluid phase (*e.g.* Metz and Winkler, 1963). The temperature of the reaction ranges from 425° at a mole fraction X_{CO_2} = 0.3 to 480°C at X_{CO_2} = 0.7, at a constant fluid pressure of 2000 atmospheres. A drop in fluid pressure to 1000 atmospheres reduces these temperatures by about 20°C. The presence of talc thus indicates a temperature probably slightly in excess of 400°C.

Phlogopite is also abundant and occurs in alternating bands with talc in some specimens (*e.g.* 1532). The phlogopite occurs as sheaves of varying size, a maximum of 1 mm in length, which replace carbonates and sellaite. It is faintly pleochroic from pale brown to colourless. X-ray diffraction of several specimens indicates a consistent pattern which is most similar to synthetic fluor-phlogopite (*e.g.* Kohn and Hatch, 1955). A semi-quantitative, X-ray spectrographic analysis confirms the presence of about 5% F in the phlogopite. It is probable that it is a fluor-phlogopite; Deer *et al.* (1964, vol. 3) also recorded naturally occurring phlogopites with up to 6.74% F. Deer *et al.* (1964, vol. 3) recorded that phlogopite typically occurred in metamorphosed dolomites and that the potassium may have been derived from reaction of dolomite with muscovite and/or K-feldspar. In this instance the potassium may be derived from the ore-bearing fluids, which, on the basis of fluid inclusion studies appear to have an abnormally high K/Na ratio.

Other micaceous minerals include chlorite, which occurs rarely as rims around pyrrhotite in serpentine-rich specimens. The serpentine occurs as fibro-lamellar aggregates replacing chondrodite (plate 8). X-ray diffraction indicates that it is an extremely poorly crystallised serpentine with only four distinguishable reflections. The occurrence of serpentine indicates at least locally the presence of an aqueous fluid phase containing less than 5% CO_2, as at greater concentrations the serpentine will react to form magnesite + quartz or talc + magnetite (Winkler, 1967, p. 29). Muscovite occurs rarely as rosettes associated with carbonates and quartz.

The pre-sulphide minerals have probably formed during metasomatism associated with intrusion of the porphyry dykes, with introduction of fluorine, boron, silica and minor H_2O at temperatures in excess of 400°C. These minerals have become unstable during later hydrothermal activity involving sulphide deposition and have been generally replaced by hydrous micaceous minerals. Introduction of iron, manganese, potassium, silica and base metals with sulphur, fluorite and H_2O and removal of calcium, CO_2 and to a lesser extent



COMPOSITION OF DOLOMITE AND CARBONATE — MT BISCHOFF

Figure 50.

magnesium, has been widespread. The calcium and CO_2 may have been redistributed in the rocks above the replacement horizons as the greywackes and mudstones of the Waratah River sequence in particular contain abundant calcite and ankerite veins. Some of the potassium and silica may have been derived from alteration of porphyry dykes (see later section).

Fissure Veins

The vissure veins commonly consist essentially of quartz (*e.g.* Giblin Lode, North Valley Lode), carbonate (Thompson's Lode and most Pb-Zn-Ag lodes) or fluorite (Fooks Lode). They are commonly banded. Minor components include tourmaline, topaz, muscovite, biotite and apatite. The quartz occurs at all stages throughout deposition. The fluorite and carbonate are penecontemporaneous with, or later than, the sulphides. Analyses of typical carbonates from Thompson's Lode and Silver Cliffs Lode (table 9) indicate that they are manganiferous siderites or manganosiderites containing small amounts of $MgCO_3$ and very minor $CaCO_3$. They contain no detectable Sr in contrast with the carbonates from the replacement deposit.

Small quartz-cassiterite veins exhibit well developed 'tooth-comb' structure with marginal crystals of cassiterite growing towards the centre of the veins. Small veins containing beryl as euhedral hexagonal crystals up to 5 cm in length, in association with quartz, muscovite and minor cassiterite occur in the Desert Face (fig. 47).

GEOCHEMICAL STUDIES, GEOBAROMETRY, GEOTHERMOMETRY AND THE PROBLEM OF ZONING IN THE MT BISCHOFF ORES

General Geochemical Studies

Several trace element studies have been carried out on sulphides from mineral deposits in western Tasmania (*e.g.* Loftus-Hills and Solomon, 1967; Groves and Loftus-Hills, 1968; Loftus-Hills *et al.*, 1969). This has been done mainly in an attempt to provide additional empirical evidence to support the grouping of mineral deposits in western Tasmania into two main groups as discussed previously: (*a*) the hydrothermal tin and Ag-Pb-Zn ores related to late Devonian-early Carboniferous granitic activity (*e.g.* Mt Bischoff, Renison Bell, Cleveland, Zeehan, Dundas, Waratah), and (*b*) the volcanic ores of Cambrian age (Rosebery-Hercules, Mt Lyell). The main relevance of the data to this study is the demonstration, or otherwise, that deposits belong to the same phase of mineralisation before phenomena such as zoning can be discussed. The main conclusions of these studies are summarised below.

The uniform distribution of low Co and Ni in Fe-sulphides, normal Cd in sphalerites, normal Se in sulphides, and total high Mn in Devonian Ag-Pb-Zn and cassiterite-sulphide deposits in western Tasmania supports the concept of a single metallogenic province in that area in the late Devonian-early Carboniferous. The Zeehan and Waratah areas may represent sub-provinces with respect to Cd (Groves and Loftus-Hills, 1968).

The trend towards high Co contents in pyrites from pyritic-Cu and banded Pb-Zn deposits in Cambrian volcanic rocks and pyrites from unmineralised portions of these volcanic rocks, and the low Cd contents of sphalerites from the banded Pb-Zn deposits suggest a genetic association between ore deposition and vulcanism. It is probable, however, that factors controlling the distribution of Pb and Zn relative to Cu in these deposits may also control the distribution of Co, Ni, Se and possibly Cd.

The Magnet deposit groups empirically with the Devonian deposits in

the Waratah area on the basis of Cd and Se in sphalerite, and the Tullah deposits are similar to the Devonian deposits with respect to Cd in sphalerite and Co and Ni in pyrite but are dissimilar to both the Devonian and Cambrian deposits with respect to Se in sulphides.

Geobarometry

Geological Considerations

The estimation of pressure from geological considerations can only be approximate due (*a*) to difficulties in applying accurate geological reconstructions and (*b*) to uncertainties as to whether the loads are lithostatic or hydrostatic. The maximum thicknesses of overburden above the deposits at Mt Bischoff at the time of mineralisation are calculated below from considerations discussed previously.

The maximum thickness of rocks which could have been deposited above the dolomite horizon are:

	m
Siluro-Devonian	1,830
Ordovician	30
Cambrian	5,500
Proterozoic - Lower Cambrian	300
	7,660

The thickness of approximately 7,660 m may be far too high. The Cambrian sequence may be considerably reduced as Mt Bischoff was probably a structural high during the Cambrian, and fossiliferous rocks similar to those of the Dundas Group (maximum thickness of 2,700 m in the Dundas area) have not been found in this area. It is also probable that mid-Devonian folding and uplift (Tabberabberan Orogeny) which preceded late Devonian-early Carboniferous granite intrusion was accompanied by severe erosion which reduced the thickness of the overburden present in Siluro-Devonian times (*e.g.* Solomon, 1962).

The maximum overburden of 7,660 m is approximately equivalent to a lithostatic load of 1950 atmospheres and a hydrostatic load of 775 atmospheres.

The occurrence of tensional fractures prior to mineralisation and of open spaces during ore deposition at Mt Bischoff is not inconsistent with the estimated overburden. The lower depth limit of tensile failure for quartzite is approximately 3,000 m (Secor, 1965) although quoted values of tensile strength vary considerably and do not allow an exact calculation. However Secor (1965) has shown that this limiting depth of tensile failure is a minimum which increases with increasing fluid pressure, and that this increase is significant for fluid pressure-overburden weight ratios such as those observed in deep oil wells. Secor (1965) has shown that open fractures can occur at depths of over 6,000 metres and may in fact represent earlier fractures which may have been opened up by high fluid pressure at depth. This may have been an important process operating during intrusion of the acid rocks at Mt Bischoff and Renison Bell (Groves and Solomon, 1969).

Fluid inclusion estimates

If contemporaneous primary CO_2 and aqueous inclusions can be identified in a mineral, then it is possible to estimate the effective pressure using

the density of CO_2 (Holden, 1925; Roedder, 1965; Koltun in Yermakov, 1965).

Groves and Solomon (1969) found coexisting CO_2 and aqueous inclusions in one specimen from Happy Valley Face. They estimate that the inclusions formed at about 770 ± 100 atmospheres from corrected filling temperatures of the aqueous inclusions and the density of the CO_2 inclusions. These pressures are only approximate because the composition of the aqueous fluid is unknown and there could be minor impurities in the CO_2. It is also not clear whether the Happy Valley results can be applied throughout the area; presumably the pressure could vary for substantial periods between different fractures and ore zones. There is some evidence (Groves and Solomon, 1969) for locally lower pressure zones in Greisen Face.

If the figure of 770 atmospheres is accepted as the prevailing pressure it could be greater than, equal to, or less than the overburden pressure. At Mt Bischoff the figure calculated for the CO_2 density is approximately equal to the maximum hydrostatic load (775 atmospheres). The overburden would have to be reduced to 3,000 m by the late Devonian for the calculated pressure to approach a lithostatic load. It is probable that the pressure was largely intermediate between the hydrostatic and lithostatic extremes.

Geothermometry and Zoning

Introduction

Several authors (*e.g.* Park, 1955) have drawn attention to the parallelism which might be expected between zonal sequence and paragenetic sequence in an idealised hydrothermal vein system, and Both and Williams (1968) have recently summarised many of the features of hydrothermal zoning.

A high probability that the cassiterite-sulphide deposits and surrounding Pb-Zn-Ag fissure deposits belong essentially to the same phase of mineralisation in a common metallogenic province has been demonstrated. The cassiterite-sulphide deposits are considered to have been deposited at an initially higher temperature because the earliest formed minerals (*e.g.* cassiterite, wolframite, arsenopyrite, tourmaline, topaz) are virtually restricted to these deposits (Groves and Solomon, 1964). The Pb-Zn-Ag deposits contain minerals of intermediate or late position in paragenesis (*e.g.* sphalerite, galena, jamesonite).

At Mt Bischoff lateral mineralogical zoning can be demonstrated on a local scale. The replacement deposit in the central area of the mine (*e.g.* Brown Face, Slaughteryard Face) consists of massive pyrrhotite with cassiterite, arsenopyrite, pyrite and wolframite with extremely rare sphalerite and no galena or jamesonite. Further south (*e.g.* Greisen Face, Pig Flat) pyrite, sphalerite, galena and jamesonite become more abundant although pyrrhotite is still predominant, and in the south end of the open cut and Happy Valley Face pyrite is predominant with sphalerite and some galena in largely unmineralised dolomite. The surrounding fissure veins (*e.g.* Giblin and North Valley Lodes) contain cassiterite with predominantly pyrite and common sphalerite, galena and jamesonite for a radius of approximately 1,000 m from Mt Bischoff Peak, and outside this area they contain predominantly galena and sphalerite with jamesonite (*e.g.* Silver Cliffs, Magnet). Fook's Lode which is approximately 1,500 m south-east of Mt Bischoff Peak (fig. 40) is unusual in that it is similar to the vein deposits within the inner circular area. Mineralogical zoning has also been recorded within the fissure veins, Weston-Dunn (1922) recording abundant cassiterite and wolframite and rare galena in the Giblin Lode near the summit of Mt Bischoff and abundant sphalerite, galena and jamesonite towards its south-west extremity.

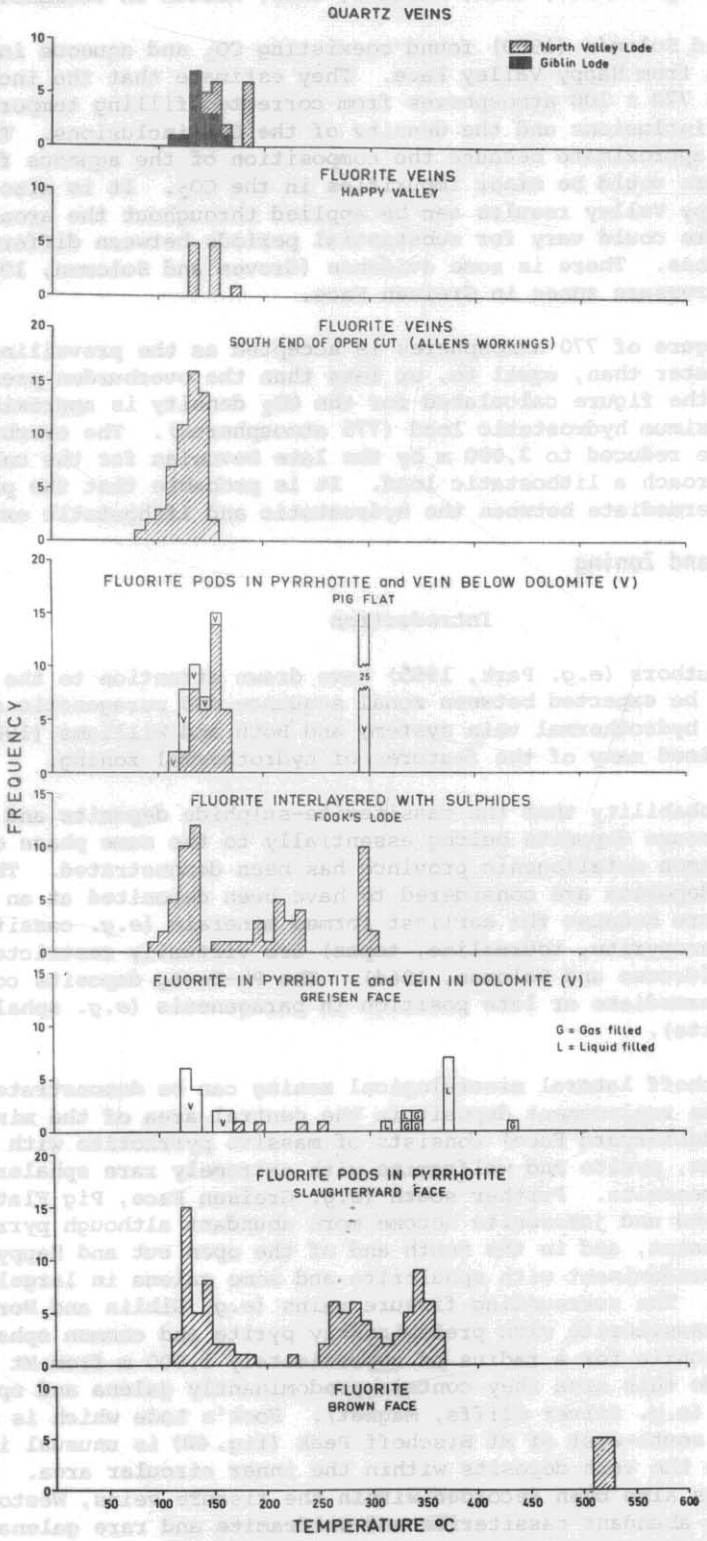


Figure 51. *Frequency distribution of filling temperatures of inclusions in fluorite and quartz from Mt Bischoff [Groves and Solomon, 1969]*

5 cm

The mineralogical zoning at Mt Bischoff has been attributed, at least in part, to the existence of a temperature gradient from the centre of Mt Bischoff outwards (Groves and Solomon, 1964). In this section an attempt has been made to quantify the temperature and other parameter variations at Mt Bischoff using fluid inclusion data and sulphide systems. The scope of the investigation has been partly restricted by the comparatively rare occurrence of suitable minerals, and by their occurrence commonly late in paragenesis.

Fluid Inclusion Studies

Fluid inclusion studies on Mt Bischoff ores have been described by Groves and Solomon (1969), and the conclusions of this study are presented below. Formation temperatures of minerals have been derived from filling temperatures of inclusions (fig. 51) by applying a correction for pressure (c. 750 atmospheres) and salinity of inclusions (Klevstov and Lemmlein, 1959).

Formation temperatures of fluorite and quartz support the concept of a generalised temperature decline outwards from the centre of Mt Bischoff. Initial temperatures of deposition of fluorite in the Brown Face were up to 580°C and decreased to approximately 200°C in the marginal zone some 600 m away. Temperatures of formation of fluorite and quartz in later marginal fissure veins range from 170° to 380°C, with the majority between 200°C and 250°C.

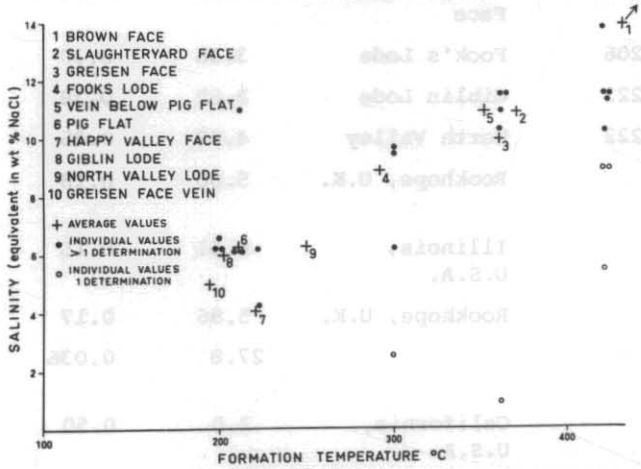


Figure 52. *Plot of salinity against temperature of formation, using average values of primary inclusions and values for individual primary inclusions for which both parameters were measured [Groves and Solomon, 1969].*

A positive correlation exists between salinity and temperature of formation for minerals from different localities (fig. 52). A possible mechanism to explain (*a*) this correlation (*b*) the great variation of salinity within one area (*c*) the temperature decline and (*d*) the high salinities of inclusions from the Brown Face (fig. 52) is for initially hot, highly saline hydrothermal solutions to be mixed with cooler, less saline meteoric and connate waters, in conjunction with heat loss from the solutions to the wall rocks. The occurrence of meteoric and connate waters at depths of over 3,000 m is possible (*e.g.* White, 1968), particularly in an environment with open fractures and carbonate beds.

Low Na/K and Na/Li ratios of fluid inclusions from Brown Face and Slaughteryard Face (table 10) probably result from addition of excess K and

5 cm

Li over Na from hydrothermal alteration of the porphyry dykes which may have been feeders to the mineralisation. The Na/K ratio of the ore solutions increased with declining temperature and salinity possibly due to admixing with relatively Na-rich meteoric and connate waters; this increase may have been accentuated by fixation of K in micaceous minerals (*e.g.* phlogopite). A possible local source of solutions is indicated for deposition of fluorite at Fook's Lode which has a relatively high temperature of formation and salinity but similar Na/K ratios to the other marginal deposits.

Table 10. ALKALI WEIGHT RATIOS OF FLUID INCLUSIONS FROM MT BISCHOFF AND OTHER OCCURRENCES

Mineral	Number	Locality	Na/K	K/Na	Li/Na
Fluorite	13511	Brown Face	1.12	0.89	0.0210
Fluorite	100204a	Slaughteryard Face	1.44	0.70	0.0077
Fluorite	100204b	Slaughteryard Face	1.41	0.71	0.0129
Fluorite	100205a	Slaughteryard Face	1.37	0.73	
Fluorite	100205b	Slaughteryard Face	1.12	0.90	0.0313
Fluorite	100206	Fook's Lode	3.36	0.30	0.0117
Quartz	100223	Giblin Lode	2.62	0.38	0.0105
Quartz	100222	North Valley	4.03	0.25	0.0120
Green Fluorite		Rookhope, U.K.	5.60	0.18	0.0035
Purple Fluorite		Illinois, U.S.A.	6.51	0.15	0.0004
Quartz		Rookhope, U.K.	5.86	0.17	0.0027
Ocean Water (1)			27.8	0.036	0.00001
Salton Sea (2)		California, U.S.A.	2.0	0.50	0.0059

Figures for (1) and (2) are from White (1965, table 1). From Groves and Solomon (1969)

Sulphide Systems

The following discussion is taken from Solomon *et al.* (in press) which represents research described by Groves (1968) together with more recent research on sphalerite compositions by Drs M. Solomon, K.L. Williams and D.I. Groves.

Fe-S System

The pyrrhotites of the replacement orebodies were determined as hexagonal (probably 2A, 5C) or monoclinic (2B, 4C), using the criteria of Desborough and Carpenter (1965). The pyrrhotites were ground and X-rayed under shellite to avoid changes due to oxidation.

Discrete pyrrhotite samples taken for X-ray diffraction are largely hexagonal or largely monoclinic, but a number contain both types in prominent amounts, and in several polished sections alternating lamellae of hexagonal and monoclinic types were shown by selective etch reactions.

There are variations within hand specimens and between samples from the same deposit (table 11). However the pyrrhotites from Brown, Slaughteryard and Greisen Faces and for Pig Flat are generally mixtures of hexagonal and monoclinic types. The monoclinic samples from Brown Face are rich in colloform pyrite and the abundance of monoclinic structures there may be related to supergene alteration. At the southern end of the open cut, and in the fissure lodes, the pyrrhotites are entirely monoclinic, and these samples are relatively rich in pyrite probably derived from hypogene alteration of pyrrhotite.

Table 11. SYMMETRY AND COMPOSITION OF PYRRHOTITES FROM MT BISCHOFF

Locality	Specimen No.	Symmetry	$d_{(210)}$ Å ±0.0007	Atomic % Metals ±0.30	Ratio 1st Peak Both Peaks
Brown Face	100059	H	2.0632	47.06	1.0
	100060a	H	2.0669	47.37	1.0
	100060b	M	2.0674	n.d.	0.5
	100060c	M	2.0705	n.d.	0.4?
	100060d	M	2.0678	n.d.	0.5
	100060e	M	2.0597	n.d.	0.5
Slaughteryard Face	100061a	H	1.0666	47.35	1.0
	100061b	H	2.0673	47.40	1.0
	100062	H + M	2.0612	n.d.	0.66
	100063	H	2.0649	47.20	1.0
	100065a	H	2.0612	46.89	1.0
	100065b	H	2.0680	47.45	1.0
	100065c	H	2.0687	47.50	1.0
	100065d	H	2.0632	47.06	1.0
	100065e	H	2.0639	47.12	1.0
	100066	H + M	2.0582	n.d.	0.63
	100067	H	2.0595	46.76	1.0
Greisen Face	100068	M	2.0632	n.d.	0.5
	100069	M	2.0605	n.d.	0.5
	100070	M	2.0635	n.d.	0.5
	100071	M	2.0585	n.d.	0.5
	100072	H	2.0568	46.54	1.0
	100073	M	2.0568	n.d.	0.4?
	100074	M	2.0652	n.d.	0.5
	100075	M	2.0571	n.d.	0.5

Table 11. - Continued

Locality	Specimen No.	Symmetry	$d_{(210)}$ Å ±0.0007	Atomic % Metals ±0.30	Ratio 1st Peak Both Peaks
Greisen Face	100127b	M	2.0633	n.d.	0.4?
Pig Flat	100076a	H	2.0622	46.97	1.0
	100076d	H + M	2.0588	n.d.	0.63
	100076f	H + M	2.0619	n.d.	0.6
	100077a	H + M	2.0673	n.d.	0.66
	100077b	H + M	2.0669	n.d.	0.66
	100078a	H	2.0680	47.45	1.0
	100078b	H + M	2.0592	n.d.	0.7
	100079	H	2.0601	46.81	1.0
	100080	H	2.0612	46.89	1.0
	100081	H	2.0619	46.95	1.0
	100082	M	2.0558	n.d.	0.5
	100083	M	2.0571	n.d.	0.5
South of Pig Flat	100127a	M	2.0642	n.d.	0.5
	100124	M	2.0732	n.d.	0.5
	100125	M	2.0719	n.d.	0.5
	10936	M	2.0696	n.d.	0.5
	10937	M	2.0714	n.d.	0.5
	100129	M	2.0759	n.d.	0.5
	100133a	M	2.0669	n.d.	0.4?
	100130	M	2.0705	n.d.	0.5

Samples were crushed and analysed under shellite to avoid oxidation

Because of the lack of critical lines on the X-ray diffraction patterns it is not known whether the hexagonal pyrrhotites are high or low temperature types but the rapidity with which inversion takes place experimentally suggests they are the low-temperature polymorphs.

The total metal content of the hexagonal pyrrhotites was determined using the method of Arnold and Reichen (1962). This gives an uncertainty of approximately ±0.30%. Arnold and Reichen (1962) suggested that the method would be unsuitable for pyrrhotites containing more than a total of about 0.6% of Co, Ni and Cu. The maximum concentrations of Co and Ni in the pyrrhotites measured by Loftus-Hills (1968) is 29 and 109 ppm and the Cu content as determined on three samples by X-ray fluorescence is less than 0.3% even though some free chalcopyrite is present.

The metal contents are variable (table 11), but indicate that the Pig Flat - Greisen pyrrhotites may generally have a lower metal content than the Brown Face - Slaughteryard types.

Use of the pyrrhotite geothermometer to yield a minimum temperature of

formation requires that the pyrrhotite continually re-equilibrates during cooling along the pyrrhotite-pyrite solvus of the Fe-S system, until quenching or some other factor prohibits further change of composition. The position of the pyrite - pyrrhotite solvus as determined in anhydrous conditions by Arnold (1957) was confirmed by Chernyshev *et al.* (1968) for hydrous systems except that the solvus apparently continues below the β-inversion temperature of 308°C, at least to 250°C. If pyrite-pyrrhotite equilibrium existed at Mt Bischoff then minimum temperatures of about 250° to 450°C would be indicated for the Slaughteryard - Brown Face area, and 500°C for Greisen Face. The fluid inclusion data for fluorite from these areas indicate temperatures of deposition ranging from a possible 285 to about 590°C(?) for the Brown and Slaughteryard Faces, and 300 to 430°C for Greisen Face (fig. 51). Hence the data are broadly compatible for the Brown Face - Slaughteryard pyrrhotites and possibly also for those from Greisen Face (in view of the poor precision of the d-spacing measurements), indicating that cooling could have occurred along the pyrite-pyrrhotite solvus. However, the scarcity of pyrite apparently exsolved from pyrrhotite does not confirm this and the pyrrhotites may well have remained within the hexagonal pyrrhotite field, with sulphur fugacity declining during cooling. Whatever detailed cooling path was followed it is probable that there was an overall decline in sulphur fugacity during cooling (*e.g.* Holland, 1965).

The composition of the pyrrhotites south of Greisen Face is mainly consistent with the experimental data of Clark (1966) for pyrrhotites formed below 300°C in anhydrous systems. The metal contents fall within the monoclinic and hexagonal field at temperatures near 200°C, with most samples clustering near the (hexagonal + monoclinic) - monoclinic boundary.

Secondary, hypogene pyrite is common south of Greisen Face, and also in the fissure lodes. This phase of pyrrhotite alteration may have locally affected the composition of the adjacent pyrrhotites and might account for some of the spread in the results.

Fe-Zn-S System

Sphalerite is rare in the replacement ores of Brown and Slaughteryard Faces but is more common south of Greisen Face, where it is commonly associated with pyrite, pyrrhotite, and carbonate. It also occurs in the altered porphyry dykes where it has replaced orthoclase phenocrysts and is generally free of other sulphides. In the fissure lodes it occurs most commonly with pyrite, stannite, fluorite and carbonate and in the lead-zinc lodes with galena, pyrite and jamesonite. The sphalerite commonly appears to be replacing iron-sulphides though in some examples (*e.g.* Fook's Lode) pyrite and sphalerite appear contemporaneous.

Most of the sphalerites examined were analysed for Fe, Mn and Cd by electron microprobe techniques (Williams, 1967) but some larger samples were analysed in bulk for Fe by X-ray diffraction (using cell-size formulae based on Skinner *et al.* (1959) and for Cd and Mn by X-ray fluorescence spectroscopy. Analytical procedures and precision of analyses are quoted by Groves (1968) and Solomon *et al.* (in press) and the results are shown in Table 12 and Figure 53.

The results have been grouped into those from (*a*) porphyry dykes (*b*) replacement ore (*c*) cassiterite-bearing fissure lodes and (*d*) galena-sphalerite lodes (table 12). The FeS contents (fig. 53), of the last three groups are significantly different with successively lower means (20.66, 16.74, 12.99 wt% FeS) and the first two are similar (21.17, 20.66 wt% FeS). For CdS and MnS, the first two are similar and differ from the last two, which

Table 12. COMPOSITION OF SPHALERITES FROM DIFFERENT ENVIRONMENTS, MT BISCHOFF AND WARATAH DISTRICT

Analyses for which a single result is shown represent analyses from Groves (1968) with CdS and MnS determined by X-ray fluorescent spectroscopy and FeS by X-ray diffraction. Analyses for which two or more analyses are given (with the average in brackets) were determined by electron-probe microanalysis under the supervision of K.L. Williams at the Australian National University (from Solomon, Groves, Rafter and Williams, in press).

