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REPORT TO DEPARTMENT OF MINES, TASMANIA

ON THE BEACONSFIELD GOLD TAILINGS PROJECT

BY BMI MINING PTY. LIMITED

JUNE, 1980 TO MAY, 1981

OPEN FILE

D. J. SMITH
MINING ENGINEER

22 MAY, 1981

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3. Metallurgical Testwork

1. INTRODUCTION

This report serves to inform the Director of Mines of the work being performed by BMI Mining Pty. Limited, on the Beaconsfield gold dumps.

The tailings from the Tasmania Mine lie in and adjacent to the Middle Arm cove of the Tamar River. The tailings in the tidal area are held by BMI Mining Pty. Limited under four mineral leases. A fifth lease has not been granted, due to a complication in compensation to owners of private land within the lease.

An exemption of labour conditions has been granted by the Director, for the period 1 December, 1980 to 1 June, 1981. This report fulfils the stipulation, whereby a detailed report was required at the end of the six months exemption, to ascertain the necessity of a further exemption.

The report covers the twelve months ending May, 1981. The scope of activity during that period has concentrated on refining the metallurgical processing. A series of tests on the dissolution phase were carried out by the Department of Mines laboratory, in Launceston in 1980. Present testwork is examining the suitability of the carbon-in-pulp process of gold recovery. This testwork, if successful, will be followed by detailed flowsheet and plant design in early 1982. Environmental approvals would then be sought, to allow plant construction and subsequent mining and processing.

2. LEASES

The four mineral leases 61M/73 (21 ha.), 62M/73 (11 ha.), 63M/73 (15 ha.) and 96M/74 (100 ha.) were granted on 1 August, 1976 for a period of twenty-one (21) years.

Lease 969P/M has not been granted, but remains in application status. This lease application covers the surface stockpile of tailings adjacent to lease 63M/73. The area applied for covers private land and some Crown land, and requires compensation to be paid to the owners of the private lands. The private land comprises part of Lot 10439 and part of Lot 7300. A deeds search in 1977 found that certain owners of these two lots could not be traced. Section 73 of the Mining Act, 1929 requires compensation be secured with the owner of private lands within a lease. BMI has instructed its solicitors in Tasmania, Shields Heritage Stackhouse and Martin, to initiate action under Section 74 of the Mining Act, 1929. This allows the Warden's Court to determine the appropriate compensation, to obtain the order required to enable the lease to be granted. Our solicitors are determining the current land ownership as the first stage of investigation. A Warden's Court hearing will then be sought.

3. METALLURGICAL TESTWORK

3.1 Introduction

In May, 1980, BMI commissioned Mitchell Cotts Projects (Australia) Pty. Limited to appraise the previous metallurgical testwork. Mr. Colin Harris, of Mitchell Cotts devised a comprehensive test programme to evaluate various procedures (pre-washing; cyanidation of untreated ore; sulphide flotation and cyanidation; roasting sulphide concentrate) on representative samples. Gold dissolution with time was to be examined, to determine the effect of reprecipitation of gold out of the cyanide solution.

For the first time, the dump was sampled and tested as three distinct areas, based on location and grain size. The coarse surface stockpile was treated as one sample. The tidal flats were divided in two, with the northern, downstream area being finer and richer in gold values. The enclosed map shows sampling locations. These areas represent approximately equal tonnages of tailings.

The programme was decided between Colin Harris and Hugh Wellington, Chief Metallurgist of the Department of Mines Laboratories in Launceston. The testwork was performed in these laboratories, and commenced in June, 1980, immediately following the bulk sampling. Minor breakdowns and laboratory workload delayed the programme, which was completed in October, 1980. The results are published in Department of Mines Report R792, October, 1980. A copy of R792 and the Mitchell Cotts appraisal are included in this report. A mineralogical report on the carbon float concentrate from R792 is also included (Pontifex report No. 2949, July, 1980).

