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ABERFOYLE TIN N. L.

# ABERFOYLE MINING AND TREATMENT OPERATIONS

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# ABERFOYLE MINING AND TREATMENT OPERATIONS

## 1. Abstract

### 1.1 Introduction

This report summarises mining and treatment operations at Aberfoyle Tin N.L., production scales, ore grade and profitability. Comment is made on the installation of an H.M.S. treatment plant about to be commissioned, conversion of mining methods and their anticipated effect on operations. A concise summary of ore reserves is submitted. Included is an approximate assessment of the Kookaburra prospect recently revealed by diamond drilling one-half mile north of Aberfoyle. Appended is a detailed report on this prospect by the Senior Geologist Mr C.J. Kingsbury.

### 1.2 Summary

The Aberfoyle mine has enjoyed until recent times a high level of prosperity in the main due to abundance of high grade ore and enhanced at certain periods by artificially inflated metal prices.

This background evidently stifled any necessity to introduce modern methods to reduce costs or improve metallurgical recovery. This report is not intended to cast aspersions on any person or group of persons but rather to present facts as they are known.

Several years ago it became evident that because of falling ore grade, rising costs and a sharp drop in tungsten price a critical assessment of the Aberfoyle operation had to be made. Because of limited reserves the assignment became "what can be done out of what has to be done to keep Aberfoyle at an acceptable level of profitability?"

It is shown that earnings for the current year are drastically reduced due to an alarming drop in grade notwithstanding significant cost reductions. Certain rearrangements and further economics are currently producing an improved rate, viz. £70,000 - £80,000, per annum not allowing for the recent upward trend in the tin price to £stg.900.

It is further shown that with the imminent commissioning of an H.M.S. plant and conversion of mining method the mine is on the threshold of returning to a profitability level of about £150,000 per annum; again with no allowance for any increase in the price of tin.

The Kookaburra-Lutwyche prospect adjacent to the Aberfoyle mine is revealed as a promising new deposit and a possible contribution to metal production within a few years.

## 2. Operations

### 2.1 Mining

The Aberfoyle mine is an underground operation producing tin and tungsten from quartz veins varying in width from a few inches to 4 feet. The average width of vein material in working places is about 20 inches and stope width 7½ ft. The tin and tungsten is present as cassiterite and wolframite freely distributed without intimacy with gangue minerals.

Vein dip averages $60^{\circ}$ and the number of veins in any particular stope varies from one to three veins with occasionally more than 3 over stope widths from 3 to 12 feet. The richer veins are associated with faults and these often dictate the physical conditions of stopes.

As a general rule walls and orebody are weak with a consequence that the most common mining method employed is flat back cut and fill. Broken ore contains a high proportion of wall rock which is hand sorted and placed as rough fill.

Ore is hand trucked in ½ ton box trucks to the main shaft where it is hoisted to the surface in double deck cages and fed to a coarse ore bin.

Mining is widely spread from the No.1 to No.11 levels and apart from pillar reclamation is confined to the lateral extremities of the mine. At the shaft section the vertical extent of the veins is delineated by an aplite cupola some 1050 feet below the surface.

## 2.2 Exploration

Recently completed was a campaign of underground and surface diamond drilling. A total of 9146 feet was drilled underground and 6892 feet on the surface.

Underground drilling revealed no information of major significance but only confirmed the extension of known veins and disclosed thin veins of minor importance.

The only phase of underground drilling remaining is long hole core drilling to the south east. This is purely speculative but should be done whilst underground sites are available.

Surface drilling has been confined to known structures revealed by mapping and of the targets tested the Kookaburra-Lutwyche complex (hereinafter referred to as the Kookaburra) shows up as a very promising prospect. An assessment of this is made later in this report.

## 2.3 Mineral Dressing

The ore is crushed to minus ⅜" and treated in sized fractions. The -⅜" + 7/32" + 10 mesh and -10 mesh + 30 mesh fractions are treated in Harz jigs whilst finer hydraulically sized fractions are treated by Wilfley sand and slime tables.

Crude concentrates produced by the above treatment are dried and the Wolfram removed by magnetic separation methods. The fraction remaining after magnetic separation is ground to separate sulphides from the cassiterite.

