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A PROGRESS REPORT ON MINING LEASES 32M/81 & 30M/74

MATHINNA TASMANIA

FOR PERIOD 15th JUNE 1984 - 15th SEPTEMBER 1984

PREPARED FOR

EPOCH MINERALS EXPLORATIONS NL

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Report 1005/B
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1. INTRODUCTION

1.1 This report covers exploration activity on Mining Leases 32M/81 and 30M/74 at Mathinna in Eastern Tasmania, for the period 15th June - 15th September 1984.

1.2 The leases in question are held by "Tasminex NL". Epoch Minerals Exploration NL took up an option to acquire these 2 leases in May 1984. Exploration in the first quarter (see report 1005A) concentrated on the re-drilling of tailings on this lease and adjoining lease 3M/83 then held by Mr J. Taylor and others.

1.3 During the latest quarter the results of the drilling have been assessed; the reserves available have been calculated and metallurgical test work carried out.

1.4 The results overall were fairly disappointing and suggested that a price rise of some $100 - $200 per ounce ofgold would be necessary before the retreatment of tailings on 32M/81 and 30M/74 and adjoining 3M/83 would be viable, with present technolegy.

1.5 At this stage there are no plans for bulk testing the deposit, until at least some encouragement is gained on a laboratory scale.

2. GEOLOGY & RESERVES OF TAILINGS

2.1 The geology of the Mathinna Goldfield and the general setting of the mineralization within the area has already been described in an report 1005 and it is not re-included herein.

2.2 In particular the shafts associated with the New Golden Gate Mine occur on the leases in question. Total production from this mine was 267,140 tonnes of quartz, producing 222,755 of gold. Tailings from this mining operation now occur at surface on the two leases in question and an adjoining lease 3M/83.

2.3 The tailings were original drilled by the Tasmanian Dept. of Mines in 1948 (Hughes). Check drilling was carried out by Epoch Minerals in the March - June quarter 1984. The relative position of each of the Epoch and Hughes boreholes are shown on Plate 1. Plate 1 also covers Lease 3M/83. In the Hughes programme one sample was assayed for the whole hole. In the Epoch programme samples were split at one metre intervals and seperately assayed. All grades quoted on Plate 1 are in gms/tonne, and depths are in metres.

2.4 The reserves were calculated by constructing representatives squares with one of the existing drillholes at each corner of the square. The depth from the 4 holes was averaged and calculated by the area of the square. This value was considered representative of the volume of tailings available within the square. The tonnage was calculated by assuming a density of 1.8 gms/cc.

2.5 The representative grade was calculated by multiplying grade by depth for each hole, adding them together around the square and then dividing the sum by the total depth of the 4 holes.

2.6 The calculated reserves and grade for each small block were then added together. In preforming this task the reserves were divided into 3 groups:

a) higher grade
b) medium grade
c) lower grade
On this basis, the reserves of tailings on the Tasminex's leases have been calculated to be:
a) higher grade 21,000 tonnes at 2.0 gms/tonnes
b) medium grade 40,000 tonnes at 1.6 gm/tonnes
c) lower grade 118,000 tonnes at 1.2 gm/tonnes
If we mix the higher grade (a) and medium grade (b) together then we have a composite of 61,000 tonnes at 1.74 gm/tonnes.

2.7 On the adjoining half of the dump, (until recently held by Taylor), the calculated reserves are:
a) Higher grade 31,000 tonnes at 2.0 gm/tonnes
b) Medium grade 37,000 tonnes at 1.65 gm/tonnes
c) Lower grade 60,000 tonnes at 1.25 gm/tonnes
Again mixing the higher grade (a) and medium grade (b) together then we have a composite of 68,000 tonnes at 1.85 gm/tonnes.

2.8 It really would not be feasible to try to work only half of this dump,(ie. say the Tasminex Leases), without also being able to work Taylor's ground as well. Therefore a preliminary feasibility study is based on ultimately having access to both halves of the dump.

2.9 Combining the higher and medium grade from both halves, the reserves are 129,000 tonnes at 1.8 gms/tonnes with 178,000 tonnes of low grade material at 1.22 gms/tonnes.

3. METALLURGICAL RESULTS

3.1 A number of organisations have investigated the likely recovery of gold from the tailings. All known previous investigations were summarized in our last progress report.

3.2 We have had our own investigations carried out by Robertson Research and their report is attached. The material supplied to Robertson Research for their investigations, comprised a representative split of all the recent holes drilled on the Tasminex Leases by Epoch.

