Koolamara Mining Company

Report No. 18.053.1/3

Testing of Tin-Tantalum Ore

From Mt Finniss Mine, N.T.

S. K. Pennyquick
August, 1979

Mineral Deposits Limited
Research and Engineering Department
Southport Queensland
Australia
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</tbody>
</table>
1. INTRODUCTION

This is a report prepared for Mr. John Walton of Koolamara Mining Company concerning the amenability to treatment of a parcel of 729 Kg (dry weight) of eluvial ore from the Mount Finnis area. The elements to be recovered were primarily Tin and secondarily Tantalum.

The sample was taken by Mr. Walton and processed at our Southport ore testing facility.

Analysis of samples was carried out by Australian Laboratory Services, Brisbane, and some check analyses of Tantalum and Niobium were carried out by A.M.D.E.L. in Adelaide.

1.1 TERMS OF REFERENCE

Authority for this work is contained in Mr. Walton's letter of 19th January, 1979 and specification of services offered in Mineral Deposits Limited's letters, ref SKP:ML:18.058.1 of 2nd January and 18th January.

1.2 SUMMARY AND RECOMMENDATIONS

The material is amenable to concentration by wet gravity and dry magnetic methods.

The tests show that 1 ton of "feed" would yield 220 grams of Tantalite concentrate (25.03 Ta₂O₅, 27.46% Nb₂O₅) and 300 grams of Tin in a Tin concentrate (at approximately 60% Sn). Allowing for the fact that the "feed" represents 58% of the Total ore to be mined (i.e. 58% of weight is "feed" and used as feed in these tests).

The yield per tonne of ore would be much less, viz.

<table>
<thead>
<tr>
<th>Material</th>
<th>Grams</th>
</tr>
</thead>
<tbody>
<tr>
<td>Tantalite concentrate</td>
<td>127.6</td>
</tr>
<tr>
<td>Tin in concentrate</td>
<td>174</td>
</tr>
</tbody>
</table>
Using pricings in the Metal Bulletin, it is estimated that the value of the products per tonne of ore would be, combined, $5.50/tonne.

It is projected that a plant would have a mining rate of approximately 40 tonnes/hour ore treated. This would amount to about 300 tonnes/day and be worth = $1,650.00 per day in revenue.

The estimated cost of the treatment plant would be $55,000 and total capitalisation would be approximately $200,000.00.

Since the amount of ore is only 40,000 cubic yards, or, say, 60,000 tonnes, the cost/tonne of establishment would be approximately $3.30/tonne.

Operating costs would be approximately $7,500/week, and Capital repayment would be approximately $5,000/week Total Cost $12,500/week

At these costs the project is hardly marginal.

S. K. PENNYCUICK
B.E.(Min.), A.M.Aus.I.M.M.
CHIEF MINING ENGINEER
8th August, 1979
SKP:ML
2. SAMPLING AND SIZE ANALYSIS

2.1 DESCRIPTION OF SAMPLE

The sample was in 3 x 200 litre drums weighing 866 Kg wet.

The moisture content was determined by drying two 7 Kg samples and found to be 13.4%. The drums, although covered, were affected by heavy rain in transit and before the testing. The material was an eluvial reddish brown sandy clay type material which had been already screened at approximately 15mm in the field by the client. It was very thick and difficult to handle owing to the presence of moisture. It required some mixing and pre-drying before it could be become sufficiently mobile for free flow through the automatic sampler.

2.2 TESTWORK

The size analysis required the taking of a sample by means of a dry rotary automatic sampler. Two samples of \( \frac{1}{100} \) each were taken simultaneously.

2.2.1 Sizing Analysis

Both samples were given sizing analysis in 8 sizings down +8mm through to -53 micron (slimes). The averaged results of weight distribution is given in Table 2.1. This sizing was carried out wet using laboratory sieves with the exception of the sizes -1mm +53 micron, which material was dried and dry screened.
TABLE NO. 2.1