3.2 Results

The scope of the R792 testwork dealt solely with the gold-cyanide dissolution phase, as did most of the previous Department of Mines tests. The work was to confirm the past work, and to identify the optimum processing techniques suited to a small mining operation.

R792 testwork concentrated on two mainstreams:

Test A: cyanidation of material ground to minus 75 µm.

Test B: sulphide flotation of material ground to minus 75 µm, and cyanidation of the various products.

Gravity concentration methods of tabling and jigging of ground material were not considered, because of poor recoveries from cyanidation of such concentrates in earlier test R655 pt II. Also cyanidation of unground, as received material was not considered, because past tests (R697 and R717) showed gold dissolution to be less than that of ground material, and not greater than 54%, though reprecipitation was never considered in these tests.

The straight cyanidation tests (A) maintained the three individual sample areas. The samples were grouped in the sulphide flotation tests (B). In both cases, by sampling at three hourly intervals, the premise of reprecipitation of dissolved gold was investigated. This previously unexamined aspect showed in Test A that the optimum cyanide contact time was nine to twelve hours. Reprecipitation occurred after this time, and one test showed re-dissolution of gold after twenty-four hours. These tests ceased at twenty four hours, thus the dissolution/reprecipitation cycles were not examined beyond the first dissolution peak. Test B, however, progressed for only fifteen hours, and thethree hourly samplings indicated that the dissolution peak may not have been reached. But no significant increases in dissolved gold values could be expected, as rates of dissolution had generally eased.

Test A consisted of four strands. In addition to the differences in physical location of three samples, a fourth sample was subjected to flotation to remove charcoal. The trends that emerged were:

i) optimum cyanide contact time for the three samples was nine to twelve hours.

ii) there is a pronounced relationship between tailings grain size, gold head grade and gold recovery. As one progresses downstream towards the finer tailings area, the gold values are higher, and give higher recoveries.

iii) an advantage of preconditioning by carbon flotation, may conservatively increase recoveries by 7.8%. In this case, the average gold recoveries become 69.9%.

Test B involved sulphide flotation in four stages. Cyanidation of all concentrates and tails gave poor gold recoveries. Calcination of the final sulphide concentrate did not markedly improve gold recovery. Any flotation circuit would require cyanidation of the tailings, because the final concentrate, while having a gold grade of 51.7 gm/tonne, contained only 20.5% of the total gold. Flotation produced two concentrates and three tailings, each containing approximately 20% of the total gold. The best result, from cyanidation of all products, was 44.7% gold recovery. The conclusion was that sulphide flotation is not amenable to the Beaconsfield material.

Neither series of tests tackled the problem of contaminants, Hg, Sb, and Cu in the process effluent.

3.3 Conclusions

The most efficient extraction of gold from a feed of the Beaconsfield material will require grinding to minus 75µm, carbon flotation and controlled cyanide contact. The quantity of gold going into solution, and reagent consumption have been calculated on the arithmetic average of individual sample results. Laboratory figures are 69.9% gold recovery, with cyanide and lime consumption at 6.2kg/t. and 2.3kg/t. respectively. However, the preflotation indicated a saving in reagent usage.

The plant circuit suggested by C. Harris was cyanidation under the conditions mentioned above. Harris then suggested two stage filtration for recovery of the pregnant cyanide solution, followed by zinc dust precipitation.

However, BMI wish to consider the carbon-in-pulp method of gold extraction. This technology of absorption/desorption/electrolysis, has potential advantages of savings in plant capital, operating expenses, and increased gold recoveries. Plant simplicity is achieved by the absence of the two filtration stages prior to precipitation. Once the gold is absorbed into the activated carbon, then the carbon may be safely transported to another location for gold stripping and reactivation. A strong advantage is the mechanism of the carbon absorbing the gold from the cyanide solution. As the gold goes into solution, the activated carbon absorbs it, thus denying the other carbon contaminants which are responsible for the reprecipitation in test A. In addition, the removal of gold from the cyanide solution may be expected to drive dissolution chemical action forward, resulting in an increase in recovered gold in the order of five percent. The gold solution peaks which occur at different times (nine and twelve hours) are thus never reached.