## 3. Elements of Production Targets

### 3.1 Ore Tonnage

The limiting factor in ore production is the capacity of the treatment plant. This plant is nominally a 10 tons per hour plant but for many years has operated in the range 13 to 15 tons per hour. This practice has not helped mechanical or metallurgical efficiency but no doubt developed with the necessity to increase ore production. This necessity still exists as will be revealed below and explains one dominant reason leading to the decision to instal a heavy media separation plant between the mine and existing gravity plant.

Tonnage capacity in the gravity mill therefore limits weekly throughput to the range 1600-1650 tons per week in normal hours and by working penalty shifts in mine and mill may be built up to about 1800 tons per week. In certain circumstances the economics of this practice are doubtful due to ore storage limitations both underground and on the surface.

### 3.2 Production Summary

Table I shows a production, grade and financial summary for 1933 through to present date.

Two important features will be noted.

(a) the persistent and steady decline of metal grades

(b) the corresponding incline of ore production until the present plateau is reached.

# ABERFOYLE TIN N. L.

## Production Costs & Profit Summary

### TABLE I

| YEAR | TONS MILLED | CONCENTRATES PRODUCED TIN (TONS) | CONCENTRATES PRODUCED WOLFRAM (TONS) | METAL CONTENTS TIN (TONS) | METAL CONTENTS WOLFRAM (TONS) | RECOVERED ORE GRADE TIN % | RECOVERED ORE GRADE WOLFRAM % | RECOVERED ORE GRADE C.M.U. % | PRODUCTION REVENUE | NET COSTS | NET PROFIT |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1933 | 9,067 | 191.12 | 13.70 | 137.61 | 9.86 | 1.52 | 0.11 | 1.63 | 25,827 | 17,178 | 8,649 |
| 1934 | 10,234 | 213.90 | 9.74 | 154.01 | 7.01 | 1.50 | 0.07 | 1.57 | 34,373 | 23,688 | 10,685 |
| 1935 | 13,218 | 301.76 | 26.31 | 217.27 | 18.94 | 1.64 | 0.14 | 1.78 | 55,795 | 30,026 | 25,769 |
| 1936 | 14,791 | 313.69 | 37.26 | 225.86 | 26.83 | 1.53 | 0.18 | 1.71 | 53,805 | 35,480 | 18,325 |
| 1937 | 13,289 | 268.68 | 33.68 | 193.45 | 24.25 | 1.46 | 0.18 | 1.64 | 60,629 | 36,434 | 24,195 |
| 1938 | 13,686 | 263.20 | 36.18 | 189.50 | 26.05 | 1.38 | 0.19 | 1.57 | 49,036 | 39,553 | 9,483 |
| 1939 | 14,725 | 311.00 | 32.13 | 223.92 | 23.13 | 1.52 | 0.16 | 1.68 | 61,947 | 47,894 | 14,053 |
| 1940 | 16,327 | 344.64 | 25.05 | 248.14 | 18.04 | 1.52 | 0.11 | 1.63 | 73,599 | 56,316 | 17,283 |