<table>
<thead>
<tr>
<th>SIZING</th>
<th>WEIGHT DISTRIBUTION</th>
<th>AVERAGE ASSAY</th>
<th>UNITS TIN</th>
<th>DISTRIBUTION TIN</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>%</td>
<td>% Sn</td>
<td>% Ta</td>
<td>% Sn</td>
</tr>
<tr>
<td>+8mm</td>
<td>15.90</td>
<td>trace</td>
<td>.13</td>
<td>Nil</td>
</tr>
<tr>
<td>-8mm +3mm</td>
<td>22.90</td>
<td>0.007</td>
<td>.001</td>
<td>0.1603</td>
</tr>
<tr>
<td>-3mm +1mm</td>
<td>12.10</td>
<td>0.05</td>
<td>.006</td>
<td>0.6050</td>
</tr>
<tr>
<td>-1mm +500μ</td>
<td>2.15</td>
<td>0.51</td>
<td>.052</td>
<td>1.0965</td>
</tr>
<tr>
<td>-500μ +250μ</td>
<td>3.00</td>
<td>0.44</td>
<td>.015</td>
<td>1.3200</td>
</tr>
<tr>
<td>-250μ +125μ</td>
<td>8.35</td>
<td>0.12</td>
<td>.008</td>
<td>1.0020</td>
</tr>
<tr>
<td>-125μ +53μ</td>
<td>8.45</td>
<td>0.06</td>
<td>.004</td>
<td>0.5070</td>
</tr>
<tr>
<td>-53μ(slimes)</td>
<td>27.15</td>
<td>0.02</td>
<td>trace</td>
<td>0.5430</td>
</tr>
<tr>
<td>TOTAL</td>
<td>100.00</td>
<td>0.05</td>
<td>N.A.</td>
<td>5.2338</td>
</tr>
</tbody>
</table>

2.2.2 Assaying of Sizings

Chemical analysis of the sized fractions was carried out in relation to Tin and Tantalum, and results are shown in Table 2.1. A size distribution of the Tin content is also given in this table.

2.2.3 Heavy Liquid Analysis of Sizings

In order to test the amenability of the material to gravitational separation, analyses of the fractions was made, firstly in Bromoform, S.G. 2.85, and secondly in Clerici Solution, S.G. 4.05.

Results of these separations and their chemical analyses for Tin and Tantalum are given in Table No.2.
2.3 SUMMARY

From Table No.1, it can be deduced that:

- The fractions -1mm to +53 microns contain 21.95% of the weight of the sample and 75% of the contained Tin.
- The grade of this fraction is 0.17% Sn.
- The Tantalum content is very low. However, subsequent examination of the large scale sample revealed that there is recoverable tantalite/columbite in the +3mm sizing. This will be discussed later.

From Table No.2, it can be seen that:

- The Tin in the -1mm +53 microns fraction should be recoverable by gravity methods to produce a 45% Sn concentrate.
- The Tantalum present in these sizings is low and would not be upgraded by gravity to an economic grade.
3. LARGE SCALE TESTWORK

The results of the above work indicated that the Tin would be best recovered by washing and screening the total sample at 3mm and rescreening the undersize at 1mm and cycloning the -1mm material to remove the -53 micron slimes.

3.1 PROCESSES

3.1.1 Sizing

The process used was that of washing in a 4-ft diameter rotary scrubber fitted with a 3mm trommel screen at the outlet.

The undersize was pumped over a 1mm 4-ft x 2-ft vibrating screen, and the undersize was pumped through a Krebs D6B cyclone, from which the underflow gravitated into a transfer bin, and the overflow gravitated to slime dam. A diagram of the process is given in figure No.3.2.

Results of the sizing are given in Table No.3.1.

<table>
<thead>
<tr>
<th>TABLE NO. 3.1</th>
<th>WEIGHT</th>
</tr>
</thead>
<tbody>
<tr>
<td>SIZING</td>
<td></td>
</tr>
<tr>
<td>+3mm product</td>
<td>276 Kg</td>
</tr>
<tr>
<td>+1mm product</td>
<td>85 Kg</td>
</tr>
<tr>
<td>-1mm product</td>
<td>256 Kg</td>
</tr>
<tr>
<td>Slime overflow loss</td>
<td>112 Kg</td>
</tr>
<tr>
<td>Total Feed to Washing and Screening Plant</td>
<td>729 Kg</td>
</tr>
</tbody>
</table>
RESEARCH AND ENGINEERING DEPARTMENT
MINERAL DEPOSITS LTD.—SOUTHPORT, QUEENSLAND

KOOLAMARA MINING COMPANY

DIAGRAMMATIC ARRANGEMENT CYCLONE/SCRUBBER
TROMMEL SYSTEM FOR FEED PREPARATION TESTWORK

FIGURE 3.2

DATE: 8/8/79
SCALE: None
3.1.2 Gravity Separation - Tin

The -1mm (cyclone underflow) product was tested in a seven turn x 14½ inch pitch spiral concentrator.

The concentrate from the Spiral was successively reconcentrated in three stages on a shaking table to produce a concentrate assaying 47% Sn.