3.4 Present Programme

The present testwork is aimed at determining the applicability of the carbon-in-pulp process. Tests in the past (R675 and R697) showed fair recoveries of gold from cyanide solution, but are not considered indicative of the method's potential.

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A series of preliminary tests are being carried out in May, 1981 by the metallurgical testing company, Miller-Kappes Company of the United States. This company, headed by Mr. D. Kappes, has a large established laboratory in Reno Nevada for gold metallurgy testwork. Currently the company is supervising a carbon-in-pulp plant being erected at the Meekatharra gold dumps in the Murchison Goldfields, W.A. Kappes is doing this work in his role as principal consultant for Argosy Gold Mines N.L.

Mr. Kappes has studied all the previous testwork, and has formulated two stages of testwork. The first stage is aimed at indicating the maximum overall percent recovery that could be obtained in a very conservatively designed system. If this result is encouraging, then full scale tests will follow.

The bulk sampling was completed in March, 1981. The dump has been split into three areas, on the lines of R792 testwork. The sampling points were almost identical, and are located on the accompanying map. An additional sample was taken to include the plant matter growing in the tidal flats, to determine its degree of carbon activation. A sample from the creek bed was also included. The first stage work commenced in May, 1981, and is due to be completed by the end of the month. The cost of this testwork is $1,400 (exclusive of sampling and airfreight costs).

The second stage testing will be much more detailed, and will take three months at a cost of $72,000 (exclusive of sampling and airfreight costs). The extent of these tests will be:

a) grinding times followed by C-in-P leaches.

b) pre-oxidation to eliminate organic carbons and soluble polysulphides, both of which can inhibit gold dissolution and carbon absorption.

c) optimum chemical composition and leaching temperature.

d) build up of base metals in solution (Hg, As and base metals).

e) methods of base metals removal to environmentally acceptable limits (particularly Cu).

f) effect of sea water in absorption/desorption behaviour of activated carbon, and the electrolytic recovery section.

3.5 Future Work

Certain additional tests may be required as the programme develops. However, this testwork has been designed to provide sufficient data for a complete plant design. The ensuing stages of plant design, construction and commissioning are estimated to require twelve months. The cost of these subsequent stages will depend on the plant specifications, but are expected to be in the order of $1.5 million.

Additional exploration is planned to delineate the extent of the gold bearing tailings in lease 96M/74. In addition, check drilling will confirm the 1973 augering of the tidal leases 61M/73, 62M/73 and 63M/73.

4. ENVIRONMENTAL CONSIDERATIONS

Before BMI's four mineral leases were granted in 1976, an environmental impact statement was prepared by the Port of Launceston Authority. Several aspects of this study need to be revised. The original proposal was to treat the dump material with no size reduction, then use the tailings in foreshore reclamation. However, the most recent testwork requires grinding of all the material to minus 75 microns. The solids portion will be too fine for the previous reclamation scheme, and new disposal methods are being investigated. Previous tests have shown that the liquid refuse contains excessive copper. Liquid emissions must conform to the standards set in the Environment Protection (Water Pollution) Regulations, 1974. The second stage Miller-Kappes testwork will address the whole spectre of base metals in solution.

Additional changes are envisaged in both the winning and processing methods. The 1975 submission proposed excavation by a land-based Sauerman scrapper type operation for the river work. But present thinking is towards a cutter-suction dredge, to enable more precise extraction of the variable thickness tailings. The current proposal, involving a ball milling circuit, will need to be examined with regard to noise emission.