University of Tasmania Sample No.	Details	Locality	Wt% FeS	Wt% MnS	Wt% CdS
Dolomite horizon					
100059	In massive pyrrhotite	Brown Face	22.0 ± 3.8		
100135	In massive pyrrhotite with chalcopyrite	Greisen Face	22.9,22.7,21.1 (22.2)	0.32,0.32,0.40 (0.34)	0.64,0.63,n.d. (0.63)
101053A	With pyrite in secondary carbonate	Greisen Face, borehole B5 at 42 ft	17.6,16.2 (16.9)	0.49,0.58 (0.53)	
101053B	With pyrite in secondary carbonate	Greisen Face, borehole B5 at 42 ft	24.7,20.9,24.5 (23.4)	0.73,0.61,0.73 (0.69)	
100768	In massive pyrrhotite	Greisen Face, borehole B6 at 56 ft	21.1,21.0,21.5 (21.2)	0.42,0.40,0.38 (0.40)	0.46,0.51,n.d. (0.48)
101038	With pyrite in secondary carbonate	Greisen Face, borehole B6 at 57 ft	20.2,19.7,19.1 (19.7)	0.39,0.41,0.44 (0.41)	
101040	With pyrite in secondary carbonate	S of Greisen Face, borehole B9 at 121 ft	20.8,21.6,19.3 19.8,22.6,23.0 (21.2)	0.85,0.91,0.81 0.88,0.87,0.66 (0.83)	
100011	In pyrrhotite, pyrite and carbonate	Pig Flat	12.1 ± 1.6	0.09	0.89
100012	In pyrrhotite, pyrite and carbonate	Pig Flat	13.1 ± 1.6	0.22	0.37
100774	In dolomite	S of Greisen Face, borehole B23 at 123 ft	20.4,19.2,19.0 (19.5)	0.17,0.17,0.17 (0.17)	
100773	With pyrite in dolomite	W side of Allens Workings, borehole B22 at 9 ft	22.2,23.4,23.5 24.0 (23.3)	0.50,0.55,0.52 0.56 (0.52)	nd.,n.d.,0.51 n.d.
101047	With pyrite in dolomite	W side of Allens Workings, borehole B22 at 50 ft	22.7,22.3,22.5 (22.5)	0.73,0.86,0.91 (0.83)	
10577	With pyrite and secondary carbonate	Centre of Allens Workings, at surface	25.1,25.0,23.0 (24.4)	0.53,0.66,0.93 (0.71)	

100146	With pyrite in carbonate	Centre of Allens Workings, at surface	14.1 ± 3.6		
100762	With pyrite in dolomite	Happy Valley Face	20.0,18.7,20.0 (19.6)	0.58,0.46,0.54 (0.53)	n.d.,0.39,0.44 (0.41)
100825	With pyrite in dolomite	Happy Valley Face	23.0,24.2,21.8 (23.0)	0.92,0.55,0.51 (0.66)	n.d.,0.32,0.39 (0.36)
100134	With pyrite and stannite in dolomite lens below main dolomite bed	S of open cut, borehole B14 at 211 ft	24.2,22.0,24.5 (23.5)	0.70,0.48,0.45 (0.54)	
		MEAN	20.7	0.55	0.50
		STANDARD DEVIATION	2.9	0.22	0.16
Porphyry dyke					
100827	With tourmaline in metasomatised White Face dyke	White Face	20.9,22.3,22.7 21.0 (22.0)	1.11,0.88,0.73 1.10 (0.95)	0.40,0.35,n.d. n.d. (0.37)
100778	Replacing feldspar (?) in White Face dyke	S of open cut, borehole B24 at 120 ft	23.7,23.6,23.4 (23.6)	0.64,0.63,0.61 (0.63)	n.d.,0.63,0.50 (0.56)
101050	Replacing feldspar (?) in White Face dyke	S of open cut, borehole B24 at 126 ft	18.1,18.0,15.5 (17.2)	0.47,0.50,0.22 (0.40)	
101051	Replacing feldspar (?) in White Face dyke	S of open cut, borehole B24 at 156 ft	22.6,20.5,20.7 (21.3)	0.66,0.50,0.22 (0.46)	
100767	With pyrite in altered White Face dyke	S of open cut, borehole B19 at 234 ft	21.2,21.6,24.2 (22.3)	0.81,0.28,0.71 (0.60)	n.d.,n.d.,0.55
1433	Replacing feldspar (?) in White Face dyke	S end of Allens Workings, borehole B12 at 87 ft	18.7,22.4 (20.6)	0.15,0.67 (0.41)	
100791	With pyrite replacing feldspar (?) in White Face dyke	S of open cut, borehole B38 at 312 ft	16.2,13.5,16.0 (15.2)	0.06,0.03,0.07 (0.05)	
		MEAN	21.2	0.61	0.49
		STANDARD DEVIATION	2.2	0.25	0.11
Cassiterite fissure lodes and veins					
101060	With pyrite, chalcopyrite, marcasite, stannite, carbonate and quartz in thin vein shale	W of and below, Brown Face, borehole B51 at 287 ft	13.7,14.0 (13.8)	0.34,0.36 (0.35)	

Table 12. COMPOSITION OF SPHALERITES FROM DIFFERENT ENVIRONMENTS, MT BISCHOFF AND WARATAH DISTRICT - continued

University of Tasmania Sample No.	Details	Locality	Wt% FeS	Wt% MnS	Wt% CdS
101015A	With pyrite, pyrrhotite, chalcopyrite, stannite in marcasite in thin vein in shale	Below Brown Face, borehole B54 at 20 ft	17.2,17.9,17.1 16.9,16.1 (17.0)	0.23,0.25,0.24 0.23,0.23 (0.23)	
100803	With pyrite, stannite, quartz and carbonate in thin vein in shale	Below Brown Face, borehole B54 at 23 ft	19.1,19.5,19.1 (19.2)	0.09,0.21,0.20 (0.17)	
100805	With pyrite, stannite, marcasite and carbonate in thin vein in shale	Below Brown Face, borehole B54 at 187 ft	13.0,15.7,11.4 (13.4)	0.09,0.19,0.07 (0.12)	
100808	With pyrite, stannite, jamesonite, quartz and carbonate in thin vein in shale	Below Brown Face, borehole B54 at 234 ft	19.8,17.6,17.5 20.1 (18.7)	0.17,0.08,0.13 0.18 (0.14)	
100811	With jamesonite in thin vein in shale	Below Brown Face, borehole B54 at 362 ft	11.2,11.8,12.3 (11.8)	0.02,0.02,0.02 (0.02)	0.64,0.46,n.d. (0.55)
100794	With pyrite, stannite, cassiterite, quartz and carbonate in vein in shale	W of Brown Face, borehole B49 at 76 ft	19.8,20.0,20.7 (20.1)	0.25,0.24,0.26 (0.25)	n.d.,0.72,0.73 (0.72)
100140	With pyrite, pyrrhotite, stannite, chalcopyrite and jamesonite in replacement (?) lode in Western dyke	S of Slaughteryard Face, in dyke	14.0,13.9,13.1 18.0 (14.8)	0.19,0.21,0.08 0.19 (0.17)	n.d.,0.32,0.33 0.44 (0.54)
100138	With pyrite, pyrrhotite chalcopyrite, arsenopyrite, jamesonite and galena in lode in Western dyke	S of Slaughteryard Face, in dyke	18.0,18.5,18.5 (18.3)	0.09,0.22,0.15 (0.15)	
100149	With pyrite, arsenopyrite, stannite, cassiterite and carbonate in lode in Western dyke	S of Slaughteryard Face, in dyke	17.9,13.4,18.9 (16.7)	0.16,0.04,0.21 (0.13)	

100133	With pyrrhotite, pyrite, stannite, jamesonite, carbonate and quartz in thin vein in dyke	S of Greisen Face, borehole B10 at 270 ft	18.5,18.7,18.7 (18.6)	0.17,0.18,0.19 (0.18)	
100148	With jamesonite, galena and quartz in vein in shale	S of Greisen Face, near Main Tunnel portal	11.0,13.2,15.1 17.6 (14.2)	0.02,0.09,0.09 0.07 (0.07)	
100770	With carbonate in vein in shale	Between Pig Flat and Happy Valley Face, borehole B20 at 330 ft	14.1,17.4,15.6 (15.7)	0.06,0.20,0.16 (0.14)	
100766	With arsenopyrite, fluorite and carbonate in shale	SW of Allens Workings, borehole B18 at 82 ft	15.0,17.2,17.1 17.0 (16.6)	0.13,0.07,0.17 0.13 (0.12)	
100768		SW of Allens Workings, borehole B19	17.3,16.8,18.8 (17.6)	0.15,0.15,0.14 (0.15)	
100780	With pyrite, stannite and carbonate in vein in shale	S of Allens Workings, borehole B25 at 266 ft	19.4,19.3,20.7 (19.8)	0.22,0.22,0.25 (0.23)	
100785	With pyrite and carbonate in vein in shale	S of the open cut, borehole B29 at 153 ft	18.6,18.5,18.9 (18.7)	0.25,0.25,0.24 (0.25)	n.d.,0.55,0.44 (0.50)
100792	With pyrite and carbonate in vein cutting chert	S of open cut, borehole B38 at 501 ft	16.4,16.3,15.7 (16.1)	0.11,0.12,0.12 (0.12)	0.47,0.72,n.d. (0.59)
100017	With pyrite, jamesonite, galena, fluorite and carbonate in lode	Thompsons Lode, SW of open cut	17.5,17.5,16.8 (17.3)	0.15,0.14,0.13 (0.14)	
100153	With pyrite, jamesonite, galena, fluorite and carbonate in lode	Thompsons Lode, SW of open cut	13.9,14.3,17.5 14.5 (15.0)	0.06,0.09,0.14 0.10 (0.10)	0.50,0.47,0.60 0.54 (0.52)
100761	With pyrite and carbonate in vein in sandstone	1/4-mile E of Happy Valley Face, borehole B8 at 250 ft	19.1,16.4,15.4 (17.0)	0.40,0.23,0.82 (0.48)	
		MEAN	16.7	0.17	0.53
		STANDARD DEVIATION	2.5	0.11	0.01
Galena - sphalerite lodes					
5-27	With galena, jamesonite, stibnite, pyrite, quartz and fluorite	Antimonial lode, Tinstone Creek	9.0,7.5,6.7 (7.8)	<0.01	n.d.,0.89,0.69 (0.79)

Table 12. COMPOSITION OF SPHALERITES FROM DIFFERENT ENVIRONMENTS, MT BISCHOFF AND WARATAH DISTRICT - continued

University of Tasmania Sample No.	Details	Locality	Wt% FeS	Wt% MnS	Wt% CdS
56-3	With galena, jamesonite, stibnite, pyrite, quartz and fluorite	Antimonial lode, Tinstone Creek	17.2,16.9,16.8 (16.9)	0.17,0.15,0.18 (0.17)	
56-4	With galena, jamesonite, stibnite, pyrite, quartz and fluorite	Antimonial lode, Tinstone Creek	18.4,15.1,17.1 (16.9)	0.15,0.08,0.10 (0.11)	
56-5	With galena, jamesonite, stibnite, pyrite, quartz and fluorite	Antimonial lode, Tinstone Creek	17.0,16.8,18.1 (17.3)	0.16,0.22,0.13 (0.17)	
56-6	With galena, jamesonite, stibnite, pyrite, quartz and fluorite	Antimonial lode, Tinstone Creek	15.7,17.4,18.3 (17.1)	0.17,0.29,0.25 (0.24)	0.46,0.40,n.d. (0.43)
100027	With jamesonite, galena and carbonate	Silver Cliffs Mine	14.8 ± 1.6	1.11	1.25
100028	With jamesonite, galena and carbonate	Silver Cliffs Mine	9.8	0.84	0.38
10575	With jamesonite, galena and carbonate	Silver Cliffs Mine	11.0,10.8,9.0 8.8 (10.2)	0.04,0.04,0.03 0.04 (0.04)	n.d.,0.72,0.72 n.d. (0.72)
100031	With jamesonite, galena and carbonate	Silver Cliffs Mine	10.9,12.1,9.7 (10.9)	0.06,0.06,0.03 (0.05)	n.d.,0.50,0.47 (0.48)
100029a	With jamesonite, galena and carbonate	Silver Cliffs Mine	8.3 ± 1.7	0.08	0.42
100029b	With jamesonite, galena and carbonate	Silver Cliffs Mine	8.4 ± 1.7	0.08	0.40
100030	With jamesonite, galena and carbonate	Silver Cliffs Mine	13.3 ± 1.6	0.20	0.26
100007	With jamesonite, galena and carbonate	North Valley lode	9.5 ± 1.7	0.08	0.37
100008	With jamesonite, galena and carbonate	North Valley lode	8.7 ± 1.7	0.13	0.39
100009	With jamesonite, galena and carbonate	North Valley lode	7.9 ± 1.8	0.09	0.37

100010	With jamesonite, galena and carbonate	North Valley lode	9.9 ± 1.6	0.18	0.39
100159	With galena, jamesonite, pyrite and carbonate	Lode 3500 ft N of Mt Bischoff summit	14.6,16.4,15.8 (15.6)	0.26,0.26,0.27 (0.26)	
100161	With galena, jamesonite, pyrite and carbonate	Lode 3500 ft N of Mt Bischoff summit	10.7,7.1,11.6 (10.5)	0.04,0.02,0.04 (0.03)	n.d.,0.59,0.65 (0.62)
100822	With pyrite and stannite in shales	1/4-mile E of Happy Valley Face, possibly on North Valley lode extension	14.0,15.9,6.8 7.5 (11.1)	0.23,0.60,0.78 0.49 (0.53)	0.47,0.50,0.41 n.d. (0.45)
100017		Thompsons Lode	15.2 ± 1.6	0.38	0.40
100137	With pyrite, fluorite and carbonate	Fooks Lode	16.6,18.2,16.7 (17.2)	0.29,0.20,0.28 (0.26)	
100014	With pyrite, fluorite and carbonate	Fooks Lode	19.0,15.1,16.6 16.7 (16.4)	0.22,0.12,0.14 0.15 (0.16)	n.d.,0.27,0.39 0.55 (0.40)
100154	With pyrite, galena, chalcopyrite, fluorite and carbonate	Fooks Lode	16.1,15.4,16.7 18.0 (16.6)	0.16,0.11,0.14 0.14 (0.14)	
100015	With pyrite, fluorite, and carbonate	Fooks Lode	14.7 ± 1.6	0.13	0.42
100016	With pyrite, fluorite and carbonate	Fooks Lode	14.7 ± 1.6	0.11	0.42
100155	Banded with carbonate	Magnet Mine	8.7 ± 3.1		
100156	Massive with galena and carbonate and a little pyrite	Magnet Mine	13.6,12.7,11.6 (12.6)	0.04,0.03,0.03 (0.03)	0.82,1.17,1.37 (1.12)
100157	Massive with galena and carbonate and a little pyrite	Magnet Mine	6.4,10.1,1.3 (5.9)	<0.01	0.50,2.70,1.04 (1.41)
100158	Breccia with galena and carbonate	Magnet Mine	12.9,12.0,12.5 (12.5)	0.03,0.04,0.03 (0.03)	0.83,0.45,0.67 (0.65)
		MEAN	12.9	0.17 (0.14)	0.64 (0.73)
		STANDARD DEVIATION (Probe analyses only)	3.9	0.20 (0.15)	0.43 (0.48)

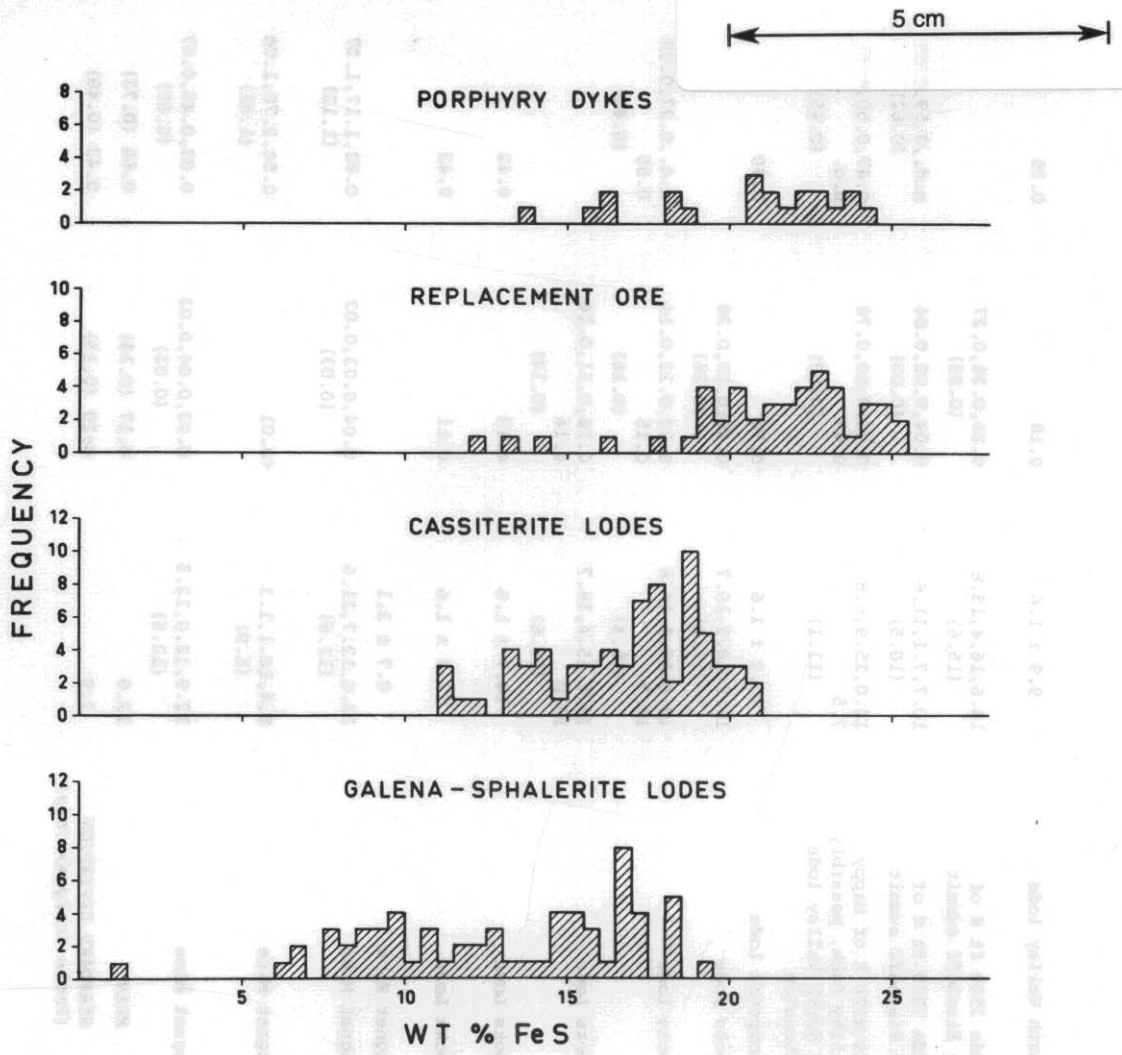


Figure 53. *Histogram of FeS contents (wt %) of sphalerites from different environments, Mt Bischoff [Solomon et al., in press].*

are similar even if the microprobe analyses are treated separately from X-ray analyses.

Within the replacement ore and the porphyry dykes there is no significant spatial variation in Fe content of sphalerite and no variation related to the proximity or otherwise of pyrite or pyrrhotite. Similar results are found for sphalerites from the other two groups, except for Fook's Lode which is anomalous in several respects. However, the gross variation in Fe-content of sphalerites between the replacement orebodies, the cassiterite-fissure lodes and the galena-sphalerite lodes correlates with the variation in total Fe-content of the ore-types, expressed in the relative proportions of iron sulphides (table 7).

It is not clear whether the sphalerites of the replacement ore were deposited in equilibrium with pyrrhotite and/or pyrite or not because of uncertainty in interpretation of microscopic textures. Even if true depositional equilibrium was not attained, it seems possible that surface equilibrium with pre-existing pyrrhotite and pyrite was reached via the ore solutions

or that equilibrium was achieved after deposition during slow cooling. However, the intra-specimen variation in the results both in and between crystals, indicates that equilibrium was only locally, if ever, achieved. That equilibrium between sphalerite, pyrrhotite and pyrite may have been reached in a few cases within the higher temperature areas is indicated by the approximate correspondence of some high Fe values (*e.g.* 100059 and 101053) with curves for sphalerite composition as calculated by Einaudi (1968) and determined experimentally by Chernyshev *et al.* (1968). However if such equilibrium had been generally attained in the replacement orebody then the Fe-content of the sphalerites should have increased from the Brown - Slaughteryard Face area towards the south (*i.e.* down the temperature gradient) but no such trend is apparent. The relatively high lower limit of FeS content of sphalerites from the porphyry dykes (15.5 wt% FeS) is consistent with sphalerite having approached at least surface equilibrium with pyrite during cooling accompanied by a decrease of sulphur fugacity (*e.g.* Chernyshev *et al.*, 1968). The much lower limit of FeS contents of sphalerites from the outer lead-zinc fissure lodes, particularly Magnet (1.3 wt% FeS), is consistent with the absence of pyrrhotite and the sporadic occurrence of pyrite in these deposits. For the lode deposits there is no indication that equilibrium with pyrite and pyrrhotite was ever achieved at any scale. The hypogene development of pyrite from pyrrhotite does not appear to have had a significant effect on sphalerite composition but may have caused localised variation.

Experience with other studies of pyrite-pyrrhotite-sphalerite occurrences (and assuming the applicability of the latest experimental data) suggests that equilibrium is seldom attained, even where the field occurrence suggests it is likely (*e.g.* Campbell and Williams, 1968; Arnold *et al.*, 1962). However, there are cases (*e.g.* from New Mexico, Rose, 1967) in which individual sphalerites co-existing with pyrite and pyrrhotite have Fe contents as high as 24.1 wt% FeS, indicating that localised equilibrium may occur at the crystal scale.

There is a marked positive correlation between the FeS and MnS contents of the sphalerites within each group and similar correlation has also been demonstrated by Stoiber, 1940; Fryklund and Fletcher, 1956; Bradbury, 1961; and Both and Williams, 1968. The higher MnS contents, and stronger correlation between MnS and FeS, of sphalerites from the pyrrhotite ores and porphyry dykes relative to the outer tin and lead-zinc fissure lodes is probably related to the relative abundance of sulphides and carbonates in the different groups. Manganese is strongly lithophile in all but strongly reducing environments and thus preferentially enters the carbonate structure rather than the sphalerite structure if both are present. In the outer veins the carbonate/sulphide ratio is high and the Mn content of the carbonate is high (table 9), whereas in the pyrrhotite ore and porphyry dykes the carbonate/sulphide ratio is low and the Mn-content of secondary carbonate is erratic but generally relatively low (table 9). Williams (1968) suggested a similar process to explain high MnS contents of sphalerite in pyritic ores and low MnS in sideritic ores from the Zeehan field.

There is a weak negative correlation between CdS and FeS contents of sphalerites for all groups taken together, similar to that found in the Zeehan field by Williams (1968) and in a regional survey of Tasmanian ores by Groves and Loftus-Hills (1968).

Fe-As-S System

Arsenopyrite occurs as very fine-grained cracked euhedral crystals in replacement ore south of Greisen Face and as larger euhedral crystals in the cassiterite fissure lodes. Its relationship to pyrite is obscure, mainly because the two minerals are seldom in contact.

Clark (1960) has shown than in an anhydrous Fe-As-S system although pyrite is stable up to 743°C and arsenopyrite up to 702°C, the two minerals are not stable as a pair above about 491°C, and if pyrite coexists with arsenopyrite in the presence of vapour at least one of the minerals was deposited below 491°C. This temperature is not greatly affected by confining pressure or the presence of pyrrhotite, but the effects of aqueous solutions are unknown. Below 491°C, Morimoto and Clark (1961) found that the sulphur content of arsenopyrite varied with temperature and that the $d_{(131)}$ spacing could be used as a measure of the sulphur content.

Measurements of $d_{(131)}$ spacings made on four arsenopyrites from minor veins in sediments below Brown Face and two arsenopyrites from a replacement lode in Western dyke, range from 1.6285-1.6319Å. For a confining pressure of about 1000 atmospheres, these figures indicate a maximum temperature of deposition of 450°C. There are no fluid inclusion data for these particular veins, but fluorite in the contiguous pyrrhotite ores yielded 590°C(?) in Brown Face and 285°C to 470°C in Slaughteryard Face. Unfortunately, insufficient arsenopyrite was available from other environments for X-ray analysis.

Sulphur Isotope Data

Preliminary results of sulphur isotope investigations of sulphides from Mt Bischoff have been given by Rafter and Solomon (1967), and a more complete investigation is continuing (Solomon *et al.*, in press). The broad trends evident at this stage are mentioned below.

The $\delta^{34}S$ value of sulphur in the 'primary' solutions was probably close to zero because pyrite in the quartz porphyry dykes averages +0.8. In the pyrrhotite ores of the replacement deposit the $\delta^{34}S$ values are close to zero in the central zone of mineralisation, and increase towards the southern extremities. $\delta^{34}S$ values for the pyrrhotite ore in the main zone of mineralisation (Summit, Brown, Slaughteryard and Greisen Faces) average 0.2 and for the ore to the south the average is +2.4.

The trends observed are not those expected from diffusion outwards from the centre of mineralisation, or if sulphide sulphur was undergoing continuing re-equilibration with another sulphur species (*e.g.* $SO_4^{=}$) along a declining temperature gradient.

Pyrite Geothermometer

Hill and Green (1962) have related the thermoelectric potential of pyrites from Mt Bischoff and Renison Bell to temperatures of formation as suggested by Smith (1947). Low resistivities (positive thermoelectric potentials) were obtained for both groups of pyrites and were related to high temperatures of formation. Slightly lower resistivities were obtained for pyrite at Mt Bischoff than at Renison Bell, and a higher temperature of deposition was suggested. Some local correlation between resistivity and Cu and/or Mo content was suggested on the basis of 5 analyses of pyrite, but the analyses do not justify this correlation.

It is now evident that the trace or minor element content of the pyrite is the principal factor governing the thermoelectric potential (*e.g.* Suzuki, 1963), and that this is probably a function of the nature of the ore-forming fluid. The similar positive thermoelectric potential of pyrites from Mt Bischoff and Renison Bell may be a function of their similar trace element content. This is partly substantiated by the similar, low Co and Ni contents of pyrites from these deposits (Loftus-Hills, 1968).

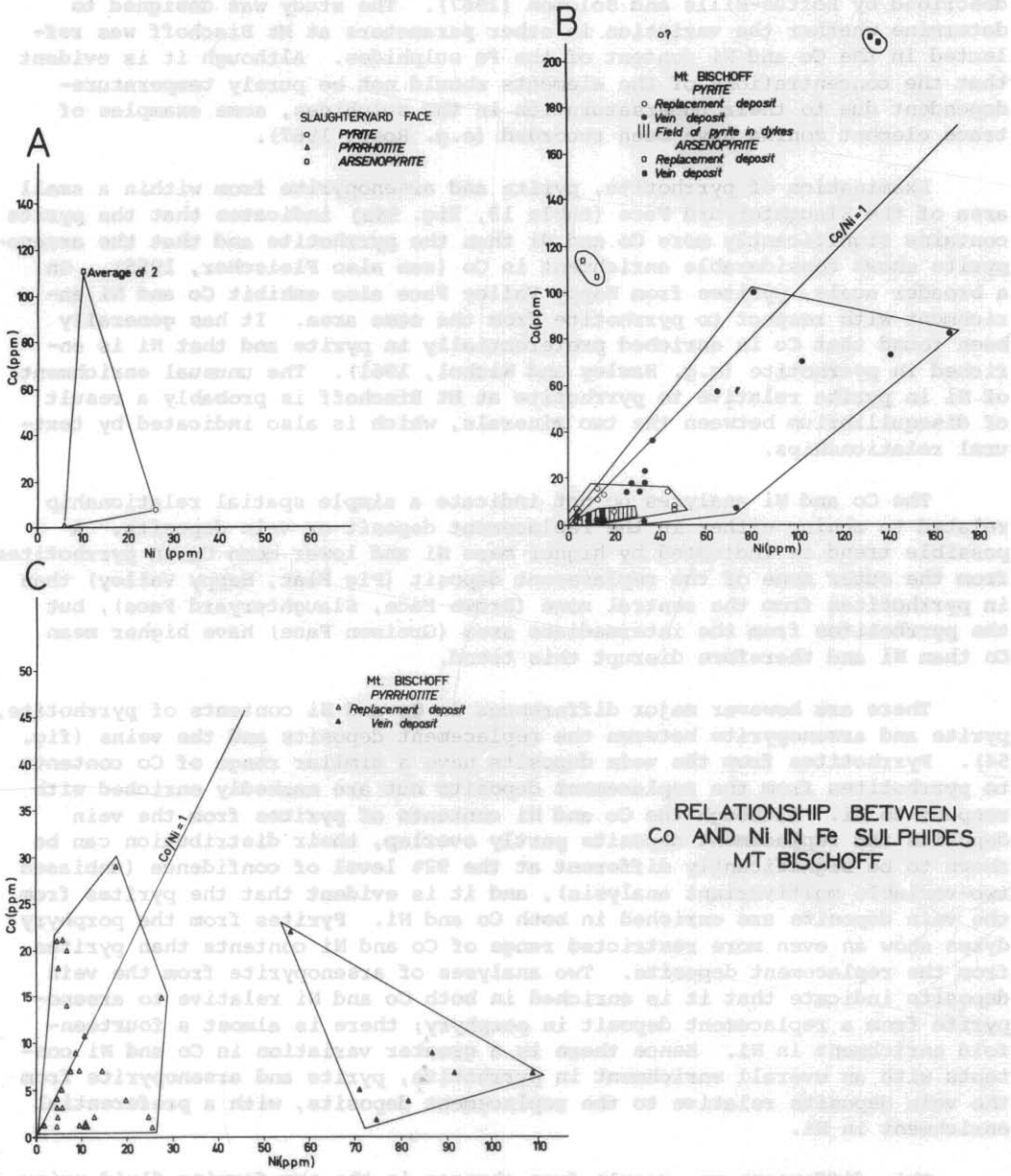

A
SLAUGHTERYARD FACE
PYRITE
PYRRHOTITE
ARSENOPYRITE
Average of 2
Co(ppm)
Ni (ppm)
B
Mt BISCHOFF
PYRITE
Replacement deposit
Vein deposit
Field of pyrite in dykes
ARSENOPYRITE
Replacement deposit
Vein deposit
Co/Ni = 1
Co(ppm)
Ni(ppm)
C
Mt. BISCHOFF
PYRRHOTITE
Replacement deposit
Vein deposit
Co/Ni = 1
Co(ppm)
Ni(ppm)
RELATIONSHIP BETWEEN
Co AND Ni IN Fe SULPHIDES
MT BISCHOFF
5 cm

Figure 54.

Trace Element Studies

Analyses of Co and Ni in Fe sulphides at Mt Bischoff have been carried out by Loftus-Hills (1968) and Groves (1968), using the analytical technique described by Loftus-Hills and Solomon (1967). The study was designed to determine whether the variation in other parameters at Mt Bischoff was reflected in the Co and Ni content of the Fe sulphides. Although it is evident that the concentrations of the elements should not be purely temperature-dependent due to their undersaturation in the sulphides, some examples of trace element zoning have been recorded (*e.g.* Rose, 1967).

Examination of pyrrhotite, pyrite and arsenopyrite from within a small area of the Slaughteryard Face (table 13, fig. 54a) indicates that the pyrite contains significantly more Co and Ni than the pyrrhotite and that the arsenopyrite shows considerable enrichment in Co (see also Fleischer, 1955). On a broader scale, pyrites from Happy Valley Face also exhibit Co and Ni enrichment with respect to pyrrhotite from the same area. It has generally been found that Co is enriched preferentially in pyrite and that Ni is enriched in pyrrhotite (*e.g.* Hawley and Nichol, 1961). The unusual enrichment of Ni in pyrite relative to pyrrhotite at Mt Bischoff is probably a result of disequilibrium between the two minerals, which is also indicated by textural relationships.

The Co and Ni analyses do not indicate a simple spatial relationship related to zoning either in the replacement deposit or vein deposits. A possible trend is indicated by higher mean Ni and lower mean Co in pyrrhotites from the outer zone of the replacement deposit (Pig Flat, Happy Valley) than in pyrrhotites from the central zone (Brown Face, Slaughteryard Face), but the pyrrhotites from the intermediate area (Greisen Face) have higher mean Co than Ni and therefore disrupt this trend.

There are however major differences in Co and Ni contents of pyrrhotite, pyrite and arsenopyrite between the replacement deposits and the veins (fig. 54). Pyrrhotites from the vein deposits have a similar range of Co contents to pyrrhotites from the replacement deposits but are markedly enriched with respect to Ni. Although the Co and Ni contents of pyrites from the vein deposits and replacement deposits partly overlap, their distribution can be shown to be significantly different at the 92% level of confidence (unbiased two-variable multivariant analysis), and it is evident that the pyrites from the vein deposits are enriched in both Co and Ni. Pyrites from the porphyry dykes show an even more restricted range of Co and Ni contents than pyrites from the replacement deposits. Two analyses of arsenopyrite from the vein deposits indicate that it is enriched in both Co and Ni relative to arsenopyrite from a replacement deposit in porphyry; there is almost a fourteen-fold enrichment in Ni. Hence there is a greater variation in Co and Ni contents with an overall enrichment in pyrrhotite, pyrite and arsenopyrite from the vein deposits relative to the replacement deposits, with a preferential enrichment in Ni.

This difference may result from changes in the ore-forming fluid prior to vein formation or from differences in the conditions of deposition in the two environments. The relatively small variation in Co and Ni in the replacement deposits may reflect stabilisation of the physico-chemical conditions of the ore-forming fluids due to the large effect of the dolomitic host rocks. Rose (1967) also recorded a greater variation in trace elements in vein deposits in part of southern U.S.A.

Table 13. COBALT AND NICKEL CONTENTS OF IRON SULPHIDES FROM MT BISCHOFF

Locality	Mineral	No. of Analyses	Range (ppm) Co	Range (ppm) Ni	Mean (ppm) Co	Mean (ppm) Ni	Co/Ni
White Face Dyke	po	2	0-11	14-74	6	44	0.14
White Face	py	12	1-6	8-28	3.5	13	0.26
North-eastern Dyke	py	10	1-6	5-25	3.5	12	0.28
Brown Face Dyke	po	2	10-13	11-13	12	12	1.00
Vein in Brown Face Dyke	py	2	10-11	46-51	10	49	0.20
Vein in White Face Dyke	py	1	-	-	13	8	1.65
Brown Face	po	6	1-20	5-11	8	8	1.00
Slaughteryard Face	po	6	1-21	2-15	6	8	0.75
	py	2	7-8	4-47	8	26	0.31
	apy	2	107-114	7-13	111	10	11.1
	py (sec)	3	6-16	17-42	9.5	30	0.31
Greisen Face	po	5	9-29	5-28	19	13	1.46
Pig Flat	po	6	1-11	5-26	4.5	12	0.38
South end of open cut	py	4	1-16	5-13	9	9	1.00
Happy Valley	po	3	1-2	11-25	2	16	0.13
	py	4	2-13, 210	16-44	61	31	1.97
Veins below Brown Face	po	3	2-22	56-82	9	71	0.13
	apy	2	207-209	134-137	208	136	1.53
	py	2	14-18	26-28	16	27	0.58
Vein beneath Greisen Face	po	4	5-9	71-109	7	90	0.08
Giblin Lode	py	2	73-82	142-168	78	155	0.50
Vein near Don Hill	py	2	4-5	15	5	15	0.33
North Valley Lode	py	4	7-100	34-82	47	64	0.73
Antimonial Lode	py	3	2-3	4-9	3	7	0.36
Lode north of Waratah River	py	1	-	-	2	16	0.12
Fooks Lode	py	4	2-70	13-103	28	45	0.62

po - pyrrhotite; py - pyrite; apy - arsenopyrite; sec. - secondary

Analysed by G. Loftus-Hills, 1966-68 and D. Groves and W. Baker, 1968

Summary

The Mt Bischoff deposit provides an opportunity to compare mineralogical and isotopic variation in sulphides with temperatures of deposition determined

from fluid inclusion studies. However, there is the possibility that the transparent minerals formed at a different time and temperature to the enclosed sulphides, so such comparisons have to be interpreted carefully.

Although there is little mineragraphic evidence that the pyrrhotites formed above about 300°C (from fluid inclusion data) were in equilibrium with pyrite, the temperatures derived by making this assumption are broadly compatible with the fluid inclusion data for fluorites contained within the pyrrhotite ore. Pyrrhotites formed below about 250°C have metal contents and structural distributions compatible with the hexagonal-monoclinic - monoclinic boundary determined by Clark (1966).

The sphalerites formed in pyrrhotite-pyrite ore probably achieved local surface equilibrium for only very short periods, thus giving a fairly wide spread of results with a few values approaching the experimental values found by Chernyshev *et al.* (1968). There is no clear correlation of iron content and temperature, and the distribution of iron in sphalerite is probably controlled largely by its activity in the different ore environments.

The distribution of $^{32}S/^{34}S$ ratios in the orebodies is not that expected from continual re-equilibration along a temperature gradient and is more probably related to isotopic fractionation resulting from kinetic rate effects, or to high-temperature equilibration with the initial sulphides forming from a species relatively rich in ^{32}S (Solomon *et al.*, in press).

Trace element distribution does not appear to be primarily controlled by temperature variations, and differences appear largely due to bulk differences in environment of deposition.

It would appear that a combination of conditions at the environment of deposition is more important in determining variations in sulphide compositions than the variation in temperature alone at the deposition sites.