| 1941 | 16,270 | 322.87 | 25.95 | 232.47 | 18.68 | 1.43 | 0.11 | 1.54 | 70,211 | 55,858 | 14,353 |
| 1942 | 16,732 | 376.31 | 60.17 | 270.94 | 43.32 | 1.62 | 0.26 | 1.88 | 104,145 | 77,540 | 26,605 |
| 1943 | 16,503 | 334.98 | 69.19 | 241.19 | 49.82 | 1.46 | 0.30 | 1.76 | 109,739 | 76,427 | 33,312 |
| 1944 | 16,553 | 327.57 | 65.98 | 235.85 | 47.51 | 1.42 | 0.29 | 1.71 | 105,575 | 70,074 | 35,501 |
| 1945 | 19,191 | 334.69 | 56.78 | 240.98 | 40.88 | 1.26 | 0.21 | 1.47 | 101,344 | 64,342 | 37,002 |
| 1946 | 21,443 | 338.70 | 59.77 | 243.86 | 43.03 | 1.14 | 0.20 | 1.34 | 100,122 | 79,223 | 20,899 |
| 1947 | 27,829 | 440.56 | 67.44 | 317.20 | 48.56 | 1.14 | 0.17 | 1.31 | 147,180 | 106,043 | 41,137 |
| 1948 | 28,002 | 460.07 | 83.43 | 331.25 | 60.07 | 1.18 | 0.21 | 1.39 | 211,666 | 134,276 | 77,390 |
| 1949 | 25,362 | 424.90 | 90.44 | 305.93 | 65.12 | 1.21 | 0.26 | 1.47 | 213,845 | 162,919 | 50,926 |
| 1950 | 29,579 | 455.62 | 119.90 | 328.05 | 86.33 | 1.11 | 0.29 | 1.40 | 267,005 | 186,375 | 80,630 |
| 1951 | 33,228 | 451.48 | 143.30 | 325.07 | 103.18 | 0.98 | 0.31 | 1.29 | 713,169 | 337,606 | 375,563 |
| 1952 | 36,784 | 453.58 | 172.56 | 326.58 | 124.24 | 0.89 | 0.34 | 1.23 | 1,029,175 | 528,096 | 501,079 |
| 1953 | 44,847 | 580.60 | 212.58 | 418.03 | 153.06 | 0.93 | 0.34 | 1.27 | 1,032,501 | 555,135 | 477,366 |
| 1954 | 52,875 | 630.17 | 181.03 | 453.72 | 130.34 | 0.86 | 0.25 | 1.11 | 874.055 | 539,629 | 334,426 |
| 1955 | 60,353 | 715.87 | 163.92 | 515.43 | 118.02 | 0.85 | 0.20 | 1.05 | 904,103 | 570,139 | 333,964 |
| 1956 | 63,452 | 633.21 | 236.15 | 455.91 | 170.03 | 0.72 | 0.27 | 0.99 | 827,854 | 561,069 | 266,785 |
| 1957 | 59,452 | 581.10 | 294.65 | 418.39 | 212.15 | 0.70 | 0.36 | 1.06 | 630,867 | 474,980 | 155,887 |
| 1958 | 66,555 | 636.99 | 293.75 | 458.63 | 211.50 | 0.69 | 0.32 | 1.01 | 497,483 | 428,192 | 69,291 |
| 1959 | 76,118 | 805.56 | 312.14 | 580.06 | 224.74 | 0.76 | 0.30 | 1.06 | 640,669 | 519,936 | 120,733 |
| 1960 | 76,898 | 809.66 | 333.23 | 582.96 | 239.93 | 0.76 | 0.31 | 1.07 | 769,847 | 631,465 | 138,382 |
| 1961 | 65,510 | 746.73 | 286.34 | 537.65 | 206.16 | 0.82 | 0.31 | 1.13 | 713,000 | 617,046 | 95,954 |
| 1962 | 78,654 | 802.00 | 245.60 | 577.44 | 176.83 | 0.73 | 0.22 | 0.95 | 764,411 | 629,287 | 135,124 |
| 1963 | 61,728 | 579.50 | 144.00 | 420.02 | 101.24 | 0.68 | 0.16 | 0.84 | 466,300 | ~~462,000~~ 454,300 | ~~4,300~~ 12,000 |