The Department of the Environment has issued to BMI a licence (No. 1289) to operate at Beaconsfield. The matter of changed working conditions has been discussed with this Department. They require a submission to obtain a modification to the existing operating licence. Changes in working methods will not be decided until the detailed Stage II testwork has been completed. Applications will be placed with the Department of Environment at the appropriate time, when metallurgical and plant design details are completed.

5. EXPENDITURE

The figures below are the expenditures by BMI Mining Pty. Limited on the Beaconsfield leases. These figures include testwork and associated activities, but exclude BMI Mining staff costs or administration costs.

Costs for 12 months, June, 1980 to May, 1981 = $ 22,000

Budgetted for 12 months, June, 1981 to May, 1982 = $ 116,000

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APPENDIX I

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DEPARTMENT OF MINES—TASMANIA

TELEPHONES:
Metallurgical Research .. .. } 44 2431-2
Laboratory .. .. .. } (2 lines)
Mines Inspection .. .. ..
Explosives & Inflammable Liquids

LAUNCESTON OFFICES
287 WELLINGTON STREET
SOUTH LAUNCESTON 7250

15th October 1980

Mitchell Cotts Projects (Aust) Ltd,
246 St. Georges Terrace,
Perth W.A. 6000

Attention Mr. C. Harris.

R792

Dear Col,

Please find herewith above report. The samples confirm previous sampling grades and the test work the recoveries obtained in R697, R717 & R763, and also confirm the poor recoveries from sulphides.

RECEIVED
20 OCT 1980

Yours faithfully,

(H. K. Wellington)
Chief Chemist & Metallurgist.

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DEPARTMENT OF MINES—TASMANIA

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LAUNCESTON OFFICES
287 WELLINGTON STREET
SOUTH LAUNCESTON 7250

TELEPHONES:
Metallurgical Research
Laboratory
Mines Inspection
Explosives & Inflammable Liquids
44 2431-2 (2 lines)

R792

14th October 1980

GOLD RECOVERY TESTS ON BEACONSFIELD TAILINGS TAKEN FROM MIDDLE ARM.

Introduction

Three samples were submitted by B.M.I. Mining, taken from three areas, fine area, medium area, and coarse area. The fine and medium areas were in the tidal flats of Middle Arm, while the coarse area was "The Stockpile", situated on the northern side of Middle Arm Creek where it flows into the tidal flats.

Details of the three samples were as follows:-

Description	Reg. No.	Site No.	Wet Mass (kg.)
Fine Area - Sample 1	801579	1, 2, 3, 8, 9	68.1
Medium Area - Sample 2	801580	4, 5, 6, 7, 10	63.2
Coarse Area - Sample 3	801581	11, 12, 13, 14, 15, 16, 17	95.7

The method of treatment of the samples was devised in consultation with Mr. C. Harris of Mitchell Cotts Projects (Australia) P/L.

Sampling

Each sample, in its original wet condition was pulped with the addition of water and wet screened on a 457mm dia. Sweco screen fitted with a 1.24mm screen cloth. The screen undersize was pumped to a Galigher automatic sampler operating continuously. This initial sample was fed to a small rotary distributor with six outlets, providing six samples.

The screen oversize from each sample is described in the following table:-

	+ 1.24mm Fractions		
Sample Reg. No.	801579	801580	801581
% Mass	<0.4 %	<0.1 %	0.3 %
contents	mostly shell charcoal small stones	shell charcoal root fragments wood chips	small stones charcoal twigs some shell

One of the samples from the rotary distributor was sized and a gold and sulphur distribution determined in each case.

All samples were kept in the pulped condition.

Sizings with Gold and Sulphur Distribution.

The sizing, gold and sulphur distributions for the three samples are shown in the following tables.