GEOCHEMISTRY OF GRANITIC ROCKS

Introduction

It is evident that the tin deposits at Mt Bischoff are one of a series of similar deposits (Renison Bell, Cleveland, Razorback, Mt Lindsay, Mt Heemskirk) that are associated with several post-kinematic high-level granitic rocks of late Devonian-early Carboniferous age that intrude Lower and Middle Palaeozoic rocks in western Tasmania (fig. 36).

Chemical analyses of granitic rocks from the Meredith Granite and Mt Bischoff have been compared with analyses of granitic rocks from the Heemskirk Granite and the Pine Hill Complex at Renison Bell (fig. 36). This has been done to determine the chemical characteristics and fractionation trends of the tin-bearing granitic rocks and to associate these chemical trends with geological evidence of depth of intrusion, degree of unroofing etc. Chemical changes during alteration of porphyries at Mt Bischoff are also discussed.

Chemical analyses of granitic rocks from Meredith Granite, Mt Bischoff and Pine Hill are given in Tables 14-19 inclusive and data for the Heemskirk Granite is taken from Blissett (1962) and Brooks and Compston (1965). The analytical procedure, precision and accuracy of the results are described in detail by Groves (1968, appendix A4). The study is made difficult by the inaccessibility of much of the Meredith Granite which makes systematic sampling virtually impossible. The extreme alteration of most rocks at Pine

Hill and Mt Bischoff also makes sampling difficult.

The geologic setting of the Meredith Granite and the quartz-feldspar porphyries of Mt Bischoff are described previously.

The structure and petrography of the Heemskirk Granite have been described by Green (1966). The granite is a large stock or small batholith which is roughly concordant on a regional scale but markedly discordant locally. It is a multiple intrusion consisting essentially of two main types which have been grouped as 'red granite' and 'white granite' although they are generally adamellites (Brooks and Compston, 1965). These granites form gently undulating sub-horizontal layers (Klomínský pers. comm.)

The 'red granite' is generally coarse-grained and consists chiefly of quartz and perthitic pink orthoclase with acid andesine, biotite and rare tourmaline. The 'white granite' is mineralogically similar to the 'red granite' but contains white orthoclase and essential accessory tourmaline (schorlite), and is the host rock for most of the abundant tourmaline- and tin-bearing bodies in the granite complex. The red and white granites are intruded by aplite and pegmatite dykes and various quartz-tourmaline bodies. Brooks and Compston (1965) recorded a higher initial Sr^{87}/Sr^{86} for the white granite than for the red granite which they interpreted in terms of contamination of the white granite by radiogenic Sr during emplacement. The Rb-Sr age for the red granite and white A granite are indistinguishable but the white B granite could be several million years older (table 3).

The Pine Hill complex is probably a small cupola with associated radial dykes. The unaltered rocks are predominantly biotite adamellites, partly porphyritic, consisting of quartz, perthitic potash feldspar and albite (An_6 Ab_{94} - An_9Ab_{91}) with minor biotite, muscovite and tourmaline. They are similar to the microgranites of the Meredith Granite. Quartz-feldspar porphyry dykes intrude these adamellites and consist of large euhedral potash feldspar phenocrysts in a fine-grained groundmass of quartz and feldspar.

The presently exposed area of the Heemskirk Granite may be in closer proximity to the roof area than the exposed area of the Meredith Granite which contains only minor tin mineralisation. This is suggested by the sub-horizontal layering and the unusually high proportion of mineralisation accompanying tourmalinisation and greisenisation within the Heemskirk Granite and the cupola-type environment suggested by Brooks and Compston (1965) to explain the high initial Sr^{87}/Sr^{87} ratios of the granites. The complex at Pine Hill contains rocks similar to late intrusions in the Meredith Granite and appears to be the roof of a small cupola, possibly similar to the protuberance at Aberfoyle in Eastern Tasmania (*e.g.* Kingsbury, 1965), and may be a relatively small upward projection of a larger granite stock. The dyke system at Mt Bischoff probably represents filling of tensional fractures above a hidden cupola-like body such as that at Pine Hill. High concentrations of boron, fluorine, sulphur and H_2O were common in the roof zones of the intrusions and resulted in the ubiquitous occurrence of tourmaline, topaz, axinite and sulphides above the granites and in their contact aureoles.

The similar radiometric ages and similar trace element concentrations of sulphides in the granitic rocks and associated ore deposits suggests that the granitic rocks are inter-related. It is envisaged that each granite, as exposed, corresponds to a different level within one or more penecontemporaneous, compositionally similar batholiths.

Major Element Geochemistry

Major element analyses are given in Table 15-19 inclusive and the

Table 14. AVERAGE COMPOSITIONS OF GRANITIC ROCKS FROM MEREDITH GRANITE, HEEMSKIRK GRANITE, PINE HILL AND MT BISCHOFF

		MEREDITH GRANITE			HEEMSKIRK GRANITE		PINE HILL COMPLEX		MT BISCHOFF
		Adamellite (3)	Porphyritic Adamellite (5)	Sodaclase Microgranite (2)	Red Granite (7)	White Granite (7)	Sodaclase Adamellite (3)	Porphyry (2)	Porphyry (5)
	SiO_2	72.2	73.3	74.35	75.52	74.44	73.9	72.7	75.3
	TiO_2	0.36	0.27	0.13	0.19	0.21	0.06	0.01	0.02
	Al_2O_3	14.10	13.40	13.50	12.73	13.44	14.75	14.95	14.25
	Fe_2O_3				0.61	0.20			
	FeO				1.13	1.66			
equiv.	FeO	2.00	2.07	2.03			1.61	3.67	1.95
	MnO	0.02	0.05	0.02	0.04	0.05	0.02	0.09	0.06
	MgO	0.85	0.65	0.8	0.25	0.18	0.6	0.65	0.6
	CaO	0.81	0.76	0.17	0.60	0.83	0.13	1.09	0.20
	Na_2O	3.1	3.2	3.6	2.93	2.77	3.2	0.55	0.14
	K_2O	4.7	4.8	4.15	5.23	5.00	4.6	4.0	4.9
	P_2O_5	0.04	0.04	0.01	0.05	0.11	0.02	0.04	0.09
	H_2O^+	0.79	0.58	0.70	0.54	0.55	1.00	1.32	1.62
	H_2O^-	0.38	0.18	0.28	0.25	0.14	0.21	0.12	0.23
	F	<0.2	<0.2	<0.2			<0.2	0.8	<0.2
	Total	99.36	99.30	99.74	100.07	99.58	100.10	99.79	99.36
	Ba	552	396	83	-	-	88	190	149
	Cu	9	6	4	-	-	7	5	4
	Li	42	45	48	-	-	120	54	93
	Ni	3?	5	4	-	-	-	10	5
	Pb	33	27	25	-	-	-	27	32
	Rb	218	243	327	358	399	754	784	906
	Sn	3	5	4	-	-	27	59	72
	Sr	80	47	14	35	38	8	6	3
	Th	21	19	18	49.2	24.1	32	43	11
	U	9	9	19	12.4	18.0	23	32	13
	Zn	54	28	22	-	-	25	92	122
	K/Rb	188	166	119	142	106	52	49	56
	Th/U	2.3	2.9	1.0	4.0	1.3	1.7	1.4	0.8

Table 15. MAJOR ELEMENT ANALYSES (WT %) AND TRACE ELEMENT ANALYSES (PPM) OF GRANITIC ROCKS FROM THE MEREDITH GRANITE

	Adamellite			Porphyritic Adamellite						Microgranite		Aplite
	1404†	1408*	1479†	1405†	1481†	1482†	64-36*	30651†	1480†	64-33†	64-39†	64-34†
SiO_2	71.1	70.0	75.6	70.1	74.5	74.8	74.3	71.2	74.9	74.1	74.6	80.4
TiO_2	0.52	0.49	0.06	0.70	0.08	0.16	0.15	0.45	0.10	0.20	0.06	0.04
Al_2O_3	14.7	14.7	13.0	15.2	12.8	13.0	13.3	13.8	12.2	13.6	13.4	12.1
Fe_2O_3		1.0					0.86				1.7	
FeO		2.1					1.3				0.6	
equiv. FeO	2.4		0.55	3.2	1.3	1.4		3.0	1.45	2.0		1.1
MnO	0.03	0.03	0.00	0.02	0.04	0.03	0.02	0.13	0.04	0.03	0.00	0.00
MgO	1.0	1.1	0.45	1.6	0.2	0.4	0.6	0.9	0.1	1.1	0.5	0.8
CaO	1.10	0.97	0.35	1.48	0.44	0.42	0.33	1.41	0.46	0.23	0.11	0.01
Na_2O	2.8	2.9	3.7	2.9	3.4	3.35	2.9	2.5	3.9	3.6	3.6	0.1
K_2O	5.2	5.2	4.2	4.5	4.8	4.8	5.4	4.7	4.6	4.3	4.0	2.7
P_2O_5	0.00	0.10	0.01	0.05	0.03	0.00	0.00	0.10	0.03	0.02	0.00	0.06
H_2O^+	0.80	0.86	0.71	0.54	0.43	0.78	0.51	0.84	0.37	0.70	0.70	1.70
H_2O^-	0.35	0.42	0.39	0.22	0.13	0.20	0.23	0.14	0.15	0.25	0.32	0.49
Total	100.00	99.87	99.02	100.51	98.15	99.34	99.90	99.17	98.30	100.13	99.59	99.50
Ba	508	595	-	506	-	-	185	498	-	121	45	40
Cu	3	9	16	6	<3	13	7	6	4	3	5	7
Li	30	-	54	28	52	47	-	30	66	48	-	225
Ni	<3	-	62?	6	4	<3	3	10	6	4	-	<3
Pb	33	-	-	19	-	-	34	-	-	25	-	-
Rb	242	240	173	210	255	221	314	213	245	230	423	158
Sn	<3	<3	<3	13	<3	3	6	3	3	6	<3	3
Sr	96	116	28	93	15	35	26	103	12	26	<3	4
Th	25	19	19	19	17	27	16	18	18	23	12	5
U	11	8	9	6	12	7	11	13	3	14	23	11
Zn	24	81	57	38	12	15	10	77	15	16	28	27
K/Rb	182	180	201	178	156	180	143	183	155	159	78	142
Th/U	2.3	2.4	2.1	3.2	1.4	3.9	1.5	1.4	6.0	1.6	0.5	0.5

Major element analyses: †analyst, D.I. Groves, 1967; *analyst, Department of Mines Laboratories, Launceston. *Trace element analyses:* analyst, D.I. Groves, 1967.

Table 16. NIGGLI VALUES AND C.I.P.W. NORMS FOR GRANITIC ROCKS FROM THE MT MEREDITH GRANITE

	Adamellite			Porphyritic Adamellite						Microgranite		Aplite
	1404	1408	1479	1405	1481	1482	64-36	30651	1480	64-33	64-39	64-34
Niggli Values												
al	44	43	49	42	48	47.5	46	43	46	45	48	65
fm	19	21	7	24	9	11	15	20	9	19	15	19
c	6	5	2.5	7	3	3	2	8	3	1	1	0
alk	31	31	41	27	40	38.5	37	29	42.5	35	36	16
si	368	350	487	329	476	463	434	377	477	416	452	727
C.I.P.W. Norms												
Quartz	29.4	28.8	36.2	28.4	34.1	34.4	34.5	32.3	32.4	33.1	37.4	67.6
Orthoclase	30.7	30.7	24.8	26.6	28.4	28.4	31.9	27.8	27.2	25.4	23.6	16.0
Albite	23.7	24.5	31.3	24.5	28.8	28.3	24.5	21.1	33.0	30.4	30.4	0.5
Anorthite	5.5	4.2	1.7	7.0	2.0	2.1	1.6	6.4	2.1	1.0	0.5	0.0
Corundum	2.5	2.8	1.8	3.0	1.3	1.5	2.1	2.3	0.0	2.7	3.0	9.1
Magnetite	0.0	1.5	0.0	0.0	0.0	0.0	1.25	0.0	0.0	0.0	1.8	0.0
Ferrosilite	3.6	2.3	0.9	4.8	2.3	2.4	1.5	5.0	2.6	3.4	0.0	2.0
Enstatite	2.5	2.7	1.1	4.0	0.5	1.0	1.5	2.25	0.3	2.7	1.2	2.0
Ilmenite	1.0	1.0	0.1	1.3	0.2	0.3	0.3	0.9	0.2	0.4	0.1	0.1
Apatite	0.0	0.2	0.0	0.1	0.1	0.0	0.0	0.25	0.1	0.1	0.0	0.0

Table 17. MAJOR ELEMENT ANALYSIS (WT %) AND TRACE ELEMENT ANALYSES (PPM) OF PORPHYRIES FROM MT BISCHOFF

	Quartz-Feldspar Porphyry							Altered Quartz Porphyry											
	30634*	1474*	1388a†	1388b	1389	1391†	1393†	1390*	1389a	1392*	1471*	1472*	1473*	1394	1395	1396	1397	1398	1399
SiO_2	73.8	79.7	74.2	-	-	73.6	75.3	66.9	-	70.2	69.0	68.6	72.3	-	-	-	-	-	-
TiO_2	0.06		0.01	-	-	0.00	0.02	0.06	-	0.09	0.05	0.00	tr	-	-	-	-	-	-
Al_2O_3	14.4	13.5	14.2	-	-	14.7	14.5	19.9	-	21.7	20.0	19.4	17.2	-	-	-	-	-	-
Fe_2O_3	0.61	0.14	-	-	-	-	-	tr	-	0.14	1.05	0.21	tr	-	-	-	-	-	-
FeO	1.57	2.08	-	-	-	-	-	0.64	-	0.35	0.19	0.29	0.58	-	-	-	-	-	-
equiv. FeO	-	-	1.5	-	-	1.6	2.3	-	-	-	-	-	-	-	-	-	-	-	-
MnO	0.09	-	0.10	-	-	0.00	0.12	tr	-	tr	tr	tr	tr	-	-	-	-	-	-
MgO	0.6	0.6	0.6	-	-	0.65	0.6	0.25	-	0.2	0.3	0.3	0.3	-	-	-	-	-	-
CaO	0.44	0.46	0.04	-	-	0.01	0.04	0.36	-	0.20	0.20	0.68	0.28	-	-	-	-	-	-
Na_2O	0.4	0.1	0.1	-	-	0.1	0.6	0.05	-	tr	tr	tr	tr	-	-	-	-	-	-
K_2O	6.0	2.7	6.0	-	-	5.8	3.9	0.1	-	0.1	tr	tr	0.4	-	-	-	-	-	-
P_2O_5	tr	-	0.09	-	-	0.09	0.09	tr	-	tr	tr	tr	tr	-	-	-	-	-	-
H_2O^+	2.15	-	1.45	-	-	1.35	1.53	0.98	-	0.77	0.69	0.96	1.04	-	-	-	-	-	-
H_2O^-	0.12	-	0.20	-	-	0.30	0.30	0.11	-	tr	0.05	0.24	0.19	-	-	-	-	-	-
F	-	-	<0.2	-	-	<0.2	2.5	6.48	-	6.63	6.86	6.46	6.15	-	-	-	-	-	-
FeS_2	-	-	-	-	-	-	-	6.62	-	2.19	4.86	5.85	3.91	-	-	-	-	-	-
Total	100.24	99.28	98.49			98.20	100.80	99.74		99.87	100.31	99.75	99.85						
Ba	180	-	192	154	96	192	81	47	50	52	-	-	-	47	47	60	37	52	62
Cu	4	-	4	4	<3	6	4	<3	<3	<3	-	-	-	48	154	178	86	130	40
Li	-	-	107	108	-	111	45	27	5	-	-	-	-	6	-	-	5	7	5
Ni	5	-	6	5	7	4	5	<3	<3	<3	-	-	-	6	4	5	4	4	4
Pb	28	-	39	44	23	21	37	-	-	-	-	-	-	-	-	-	-	-	-
Rb	858	-	846	945	1250	846	689	12	52	12	-	-	-	5	<3	<3	<3	<3	<3
Sn	30	-	32	32	144	100	95	1500	144	610	-	-	-	544	660	2342	1720	4536	1078
Sr	3	-	4	4	<3	<3	3	<3	<3	<3	-	-	-	<3	<3	<3	<3	5	3
Th	-	-	9	8	7	20	11	-	7	14	-	-	-	100	50	21	17	<3	6
U	-	-	10	10	11	22	11	-	11	<3	-	-	-	11	7	9	4	6	3
Zn	100	-	124	138	119	184	67	194	80	90	-	-	-	98	135	61	143	112	82
K/Rb	58	-	59	59	-	57	47	67	-	67	-	-	-	-	-	-	-	-	-
Th/U	-	-	0.9	0.8	0.6	0.9	1.0	-	0.6	>4.7	-	-	-	9.1	7.1	2.3	4.25	<0.5	2.0

Major element analyses: †analyst D.I. Groves, 1967; *analyst Department of Mines Laboratories, Launceston. *Trace element analyses:* analyst D.I. Groves, 1967

Table 18. MAJOR ELEMENT ANALYSES (WT %) AND TRACE ELEMENT ANALYSES (PPM) OF GRANITIC ROCKS FROM PINE HILL

	Adamellite			Porphyry		Greisen		Tourmalite		
	1424	1426	1427	1417b	1419	1415	1417a	1416	1418	1420
SiO_2	73.1	74.4	74.2	70.4	76.8	71.0	70.3	63.6	66.8	72.1
TiO_2	0.12	0.05	0.02	0.01	0.02	0.00	0.01	0.02	0.03	0.06
Al_2O_3	15.0	14.4	14.8	15.9	13.4	15.5	15.0	18.1	14.4	12.3
equiv. FeO	1.9	2.1	0.9	4.0	1.8	5.1	10.0	12.2	10.8	9.1
MnO	0.02	0.02	0.02	0.10	0.02	0.15	0.13	0.03	0.06	0.08
MgO	0.7	0.6	0.55	0.55	0.6	0.8	0.7	0.7	0.6	0.6
CaO	0.10	0.12	0.16	0.04	0.64	2.58	0.03	0.07	0.03	0.05
Na_2O	3.2	2.3	4.1	0.25	0.7	0.7	0.9	1.0	1.0	0.9
K_2O	4.8	4.5	4.5	4.2	4.9	2.9	0.9	0.1	0.1	0.4
P_2O_5	0.02	0.02	0.02	0.07	0.04	0.02	0.04	0.02	0.00	0.02
H_2O^+	0.90	1.30	0.80	1.67	0.97	2.15	1.11	1.53	1.27	1.16
H_2O^-	0.25	0.22	0.16	0.12	0.11	0.15	0.03	0.02	0.02	0.02
B_2O_3								4.8	4.5	3.2
Total	100.11	100.03	100.23	97.31	100.00	101.05	99.15	102.19	99.61	99.99
Ba	68	63	133	329	50	30	58	39	40	35
Cu	6	7	7	4	6	4	4	4	4	6
Li	38	299	24	34	74	68	50	41	35	40
Ni	-	-	-	11	9	7	9	11	5	4
Pb	-	-	-	16	38	-	9	7	18	18
Rb	824	765	672	834	734	826	115	7	5	7
Sn	15	37	27	104	14	104	87	28	103	58
Sr	7	6	12	3	8	11	7	9	6	3
Th	25	59	12	25	61	6	15	11	5	<3
U	40	22	7	17	46	7	5	17	<3	11
Zn	29	31	15	68	113	561	108	104	117	135
K/Rb	49	50	57	42	55	29	65	119	166	474
Th/U	0.6	2.7	1.7	1.5	1.3	0.9	3.0	0.6	>1.7	<0.3

Analyst: D.I. Groves, 1967

Table 19. NIGGLI VALUES AND C.I.P.W. NORMS FOR GRANITIC ROCKS FROM MT BISCHOFF AND PINE HILL

	MT BISCHOFF					PINE HILL			
	Quartz-Feldspar Porphyry					Adamellite			Porphyry
	30634	1474	1388a	1391	1393	1424	1426	1427	1419
Niggli Values									
al	53	61	58	59	59	50	51	50	53
fm	17	21	15	16	19	15	16	9	16
c	3	4	0.5	0	0.5	1	1	1	4
alk	26	14	26	26	21	35	31	40	26
si	463	614	512	500	517	413	457	430	522
C.I.P.W. Norms									
Quartz	45.6	65.2	48.4	48.5	54.0	33.4	41.0	31.3	50.3
Orthoclase	35.5	16.0	35.4	34.3	23.0	28.4	26.6	26.6	28.9
Albite	3.4	0.85	0.9	0.8	5.1	27.0	19.45	34.7	5.9
Anorthite	2.1	2.3	0.0	0.0	0.0	0.4	0.5	0.7	2.9
Corundum	6.5	9.6	7.5	8.3	9.3	4.4	5.6	2.9	5.9
Magnetite	0.9	0.2	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Ferrosilite	2.5	3.7	2.9	2.9	4.4	3.3	3.8	1.7	3.3
Enstatite	1.5	1.5	1.5	1.6	1.5	1.7	1.5	1.4	1.5
Ilmenite	0.1	0.0	0.0	0.0	0.0	0.2	0.1	0.0	0.0
Apatite	0.0	0.0	0.2	0.2	0.2	0.0	0.0	0.0	0.1

average composition of the granites in Table 14.

From these analyses it is evident that the equigranular adamellite and the porphyritic adamellite of the Meredith Granite are chemically similar, thus substantiating the earlier work by Groves (1963) and Groves and Solomon (1964) who suggested on the basis of modal analyses that they were mineralogically similar. All the granitic rocks have high average SiO_2 contents indicating that they are highly fractionated rocks. High SiO_2 content is a consistent feature of granites associated with tin mineralisation (*e.g.* Hosking, 1965). The microgranites are SiO_2-enriched relative to the adamellites of Pine Hill, and the adamellites of the Heemskirk Granite are more SiO_2 rich than the adamellites of the Meredith Granite. The SiO_2 contents of the Heemskirk Granite are similar to those of granites from Rossarden and Royal George (Ben Lomond) Granite in eastern Tasmania which are also high level bodies (Blissett, 1959; Beattie, 1967) and may also have anomalously high Sr^{87}/Sr^{86} ratios (McDougall and Leggo, 1965). The unaltered quartz-feldspar porphyry from Pine Hill (1419) has a similar SiO_2 content to the Mt Bischoff porphyries but, as predicted from the petrography, contains significantly more Na_2O and CaO than the Mt Bischoff rocks.

It is evident from Figure 55 that there is a sympathetic variation of FeO and MgO for all the granite types, MgO increasing with increasing FeO. The adamellites of the Meredith Granite have higher FeO and correspondingly higher MgO than the adamellites of the Heemskirk Granite, and appear to be relatively enriched in MgO. The rocks from Mt Bischoff and Pine Hill also appear to be slightly enriched in MgO relative to the Heemskirk Granite.

The relationship between the major alkalies is indicated in the plot of molecular quotients (x100) of K_2O and Na_2O (fig. 55). The adamellites from the Meredith Granite, Heemskirk Granite and Pine Hill complex plot close to the line of equimolar proportions of K_2O and Na_2O, with a tendency to plot on the K_2O-rich side of this line. The alkali oxides also show an inverse relationship and appear to plot about a line with the form $K_2O + Na_2O$ = constant (approx. 10). Superimposed on this major trend there is a tendency for total alkali oxides to increase slightly with increasing Na_2O/K_2O, this being a common phenomenon in granitic rocks.

The field occupied by the porphyries from Mt Bischoff and Pine Hill on the alkali diagram is unusual, and is an entirely separate field to the adamellites. The porphyries have a similar range in K_2O to the adamellites but have extremely low Na_2O values and there appears to be no correlation between K_2O and Na_2O in the limited range of values obtained. Similar depletion of Na_2O has been recorded in some late stage intrusions of fine granite and elvans from the Carnmenellis granite from south-west England (Ghosh, 1934), and it is evident in some analyses of elvans given by Exley and Stone (1966). The relationship between the alkalies explains the anomalous behaviour of K found by Brooks and Compston (1965), the K in the white granites decreasing with increasing SiO_2 the reverse of trends shown by the red granites and general trends in granites described by Nockolds and Allen (1953). It can be seen that Na has the reverse relationship in each of the granite types, thus maintaining the inverse relationship indicated in Figure 55.

The sympathetic variation of FeO-MgO and the inverse relationship of K_2O and Na_2O indicates that the most valid graphical representation of chemical variation in the granitic rocks would be obtained by plotting Niggli values, calculated for groups of chemically similar oxides, rather than single oxides. Values of *al*, *alk*, *fm* and *c* have been plotted against *si* for all the granitic rocks (fig. 56) and smoothed variation curves are given in Figure 57.

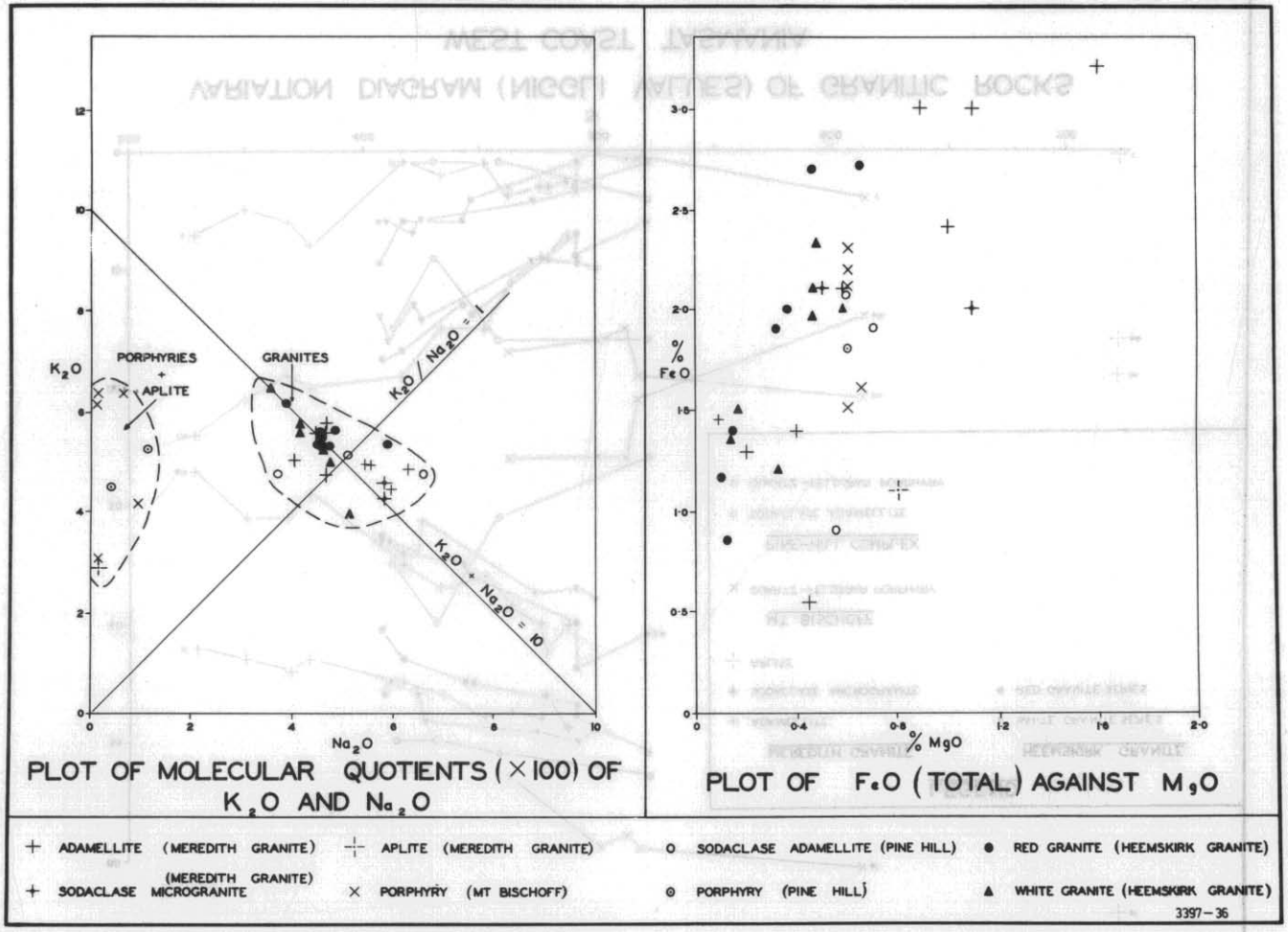

5 cm
PORPHYRIES + APLITE
GRANITES
$K_2O / Na_2O = 1$
$K_2O + Na_2O = 10$
K_2O
Na_2O
PLOT OF MOLECULAR QUOTIENTS (×100) OF K_2O AND Na_2O
% FeO
% MgO
PLOT OF FeO (TOTAL) AGAINST MgO
+ ADAMELLITE (MEREDITH GRANITE)
+ SODACLASE MICROGRANITE (MEREDITH GRANITE)
+ APLITE (MEREDITH GRANITE)
× PORPHYRY (MT BISCHOFF)
O SODACLASE ADAMELLITE (PINE HILL)
⊙ PORPHYRY (PINE HILL)
● RED GRANITE (HEEMSKIRK GRANITE)
▲ WHITE GRANITE (HEEMSKIRK GRANITE)
3397-36

Figure 55.

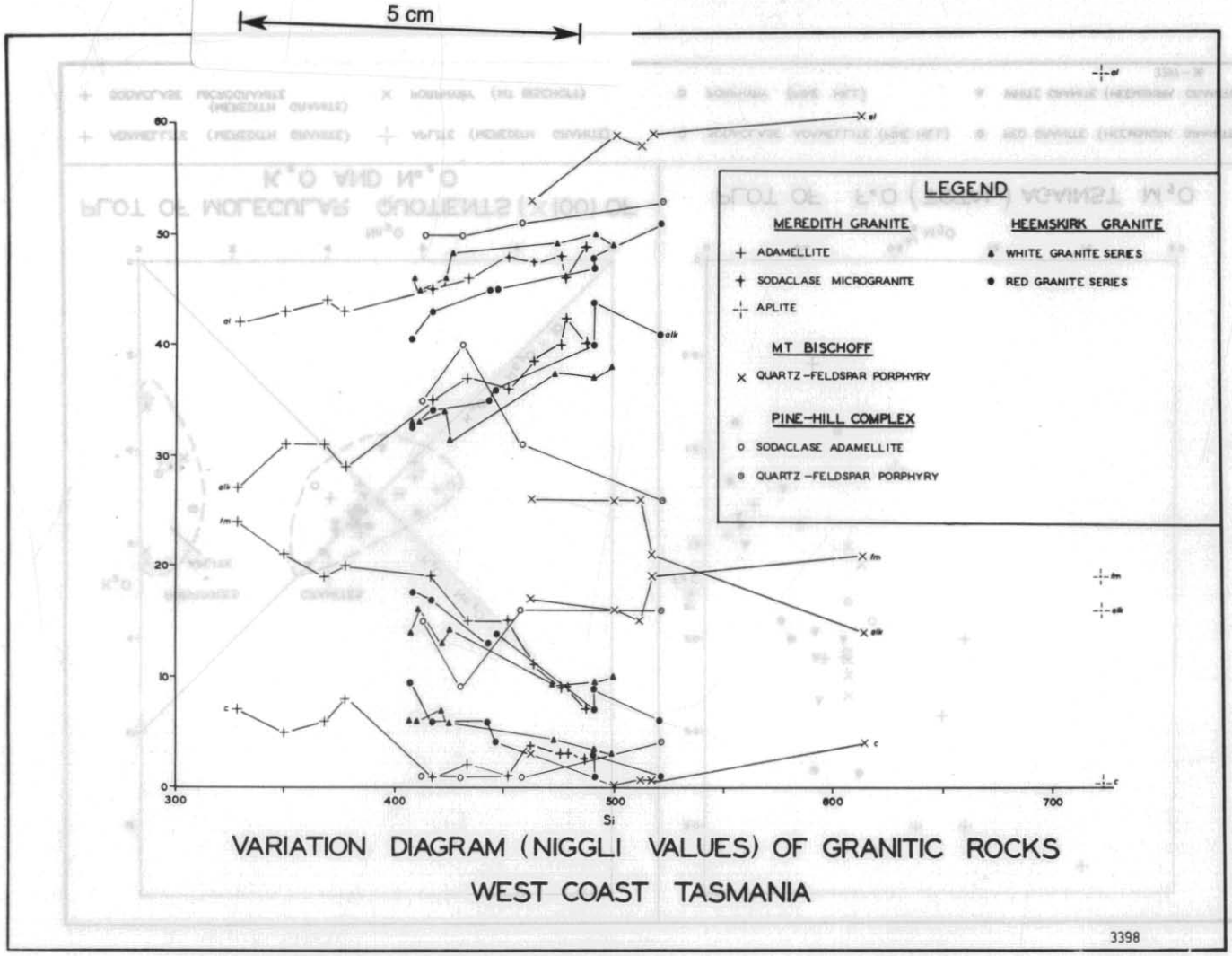

5 cm
LEGEND
MEREDITH GRANITE
ADAMELLITE
SODACLASE MICROGRANITE
APLITE
HEEMSKIRK GRANITE
WHITE GRANITE SERIES
RED GRANITE SERIES
MT BISCHOFF
QUARTZ-FELDSPAR PORPHYRY
PINE-HILL COMPLEX
SODACLASE ADAMELLITE
QUARTZ-FELDSPAR PORPHYRY
60
50
40
30
20
10
0
300
400
500
600
700
Si
af
alk
fm
c
VARIATION DIAGRAM (NIGGLI VALUES) OF GRANITIC ROCKS
WEST COAST TASMANIA
3398

Figure 56.