NOTE 1963 38 Working Weeks - on estimate to 20.4.63.

It is suggested that ore production shows this trend to offset the metal grade decline and further that this trend must continue for Aberfoyle to continue on a profitable basis. This is subject to some qualification as other components such as mining method conversion and retreatment of jig tailings will make significant contributions to earnings in the future.

3.3 Ore Grade

It may well be asked - what factors determine ore grade? - and it should be noted that throughout this report ore grade is as recovered and not feed or head grade.

Ore delivered to the mill may be broken down into three main constituents :

(a) quartz containing economic and gangue minerals.

(b) fault material containing economic and gangue minerals.

(c) valueless wall rock

The relative proportions of each depend on various factors but mainly :

(a) number of veins in stope

(b) location and relationships of veins

(c) vein widths

(d) method of mining

(e) condition of walls

With this data as a backdrop it may be postulated that ore grade is controlled by the factors following :

(a) grade of quartz mined

(b) grade of fault zone mined

(c) width of quartz vein or fault zone

(d) stope width

(e) location of stope

(f) method of mining

(g) metallurgical efficiency

Not a great deal is known about the relationship between vein width and grade but it is known that the wide veins in the lower levels are extremely poor grade. It is also known that with depth the tin-wolfram ratio decreases.

It is also known that the veins attenuate with lateral extend both north and south. Another quite definite fact is that the better grade ore is confined to a territory bounded by the No's 4 and 8 levels in the vertical dimension and about 400 ft. each side of the main shaft in the lateral dimension. Only pillar reclamation and the mining of subsidiary thin veins remains to be done in this territory.

The falling off in metal grade can be therefore attributed to three factors :

(a) depletion of high grade areas

(b) mining of narrower veins in the middle sections of the mine

(c) mining of wider, poor grade veins in depth

3.4 Current Production and Reconciliation with Target.

A comparison of actual production with original target or budget production is made :

TABLE II

| | Target | Actual Average 38 Weeks to 20.4.63. | Deviation | Total Deviation for 38 Weeks |
|---|---|---|---|---|
| Ore per week tons. | 1700 | 1624 | - 76 | -2888 |
| Recovered Metal Grade % | 0.95 | 0.84 | - 0.11 | - 0.11 |
| Sn Units | 1244 | 1108 | -136 | -5168 |
| $WO_3$ Units | 372 | 266 | -106 | -4028 |
| C.M.U's | 1616 | 1374 | -242 | -9196 |

As shown on current cost and profit statement the present budget is revised downwards from the original. The above therefore represents the worst possible picture. Current profit level is estimated to be £6000 - £7000 per month.

The effect on revenue of such a deficit in metal production is somewhat excessive and cause for considerable concern to the operators. Some encouragement is to be gained from the reduction in costs of £21,000 over the same period.

The drop in metal production revealed in Table II comes from two sources -

(a) lower ore tonnage

(b) lower ore grade

The deficiency in tonnage is a direct result of mill capacity limitations and could have been avoided if sufficient ore storage capacity were available between the mine and mill. However by far the most significant contribution to loss in metal production is the drop in grade from the forecast of 0.95% to 0.84% actual. In selecting a budget figure of 0.95% a drop of 0.02% was allowed for below the figure achieved in 1961-62. It should be noted that no sampling data is available on which to forecast ore grade. All that can be done is to combine a projection of history, and judgment.

In this case the error of judgment is most likely due to a lack of appreciation of the relative importance of 3 high grade stopes worked out early this year in the high grade territory.

This suggests that it would not only be prudent to accept the present grade figure as typical of what to expect in the future but to acknowledge the possibility of further deterioration. On this premise therefore steps are necessary to offset this situation and achieve an acceptable profit rate.

4. Heavy Media Separation.

4.1 Background

To quote from a report by the writer in October 1961 :-

> "Recent trends and problems in mining and milling operations at Aberfoyle have highlighted the necessity to investigate any means available to counter the certain inevitability of restricted production and higher costs in the future".

When a second hand H.M.S. package plant became available through an international company at this time such was the stimulus to reassess the economics of H.M.S. applied to Aberfoyle. Fortunately over a period of 10 years or more ending in 1957 a great deal of H.M.S. test work had been completed on Aberfoyle ore and accumulated jig tailings. Such test work was conducted by the Research and Development Branch of the S.A. Mines Department (now AMDL), the Tasmanian Mines Department and technical representatives of American Cyanamid. Results of the test work supported the amenability of Aberfoyle products to H.M.S. treatment and the degree of economic advantages to be so derived.

In November 1961 the decision was made to purchase and instal a second hand plant ex Africa and through George Cohen & Sons Ltd. Estimated cost was £80,000 for a plant if purchased new would have cost £250,000, or in that order.

## 4.2 The H.M.S. Plant

The plant is nominally 60 ton/hour and will accept the discharge from the existing coarse crushers at -2" sizing. After screening and crushing in closed circuit -1" material is fed to specially constructed bins. From these, ore is fed to a feed preparation screen with -10 mesh undersize to gravity mill and +10 mesh -1" to the H.M.S. vessel. This produces the float reject and the valuable sink product to be processed. Preconcentration is such that the sunk product plus -10 mesh material should represent about 25% - 33.1/3% of the original feed.