Reg. No. 801579 **Fine Area - Sample 1**

	Mass		Assays		% Distribution			
Size Fraction (µm)	%	%Cum.	Au(g/t)	S(%)	Au	Au Cum.	S	S.Cum.
+ 300	4.8	4.8	2.3	0.37	3.4	3.4	1.9	1.9
+ 212	6.4	11.2	0.8	0.30	1.5	4.9	2.1	4.0
+ 150	10.8	22.0	0.9	0.34	3.0	7.9	4.0	8.0
+ 106	8.9	30.9	1.1	0.47	3.0	10.9	4.6	12.6
+ 75	9.3	40.2	1.6	0.51	4.6	15.5	5.2	17.8
+ 53	6.5	46.7	1.7	0.52	3.4	18.9	3.7	21.5
+ 38	7.4	54.1	1.8	0.67	4.0	22.9	5.4	26.9
- 38	45.9	100.0	5.5	1.46	77.1	100.0	73.1	100.0
Calc. Head	100.0		(3.3)	(0.92)	100.0		100.0	

Reg. No. 801580 **Medium Area - Sample 2**

	Mass		Assays		% Distribution			
Size Fraction (µm)	%	%Cum.	Au(g/t)	S(%)	Au	Au Cum.	S	S.Cum.
+ 600	2.4	2.4	2.3	0.34	2.1	2.1	1.1	1.1
+ 425	9.2	11.6	2.2	0.35	7.7	9.8	4.3	5.4
+ 300	12.1	23.7	1.3	0.34	6.0	15.8	5.5	10.9
+ 212	17.7	41.4	1.4	0.35	9.4	25.2	8.2	19.1
+ 180	7.7	49.1						
+ 150	9.3	58.4	1.4	0.47	9.1	34.3	10.6	29.7
+ 106	10.0	68.4	1.7	0.76	6.4	40.7	10.1	39.8
+ 75	6.8	75.2	2.8	1.20	7.3	48.0	10.9	50.7
+ 38	6.8	82.0	4.2	1.47	10.8	58.8	13.2	63.9
- 38	18.0	100.0	6.0	1.50	41.2	100.0	36.1	100.0
Calc. Head	100.0		(2.6)	(0.75)	100.0		100.0	

14.

It was necessary to know the proportions of quartz and limestone present in the three samples for the design of the grinding circuit in a proposed plant for the re-treatment of the tailings.

Accordingly, proportionate quantities were taken from each size fraction coarser than 75 µm for each of the three samples. The +75 µm composite for each sample were then assayed for acid insol. and CO_2. The results were as follows:-

		+ 75µm Composites	
	801579	801580	801581
Acid insol. %	85.2	81.8	85.3
CO_2 %	3.9	4.4	3.6

Reg. No. 801581 **Coarse Area - Sample 3**

Size Fraction (µm)	Mass		Assays		% Distribution			
	%	%Cum.	Au(g/t)	S(%)	Au	Au Cum.	S.	S.Cum.
+ 600	9.8	9.8	1.6	0.36	6.8	6.8	4.8	4.8
+ 500	12.6	22.4						
+ 425	14.5	36.9	1.4	0.34	16.5	23.3	12.7	17.5
+ 355	11.2	48.1						
+ 300	12.2	60.3	1.7	0.41	17.3	40.6	13.2	30.7
+ 250	9.8	70.1						
+ 212	6.9	77.0	1.3	0.57	9.4	50.0	13.0	43.7
+ 150	9.7	86.7	2.1	0.83	8.8	58.8	11.1	54.8
+ 106	4.5	91.2	3.8	1.26	7.5	66.3	7.9	62.7
++ 75	3.0	94.2	5.4	1.61	7.2	73.5	6.8	69.5
- 75	5.8	100.0	10.6	3.86	26.5	100.0	30.5	100.0
Calc. Head	100.0		(2.3)	(0.73)	100.0		100.0	

Individual Cyanidation Tests

Approximately two kilogram samples were taken by using the rotary distributor, from each of the three areas. The samples were each wet ground batchwise in the 250 ml. Siebtechnik bowl and screened on a 75 µm screen. Screen oversize was returned to the Sieb mill.