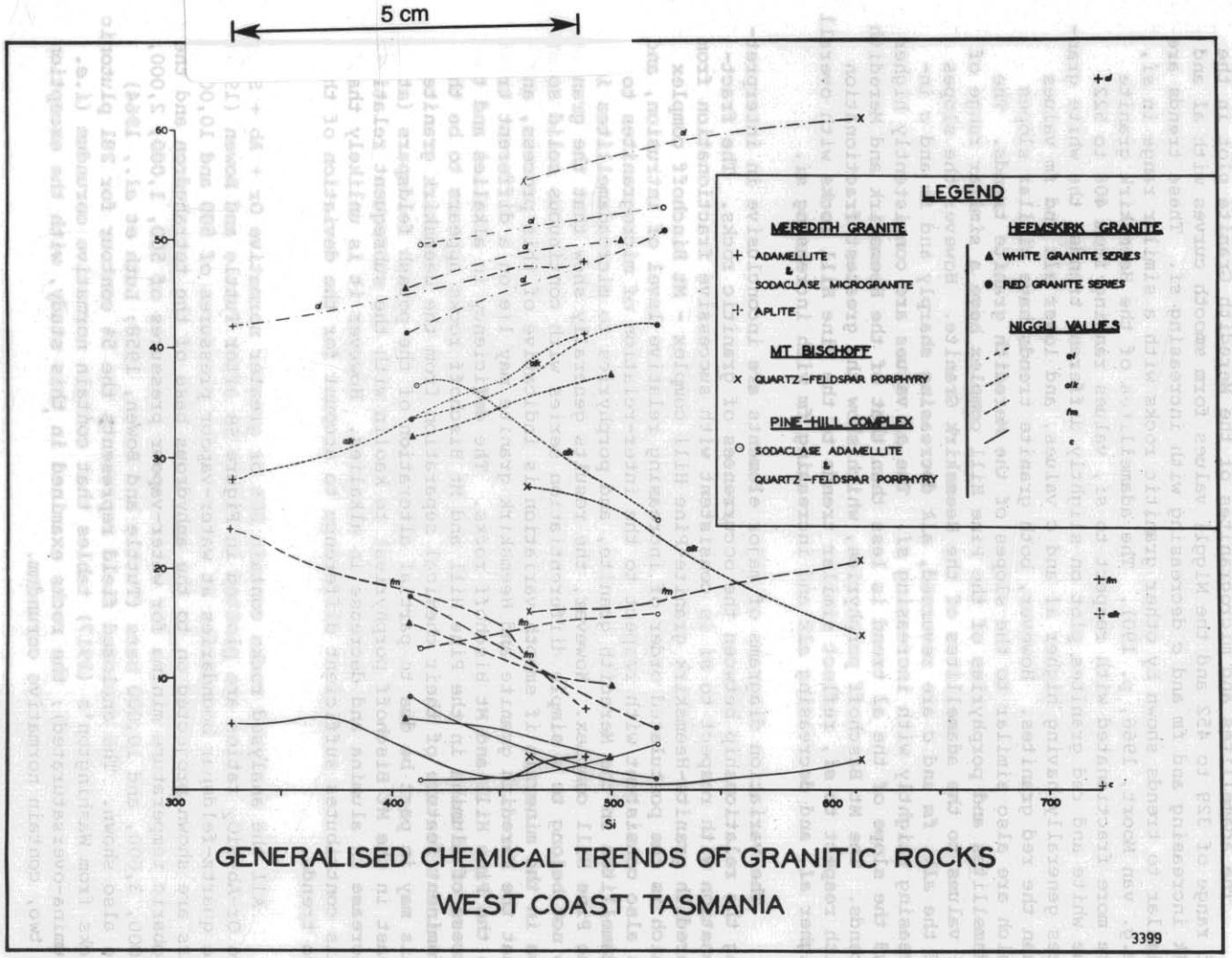

5 cm
LEGEND
MEREDITH GRANITE
+ ADAMELLITE & SODACLASE MICROGRANITE
+ APLITE
MT BISCHOFF
X QUARTZ-FELDSPAR PORPHYRY
PINE-HILL COMPLEX
○ SODACLASE ADAMELLITE & QUARTZ-FELDSPAR PORPHYRY
HEEMSKIRK GRANITE
▲ WHITE GRANITE SERIES
● RED GRANITE SERIES
NIGGLI VALUES
al
alk
fm
c
60
50
40
30
20
10
300
400
500
600
700
Si
GENERALISED CHEMICAL TRENDS OF GRANITIC ROCKS
WEST COAST TASMANIA
3399

Figure 57.

The adamellites and microgranites of the Meredith Granite plot in the *si* range of 329 to 452 and the Niggli values form smooth curves with *al* and *alk* increasing and *fm* and *c* decreasing with increasing *si*. These trends are similar to trends shown by other granitic rocks with a similar range in *si*, (*e.g.* van Moort, 1966, p. 190). The adamellites of the Heemskirk granite are more fractionated with respect to *si*, values ranging from 408 to 522. The white and red granites plot on slightly different trends, the white granites generally having higher *al* and *c* values, and lower *alk* and *fm* values than the red granites. However, both granite trends have similar slopes which are also similar to the slopes of the Meredith granite trends. The adamellites and porphyries of the Pine Hill complex have a similar range of *si* values to the adamellites of the Heemskirk Granite. However the slopes of the *alk*, *fm* and *c* are reversed, *alk* decreasing sharply and *fm* and *c* increasing slightly with increasing *si*. The *al* values are consistently higher and the slope of the *al* trend is less than that of the Heemskirk and Meredith trends. The Mt Bischoff porphyries, which show the greatest fractionation with respect to *si*, reflect similar trends to the Pine Hill rocks with overall higher *al*, and decreasing *alk* and increasing *fm* with increasing *si*.

The variation diagrams of major elements are inconclusive in interpreting the relationship between the occurrences of granitic rocks. The fractionation with respect to *si* is consistent with successive fractionation from Meredith granite-Heemskirk granite-Pine Hill complex - Mt Bischoff complex which is the postulated order of increasing relative level of intrusion, and is also consistent with respect to the inter-relation of microgranites to adamellites in the Meredith granite, and porphyries to microadamellites in the Pine Hill complex. However, the results generally show that the granites do not belong to a simple differentiation series with continuous solid solution in the minerals if smooth variation is indicative of this process, and that the Meredith granite and Heemskirk granite may lie on a different trend to the Pine Hill and Mt Bischoff rocks. The deficiency in alkalies and the excess of alumina in the Pine Hill and Mt Bischoff rocks appears to be the prominent feature of their chemical separation from the Heemskirk granite. This may in part be due to partial alteration of the potash feldspars (at least in the Mt Bischoff porphyries) to kaolin with the subsequent relative increase in alumina and decrease in alkalies. However it is unlikely that this contributes sufficient difference to account for the deviation of the two trends.

All the analysed rocks contain 80% or greater normative Or + Ab + SiO_2. The Or-Ab-SiO_2 ratios are plotted in Figure 58 after Tuttle and Bowen (1958). The quartz-feldspar boundaries at water-vapour pressures of 500 and 10,000 bars are shown projected on to the anhydrous base of the tetrahedron and the isobaric temperature minima for water-vapour pressures of 500, 1,000, 2,000, 3,000, 3,600, and 10,000 bars (Tuttle and Bowen, 1958; Luth *et al.*, 1964) are also shown. The enclosed field represents the 5% contour for 281 plutonic rocks from Washington's (1917) tables that contain normative corundum (*i.e.* alumina-oversaturated); the rocks examined in this study, with the exception of two, contain normative corundum.

It is apparent that the adamellites from the Heemskirk granite, the adamellites and microgranites from the Meredith granite and the adamellites from Pine Hill plot close to the region of the ternary minimum at 500 bars in this system. This indicates crystal-liquid equilibrium. The origin of the magma from which they have fractionated is beyond the scope of this study. However, Luth *et al.* (1964) have suggested that alumina-oversaturated plutonic rocks in general may have resulted from partial melting of sedimentary material with alumina-oversaturation reflecting earlier processes of weathering and sedimentation. The white granites of the Heemskirk granite

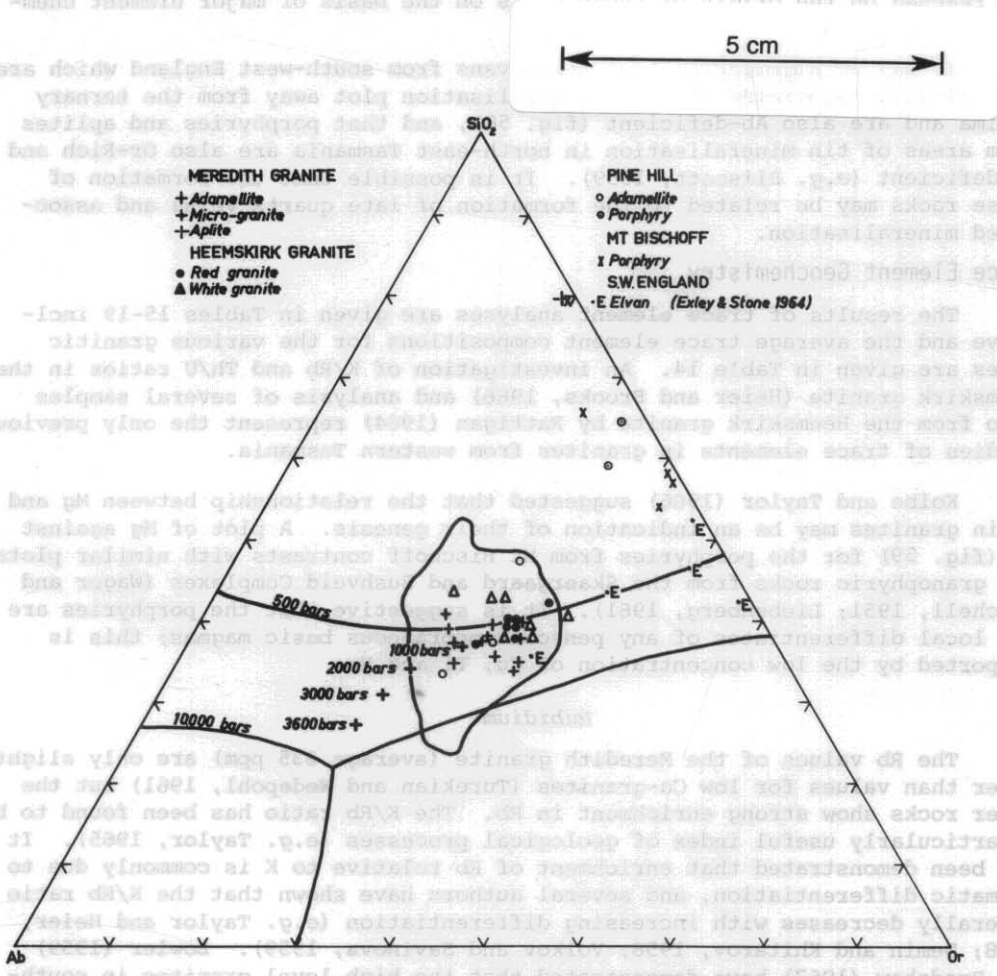


Figure 58. *Plot of normative Or-Ab-SiO_2 ratios of granitic rocks, western Tasmania. Diagram after Tuttle and Bowen (1958) and Luth et al. (1964), showing quartz-feldspar boundaries and isobaric temperature minima for different water-vapour pressures. The enclosed field represents the 5 per cent contour for 281 alumina-oversaturated plutonic rocks [Washington, 1917].*

show no tendency to plot towards an ternary minimum at higher P_{H_2O} than the red granites. This does not support Heier and Brook's (1966) suggestion that the white granites were the result of vapour phase acting on the top portion of the granite intrusion, but need not vitiate the suggestion as Luth *et al.* (1964) found only a slight shift in the contours towards the ternary minima for higher P_{H_2O} for pegmatites and aplites, presumably formed from water-rich residual liquids, relative to granites.

The quartz-feldspar porphyries from Mt Bischoff and Pine Hill plot away from the isobaric temperature minima. They contain only minor normative Ab, in agreement with petrographic determination of predominantly a single alkali feldspar comprising high Or and very low Ab. Similar compositions in keratophyres have been attributed to potash metasomatism (*e.g.* Battey, 1955), and similar potash-rich alkali feldspars are recorded for adularia of 'alpine' vein paragenesis (*e.g.* Deer *et al.*, 1962, vol. 4, p. 67) and from authigenic potash feldspar (*e.g.* Baskin, 1956). It is evident that no conclusions can

be reached on the origin of these rocks on the basis of major element chemistry alone.

It may be significant that the elvans from south-west England which are also closely associated with tin mineralisation plot away from the ternary minima and are also Ab-deficient (fig. 58), and that porphyries and aplites from areas of tin mineralisation in north-east Tasmania are also Or-Rich and Ab-deficient (*e.g.* Blissett, 1959). It is possible that the formation of these rocks may be related to the formation of late quartz veins and associated mineralisation.

Trace Element Geochemistry

The results of trace element analyses are given in Tables 15-19 inclusive and the average trace element compositions for the various granitic types are given in Table 14. An investigation of K/Rb and Th/U ratios in the Heemskirk granite (Heier and Brooks, 1966) and analysis of several samples also from the Heemskirk granite by Rattigan (1964) represent the only previous studies of trace elements in granites from western Tasmania.

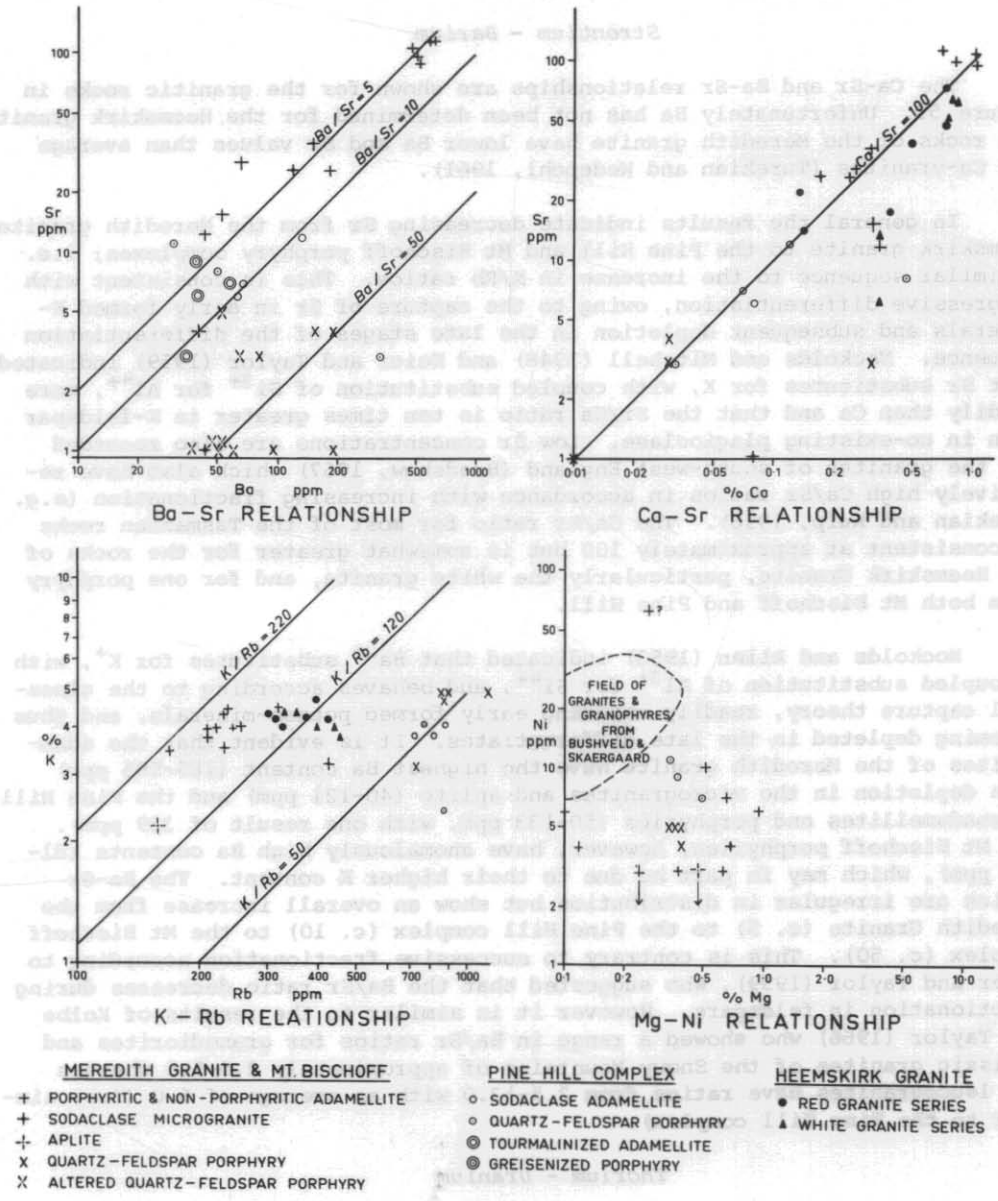
Kolbe and Taylor (1966) suggested that the relationship between Mg and Ni in granites may be an indication of their genesis. A plot of Mg against Ni (fig. 59) for the porphyries from Mt Bischoff contrasts with similar plots for granophyric rocks from the Skaergaard and Bushveld Complexes (Wager and Mitchell, 1951; Liebenberg, 1961). It is suggestive that the porphyries are not local differentiates of any penecontemporaneous basic magmas; this is supported by the low concentration of Cu, Ti and Mn.

Rubidium

The Rb values of the Meredith granite (average 235 ppm) are only slightly lower than values for low Ca-granites (Turekian and Wedepohl, 1961) but the other rocks show strong enrichment in Rb. The K/Rb ratio has been found to be a particularly useful index of geological processes (*e.g.* Taylor, 1965). It has been demonstrated that enrichment of Rb relative to K is commonly due to magmatic differentiation, and several authors have shown that the K/Rb ratio generally decreases with increasing differentiation (*e.g.* Taylor and Heier, 1958; Demin and Khitarov, 1958; Volkov and Savinova, 1959). Bowler (1959) and Bradshaw (1967) have demonstrated that the high level granites in south-west England have abnormally low K/Rb ratios.

A plot of K against Rb for the Tasmanian granitic rocks is shown in Figure 59. The adamellites of the Meredith granite have K/Rb ratios between 143 and 201 which are lower than the approximate overall average for most types of continental crust (Taylor, 1965). The microgranites and aplites show a slight enrichment in Rb with respect to K, relative to the adamellites. The rocks of the Heemskirk granite have K/Rb ratios ranging from 55 to 165 with an average around 120, indicating strong fractionation. The white granite has a lower average K/Rb ratio (106) than the red granite (142), consistent with a stronger fractionation. Similar ratios (53-208) have been recorded by Beattie (1967) for the granite at Royal George (eastern Tasmania) which is also a high level intrusion, with a tendency for lower K/Rb ratios in the microgranites. The ratio is also equivalent to the K/Rb ratio in the leucogranites of the Snowy Mountains granites (Kolbe and Taylor, 1966). The rocks of the Pine Hill and Mt Bischoff complexes have significantly lower K/Rb ratios averaging 51 and 56 respectively, indicating extremely strong fractionation.

The K/Rb ratios indicate that the granitic rocks are all strongly fractionated and are consistent with the granitic rocks being successive differentiates from a single or similar magmas, the least fractionated being the Meredith granite and the most fractionated the Pine Hill and Mt Bischoff complexes, with the Heemskirk Granite showing intermediate fractionation.



TRACE ELEMENT RELATIONSHIPS

GRANITIC ROCKS WEST TASMANIA

Figure 59.

The trend shown by plot of K against Rb is similar to that shown by Bradshaw (1967) for feldspars from British granites.

Strontium - Barium

The Ca-Sr and Ba-Sr relationships are shown for the granitic rocks in Figure 59. Unfortunately Ba has not been determined for the Heemskirk granite. The rocks of the Meredith granite have lower Ba and Sr values than average low Ca-granites (Turekian and Wedepohl, 1961).

In general the results indicate decreasing Sr from the Meredith granite, Heemskirk granite to the Pine Hill and Mt Bischoff porphyry complexes; i.e. a similar sequence to the increase in K/Rb ratios. This is consistent with progressive differentiation, owing to the capture of Sr in early formed K-minerals and subsequent depletion in the late stages of the differentiation sequence. Nockolds and Mitchell (1948) and Heier and Taylor (1959) indicated that Sr substitutes for K, with coupled substitution of Si^{3+} for Al^{4+}, more readily than Ca and that the Sr/Ca ratio is ten times greater in K-feldspar than in co-existing plagioclase. Low Sr concentrations are also recorded for the granites of south-west England (Bradshaw, 1967) which also have relatively high Ca/Sr ratios in accordance with increasing fractionation (*e.g.* Turekian and Kulp, 1956). The Ca/Sr ratio for most of the Tasmanian rocks is consistent at approximately 100 but is somewhat greater for the rocks of the Heemskirk Granite, particularly the white granite, and for one porphyry from both Mt Bischoff and Pine Hill.

Nockolds and Allen (1953) indicated that Ba^{2+} substitutes for K^{+}, with a coupled substitution of Al^{3+} for Si^{4+}, and behaves according to the classical capture theory, readily entering early formed potash-minerals, and thus becoming depleted in the late differentiates. It is evident that the adamellites of the Meredith granite have the highest Ba content (185-595 ppm) with depletion in the microgranites and aplite (40-121 ppm) and the Pine Hill microadamellites and porphyries (50-133 ppm, with one result of 329 ppm). The Mt Bischoff porphyries, however, have anomalously high Ba contents (81-192 ppm), which may in part be due to their higher K content. The Ba-Sr ratios are irregular in distribution but show an overall increase from the Meredith Granite (c. 5) to the Pine Hill complex (c. 10) to the Mt Bischoff complex (c. 50). This is contrary to successive fractionation according to Heier and Taylor (1959), who suggested that the Ba/Sr ratio decreases during fractionation in feldspars. However it is similar to the results of Kolbe and Taylor (1966) who showed a range in Ba/Sr ratios for granodiorites and gneissic granites of the Snowy Mountains of approximately 2.0-5.5 whereas the leucogranites have ratios from 2.5-13.0 with an average of 6.0 (*i.e.* similar to the Pine Hill complex).

Thorium - Uranium

The Th/U ratios determined by X-ray spectrography and those given by Heier and Brooks (1966) in which Th and U were determined by gamma ray spectrometry cannot be directly compared because of the tendency for Th to be low and U high, and hence Th/U to be low, in the X-ray analyses. However each set of analyses should show internal consistency.

The red granite of the Heemskirk granite has Th/U ratios between 3.1 and 7.6, which are typical of granitic rocks in general, while the white granite has much lower ratios between 0.9 and 2.5 and the aplites and microgranites have ratios between 0.4 and 1.9 (Heier and Brooks, 1966). The adamellites of the Meredith Granite have ratios between 1.6 and 6.0 while the microgranites and aplite have lower ratios between 0.5 and 1.6. The rocks

at Pine Hill and Mt Bischoff have much lower Th/U ratios between 0.6 and 2.7 and 0.6 and 1.0 respectively. Hence there is a trend to decreasing Th/U ratios from red granite to white granite and to aplite and microgranite in the Heemskirk granite and from Meredith granite (with lower ratios for aplite and microgranites than adamellites) to Pine Hill and to Mt Bischoff. These are similar trends to those shown by decreasing K/Rb ratios and Sr concentrations. There is no consistent trend for Th and U individually (Table 14).

It has been demonstrated that there is a general tendency for the Th/U ratio to increase with fractionation (*e.g.* Rogers and Raglands, 1961; Heier and Rogers, 1963) and this increase has been attributed to the oxidation of U. Heier and Brooks (1966) concluded that low Th/U ratios should appear (*a*) in strongly fractionated rocks that have formed under non-oxidising conditions and (*b*) in fractionated rocks in which the vapour phase is incorporated upon crystallisation. They favoured the latter conditions to explain the low Th/U ratios of the white granite and the aplites and microgranites at Heemskirk. This may also be the explanation of the low Th/U ratios at Mt Bischoff and Renison Bell where similar alteration and mineralisation has occurred to that shown by the white granite at Heemskirk.

Lithium

In the adamellites of the Meredith granite, Li averages some 44 ppm, which is about one and a half times the average abundance of Li in granitic rocks (Turekian and Wedepohl, 1961). In the microgranite the concentration is 48 ppm and in the aplite 225 ppm. In the rocks of the Pine Hill complex it varies from 24-299 ppm and is considerably enriched in the rock containing abundant mica. The Mt Bischoff porphyries are enriched in Li, the range being 45-111 ppm, although three of the four analyses are above 100 ppm. Taylor (1965) concluded that concentrations of Li greater than 100 ppm in granites are indicative of extreme fractionation and imply that the rock is a late stage, high level product. Bowden (1966) recorded high Li contents in tin-bearing biotite-granites from northern Nigeria and suggested that the high values were introduced by late magmatic and post-magmatic processes. At Mt Bischoff and Pine Hill the porphyries with high Li values are not strongly greisenised and the high Li contents are probably indicative of strong fractionation.

Copper, Zinc and Lead

Copper values in all granitic rocks analysed are remarkably consistent, without exception below 10 ppm, and generally ranging between 4 and 6 ppm. These concentrations are slightly below the average abundance of Cu in granites (Taylor, 1965). Bradshaw (1967) noted no significant increase in Cu in the granites of south-west England relative to non-mineralised granites. Zinc values are rather variable and are generally higher in the porphyries at Mt Bischoff and Pine Hill. The average value for the adamellites of the Meredith granite is 47 ppm which is approximately the average abundance in granites. The distribution of Zn values is difficult to discuss because knowledge of the geochemical distribution is unknown. Tauson and Kravchenko (1956) indicated that it occurs mainly as minute grains of sphalerite with minor substitution in biotites and feldspars. Lead has been determined only in a limited number of analyses because of the interference of As in the determination by X-ray fluorescent spectroscopy. The Pb occurs at a relatively constant concentration slightly greater than the average abundance in granites. The Pb-Sr ratio increases markedly from the adamellites of the Meredith Granite (average 0.3) to the Pine Hill porphyries (average 3.8) and the Mt Bischoff porphyries (average 10.7). Taylor (1965) indicated that such an increase is consistent with fractionation of a magma. The distribution of Pb, like Zn,

is uncertain due to the possibility of formation of the sulphide phase (Tauson and Kravchenko, 1956).

Tin

The distribution of Sn is of primary importance from an economic viewpoint. It has generally been found that granites enclosing tin deposits have greater Sn concentrations than unmineralised granites (*e.g.* Butler, 1953; Goldschmidt, 1954; Shibata *et al.*, 1960; Bradshaw, 1967). The average values of Sn concentration in silica rocks are between 3 and 4 ppm (Onishi and Sandell, 1957; Hamaguchi *et al.*, 1964) and values up to 800 ppm have been recorded for greisens (Goldschmidt, 1954). Ivanova (1963) found that the Sn content of biotite granites enclosing tin deposits (16-32 ppm) is higher than that for other granites (<5 ppm) and that the Sn concentration (60-110 ppm) increases with increasing late or post-magmatic alteration. The association of Sn with biotite is well known, and most Sn occurring in the major silicates occurs in biotite (*e.g.* Rattigan, 1964).

The granitic rocks of the Meredith Granite have low concentrations of Sn, averaging 4 ppm and ranging from 3-13 ppm. The concentrations are approximately equivalent to the average concentration of Sn in silicic rocks. Three analyses of biotites from the Meredith Granite indicate a range of Sn contents of 22-40 ppm (Table 15) and indicate that most of the Sn in the rocks must be present in the biotite. No Sn analyses are available for the total rock samples of the Heemskirk Granite although three analyses of biotites from the white granite show a higher range of Sn contents (31-118 ppm) than three samples from the red granite (9-34 ppm). Tin values are higher for unaltered rocks from Pine Hill (14-104 ppm, average 40 ppm) and Mt Bischoff (30-114 ppm, average 72 ppm). It is significant that the granitic rocks containing Sn mineralisation (Pine Hill, Mt Bischoff and the white granite, Heemskirk) show enrichment in Sn while the relatively unmineralised and less fractionated rocks (Meredith Granite and red granite, Heemskirk) have relatively low concentrations of Sn.

Summary

It is apparent that the granitic rocks show a sequence of increasing chemical fractionation which is parallel to their probable relative level of emplacement. The Heemskirk Granite which contains significant primary tin mineralisation has higher Si, K and Sn and lower Na, Mg, K/Rb and Sr than the unmineralised Meredith Granite. The porphyries of Mt Bischoff and granitic rocks of Pine Hill have high Si, Li and Sn and low K/Rb and Sr, but differ from most tin-bearing granitic rocks in their low total alkali content and their very high K/Na ratios (*e.g.* Klomínský and Groves, 1970). Their origin is difficult to determine because they appear from several of their chemical characteristics to have suffered metasomatism. However there is no convincing textural evidence to confirm this in the specimens analysed, although the majority of the adjacent rocks are strongly metasomatised.

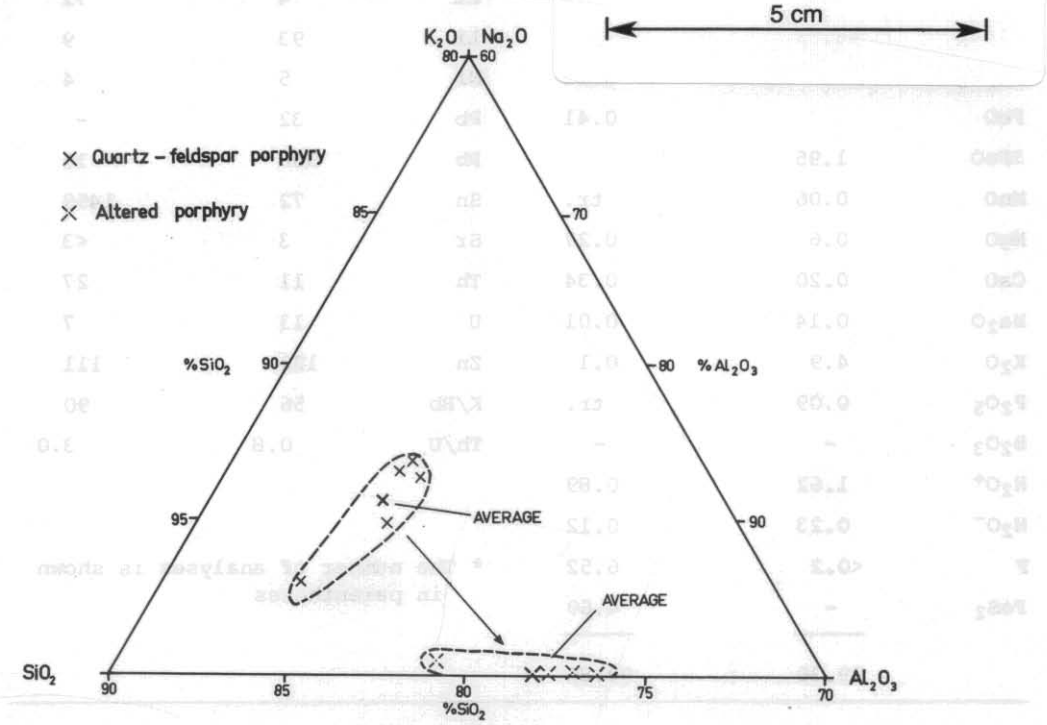
There is insufficient data available in this study on the distribution of Sn in the rocks and between minerals to add any significant contribution to the many problems of the geochemical behaviour of Sn in granitic rocks *(e.g. Flinter, 1970)*.

Geochemistry of Alteration at Mt Bischoff

There has been little systematic chemical investigation of the processes of late or post-magmatic alteration of granitic rocks. The increase of elements of economic significance such as Sn, W and Mo during greisenisation is well established (*e.g.* Ivanova, 1963) but the behaviour of other elements

has not been studied in detail.

The alteration of quartz-feldspar porphyries at Mt Bischoff involves a relative decrease in alkalies (particularly K_2O) and SiO_2 and an increase in Al_2O_3 (fig. 60) and a marked increase in F and FeS_2 (table 20). This is consistent with the petrographic evidence of replacement of orthoclase by topaz and sulphides. A decrease is shown by MgO and a small increase by CaO possibly due to introduction of calcite.



COMPOSITION OF UNALTERED AND ALTERED GRANITIC ROCKS MT BISCHOFF

Figure 60.