Upgrading such as this appeals as a cheap means of sorting and relief of pressure on costly sections of the gravity plant. As the plant at the time of writing is in the process of being commissioned it's characteristics and performance will be evident within a short time. Our plan is to work up to a tonnage of 2,300 tons of ore per week as quickly as possible and then complete the flowsheet to allow retreatment of jig tailings at a rate between 1000 and 1500 tons per week.

With few exceptions units of the plant are in superior mechanical condition and satisfactory metallurgical results are expected. The completion of the plant is about four months behind the original schedule. Complete delivery of all units and building was promised by June-July 1962 but floods in Tanganyika rendered inoperable the railway line to the Port of Dar-es-Salam. With this and other delays in transit the last shipment did not arrive on site until late December 1962. This was particularly damaging to our work and cost programs.

4.3 Projected Effects of H.M.S.

Allowing a suitable period of running in or settling down, operator training and elimination of "bugs" in the circuit the immediate effect is to permit the underground operation to deliver higher tonnage of ore. Production should reach 1900 tons of R.O.M. ore per week within a short period and then over a period of six months build up to 2300 tons of R.O.M. per week. This aspect of the program is tied up closely with conversion of mining method dealt with later in this report.

No increase in milling labor costs are expected but rather over a period of some months some decline may be expected. Maintenance labour will not increase and certainly maintenance material will decrease significantly.

Following completion of the circuit to retreat jig tailings and perfection of process technique earnings will be usefully augmented from this source. Expansion of the sand and slime section in the gravity plant to treat fines currently produced from H.M.S. offers better practice and useful gains.

The greatest effect of H.M.S. is in allowing the underground operation to convert to a mechanised operation in hydraulically filled stopes. This does not necessarily mean an increase in metal production but reduces the underground labour force by

15% to 20% meaning ultimate savings of £40000 - £50000 per annum.

Consolidating the effects of H.M.S. and estimating or converting to financial gains Table III presents the anticipated results.

## TABLE III

### EFFECTS OF H.M.S. - GAINS PER ANNUM

| | MAY | JUNE | JULY | AUG. | SEPT. | OCT. | NOV. | DEC. | JAN. 1964 | FEB. | MAR. | APRIL | MAY | JUNE | JULY |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| INCREASED TONNAGE 200 TONS/WEEK | | | £ 30000 ~~MXXXXXXX~~ | | | | | | | | | | | | |
| MILLING COSTS | | | | £ 3000 MATERIAL | | | | | | | | £ 4000 LABOUR | | | |
| JIG TAILS RETREATMENT 1200 T/W 7/6 PER TON PROFIT | | | | | | | | | £ 22000 | | | | | | |
| HYDRAULIC FILL | | | | | | | | | £ 45000 | | | | | | |
| ESTIMATED PROFITABILITY RATE PER ANNUM | £80000 | £80000 | £110000 | £114000 | £114000 | £114000 | £114000 | £114000 | £151000 | £151000 | £151000 | £155000 | | | |

N.B.

1. At Present metal prices Sn 205/- per l.t.u. $WO_3$ 80/- per l.t.u.

2. No allowance made for possible gains in other areas e.g. AN/FO and slime recovery improvement

3. No allowance made for added expenditure from industrial legislation e.g. 3 weeks annual leave

## 5. Hydraulic Filling

### 5.1 Reasons for Conversion

Earlier mention is made of the traditional Aberfoyle Mining method employing flat back cut and rock fill technique. This technique with hand sorting, hand mucking and high timber consumption has many disadvantages :

(a) it is the most costly method that could be employed

(b) it is difficult to supervise and control

(c) there are too many openings for damaging exploitation by labour

(d) it is difficult to apply satisfactory incentives

(e) risks are high for metal losses in stopes

The present management attempted wide scale implementation of more selective methods of mining such as resuing and forcing greater emphasis on hand sorting. This is still done where particular conditions are favourable but had to be abandoned as a general practice.

Therefore it was reasoned that to cut costs and improve productivity bulk mining with mechanisation had to be introduced. Such was the climate of thinking when the H.M.S. project was being considered; for bulk mining could not be undertaken without the means of treating higher tonnages of ore. Bulk mining and mechanisation predicates the use of hydraulic filling derived from the finer fractions of the mill residues.