Recoveries were good and upgrading was good.

A flowsheet and materials balance of the washing and gravity testwork is given in figure 3.3.

3.1.3 Magnetic Separation - Tin

Magnetic separation was carried out in a Carpco Roll High Intensity Laboratory magnetic separator. Speed was 200 rpm.

Field strengths were controlled by current settings, as indicated.

Cassiterite being relatively non magnetic remains in the non magnetic product at the end.

The results of these three tests are given in tables 3.4, 3.5 and 3.6.

It will be noted that further heavy liquid separation of the non-magnetic products gave enhanced grades of Tin in concentrates. This indicates that further cleaning by gravitational separation could be required after magnetic separation.

The overall results indicate that approximately 85% of the cleaner table concentrate could be recovered as marketable Tin concentrate.
TOTAL SAMPLE INPUT
Wt. Kg, % Sn, Grams Sn.
729, 0.05, 364.5

Wt. Assay Tin
Kg, % Sn, Gram
729

TROMMEL SCREEN CYCLONE

+3mm

256, 0.11, 280.3

+1mm

85, 0.05, 42.5

Cyclone Overflow 112, 0.02, 22.4

WASH Overview

164, 0.007, 11.3

SPIRAL

C 5.90, 0.06, 3.5

M 63.54, 0.003, 2.1

SCAVENGER TABLE

3.39, 0.002, 0.1

2.05, 0.005, 0.2

0.46, 0.06, 3.2

ROUGH TABLE

13.15, 0.002, 0.2

4.30, 0.06, 260.5

T 5.11, 0.04, 2.6

M 1.23, 0.003, 0.2

1.79, 0.14, 31, 0.256.2

(93.83% + 2.85 S.G.)

T 1.28, 0.32, 4.2

M 1.59, 0.2, 162.2

C 0.20, 0.47, 0, 94.0

0.20, 0.47, 0, 94.0

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

KOOLAMARA MINING COMPANY
GRAVITY SEPARATION TESTWORK

FIGURE 3.3

DRAWN: SKP:ml
DATE: 8.8.79
SCALE: None
TABLE NO. 3.4

CLEANER TABLE CONS - GRADE 14.3% Sn

<table>
<thead>
<tr>
<th>PRODUCT*</th>
<th>WT.% DIST.</th>
<th>% Sn</th>
<th>DIST.Sn%</th>
</tr>
</thead>
<tbody>
<tr>
<td>0.0a H.S.</td>
<td>11.55</td>
<td>0.45</td>
<td>0.36</td>
</tr>
<tr>
<td>0.25A M</td>
<td>17.57</td>
<td>0.28</td>
<td>0.35</td>
</tr>
<tr>
<td>0.60A M</td>
<td>37.02</td>
<td>2.37</td>
<td>6.32</td>
</tr>
<tr>
<td>1.50A M</td>
<td>2.54</td>
<td>0.71</td>
<td>0.13</td>
</tr>
<tr>
<td>1.50A N/M</td>
<td>31.32</td>
<td>41.11</td>
<td>92.84</td>
</tr>
</tbody>
</table>

* (H.S. = Highly Susceptible, M = Magnetic, N/M = Non Magnetic). (e.g. 0.25A M = Magnetic material at 0.25 Amp field current).

Separation of quartz in Non Mags product by bromoform (S.G. 2.85) indicated Heavy Mineral content of 80.30%. The grade of Tin in this was 51.2% Sn.

TABLE NO. 3.5

RECLEANER TABLE CONS - GRADE 46.6% Sn

<table>
<thead>
<tr>
<th>PRODUCT</th>
<th>WT.% DIST.</th>
<th>% Sn</th>
<th>DIST.Sn%</th>
</tr>
</thead>
<tbody>
<tr>
<td>0.0A H.S.</td>
<td>4.72</td>
<td>1.6</td>
<td>0.19</td>
</tr>
<tr>
<td>1.5A M</td>
<td>26.60</td>
<td>6.1</td>
<td>3.59</td>
</tr>
<tr>
<td>1.5A N/M</td>
<td>68.68</td>
<td>63.2</td>
<td>96.22</td>
</tr>
</tbody>
</table>

TABLE NO. 3.6

RECLEANER TABLE TAILINGS - GRADE 10.2% Sn

<table>
<thead>
<tr>
<th>PRODUCT</th>
<th>WT. % DIST.</th>
</tr>
</thead>
<tbody>
<tr>
<td>H.S.(0.0)</td>
<td>13.57</td>
</tr>
<tr>
<td>0.6A M</td>
<td>52.62</td>
</tr>
<tr>
<td>1.2A M</td>
<td>5.24</td>
</tr>
<tr>
<td>2.4A M</td>
<td>1.89</td>
</tr>
<tr>
<td>2.4A N/M</td>
<td>26.68</td>
</tr>
</tbody>
</table>

The non magnetic product was separated in Clerici Solution (4.05 S.G.). This gave 65.76% +4.05 S.G. heavy mineral. The grade of Tin in this was 50.2% Sn.