The altered porphyries show a sharp decrease in Rb and Ba, the average values decreasing from 906-11 ppm and 149-50 ppm respectively. Strontium show a possible decrease although values in the unaltered porphyries are already close to the detection limit. This decrease is due to original incorporation of Rb, Ba and Sr in feldspars which have been completely replaced, these elements being removed with the K. Selective removal of Rb relative to K is indicated by an increase in the K/Rb ratio from 56 to 90 with alteration. The retention of about one third of the Ba is difficult to explain since it substitutes only for K among the common cations. It is possible that the Ba may be fixed in minute proportions of barite although none has been seen in thin section. Rankama and Sahama (1950) suggested that Ba was leached from granites during greisenisation or alteration. Solomon (1966) found that a greisen from the Skiddaw Granite (Cumberland, England) had less than half the Ba content of adjacent granite while altered granite in the Rookhope borehole (*e.g.* Dunham *et al.*, 1965) showed enrichment in Ba relative

Table 20. AVERAGE COMPOSITIONS OF UNALTERED AND ALTERED GRANITIC ROCKS, MT BISCHOFF AND PINE HILL

	Quartz-feldspar Porphyry (5)* %	Altered Porphyry (5)* %		Quartz-feldspar Porphyry (5)* *ppm*	Altered Porphyry (5)* *ppm*
SiO_2	75.3	69.4	Ba	149	50
TiO_2	0.02	0.04	Cu	4	71
Al_2O_3	14.25	19.65	Li	93	9
Fe_2O_3		0.28	Ni	5	4
FeO		0.41	Pb	32	-
≡FeO	1.95		Rb	906	11
MnO	0.06	tr.	Sn	72	1459
MgO	0.6	0.27	Sr	3	<3
CaO	0.20	0.34	Th	11	27
Na_2O	0.14	0.01	U	13	7
K_2O	4.9	0.1	Zn	122	111
P_2O_5	0.09	tr.	K/Rb	56	90
B_2O_3	-	-	Th/U	0.8	3.0
H_2O^+	1.62	0.89			
H_2O^-	0.23	0.12			
F	<0.2	6.52	* The number of analyses is shown in parentheses		
FeS_2	-	4.69			
	99.36	99.86			

to the unaltered granite. It is probable that variations in Ba content reflect the overall mineralogical changes involved in alteration and the initial Ba content of ore fluids. Tin shows the expected marked increase during alteration from an average of 72 ppm in the unaltered porphyries to 1459 ppm in the altered porphyries, due to the formation of free cassiterite. Zinc and Ni show negligible changes during alteration but Cu shows a marked increase from an average of 4 ppm to 71 ppm, probably due to the formation of minor chalcopyrite. The increase in Th from an average of 11 to 27 ppm and the Th/U ratio from 0.8 to 3.0 suggests introduction of Th, and concentration of U relative to Th in the volatile phase present during alteration (*e.g.* Heier and Brooks, 1966), with subsequent depletion in the porphyries.

The fluids producing the alteration of dykes at Mt Bischoff have apparently introduced Al, F, Fe and S and minor proportions of Ca, Sn, Cu and Th and have become enriched in alkalies (particularly K) and Si with minor Mg, Rb, Ba and Li by reaction with the porphyries. The unusually high K/Na and Li/Na ratios of fluid inclusions from fluorite in the replacement orebody are probably partly the result of this reaction.

SUMMARY

The tin ores at Mt Bischoff occur as replacement deposits in dolomite, as fissure lodes, and as replacement deposits and joint fillings in porphyry dykes. The main replacement deposit occurs in a dolomite horizon at the top of an Upper Proterozoic-Lower Cambrian miogeosynclinal sequence of sandstones, shales and mudstones, in which the sand-grade rocks are subgreywackes and protoquartzite. Similar deposits at Renison Bell occur at the same stratigraphical level. This sequence is followed by a eugeosynclinal sequence consisting of greywackes, mudstones, cherts, spilitic lavas and mafic intrusions. This sequence contains calcareous horizons which have been mineralised at Cleveland and Mt Lindsay.

The deposits at Mt Bischoff occur mainly on the southern limb of a major anticlinal structure which is complicated by extensive faulting and flexural folding. Other more complicated structures have been suggested but there is no compelling evidence to support them. Numerous topazised and tourmalinised quartz-feldspar porphyry dykes of Upper Devonian age have intruded this structure and the tin mineralisation is restricted to the area of highest dyke frequency.

The cassiterite-sulphide deposits occur in a larger district of predominantly Pb-Zn-Ag mineralisation in fissure veins which are also considered to be genetically related to the Upper Devonian granite activity (*e.g.* Solomon, 1965). The spatial association of cassiterite-sulphide deposits with Pb-Zn-Ab deposits is typical of many Sn-rich provinces (*e.g.* Bilibin, 1955). A uniform distribution of low Co and Ni in Fe-sulphides, normal Cd in sphalerites, normal Se in sulphides, and high Mn in the Pb-Zn-Ag fissure veins and cassiterite-sulphide deposits supports the concept of a single metallogenic province in the late Devonian.

The stratiform replacement cassiterite-sulphide deposits at Mt Bischoff consist predominantly of pyrrhotite with pyrite, arsenopyrite, chalcopyrite, sphalerite and cassiterite, and minor marcasite, stannite, galena, jamesonite, hematite, tetrahedrite, wolframite and boulangerite. In all the deposits cassiterite and wolframite have generally crystallised before the earliest formed sulphides which are arsenopyrite and minor pyrite, followed by pyrrhotite and further pyrite with minor marcasite. Chalcopyrite, sphalerite and stannite were formed penecontemporaneously and exhibit extensive solid solution. They were followed by tetrahedrite and jamesonite and the final major sulphide to crystallise was galena. Some supergene alteration of pyrrhotite to pyrite and marcasite has occurred.

Pre-sulphide crystallisation of tourmaline, quartz, wollastonite, chondrodite, garnet and corundum indicates metasomatism associated with intrusion of the porphyry dykes, with introduction of F, B, SiO_2 and minor H_2O at temperatures in excess of 400°C. These minerals have become unstable during subsequent activity involving sulphide deposition, and have been generally replaced by hydrous micaceous minerals including talc, fluor-phlogopite, serpentine, and chlorite, and by Fe-Mn-Mg carbonates and fluorite. The presence of talc indicates a temperature probably slightly in excess of 400°C. Introduction of Fe, Mn, K, SiO_2 and base metals with S, F and H_2O and removal of Ca, CO_2 and to a lesser extent Mg, has been widespread.

The cassiterite-sulphide deposits are considered to have been deposited at an initially higher temperature than the surrounding Pb-Zn-Ag deposits because the earliest formed minerals (*e.g.* cassiterite, wolframite, arsenopyrite, tourmaline, topaz) are virtually restricted to these deposits. The Pb-Zn-Ag deposits contain minerals of intermediate or late position in paragenesis (*e.g.* sphalerite, galena, jamesonite).

Mineralogical lateral zoning can be demonstrated on a local scale. It is apparent the lateral zoning is generally more common than vertical zoning in tin deposits (*e.g.* Turneaure, 1960; Hosking, 1965). The replacement deposit in the central area of the mine consists of massive pyrrhotite with cassiterite, arsenopyrite, pyrite and wolframite, while further south pyrite, sphalerite, galena and jamesonite become more abundant. Similar mineralogical zonation is evident from the general distribution of the later fissure veins,, and some zonation within the veins has been recorded. Zoning may also be related to time of mineralisation as there are also major differences between the replacement deposit and the later veins.

That this zonation is at least partly a function of declining temperature is indicated by declining formation temperatures of fluorite and quartz (from fluid inclusions studies) outwards from the centre of Mt Bischoff. Initial temperatures of deposition of fluorite in the central area were as high as 580°C and decreased to approximately 200°C in the marginal zone some 600 m away. Temperatures of formation of fluorite and quartz in the marginal fissure veins range from 170° to 380°C, with the majority between 200° and 250°C.

A positive correlation exists between salinity and temperatures of formation for minerals from different localities. A possible mechanism to explain this correlation, the great variation of salinity within one area, the temperature decline and high salinities (50% equiv. NaCl) of inclusions from the central area is for initially hot, highly saline hydrothermal solutions, derived from the granitic source, to be mixed with cooler, less saline meteoric and connate waters, in conjunction with heat loss from the solutions to the wall rocks.

Low Na/K and Na/Li ratios of the fluids in fluorite from the central area are probably a function of addition of excess K and Li over Na from hydrothermal alteration of the porphyry dykes. The Na/K ratio of the ore solutions increased with declining temperature and salinity possibly due to admixing with relatively Na-Rich meteoric and connate waters; this increase may have been accentuated by fixation of K in micaceous minerals (*e.g.* phlogopite). A possible local source of solutions is indicated for deposition of fluorite at Fook's Lode (a marginal cassiterite-sulphide fissure deposit) which has a relatively high temperature of formation and salinity but similar Na/K ratios to the other marginal fissure deposits.

The fluid inclusion studies indicate that the initial ore-forming medium was predominantly a supercritical fluid.

There is limited evidence that pyrrhotites were formed in equilibrium with pyrite, but the compositions of pyrrhotites are broadly compatible with temperatures of formation derived from fluid inclusion studies if the assumption of pyrrhotite/pyrite equilibrium is made. Sphalerites in the ores probably achieved only local surface equilibrium with pyrrhotite and pyrite, as they display a wide range of FeS contents with only a few values approaching the experimental values determined by Chernyshev *et al.* (1968) at the temperatures indicated by fluid inclusion studies. The FeS content of sphalerites does not correlate simply with temperature of formation, and variations are broadly compatible with variations in the activity of iron in the different ore environments.

The $\delta^{34}S$ of Fe-sulphides from the porphyry dykes and highest temperature zone of the replacement deposit is close to zero, and increases towards the marginal zone of the replacement body and in the fissure lodes. This trend is not that expected from continual re-equilibration along a temperature gradient, and may be related to isotopic fractionation resulting from kinetic

rate effects or to high-temperature equilibrium with the initial sulphides forming from a species rich in ^{32}S.

The variation of Co and Ni in pyrite and pyrrhotite from Mt Bischoff does not simply parallel the zonal variation shown by sulphide mineralogy. However there are significant differences between Co and Ni contents of Fe-sulphides from the replacement deposit and the fissure veins, the latter being enriched in both Co and Ni, with preferential enrichment in Ni.

It appears that a combination of factors relating to the environment of deposition (probably including temperature) is more important in determining variations in sulphide compositions than the variation in temperature alone. Other important factors probably include availability of elements, activity of elements and nature of ore fluids at the depositional site and the chemistry of host rocks and penecontemporaneously deposited minerals at that site.

The mechanism of ore-metal transport and the pH of the ore-forming fluids cannot be defined by the present study. The occurrence of abundant talc and carbonates in the ore horizon suggests that the fluids must have been alkaline at the depositional stage (*e.g.* Stringham, 1952). Several authors (*e.g.* Krauskopf, 1967) have summarised the possible mechanisms of ore-metal transport and have suggested that large departures from neutrality are unlikely because of the buffering effects of the wall rocks. Most authors favour an initially slightly acid fluid which transports the ore-metals as complex ions and molecules, this resulting in an increase in metal solubility. Of these mechanisms the transport of the ore metals as chloride complexes (*e.g.* Helgeson, 1964) is attractive as chlorides are known to be abundant (from fluid inclusion studies) and deposition in the carbonate horizon can be explained as the complexes become unstable with increasing alkalinity, thus causing precipitation of the ore minerals; the sequence of attainment of instability approximates the paragenetic sequence of most sulphide ores including the cassiterite-sulphide deposits examined in this study.

The granitic rocks associated with tin mineralisation in western Tasmania are typical post-kinematic, high-level plutons. It is probable that the presently exposed area of the Heemskirk Granite is in closer proximity to the roof area of the batholith than the exposed area of the Meredith Granite. The complex at Pine Hill appears to be the roof of a small cupola and may be a relatively small upward projection of a larger granite stock. The dyke system at Mt Bischoff probably represents filling of tensional fractures above a hidden cupola-like body such as that envisaged at Pine Hill.

The similar radiometric ages of the granitic rocks and similar trace element concentrations of sulphides in the granitic rocks and associated ore deposits suggests that the granitic rocks could be inter-related. It is envisaged that each granite, as exposed, corresponds to a different level within one or more penecontemporaneous, compositionally similar batholiths. The granitic rocks are all highly fractionated, with high SiO_2 contents. The composition of the adamellites of the Meredith and Heemskirk Granites and the adamellites of the Pine Hill complex indicate that they originated by crystal-liquid equilibrium. Variation in major element chemistry of these rocks is not inconsistent with successive fractionation from a single or similar magma. The successive decrease in K/Rb ratios and Sr contents of the adamellites from the Meredith Granite to the Heemskirk Granite and to the Pine Hill complex also supports this concept of successive fractionation.

The compositions of porphyries from Mt Bischoff (and Pine Hill) are unusual. They are extremely Na-deficient but not markedly K-enriched, partly

because of incomplete kaolinisation of the feldspars. They are chemically similar to the elvans of south-west England and to aplites and porphyries from north-eastern Tasmania, both being spatially associated with tin mineralisation. Their crystallisation history is uncertain. However high SiO_2 and Li contents, and very low K/Rb ratios and Sr contents are probably indicative of extreme fractionation.

It is significant that the granitic rocks containing tin mineralisation (Pine Hill, Mt Bischoff, and the white granite, Heemskirk) show an enrichment in Sn compared to the relatively unmineralised and less fractionated rocks (Meredith Granite, and red granite, Heemskirk).

The granitic rocks adjacent to the cassiterite-sulphide deposits at Mt Bischoff are extensively altered and in places mineralised. The feldspars have been replaced by topaz, tourmaline, muscovite and sulphides. Both pre-joint and post-joint alteration have occurred although the post-joint phase is minor. At Mt Bischoff it is probable that volatiles were introduced along open brecciated walls of the dykes while the porphyries were in a semi-solidified state. The alteration involves a relative decrease in alkalies and SiO_2 and an increase in Al_2O_3, with a marked increase in F, Fe and S. The behaviour of trace elements is dependent on mineralogical changes as well as on bulk chemical changes during alteration.

SECTION 5:
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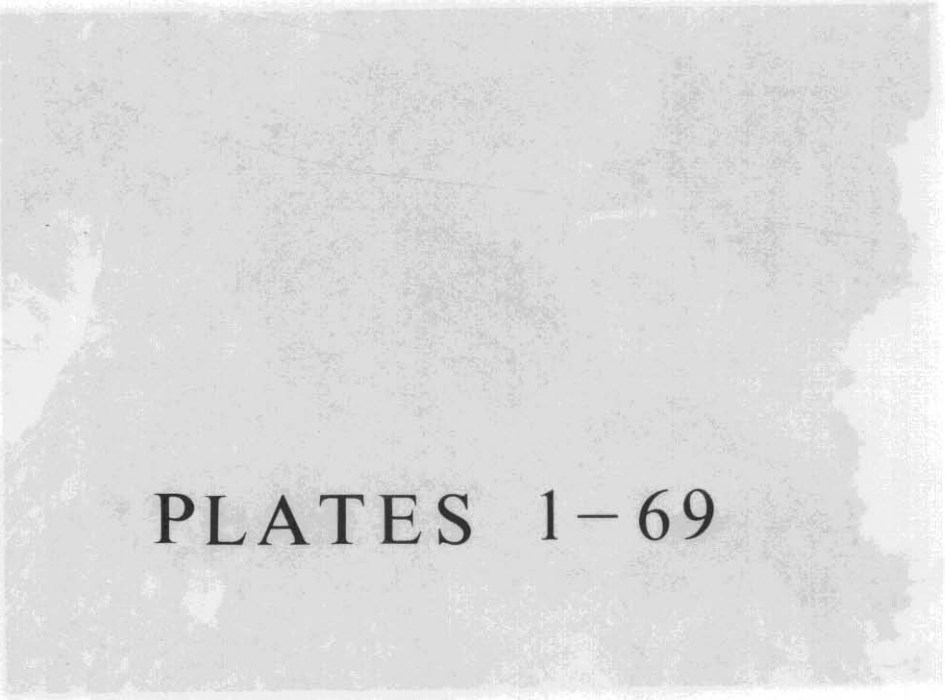
PLATES 1–69



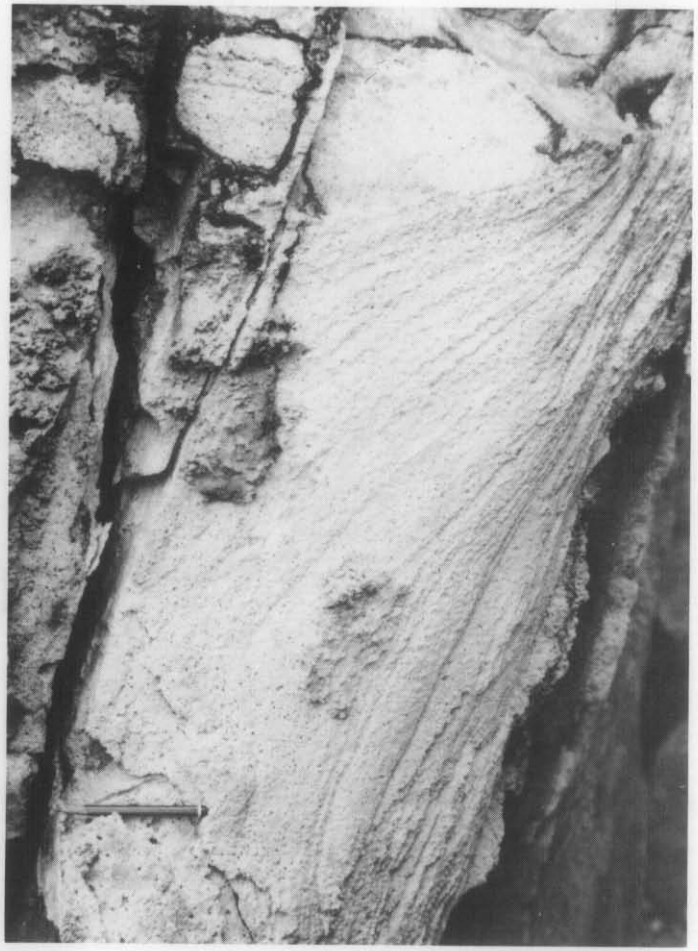
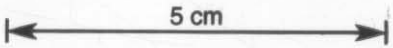


Plate 1. *Flow banding in quartz-feldspar porphyry, White Face Dyke, Mt Bischoff.*

Plate 2. *Steeply dipping Tertiary gravels, sands and silts, Don Hill, Mt Bischoff.*



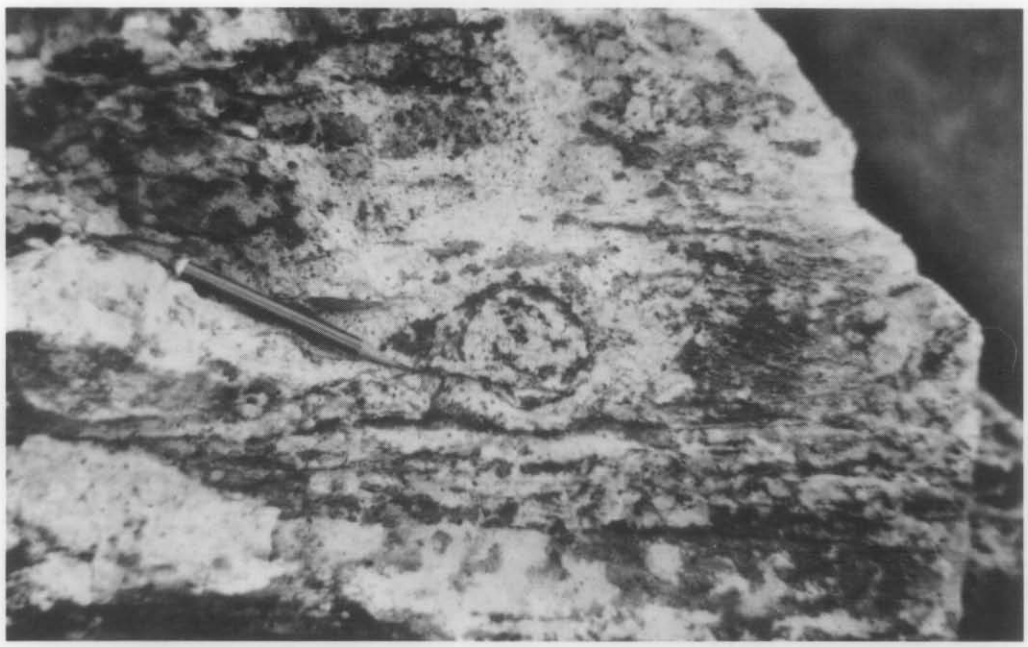


Plate 3. *Small swirl in altered quartz-feldspar porphyry, White Face Dyke, Mt Bischoff.*

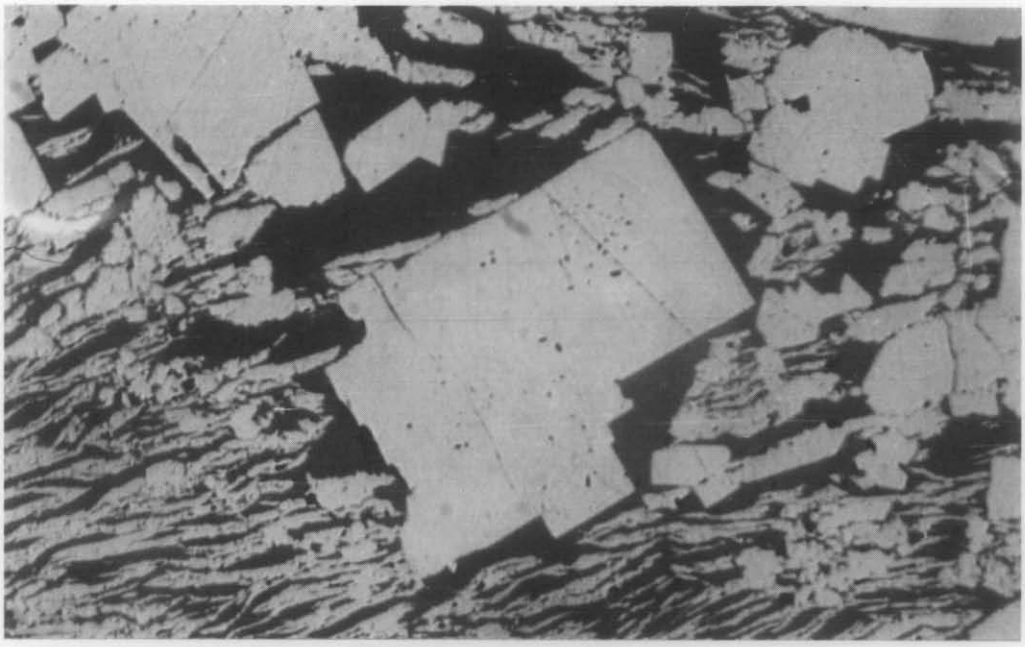


Plate 4. *Recrystallisation of second generation pyrite, and segregation of pyrite (white) and carbonate (dark grey). From replacement deposit, Mt Bischoff. Specimen No.* 100,140. x210.

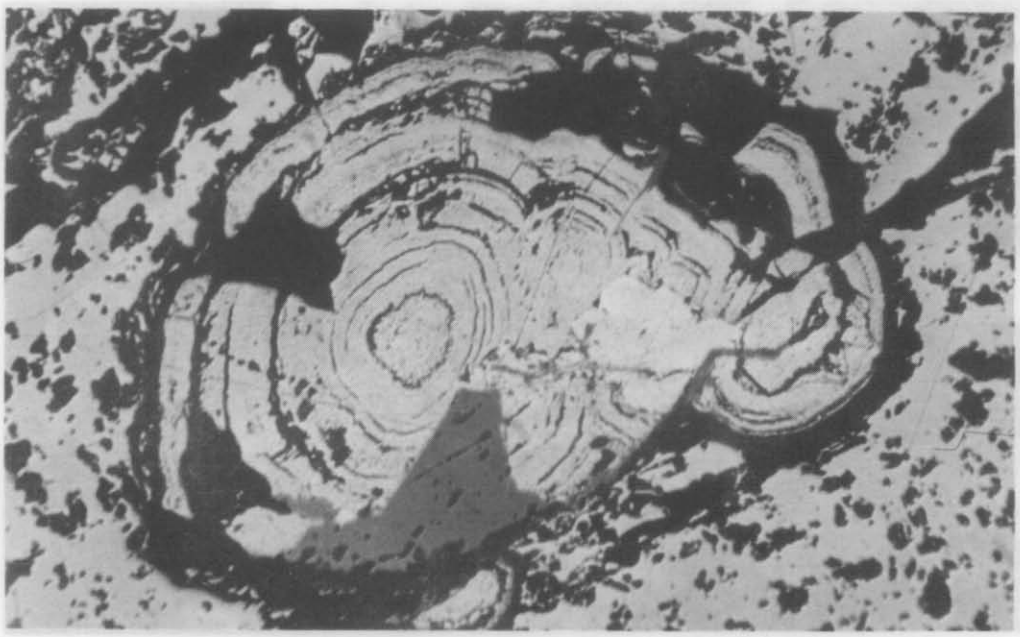


Plate 5. *Microbotryoidal supergene pyrite and marcasite forming around fracture in pyrrhotite. From replacement deposit, Mt Bischoff. Specimen No.* 100,147. x105.

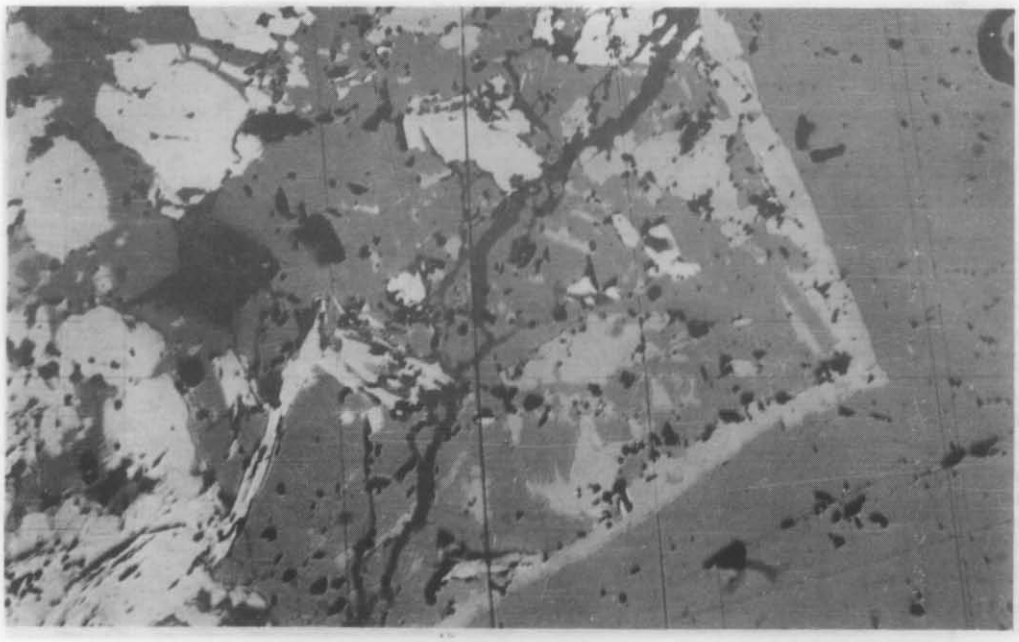


Plate 6. *Stannite (light grey) along a sphalerite (dark grey) grain boundary with other irregular blebs of stannite. From replacement deposit, Mt Bischoff. Specimen No.* 100,133b. x105.

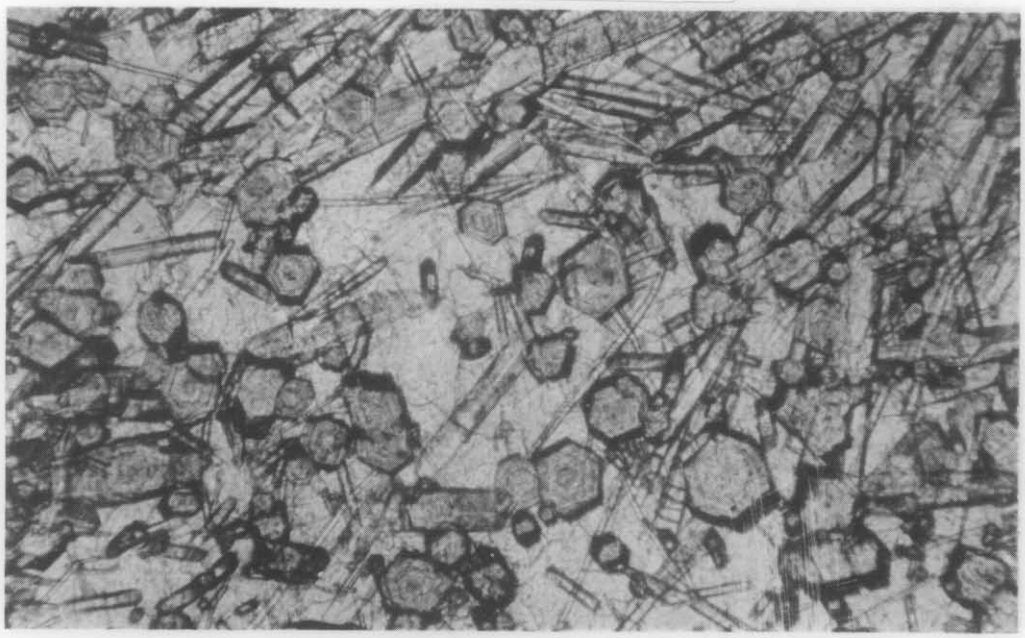


Plate 7. *Zoned tourmaline crystals in large fluorite crystal from contact of quartz-feldspar porphyry and dolomite. Southern end of open cut, Mt Bischoff. Specimen No.* 30,634D. x105.

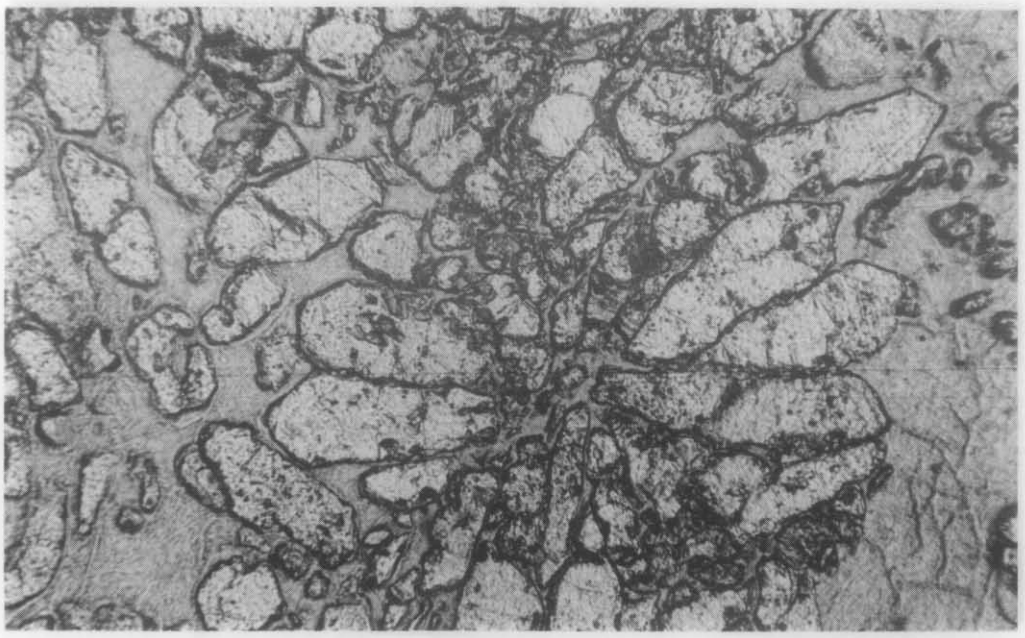


Plate 8. *Chondrodite replaced by serpentine. Replacement deposit, Brown Face, Mt Bischoff. Specimen No.* 33,453B. x77.

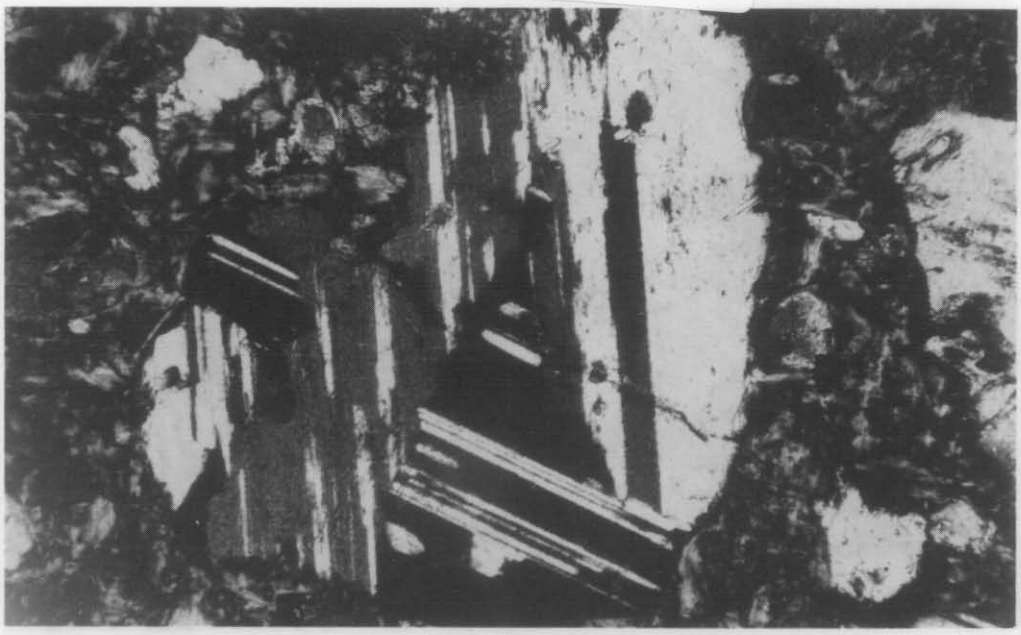


Plate 9. *Complex twinning in chondrodite crystals surrounded by serpentine. Replacement deposit, Brown Face, Mt Bischoff. Specimen No.* 33,453. *Crossed nicols,* x132.