3.3 Robertson Research head grade assay of 1.22 gms/ tonne indicate that the sample given was possibly more representative of the lower grade, than higher grade material. Their work indicated that by simple cyanide leaching, over 24 hours, the recovery was 43%. We expect that this could be improved marginly by leaching over a longer period, (say 72 to 96 hours), but the ultimate recovery is unlikely to exceed 50%. From assays of the leach residual we, susspect that there could be up to 0.6 gms/tonne that cannot be extracted by conventional cyanide leaching.

3.4 It would be unrealistic to anticipate recoveries from the higher grade material in excess of 1.8 - 0.6 = 1.2 gm/tonne. With careful and exacting work, we believe that ultimately a recovery of about 1.0 gm/tonne could be achieved? However, this figure is uncertain without a pilot plant study.

3.5 Foe the matter of convenience, we have used a recovery figure of 1.0 gm/tonne, below, for the purpose of evaluating the potential economics of retreating the dump.

4. PRELIMINARY INCOME & COSTING ESTIMATED

4.1 The capital costs associated with a small heap or vat leaching operation, including pad or vat preparation; leach sump and CIP tanks or zinc boxes, plus a loader, site vehicle and buildings etc, would be in the order of \$250 - 300.000. This does not include gold room facilities if carbon in pulp was used. This would have to be included or else the carbon would have to be shipped to the company's facility of Canbelego, near Cobar NSW. Such a facility locally would all another \$50,000 to the capital price.

4.2 Current operating costs for a heap or vat leaching operation of say 30,000 tonnes per annum are in the order of \$10 per tonne (TOTAL \$1.29 Million).

4.3 The anticipated income assuming a gold price of \$A400 per ounce would be \$12,90 per tonne (Total \$1.66 million).

4.4 On this basis the estimated profit per tonne from this operation would be

a)	Gross Income	\$12.90
b)	Minus operation costs	- \$10.00
c)	Minus Amortization	- \$ 2.32
	Profit	\$ 0.58 per tonne

or a total of only \$75,000 over the life of the higher grade reserves.

4.6 The above figures suggest that the deposit is only margina; at a gold price at \$A400 per ounce. An estimated profit of around \$1½ Million after total amortization would be the minimum required to justify development of a plant on this deposit. This is a profit of around \$3.87 per tonne. To achieve this, and income of \$16.77 per tonne would be required. At a recovery of 1 gm/tonne, the gold price required would be around \$520 an ounce. The low grade material would require an even higher gold price to be potentially viable.

CONCLUSIONS

1. The reserves of tailings available for retreatment at Mathinna have been calculated to be 129,000 tonnes at a head grade of 1.8 gms/tonnes with a further 178,000 tonnes of low grade material available at a head grade of 1.22 gms/tonnes.

2. Metallurgical testwork is likely to show that recoveries in excess of about 1.0 gm/tonnes would be unlikely for the higher grade material without grinding. Grinding is then expected to improve recoveries marginally, not sufficiently to justify the capital costs involved.

3. The capital cost of a simple heaper vat leaching operation is likely to be in the order of $250,000 to $300,000. Allowing for total amortization of capital costs over the life of the retreatment operation, the operation would probably only break-even at a gold price of $400 per ounce.

4. A stabilized gold price between $500-$600 per ounce would be required before retreatment could be considered viable. However, inflation between 1984 and the year that this occurs, must also have to be taken in consideration, in considering the viability of the project.

RECOMMENDATIONS

1. The leases be held with the view to undertaking a full scale feasibility study when gold stabilizes at $500 per ounce.

2. That during the interim, period exploration activity on the leases concentrate on the underground deposits. This work could involve electrical geophysics and drilling.

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ROBERTSON RESEARCH (AUSTRALIA) PTY. LIMITED

PROJECT NO. 2338

MEMORANDUM NO. 1375

'PRELIMINARY METALLURGICAL INVESTIGATION OF A SAMPLE OF GOLD TAILINGS FROM MATHINNA, TASMANIA'

by

K. Andrews, B.Sc., B. Econ.

August, 1984

CLIENT:

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ATTENTION: Rob Murdoch

SAMPLE DETAILS:-

One bag of damp sand-clay material of approximately 30kg weight and marked Mathinna OS/OE was received on 11th May, 1984. The sample was stated to be a composite of drilling samples of the Mathinna gold tailings dumps and was representative of the total tailings dumps.

The sample was air dried, screened at 1mm with oversize being lightly crushed and rolled to break up soft lumps then rescreened at 1mm. The oversize was weighed and portion submitted for assay. The undersize was weighed then repeatedly riffle split to provide 1000g portion for testwork and a second portion which was submitted for assay.