This represents a recovery of 86.3% of the contained Tin in the Recleaner tailings.
3.1.4 Gravity Separation - Tantalite

Considerable difficulty was experienced with the assaying of Tantalum and Niobium by our referred laboratory, Australian Laboratory Services of Balaclava Street, Woolloongabba. Some confirmatory assaying was carried out by A.M.D.E.L.

Even so, there was little encouragement in the laboratory sample. However, there was found to be a number of heavy mineral lumps in the drum of +3mm oversize from the large scale washing testwork.

The material was then jigged in a manual screen jig (a "Mexican Jig"). This produced a concentrate which was further upgraded in Clerici solution. Results are as follows:-

<table>
<thead>
<tr>
<th>Description</th>
<th>Weight</th>
</tr>
</thead>
<tbody>
<tr>
<td>Total weight +3mm material</td>
<td>276 Kg</td>
</tr>
<tr>
<td>Concentrate produced by two stages of jigging</td>
<td>0.630 Kg (630 grams)</td>
</tr>
</tbody>
</table>

The remainder was light material and discarded. The concentrate was then upgraded in 4.05 S.G. Clerici solution:-

<table>
<thead>
<tr>
<th>Description</th>
<th>Weight</th>
</tr>
</thead>
<tbody>
<tr>
<td>The concentrate weighed</td>
<td>0.189 Kg (189 grams)</td>
</tr>
<tr>
<td>The remainder weighed</td>
<td>0.441 Kg (441 grams)</td>
</tr>
</tbody>
</table>

The -3mm +1mm material was also jigged:-

<table>
<thead>
<tr>
<th>Description</th>
<th>Weight</th>
</tr>
</thead>
<tbody>
<tr>
<td>Total weight -3mm +1mm material</td>
<td>85 Kgs</td>
</tr>
<tr>
<td>Concentrate produced</td>
<td>12.9 Kg</td>
</tr>
</tbody>
</table>

The grade of this concentrate (0.058% Ta) was not encouraging to further work.
3.2 SUMMARY

3.2.1 Summary of Recovery of Tin

1. Washing Screening
   \[ \frac{280.3}{364} = 77\% \]

2. Gravity Separation
   \[ \frac{256}{280} = 91\% \]

3. Magnetic Separation
   \[ \frac{85\%}{\text{of cleaner table concentrate, as indicated in tests.}} \]
   \[ = \frac{217}{256} \]

4. Total Recoverable Tin
   \[ \frac{217}{364} = 59.6\% \]

From 1 tonne of feed we could expect:
\[ 217 \times \frac{1000}{729} = 297.7 \text{ grams} \]
say 300 grams

3.2.2 Summary of Scavenging of Tantalite

Therefore it can be assumed that of the 729 Kgs total sample, 189 \times 0.85 grams of concentrate is recoverable by multi-stage jigging, and the grade of this concentrate is 20.5\% Ta (25.03\% Ta_2O_5)
19.2\% Nb (27.46\% Nb_2O_5)
5.03\% Sn
(Recovery of 85\% of this mineral is assumed above.

1 tonne of this feed would therefore yield 220 grams of this concentrate.
3.2.3 In Summary of both Tin and Tantalum

It can be anticipated that 1 ton of feed would yield 220 grams of Tantalum concentrate (25.03 Ta$_2$O$_5$, 27.46% Nb$_2$O$_5$), and 300 grams of Tin in tin concentrate (approx. 60% Sn).
4. ECONOMICS

4.1 VALUE OF THE ORE

The concentrates can be valued in three ways, as follows:

**METAL BULLETIN INDICATIONS**

1. **Columbite Ores**
   
   Mixed Oxides \((\text{Nb}_2\text{O}_5 + \text{Ta}_2\text{O}_5)\) 10:1
   
   on 65% basis sells for $US 4.00 per lb. of Pentoxide.

   This concentrate contains 52% mixed oxides and ratio is less than 10:1. This is hardly saleable on this basis.