Plate 10. *Columnar wollastonite replaced by pyrrhotite and talc. Replacement deposit, Mt Bischoff. Specimen No.* 1535. x35.

Plate 11. *Replacement of orthoclase phenocryst by carbonate, quartz and topaz. Southern end of White Face Dyke, Mt Bischoff. Specimen No.* 1496. x44.

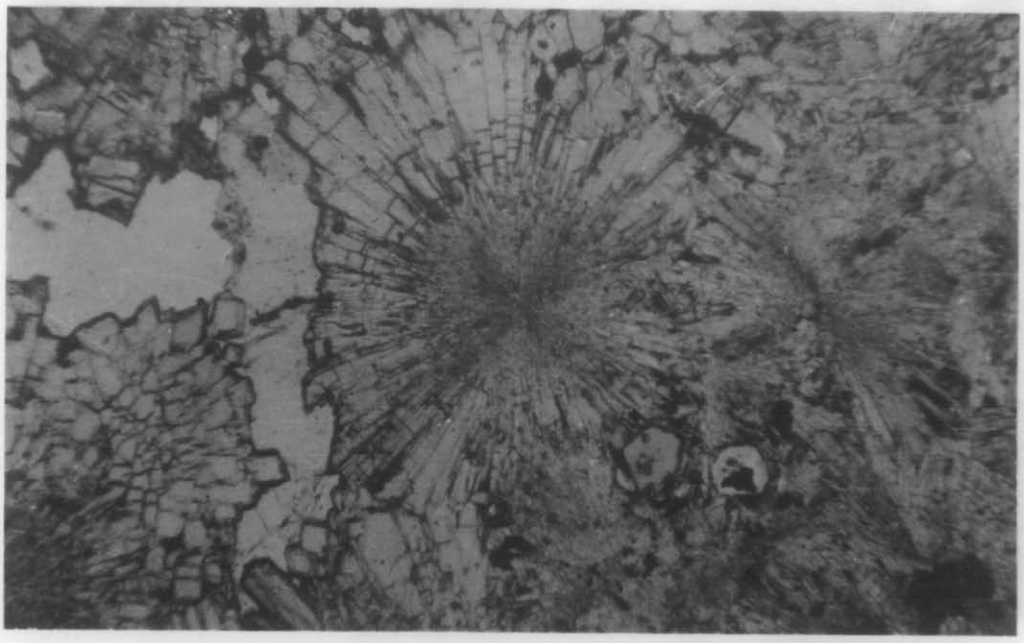


Plate 12. *Radial aggregate of columnar topaz with interstitial quartz in extensively altered porphyry. White Face Dyke, White Face, Mt Bischoff. Specimen No.* 2171. x44.

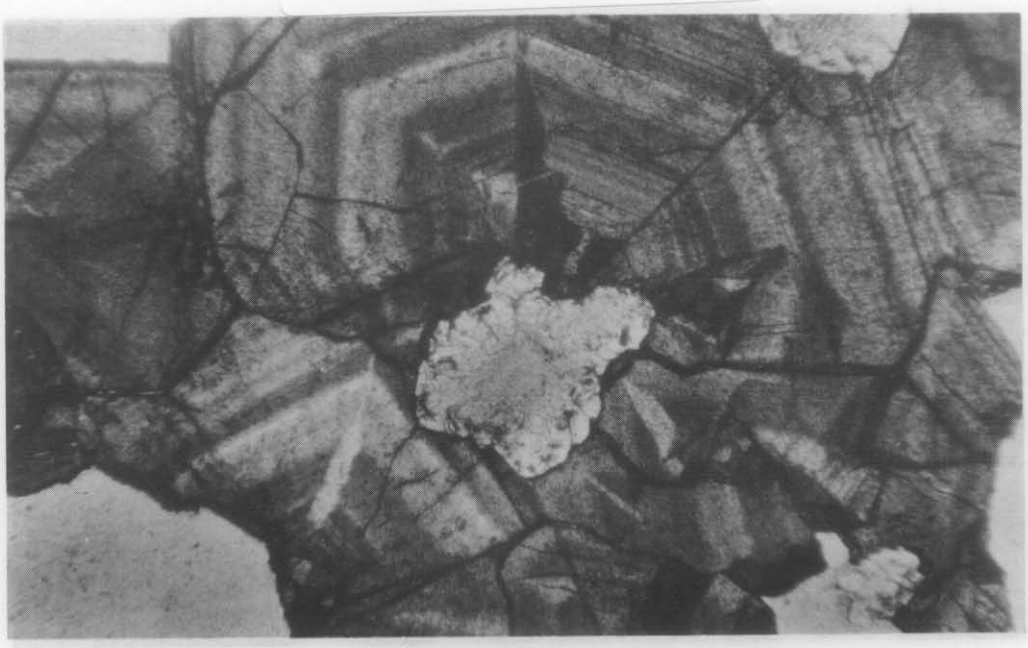


Plate 13. *Euhedral zoned cassiterite crystals with topaz in extensively altered porphyry. White Face Dyke, White Face, Mt Bischoff. Specimen No.* 2171. x44.

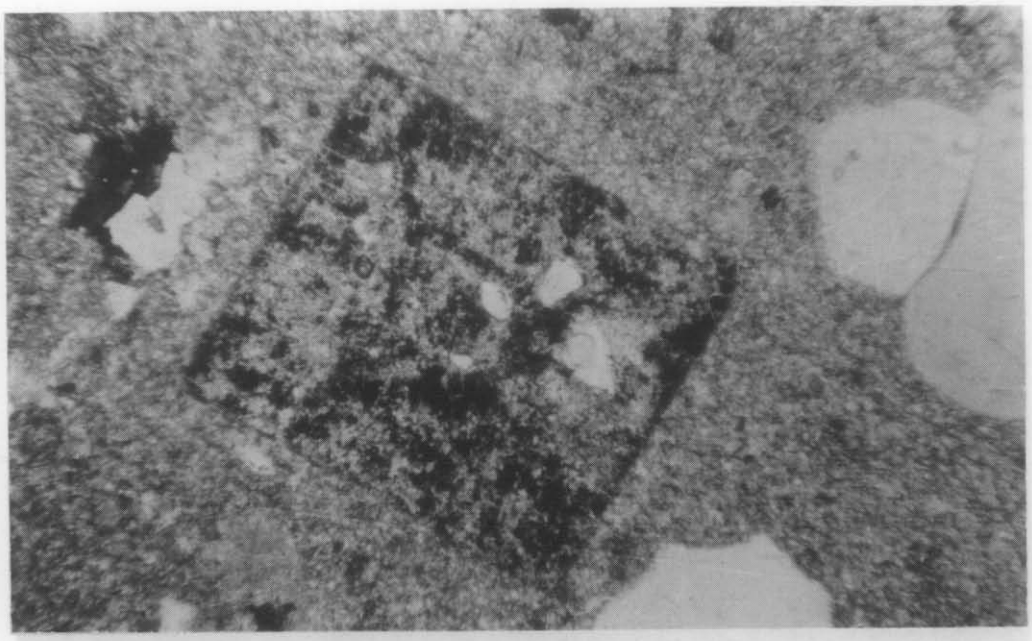


Plate 14. *Orthoclase phenocrysts in quartz-feldspar porphyry. White Face Dyke, Mt Bischoff. Specimen No.* 1388. x44.

Plate 15. *Oblique aerial view of Mt Bischoff looking NW, c. 1960.* [Rio Tinto]

Plate 16. *Main Street, Waratah and mine tramway, looking towards Mt Bischoff, 1911.*

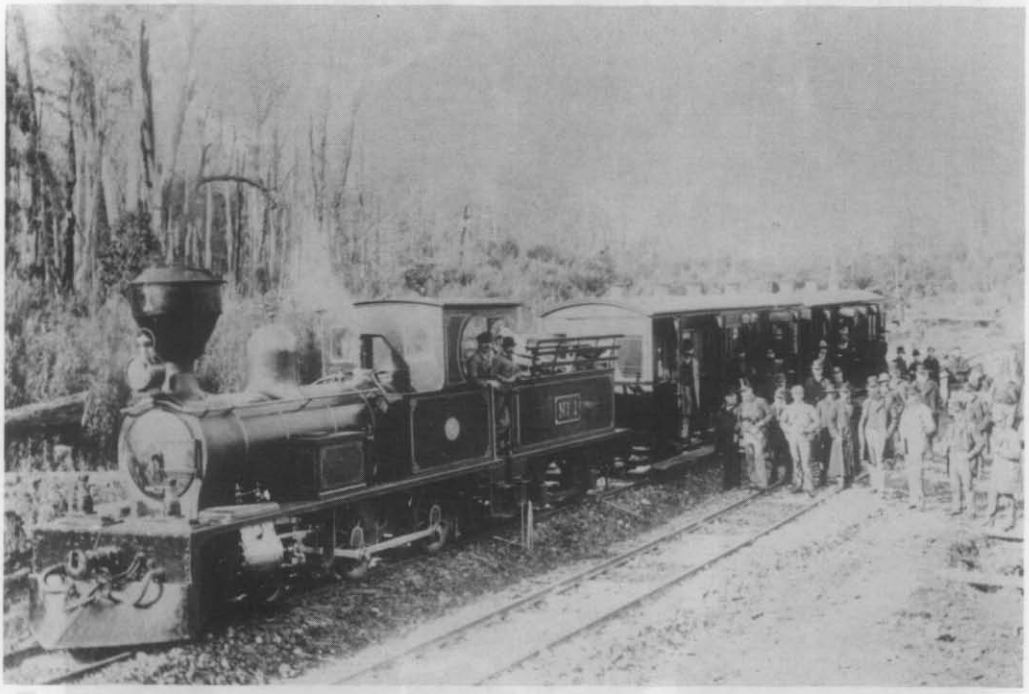


Plate 17. *Emu Bay and Mt Bischoff Railway. Locomotive No. 1 on a trial run, July 1884.* [State Libr.]

Plate 18. *An early view from the Brown Face, c. 1878.* [Winter]

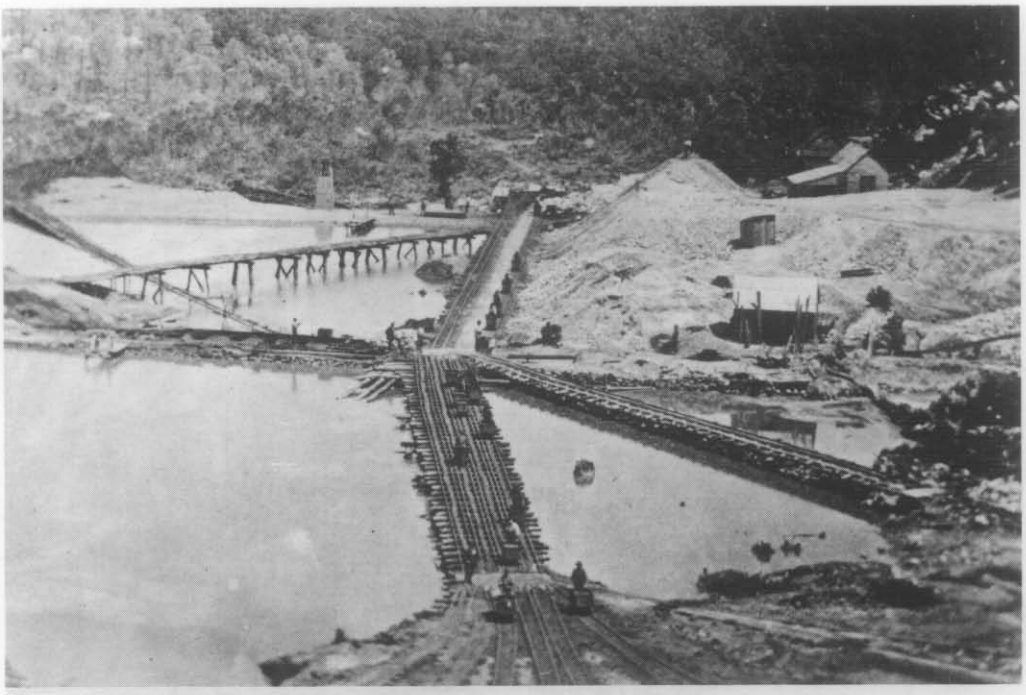


Plate 19. *View from the Brown Face, after the completion of the upper dam, 1880.* [Q.V.Mus.]

Plate 20. *Panorama from Brown Face looking*

Plate 21. *Panorama: Slaughteryard Face (left),*

towards Waratah, c. 1880. [Tasm.Mus.]

and Brown Face (right). [Q.V.Mus.]

Plate 22. *Brown Face workings, c. 1900. The ore was broken by picks and shovelled into the ore trucks.* [State Libr.]

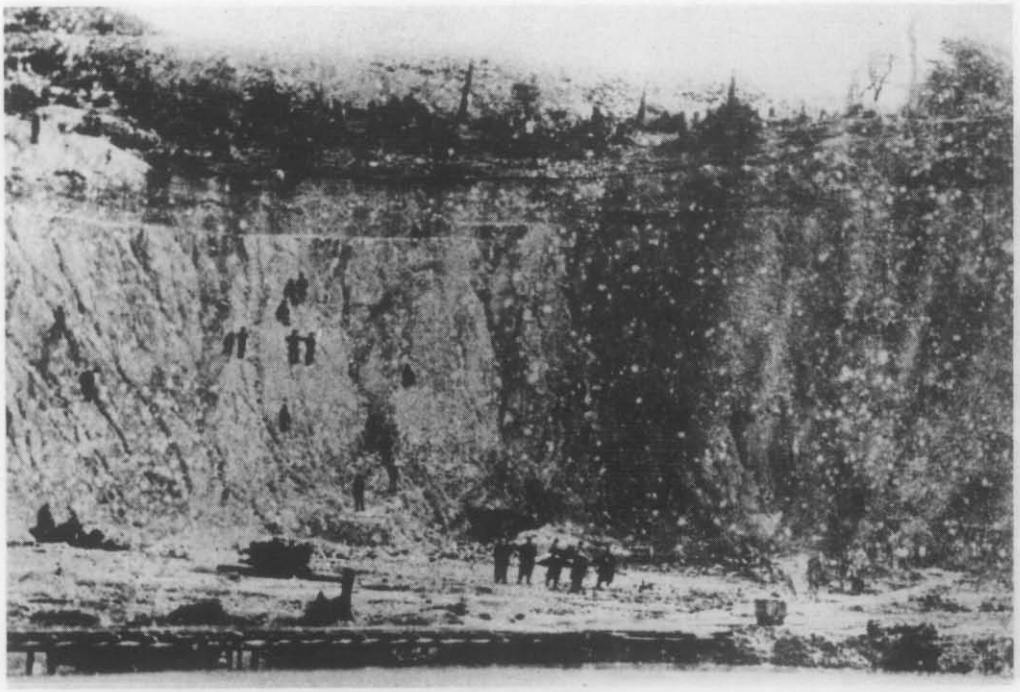


Plate 23. *Brown Face workings, c. 1880. The men, supported by ropes, broke the ore from the face.* [State Libr.]

Plate 24. *Brown Face, 1911.*

Plate 25. *Slaughteryard Face, 1880.* [Q.V.Mus.]

Plate 26. *Self acting tramway, c. 1880.* [State Libr.]

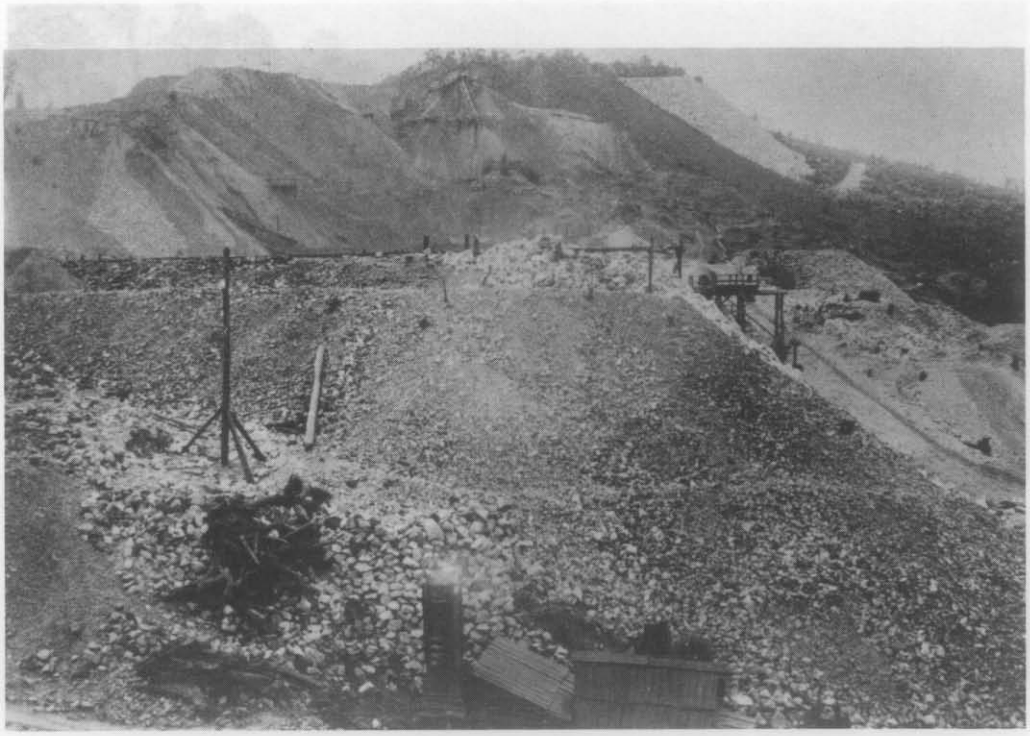


Plate 27. *White Face workings, c. 1880.* [Tasm.Mus.]

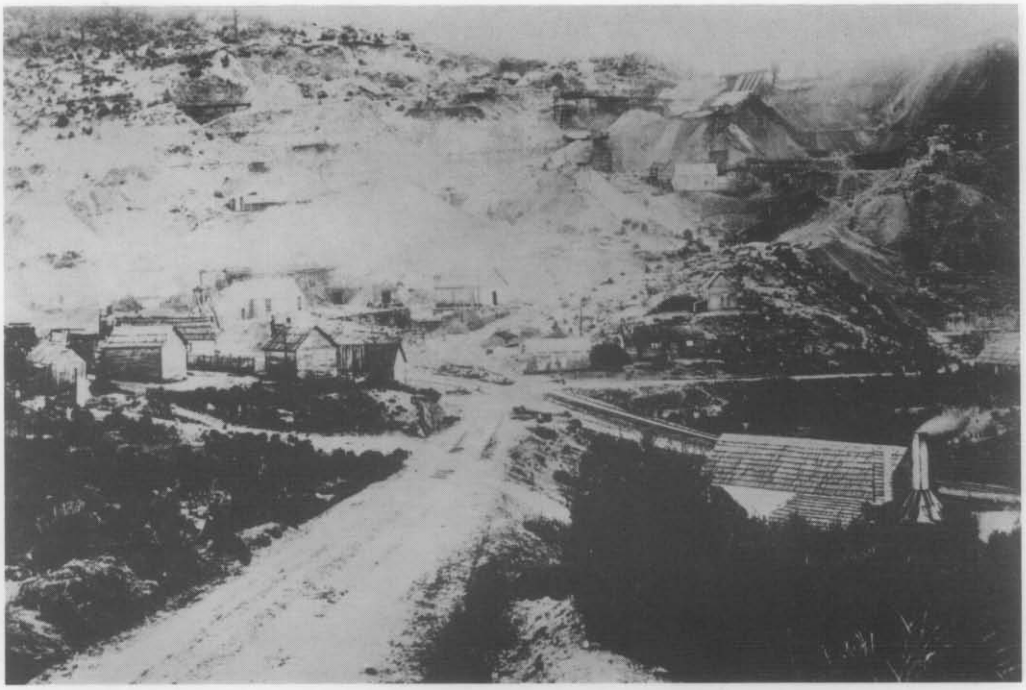


Plate 28. *White Face, c. 1880.* [Winter]

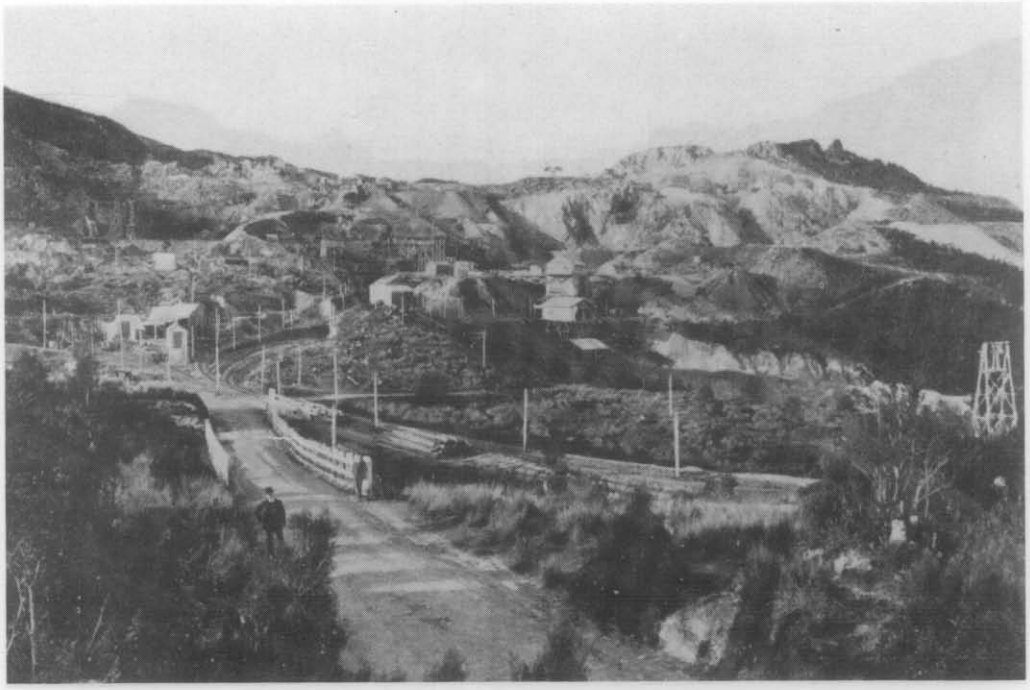


Plate 29. *Gossan Face and White Face.* [Herman, 1914]

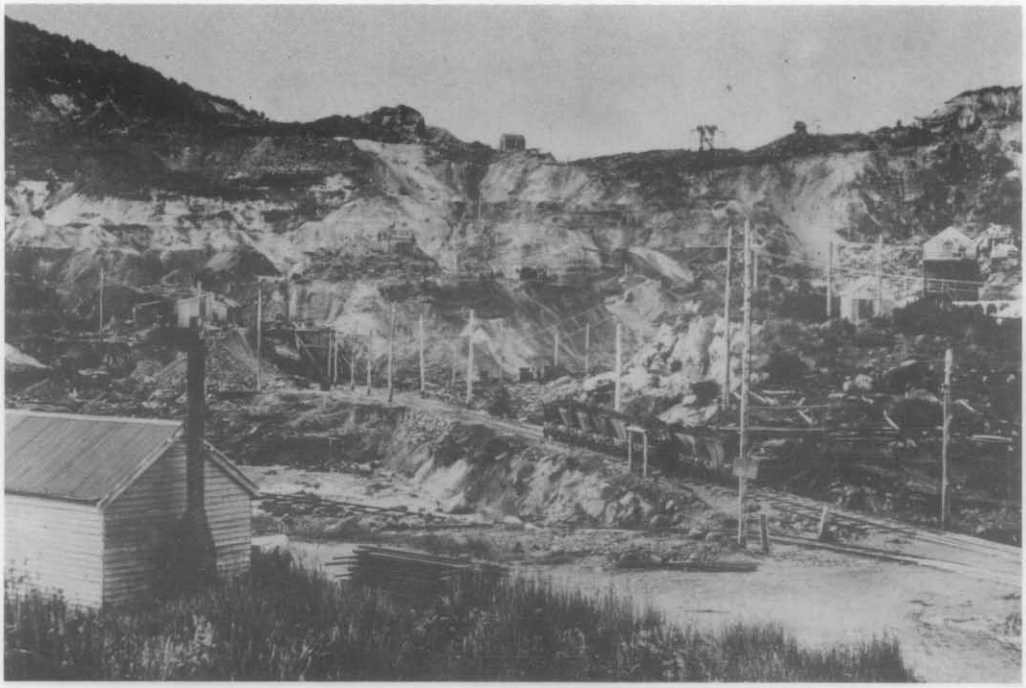


Plate 30. *White Face, c. 1915.* [J. Robinson]

Plate 31. *White Face, 1972.* [E.L. Martin]

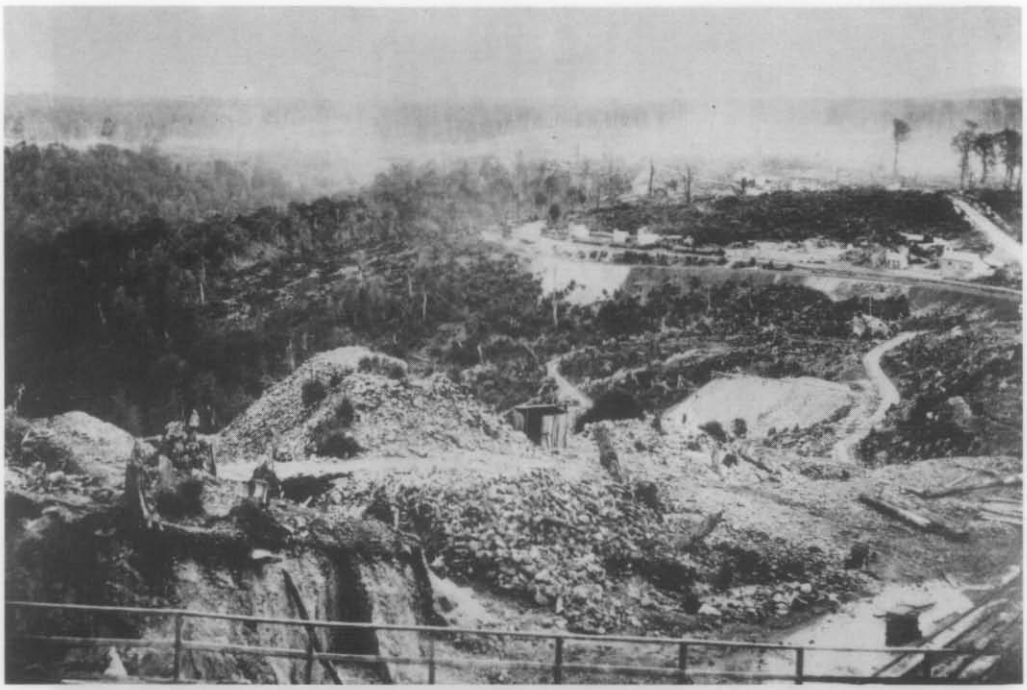


Plate 32. *View from White Face, towards Waratah, c. 1885.* [Q.V.Mus.]

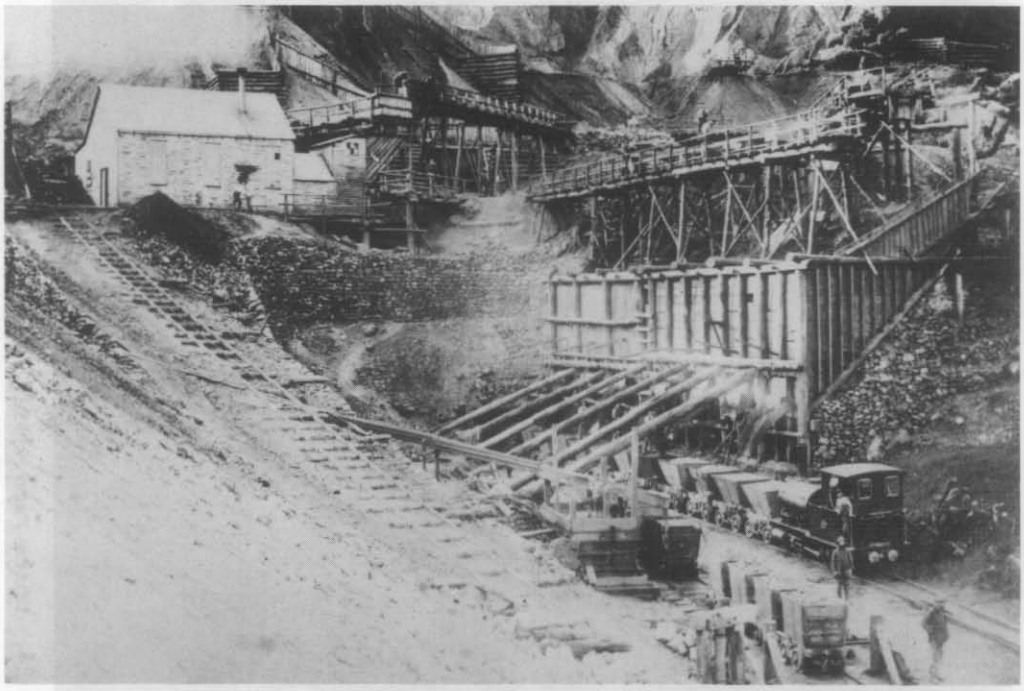


Plate 33. *White Face loading station, c. 1885.* [Q.V.Mus.]