TESTWORK:-

A 1000g representative portion of the -1mm fraction of the tailings was pulped with tap water to 49% solids plus 2g hydrated lime and 2gNaCN and agitated by rolling bottle for 8 hours. Two further 1g additions of lime were made at ½ hour and 2½hours. At 16 hours the leach pulp was well mixed then split into two approximately equal portions. 10g of screened activated coconut carbon (PICA 8 x 16) was added to one portion of pulp and both portions were agitated for a further 16 hours.

Measured volumes of solution samples were taken at 1 hour, 2 hours, 4 hours, 8 hours and 24 hours for assay and to monitor protective alkalinity and 'free' cyanide levels. Pulp densities were measured by weighing slurries and adding water as necessary. The final pulp containing the carbon was screened to recover the carbon for assay and both pulps were filtered, washed, dried, weighed, mixed, split and ½ spilts submitted for assay.

Following receipt of initial results the ½ split of the carbon-in-pulp residue from the above test was leached with conc. HNO_3, conc. HCl, and water (in approx. 1:1:1 ratio) for approximately ½ hour at boiling point then filtered and washed. Filtrate plus washings were diluted to 1000mls and the filter cake dried and weighed. The acid leach residue and acid leach solution were then submitted for assay.

RESULTS:-

Results of screening sample at 1mm were as follow:-

	Weight %	Au Assay	Au distribution, percent
+1mm fraction	11.2	0.54g/t	5.0
-1mm fraction	88.8	1.31g/t	95.0
Total Feed (calc.)	100.0	1.22g/t	100.00

The results of cyanide leach test are given below. Note that due allowance has been made for the gold removed in the intermediate solution samples and for the slight variation in the pulp density between the splits of the leach slurry at 8 hours. The calculated head assay given is based on the weighted average of the two residue assays, the carbon assay, the 24 hour leach and CIP solution assays plus the gold removed in the intermediate samples.

Results of the Cyanide leach test are given in Table 1.

The acid leach of the CIP residue yielded the following results:-

	Vol/Wt mls/g	Au Assay g/t	Au Extraction	
			g/t feed	percent
Acid Leach Soln.	1000	0.095	0.39	42
Acid Leach Residue	229.6	0.57	0.55	58
CIP Residue (Calc)	239.6	0.94		
CIP Residue Assay		0.83		

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TABLE 1

RESULTS OF CYANIDATION LEACH OF -1MM MATHINNA COMPOSITE SAMPLE

	Weight Volume g/mls	Au Assay on Solid/Soln. g/t	Au Distribution on -1mm Feed Basis		Au Distribution on Total Feed Basis		protective alkalinity as % CaO	'free' cyanide as % NaCM	Reagent Consumption kg/tonne -1mm Feed	
			g/t	Percent	g/t	Percent			CaO	NaCM
Leach Soln @ 1 hr.	1045	0.36	0.38	28	0.33	26				
Leach Soln @ 2 hr.	1045	0.34	0.38	27	0.33	26	nil(pH 10.1)	0.135	2.0	0.6
Leach Soln @ 4 hr.	1045	0.43	0.49	35	0.43	34	0.014	0.14	2.5	0.6
Leach Soln @ 8 hr.	1045	0.43	0.51	37	0.45	35				
Leach Soln @ 24 hr.	514	0.48	0.57	42	0.51	40	nil(pH 10.1)	0.09	2.6	1.0
CIP Soln @ 24 hr.	540	0.010	0.48	35	0.43	33	0.002	0.08	2.6	1.1
Carbon @ 24 hr.	9.86	20.5					(pH 10.3)			
Leach Residue	522.8	0.86	0.86	63	0.82	65				
CIP Residue	473.4	0.83	0.83	61	0.80	63				
Calc. Head	996.2*	1.37		1.28						
Assay Head			1.31		1.22					

* 1000g Feed sample was air dried only and contained approx. 0.4% moisture

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CONCLUSIONS AND RECOMMENDATIONS:-

The assays and indicated gold extractions of the leach solutions may not be highly accurate, as indicated by the discrepancy between the calculated feed assays and actual feed assays. Accordingly estimates of extractable gold content are based on fire assays of solids which are believed to be more accurate.

The material appears to be of too low a gold grade (1.22g/t) to justify the cost of grinding in any treatment operation.

Screening of the material at an early stage of treatment would probably be required and the testwork indicates that the tramp oversize material accounts for only a small proportion of the feed and is much lower in gold grade than the feed.

The cyanide leach test yielded one residue (from C.I.P. part of the test) assaying 0.83g/t Au, indicating that a gold extraction, on a total feed basis, of approximately 0.42g/t Au or approximately 34 percent is achieveable by cyanidation without grinding. Leach times required are of the order of 8 hours and lime and cyanide consumptions are relatively modest at approximately 2.6kg CaO per tonne and approximately 1.0kg NaCN per tonne.