2. **Tantalite Ores**
   
   25/40% \(\text{Ta}_2\text{O}_5\) sells for $US 80.00/lb Pentoxide on a 30% basis.

   We have 25% \(\text{Ta}_2\text{O}_5\).
   
   The value is $US 20.00/lb concentrate less deductions say, $15.00/lb C.I.F.
   
   Take off 15¢ for C.I.F.
   
   The net worth is $US 14.85/lb.

   The value of the feed is
   
   \[
   \frac{220}{453} \times 14.85 \text{ $US/tonne} = \text{ $US 7.21/tonne.}
   \]

   In Aust. currency
   
   \[
   = $6.50, \text{ say}
   \]

   This assumes the ore is acceptable and that the Columbite will compensate for its impurity penalties.
3. **Tin Ores**

60% Sn concentrate will sell for £7,000 per tonne of contained Tin less a refining charge of £300.00/tonne concentrate.

The value of concentrate is calculated as follows:

- 1 tonne of 60% concentrate
- less refining etc.

\[
\begin{align*}
&= £7,000 \times 0.6 \\
&= £4,200/tonne \\
&£300 \\
&£3,900
\end{align*}
\]

In Australian currency @ $A2 = £1

= $7,800.00

Concentrate produced/tonne of feed treated

\[
\begin{align*}
&= \frac{300}{0.6} \\
&= 500 \text{ grams} \\
\text{Value} &\quad = \frac{£7,800}{2,000} \\
&\quad = £3.90/tonne of ore treated.
\end{align*}
\]

**Total Value of Feed**

Total value of recoverable metal in feed is

\[
\begin{align*}
&= £6.50 + £3.90 \\
&= £10.40
\end{align*}
\]

Since the feed quoted is -" material and constitutes 58% of the ore, the value of the feed above will have to be discounted to give a value for the ore.

**The Value of the Ore would be:**

\[
\begin{align*}
&= £10.40 \times 0.58/tonne \\
&= £6.03/tonne \\
&\quad \text{say, less approx.10% Safety} \quad = £5.50/tonne mined.
\end{align*}
\]
4.2 ESTIMATE OF CAPITAL COST

A detailed circuit has not been drawn up, but equipment required with estimated cost is as follows:

<table>
<thead>
<tr>
<th>Description</th>
<th>Cost</th>
</tr>
</thead>
<tbody>
<tr>
<td>Ore Screening and Washing Section</td>
<td>$20,000</td>
</tr>
<tr>
<td>4 x 4 Jig</td>
<td>$4,000</td>
</tr>
<tr>
<td>1mm Screen</td>
<td>$1,000</td>
</tr>
<tr>
<td>1 4/3 Pump, Bin, Pipes, &amp;c.</td>
<td>$3,500</td>
</tr>
<tr>
<td>1 Cyclone</td>
<td>$1,500</td>
</tr>
<tr>
<td>4 Spirals</td>
<td>$8,000</td>
</tr>
<tr>
<td>1 Shaking Table</td>
<td>$6,000</td>
</tr>
<tr>
<td>1 Magnetic Separator (secondhand)</td>
<td>$5,000</td>
</tr>
<tr>
<td>Buildings and Structures</td>
<td>$6,000</td>
</tr>
<tr>
<td><strong>Infrastructure</strong></td>
<td><strong>$ 55,000</strong></td>
</tr>
<tr>
<td>Water Supply</td>
<td>$10,000</td>
</tr>
<tr>
<td>Electricity Generation</td>
<td>$25,000</td>
</tr>
<tr>
<td>Workshop Facility</td>
<td>$10,000</td>
</tr>
<tr>
<td>Messing and Accommodation</td>
<td>$50,000</td>
</tr>
<tr>
<td><strong>Mobile Equipment (secondhand)</strong></td>
<td><strong>$ 50,000</strong></td>
</tr>
</tbody>
</table>
| **Total Capital Outlay**                                 | **$200,000**

4.3 ESTIMATE OF OPERATING COST

Wages for 8 men, all up cost $10.00/hr = $3,200/week
Materials, fuel supplies etc. = $4,000
Say, $7,200
Say, $7,500/week
5. SUMMARY

The combined cost for the operation would be:-

Operating $7,500/week

Capital is estimated @ $200,000/60,000 / tonne of ore

@ 300 tonnes/day (1500/week) = \( \frac{200,000}{60,000} \times 1500/\text{week} \)

= $5,000/week

This is high because of the small orebody.

Total weekly cost = $12,500

The value of the ore is 1500 x $5.50 = $8,250

The effective loss is $4,250/week

The operation would not be economic.