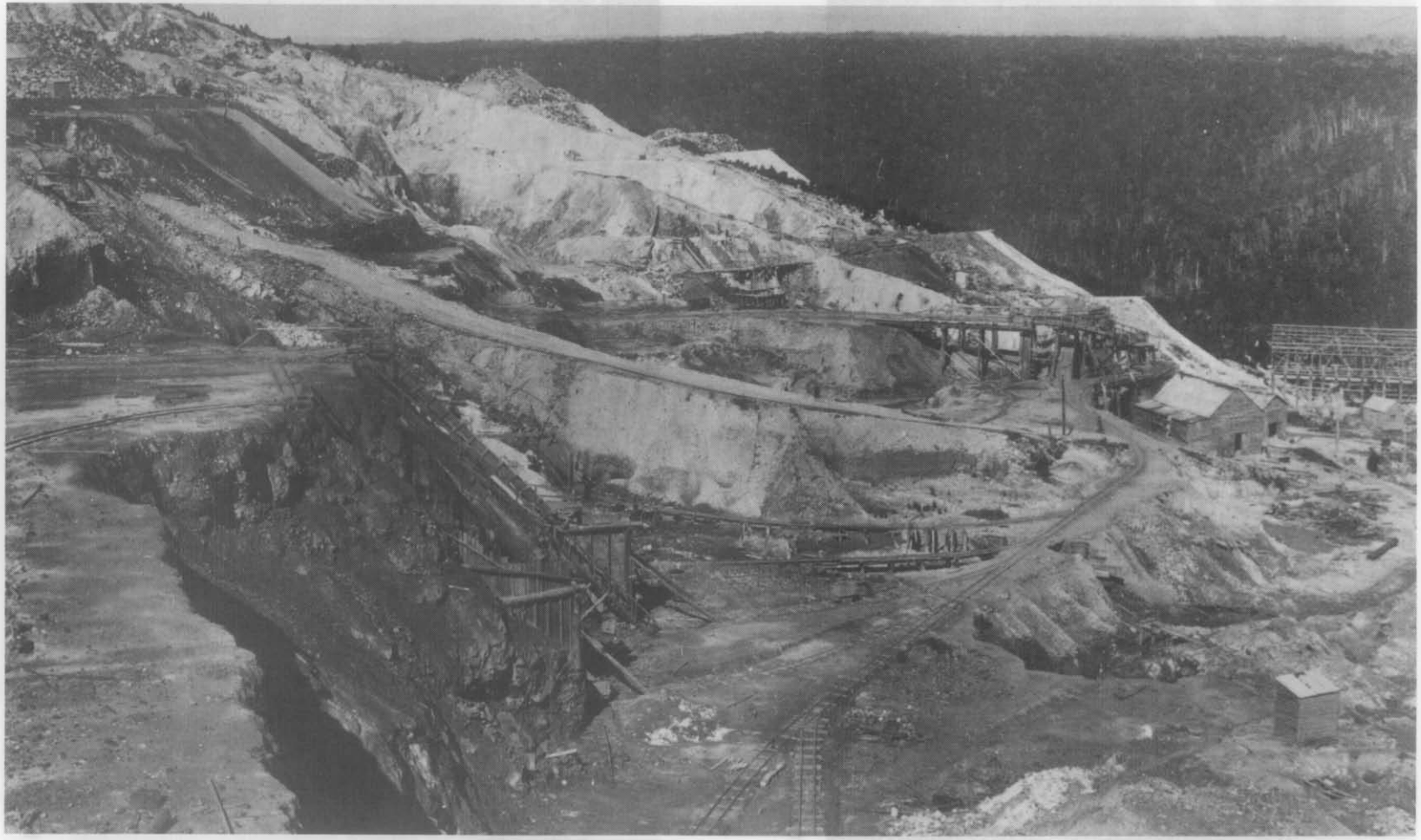


Plate 34. *White Face, 1911.*

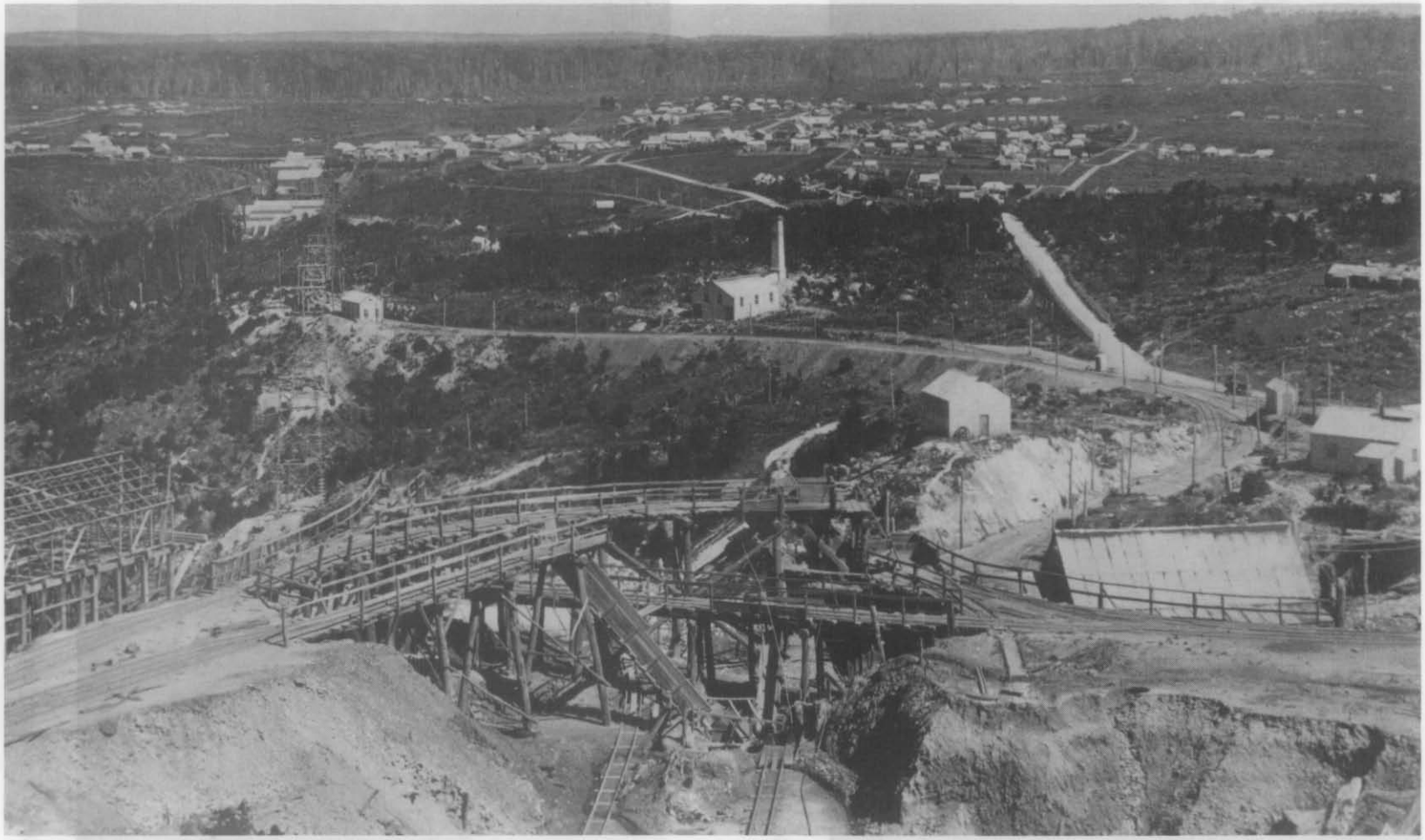


Plate 35. *View from White Face towards Waratah, showing the aerial ropeway under construction, 1911.*

Plate 36. *Bench in White Face.* [Herman, 1914].

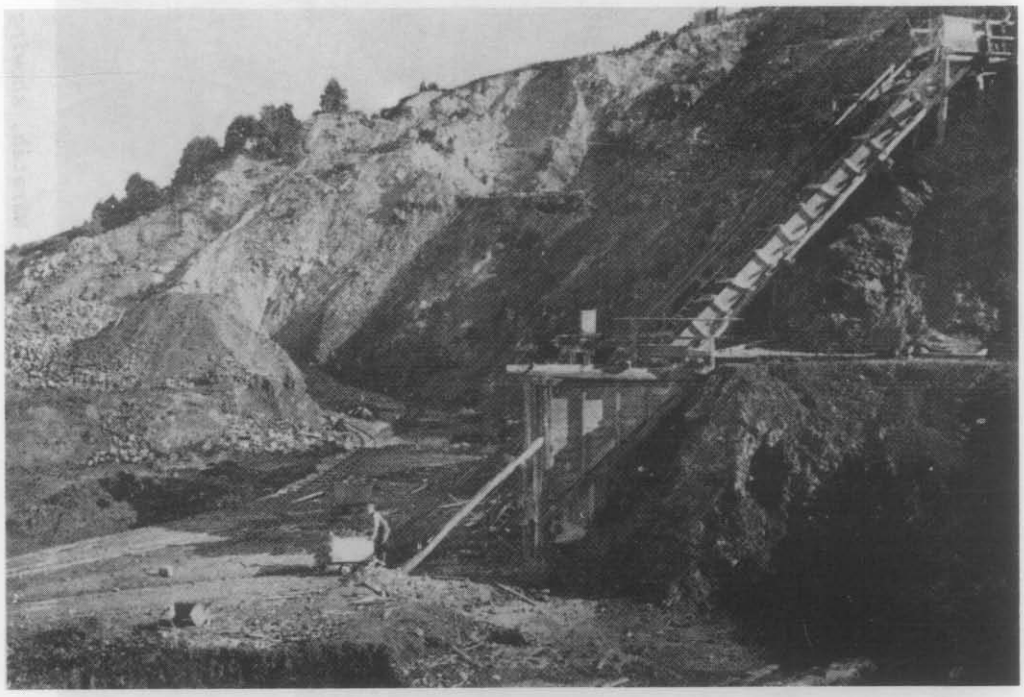


Plate 37. *Bench in Gossan Face.* [Herman, 1914]

Plate 38. *Tin sluice, Happy Valley, 1911.*

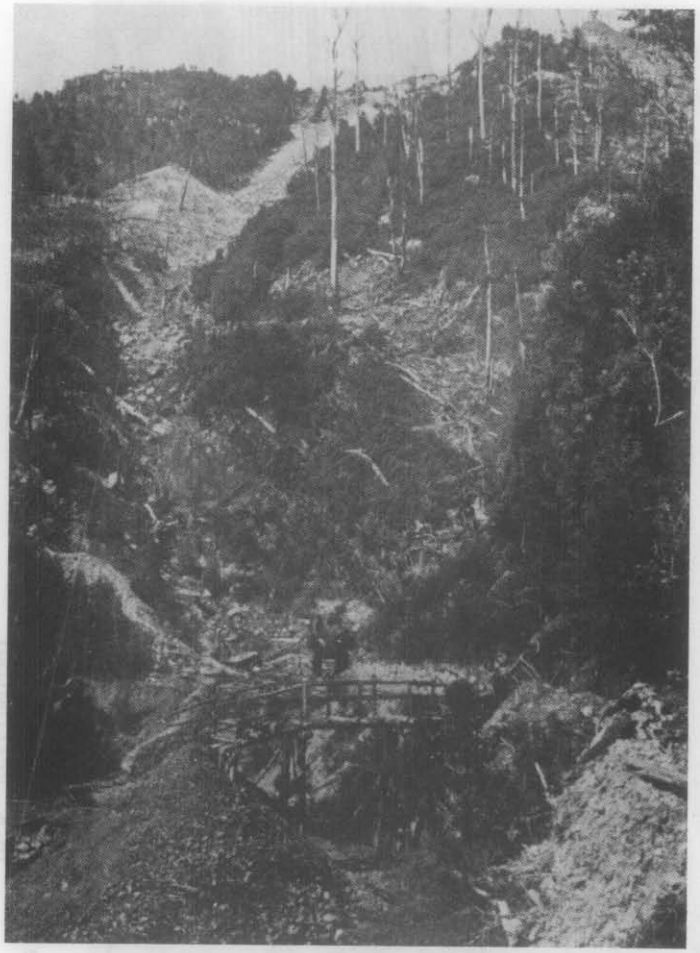


Plate 39. *Northern slopes.* [Herman, 1914]

Plate 40. *North Valley, G. Machen's sluicing operations, 1970.* [H.K. Wellington]

Plate 41. *North Valley, G. Machen's sluice, 1970.* [H.K. Wellington]

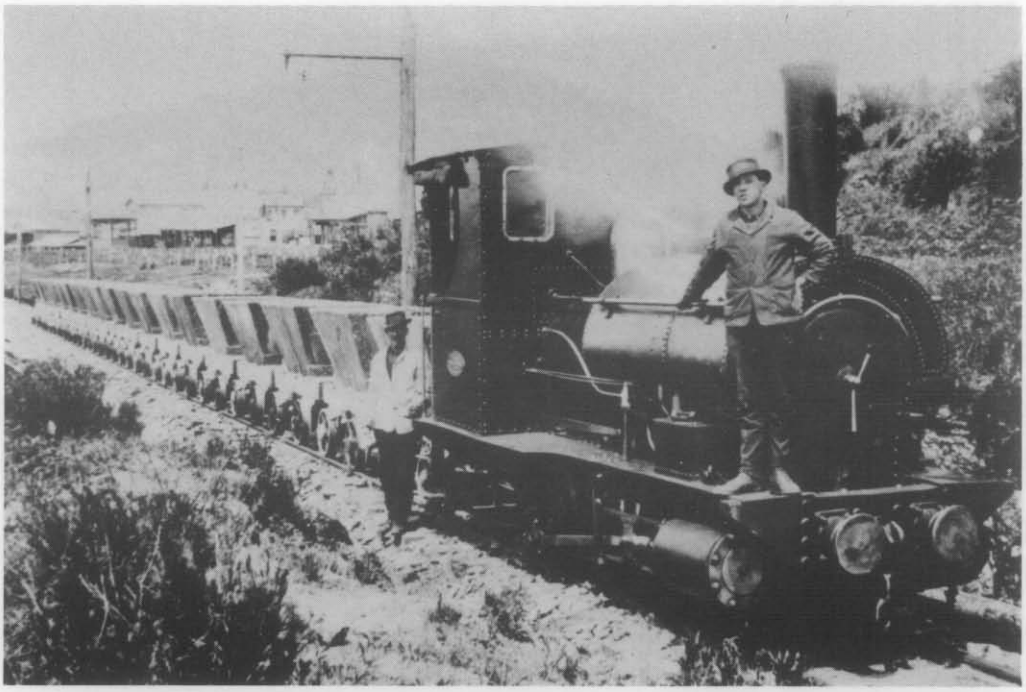


Plate 42. *Mine tramway. Steam locomotive and rake of empty ore trucks, Waratah, c. 1907.* [Winter]

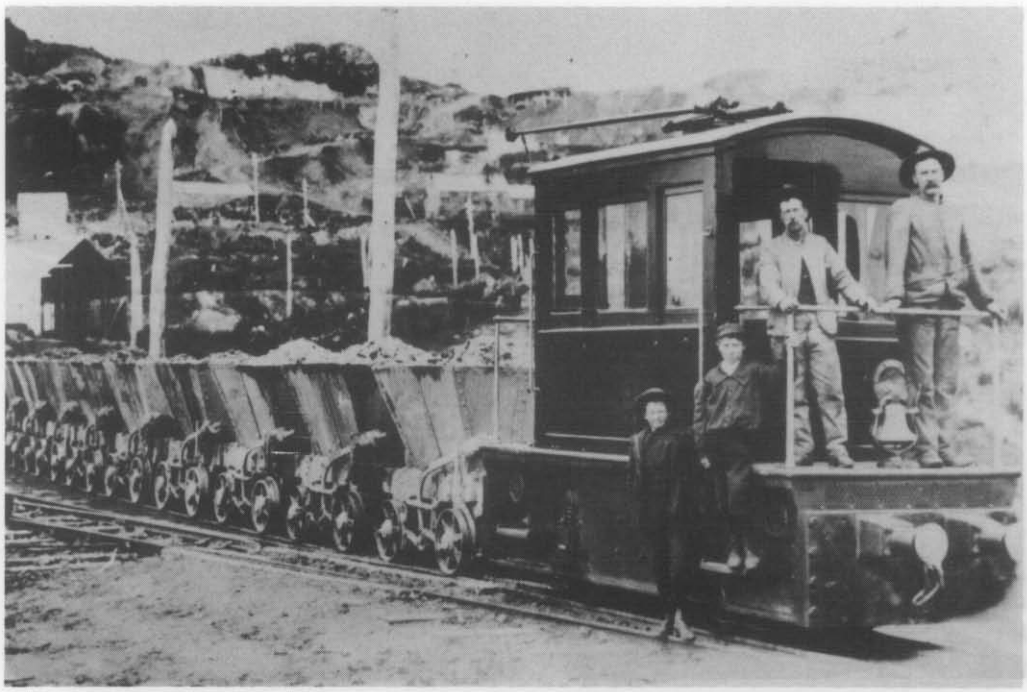


Plate 43. *Mine tramway. Electric locomotive and rake of ore trucks near White Face.* [Winter]

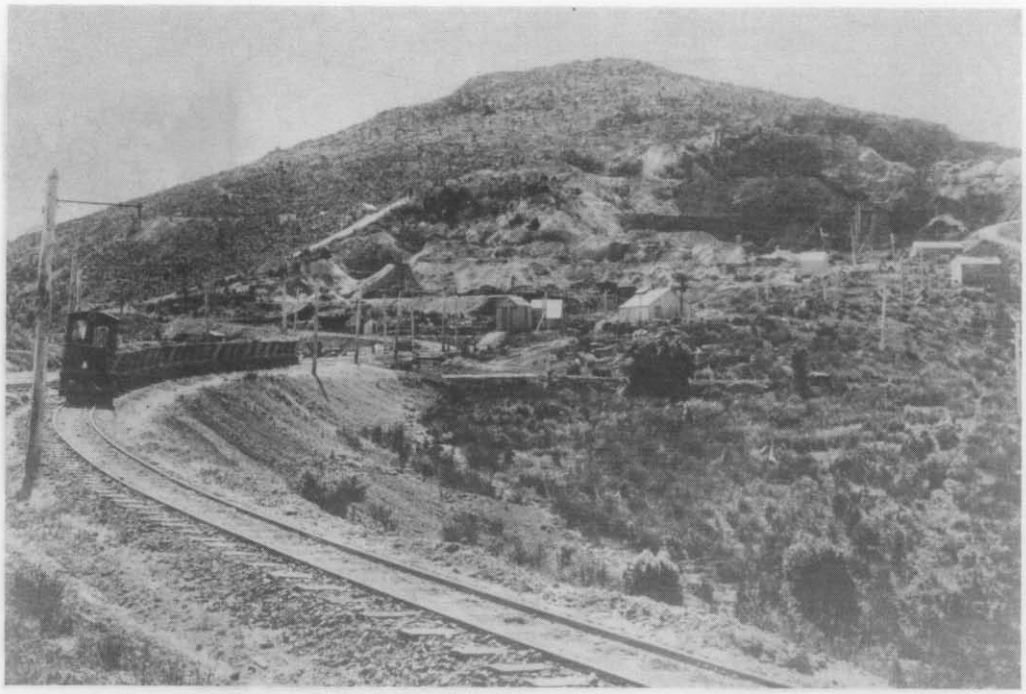


Plate 44. *Ore train near Gossan Face, 1911.*

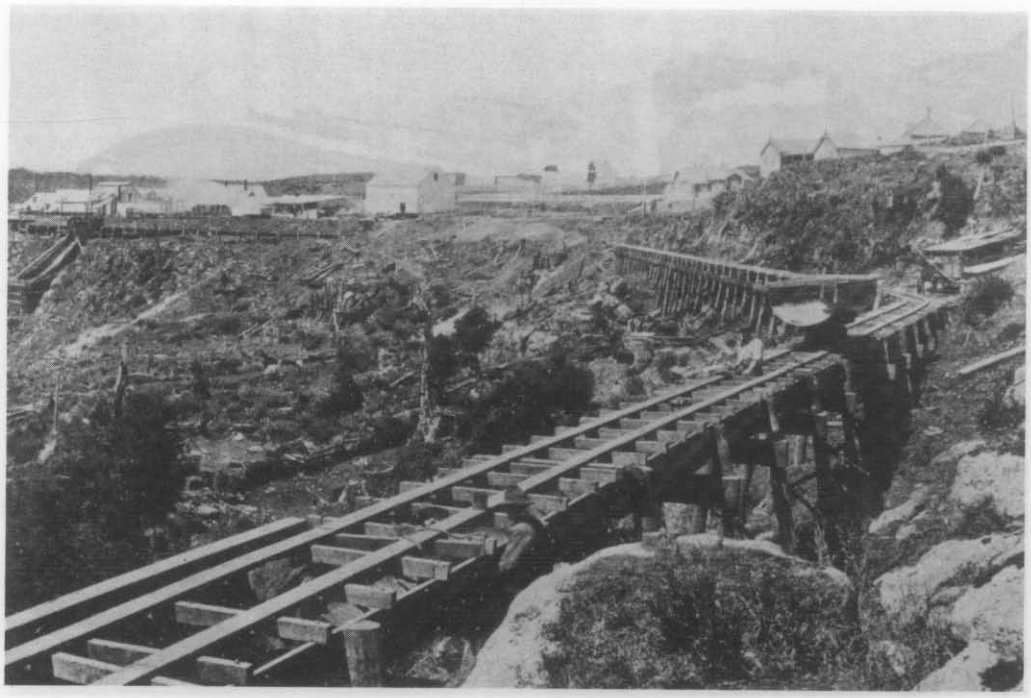


Plate 45. *Construction of water race for hydro-electric power scheme, Waratah, c. 1906. Note section of fluming on trolley.* [Herman, 1914]

Plate 46. *Pipeline to power station.* [Winter]

Plate 47. *Construction of pipeline and surge tank, c. 1906.* [Herman, 1914]

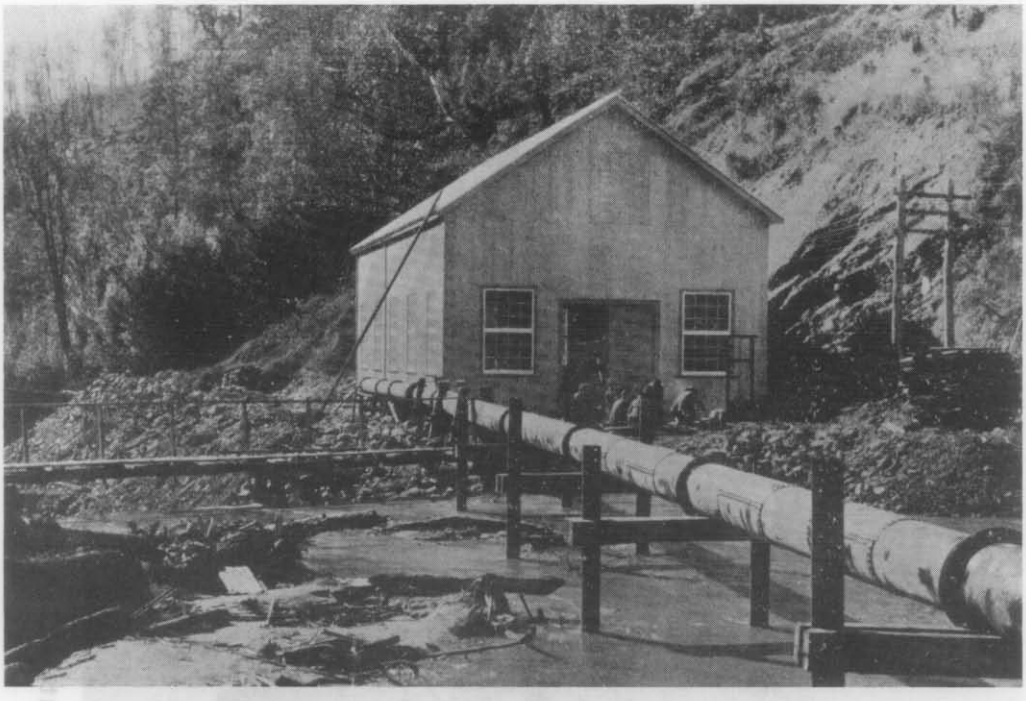


Plate 48. *Power station, c. 1907.* [Herman, 1914]

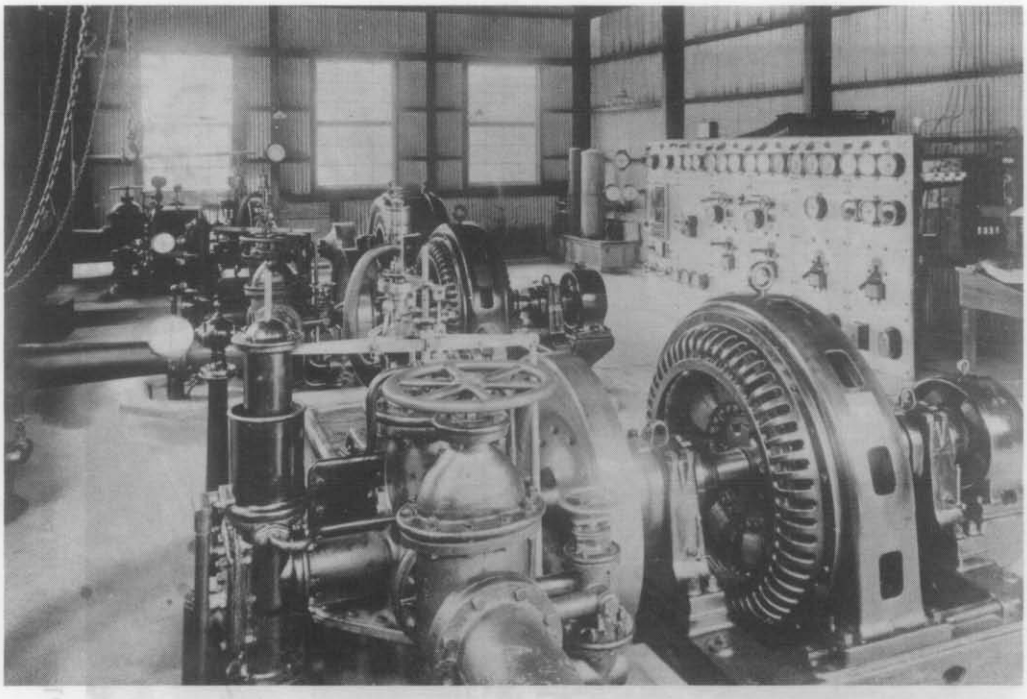


Plate 49. *Interior of power station, 1911. Two Escher Wyss turbines with 150 kw Westinghouse generators and one Voith turbine with a 375 kw A.E.G. generator are installed.*

Plate 50. *Aerial ropeway and water race, near Waratah.* [Winter]

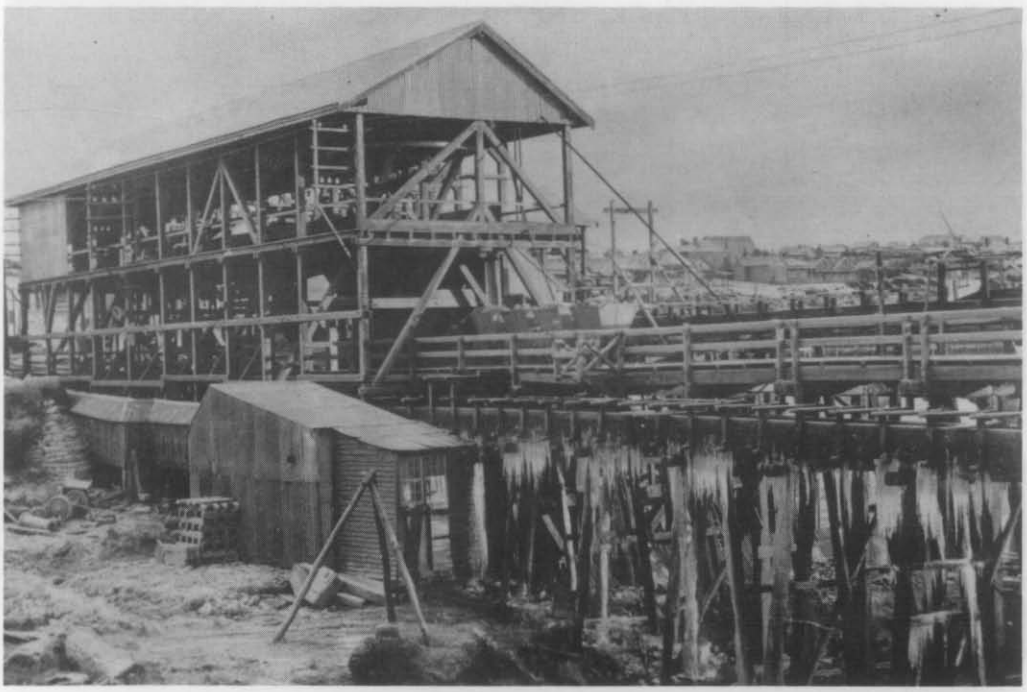


Plate 51. *Unloading station, Waratah mill. The upper floor served the aerial ropeway and the lower floor served the mine tramway.* [Winter]

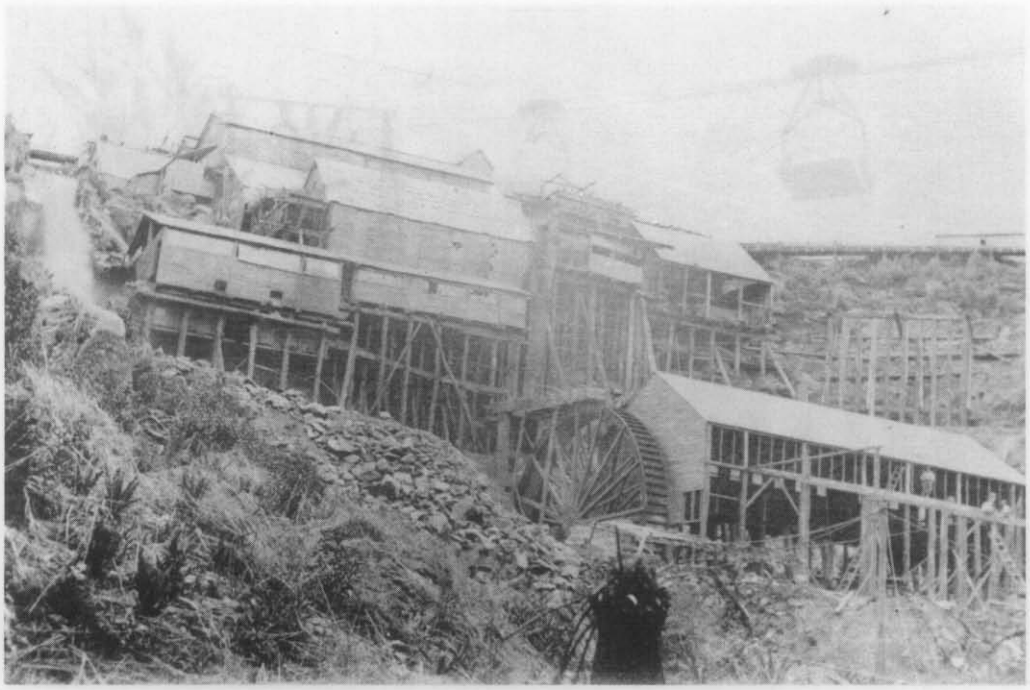


Plate 52. *Mt Bischoff Co. Mill, Waratah. 15-head mill (above) and 40-head mill under construction (below). c. 1880.* [State Libr.]

Plate 53. *Mt Bischoff Co. Mill, Waratah. 15-head mill (right centre), 40-head mill (centre) and slime sheds (right). The mill in the foreground was probably the Stanhope Co. Mill. c. 1882.* [Tasm.Mus.]

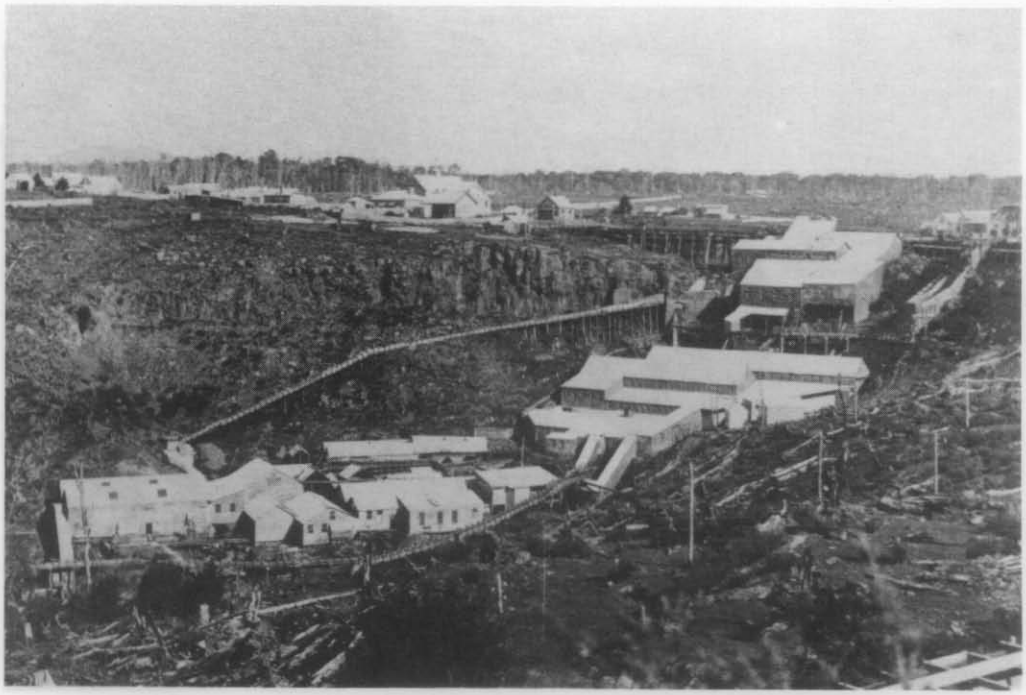


Plate 54. *Mt Bischoff Co. mill, Waratah, c. 1905. The slime sheds have been extended and the 40-head mill has been enlarged to 60 heads.*

Plate 55. *Mt Bischoff Co. mill, Waratah, c. 1912. Following electrification the belting sheds (right centre) have been removed. The aerial ropeway and unloading station have been built.* [J. Robinson]

Plate 56. *Mt Bischoff Co. mill, Waratah. New mill (foreground) and old mill (right centre, c. 1913.* [Herman, 1914]

Plate 57. *Mt Bischoff Co. mill, Waratah. Old mill, partly demolished (centre), new mill (right) and calciner (left centre), c. 1922.* [Winter]

Plate 58. *Remains of Mt Bischoff Co. mill, Waratah, 1972.* [E.L. Martin]

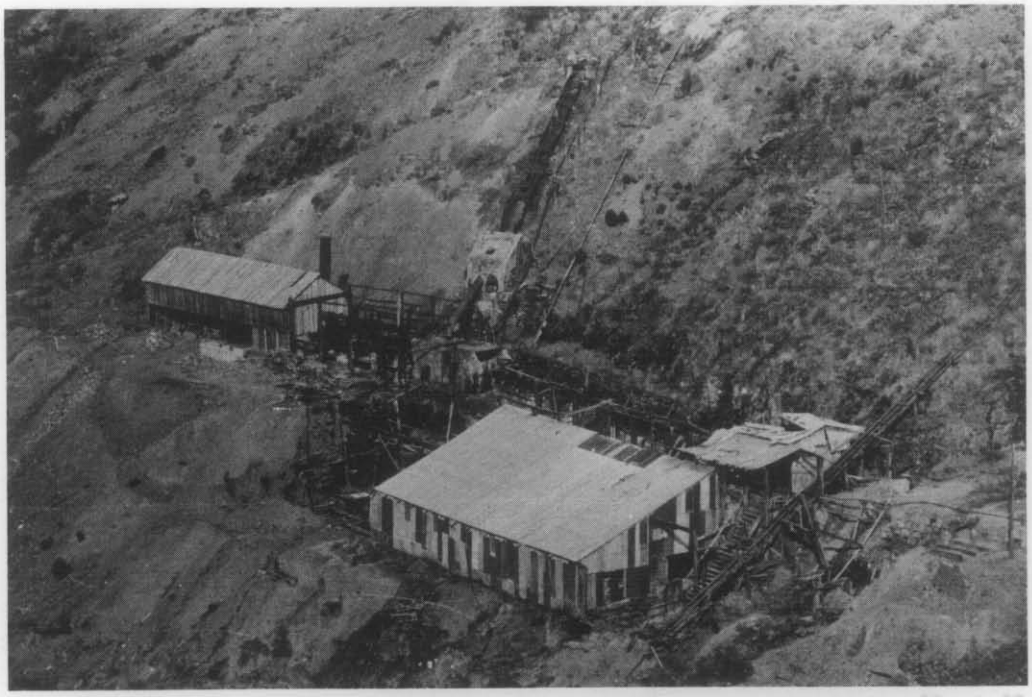


Plate 59. *Remains of calciner, 1970.* [H.K. Wellington]