All that has been done and is being done is to utilise a modern method of mining demanded by present physical and economic conditions. The necessary mechanics for conversion have been in progress for some 18 months and at the time of writing four stopes are in regular production. Surface installations and reticulation lines as yet are only partially complete and it is anticipated that about 6 months will elapse before complete conversion. At this stage estimates indicate that the method allows a total reduction in the underground labour force of 25 men.

## 5.2 Technique

Summarising the principal features of the hydraulic filling technique or more exactly the features introduced by the mining method conversion resting on the application of hydraulic filling it allows :

(a) Mechanised handling in stopes with scrapers

(b) Elimination of hand sorting

(c) Wider chute spacing

(d) Superior ground support

(e) Better conservation of metal

(f) Better utilisation of explosives

(g) More complete extraction

(h) Reduced timber consumption

These factors more than offset the added cost of hauling and hoisting a greater quantity of materials when that so occurs.

### 5.3 Performances and Economics

As a check on results an analysis was made recently on performances from the trial stopes. This analysis is reproduced in Table IV.

TABLE IV

Cost Comparison for Mining Methods

| STOPE | WEEKS ON | LABOUR COST/TON Sh. CURRENT | PROGRESSIVE | PREVIOUS METHOD |
|---|---|---|---|---|
| 76 DU | 6 | 28.4 | 32.5 | 69.0 |
| 55 DU | 16 | 18.5 | 28.8 | 55.0 |
| 69 DA | 3 | 23.2 | 95.0 | 60.7 |
| 78 DU | 10 | 19.4 | 37.1 | 44.0 |
| 1 HF | 42 | 24.0 | 40.8 | - |

NOTE.

1. Progressive costs include conversion cost
2. 69DA progressive costs include cost of 1000 tons of fill for only 3 weeks production
3. 1 HF stope absorbed the costs of all original experimental work and is in a narrow vein
4. Excepting 1 HF it is estimated that the quality of the ore produced from the stopes is little different to that produced by previous method

Original estimates indicated a stope efficiency of 10 tons per manshift broken and scraped but in fact the efficiency achieved so far is closer to 15 tons per manshift.

In converting to a bulk mining method one basic assumption is that the ore will be downgraded. Whether the degree of downgrading will be as severe as forecasted remains to be seen but it certainly will be no worse.

It is assumed therefore that 2200 tons of ore won by hydraulic fill method will yield the same metal as 1700 tons of ore from traditional method. This equation was struck giving recognition to vein widths, stope widths and material sorted.

A comparison of costs shows

| | | | |
|---|---|---|---|
| Traditional | - | 1700 tons at 96/- | £8160 |
| Hydraulic fill | - | 2200 tons at 66/- | £7260 |
| | | Cost saving per week | £ 900 |
| | | Over 251 working days = | £45000 |

which is consistent with savings to be expected from a reduction of 25 men in the labour force.

6. Kookaburra Prospect

Five holes were drilled into this prospect. Results were most encouraging especially in the deeper holes such as D.D.S.-21 yielding the following significant intersections :

1060' True width 11 inches estimated 2.4% Sn over 48 inches

1180' True width 12½ inches estimated 2% Sn over 48 inches.

Coupled with local knowledge sufficient data was obtained to draw the conclusion that any further drilling could not affect any decision regarding the desirability or otherwise of underground testing.

An inferred value of £2,000,000 is placed on the deposit and it will require approximately £120,000 to finance underground testing before it can be called a mine. It is estimated that the rate of expenditure would be £6,000 per month for 20 months with the likelihood that development would yield some revenue.

Under direction the Senior Geologist Mr C.J. Kingsbury has prepared a comprehensive report detailing geology, drilling results, evaluation and the proposed next stage in exploration. This report is appended.

7. Ore Reserves

| | March 30, 1963. | June 30, 1962 |
|---|---|---|
| Total Measured | 252,956 | 239,847 |
| Total Indicated | 63,096 | 53,965 |
| Grand Total | 316,052 | 293,812 |

These ore reserve figures are in parallel with the traditional method of mining and are actuallycomputed on a basis of 55% wall rock dilution.

In effect it represents a reserve life of 5-6 years at current metal production.

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