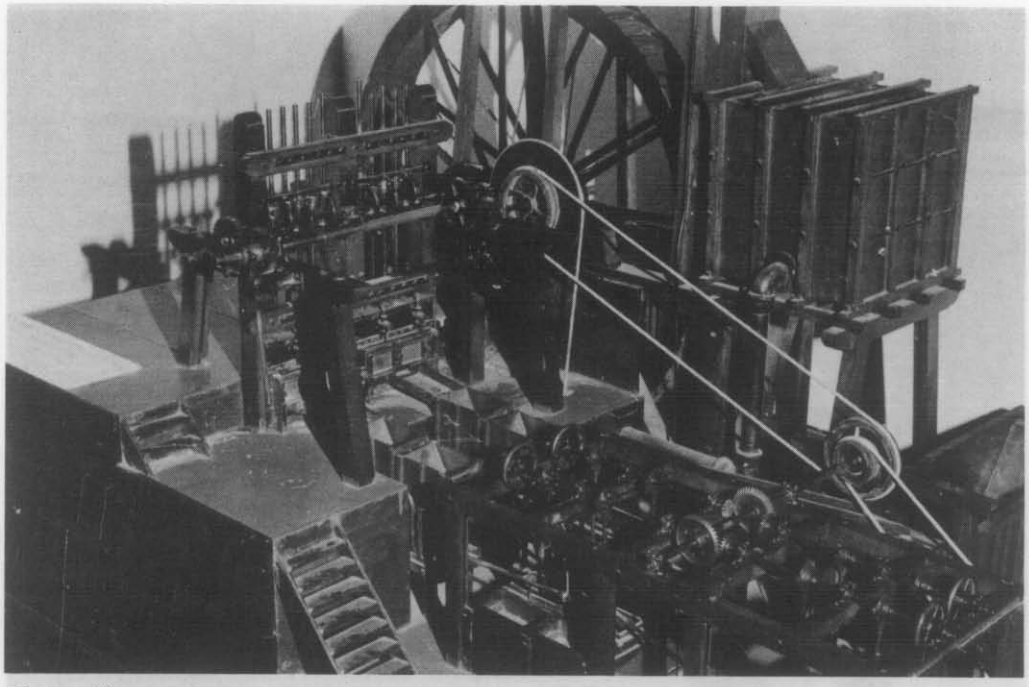


Plate 60. *Model of Kayser's Patent Oredresser at the West Coast Pioneers' Museum, Zeehan. The machinery was driven by the waterwheel (top centre); the ore was crushed in the 10-head stamp battery (left centre) and passed through a series of spitzlutte (bottom centre) and coarse and fine jigs (bottom right). A vee settling box is shown at the extreme right.* [H.K. Wellington]

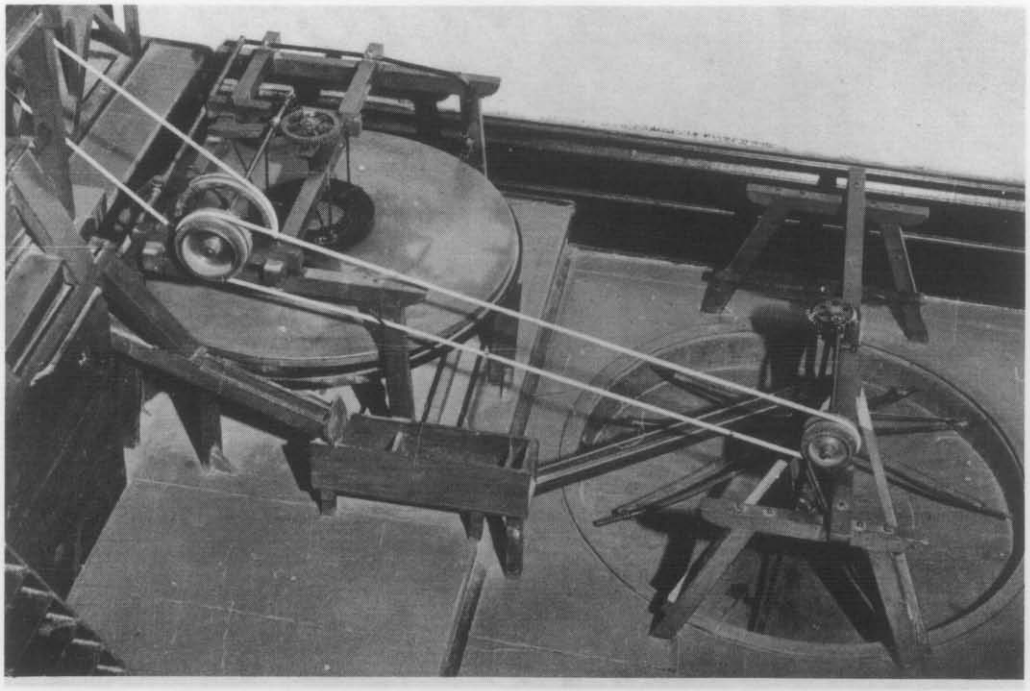


Plate 61. *Kayser's Patent Oredresser. Round table (left) and buddle (right).* [H.K. Wellington]

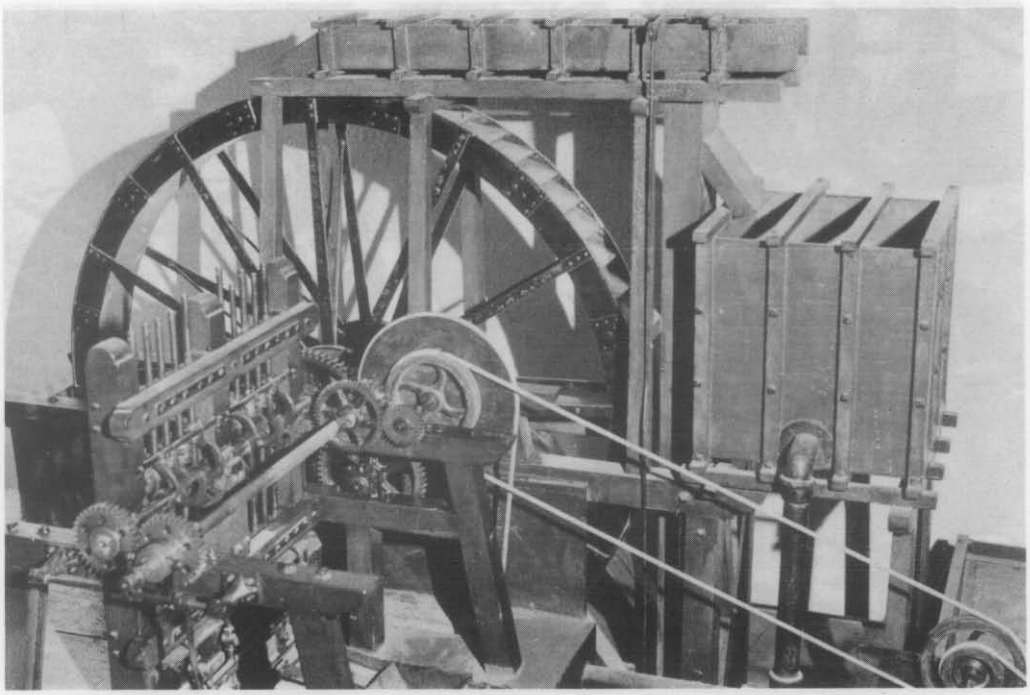


Plate 62. *Kayser's Patent Oredresser. Waterwheel, stamp battery and header tank (right).* [H.K. Wellington]

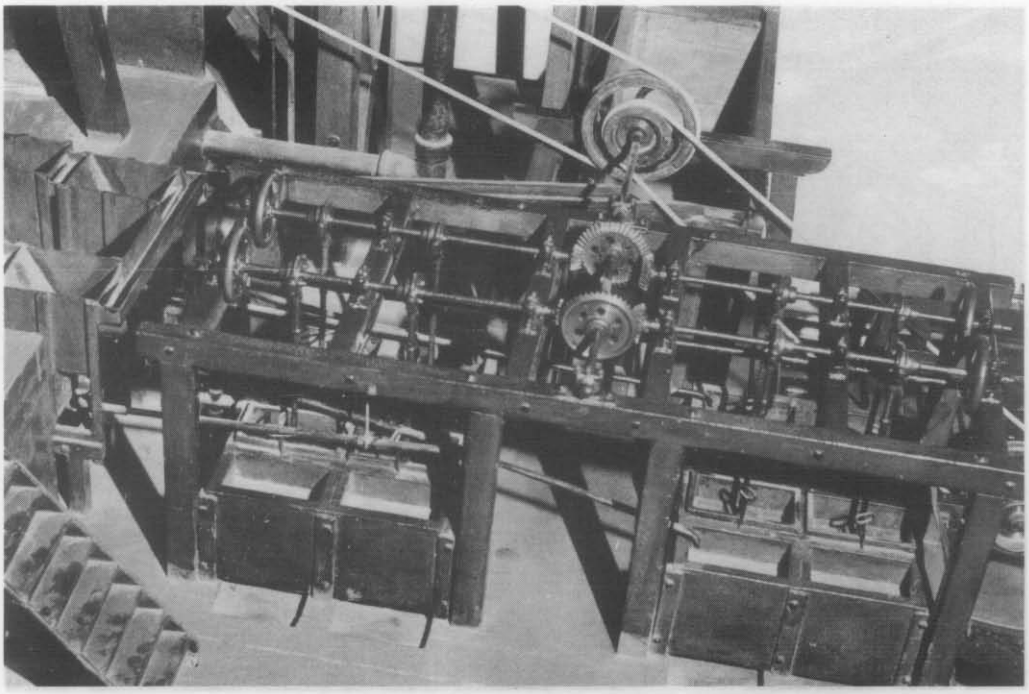


Plate 63. *Kayser's Patent Oredresser. From left to right: spitzlutte, coarse jigs, fine jigs.* [H.K. Wellington]

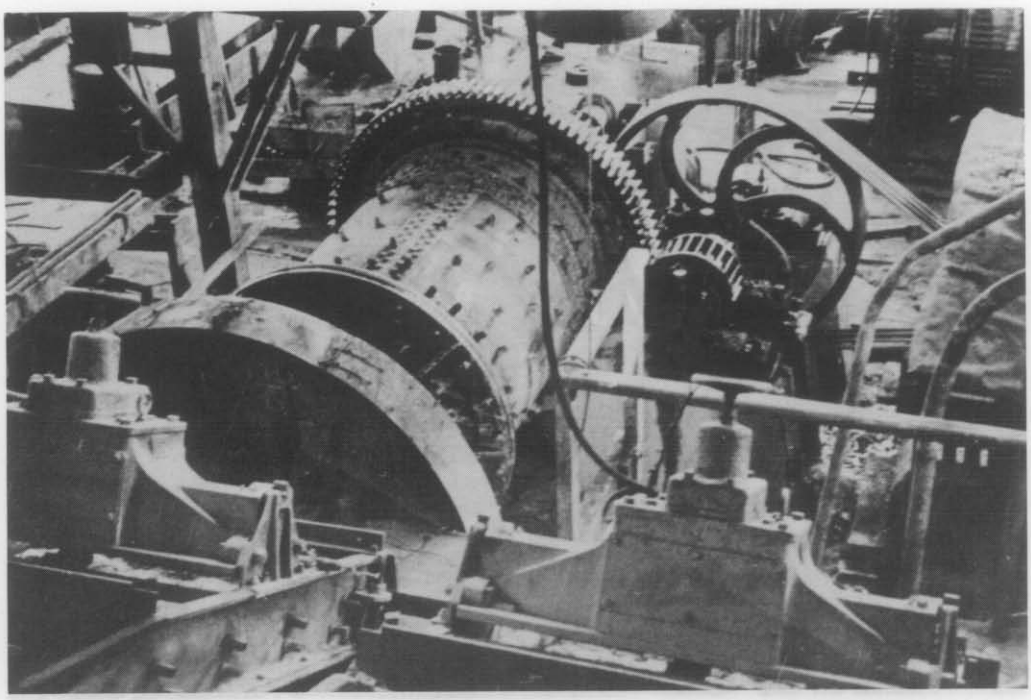


Plate 64. *Waratah mill, 1947. Grate discharge ball mill and Hummer screens.*

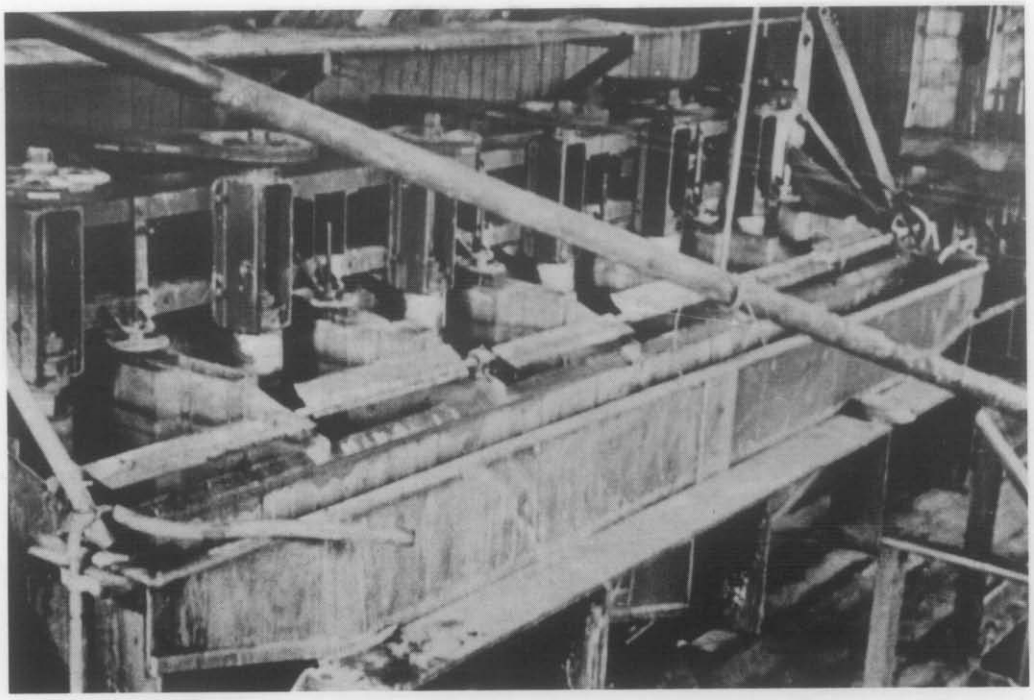


Plate 65. *Waratah mill, 1947. Denver 6-cell flotation machine.*

Plate 66. *Shaft of double round table, old mill site, Waratah, 1970.* [H.K. Wellington]

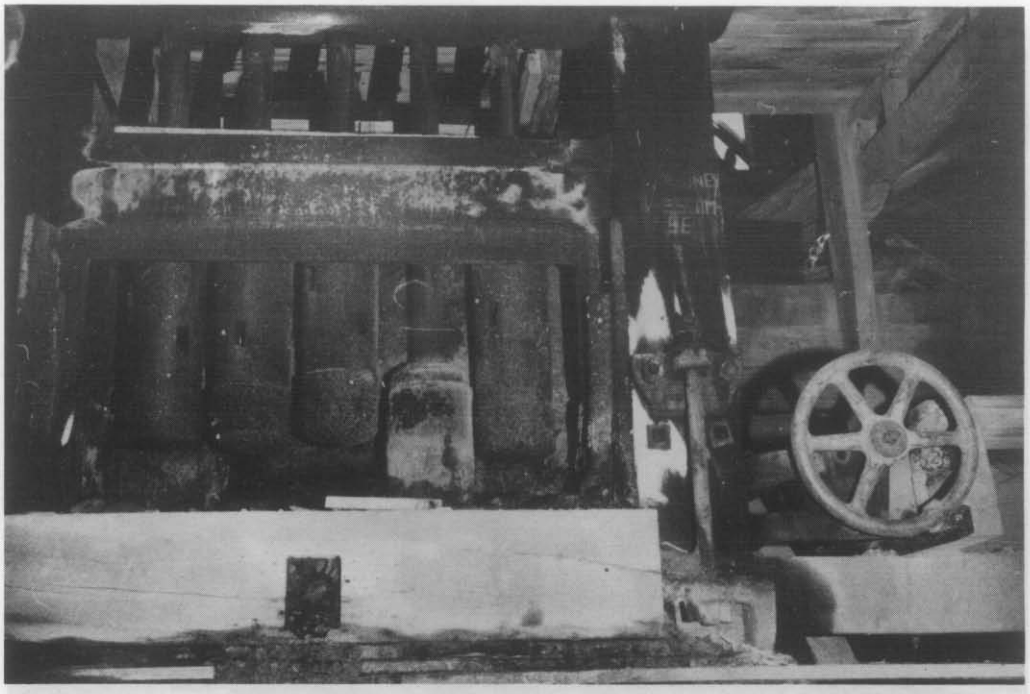


Plate 67. *Forty mill, old battery of stamps, 1970.* [H.K. Wellington]

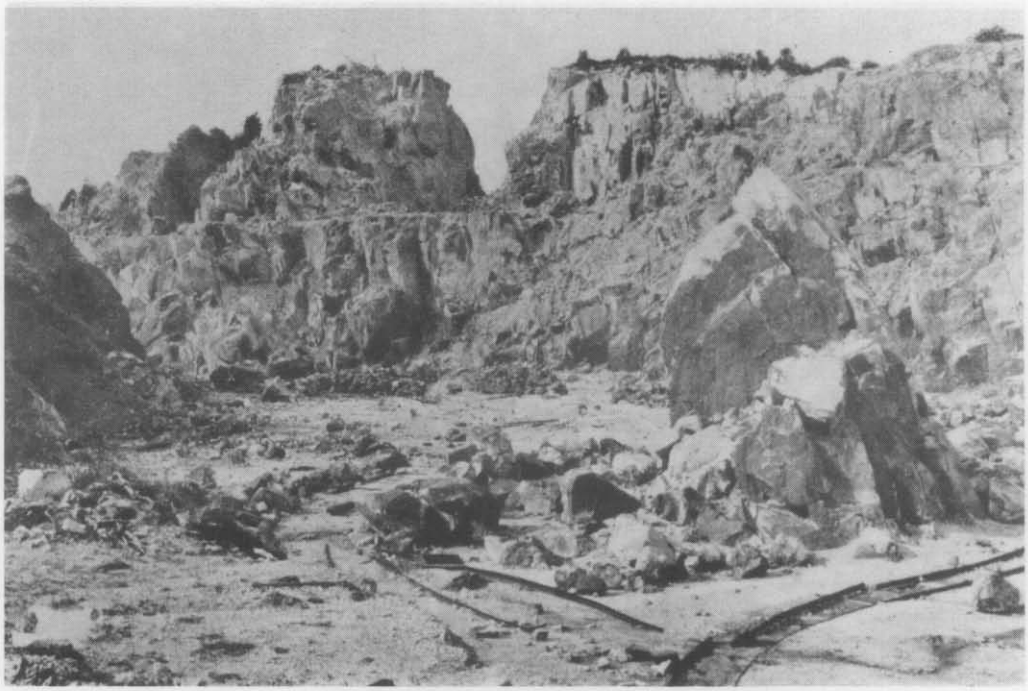


Plate 68. *Stanhope Dyke Face, c. 1920.* [J.H. Robinson]

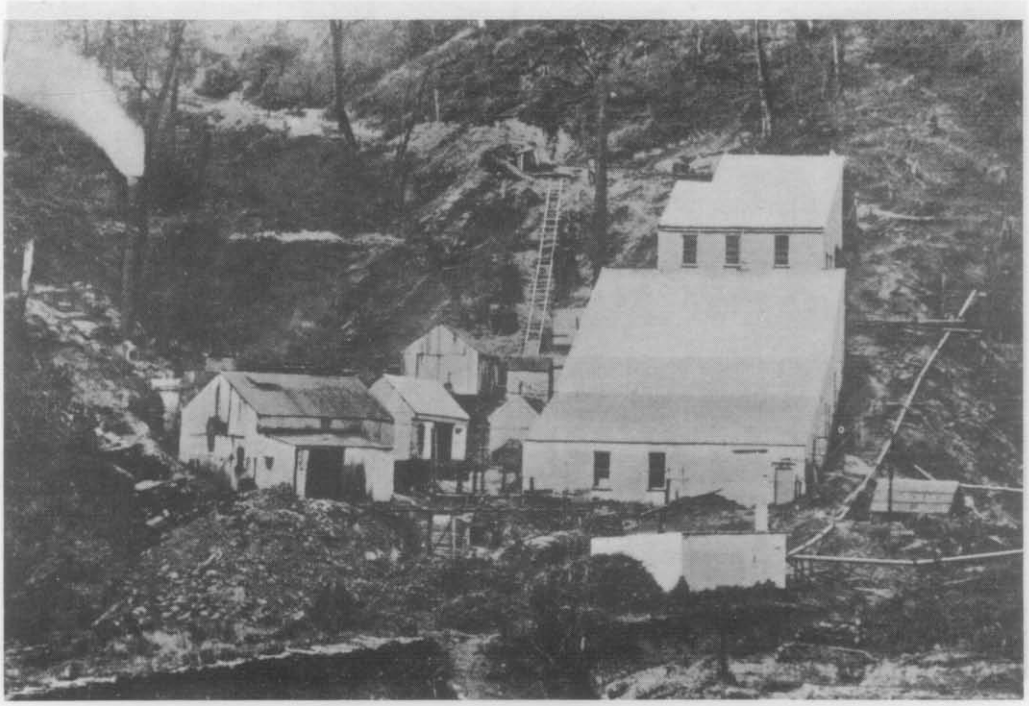


Plate 69. *Mt Bischoff Extended Co., new mill.* [Herman, 1914]

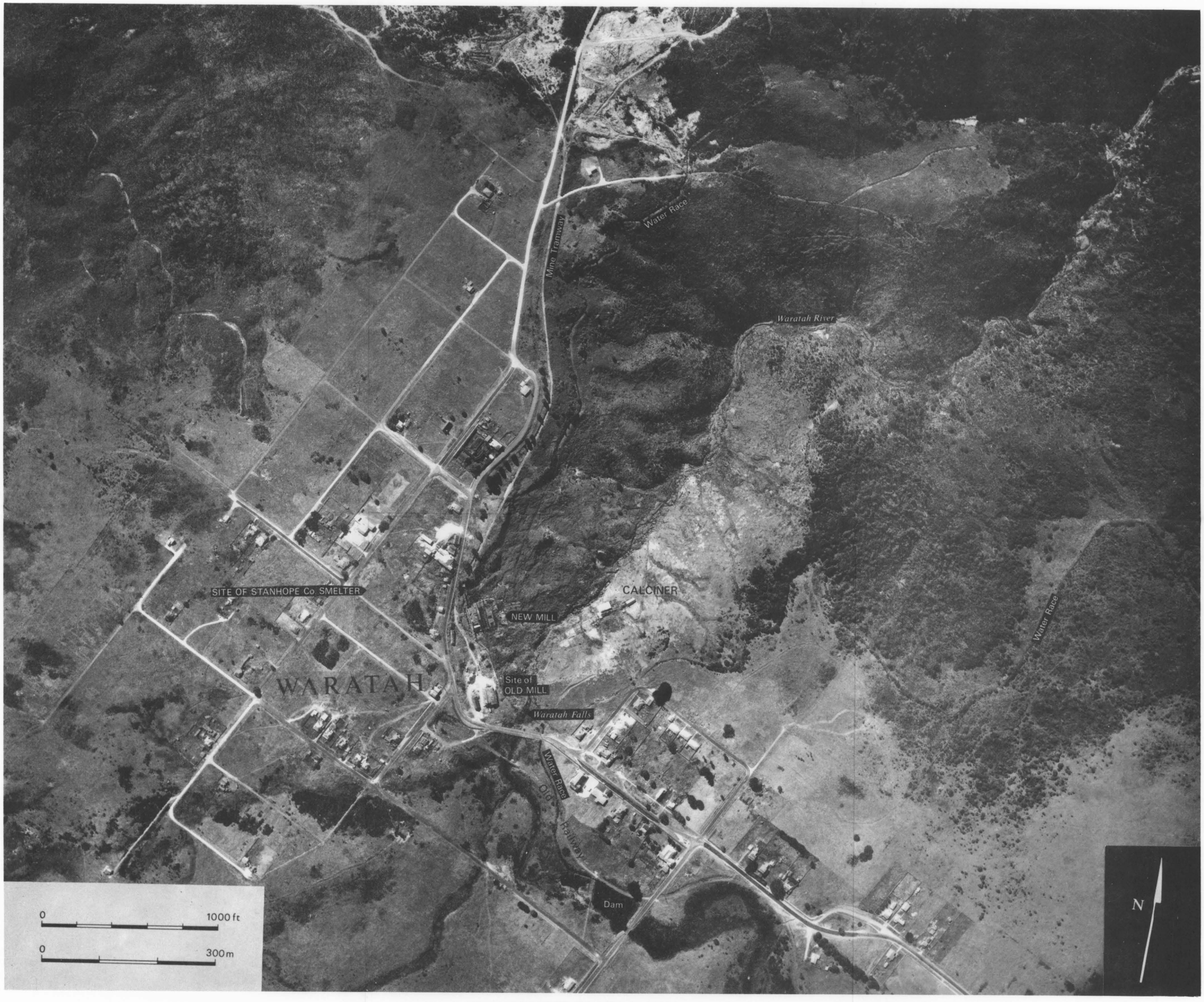


Plate 70. *Aerial photograph of the Waratah area.*

5 cm

GSB54

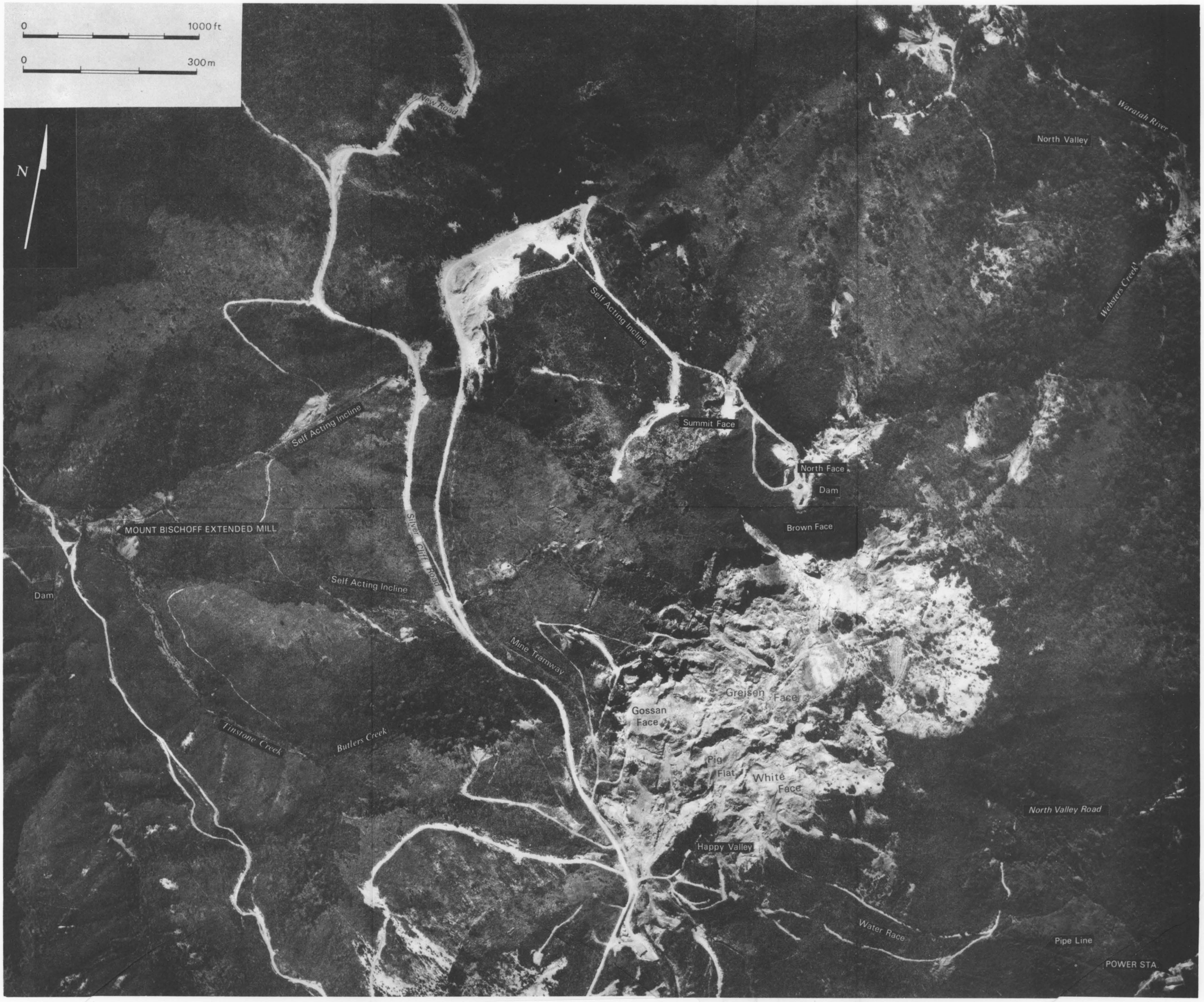


Plate 71. *Aerial photograph of the Mt Bischoff area.*

5 cm

GSB54

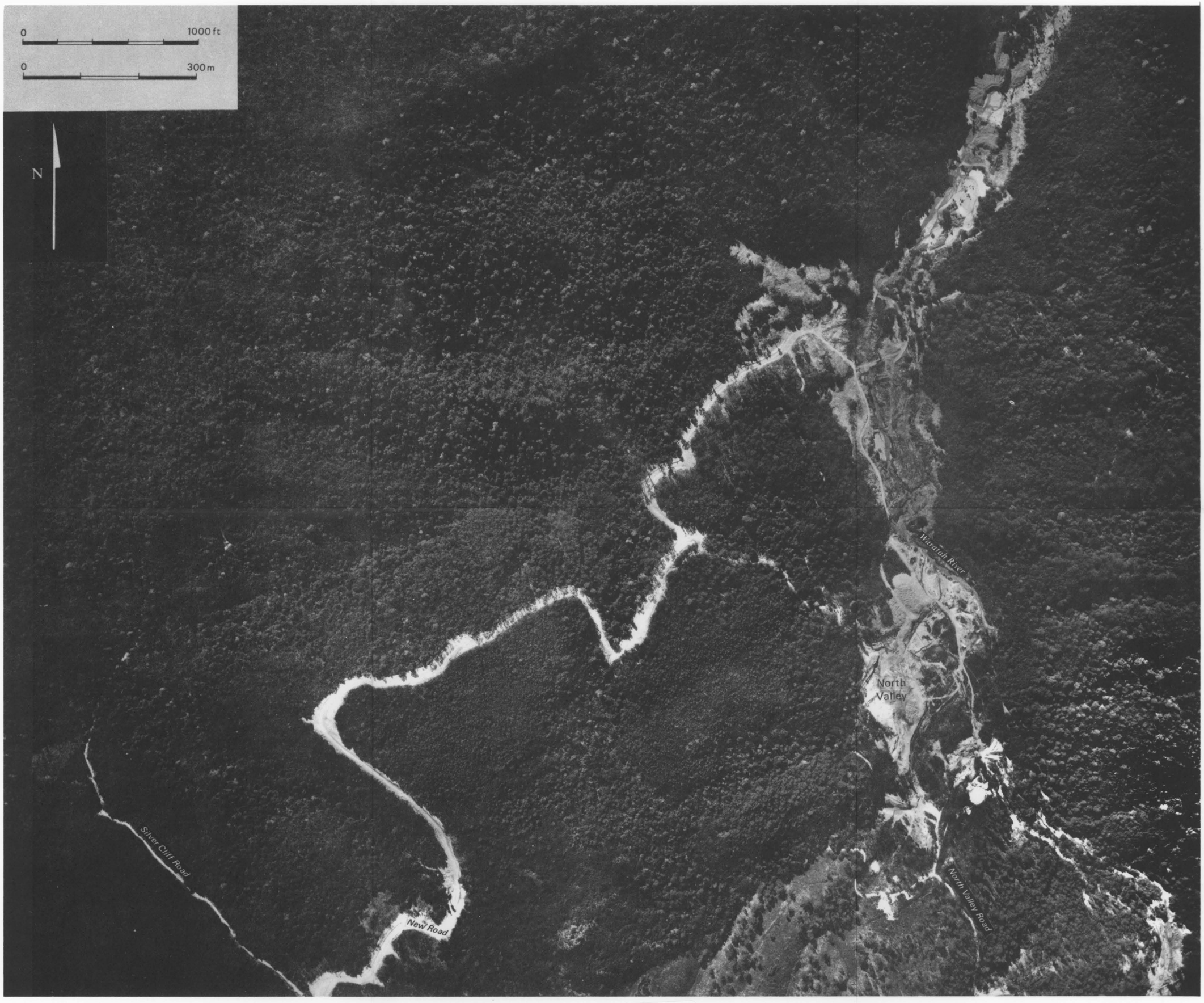


Plate 72. *Aerial photograph of the North Valley area.*

5 cm

GSB54