The presence of charcoal in the samples was cause for some concern, so a second sample of 801580 was similarly ground and subjected to flotation to remove charcoal.

20g of "Limil" to give a pH of 8.8, 0.5 mls of fuel oil and 0.1 ml of M.I.B.C. were used for the carbon float. Rougher flotation took 3 minutes. The rougher concentrate was cleaned for 3 minutes, and the cleaner concentrate was recleaned for 5 minutes. The flotation concentrate amounted to 0.28% mass and appeared to be almost all sulphide. All the flotation tails were combined for cyanidation.

The three samples and the flotation tail were subjected to cyanidation for 24 hours in Denver laboratory super agitators. Sodium cyanide addition was aimed at a 0.2% overall solution strength and the initial lime addition was 0.4g per litre.

50 ml. solution samples were withdrawn after 3, 6, 9, 12, 15, and 24 hours for checking free cyanide content, protective alkalinity, and gold content. Cyanide and lime were added at each occasion to maintain 0.2% NaCN and 0.4 g/l of CaO, and water was added to make up the volume.

The conditions for the four tests and the reagent additions made were as follows:-

Sample No. 976017

Time Hours	Condition		801579	801580	801581	801580 after pre-flotation
	Mass	kg	1.87	2.485	2.02	2.602
	Solution volume	ℓ	3.72	4.20	3.72	4.20
	Pulp density	% solids	33.5	37.2	35.2	38.3
3	Free cyanide	g/ℓ	0.22	0.82	1.16	1.16
	Protective alkalinity	g/ℓ	0.02	0.07	0.06	0.14
	NaCN added	g	-	-	-	-
	CaO added	g	-	-	-	-
6	Free cyanide	g/ℓ	0.16	1.04	1.26	1.10
	Protective alkalinity	g/ℓ	0.01	0.04	0.03	0.16
	NaCN added	g	6.85	4.03	2.75	3.78
	CaO added	g	1.5	1.5	1.4	1.0
9	Free cyanide	g/ℓ	1.76	2.05	2.14	2.39
	Protective alkalinity	g/ℓ	0.03	0.09	0.13	0.33
	NaCN added	g	0.9	-	-	-
	CaO added	g	1.4	1.3	1.0	0.3
12	Free cyanide	g/ℓ	1.73	1.86	2.18	2.20
	Protective alkalinity	g/ℓ	0.04	0.14	0.19	0.35
	NaCN added	g	1.0	0.6	-	-
	CaO added	g	1.4	1.1	0.8	0.2
15	Free cyanide	g/ℓ	1.41	2.00	1.90	1.80
	Protective alkalinity	g/ℓ	0.06	0.21	0.26	0.36
	NaCN added	g	2.2	-	0.4	0.8
	CaO added	g	1.3	0.8	0.5	0.2
24	Free cyanide	g/ℓ	1.57	1.90	1.85	1.68
	Protective alkalinity	g/ℓ	0.11	0.16	0.30	0.30
	NaCN needed	g	1.6	0.4	0.6	1.3
	CaO needed.	g	1.1	1.0	0.4	0.4
Total cyanide consumption		kg/t	10.7	5.4	5.5	5.5
Total lime consumption		kg/t	4.4	3.0	2.8	1.5

Difficulty was experienced in filtering the solution samples. The worst sample to filter was Reg. No. 801579 taken from the fine area. Sample 801580 was almost as bad.

Filtration of solution samples was abandoned and the samples were centrifuged as were all subsequent samples.

By the time the filtration difficulties of the first batch of solution samples were overcome, it was time to take the second *round of samples and so no sodium cyanide or lime additions were made at the 3 hour point in the cyanidation programme.

The solution assays were as follows. The results are reported as mg/l of gold present in the solution.

Time (Hours)	Sample 801579	 801580	 801581	 801580 after pre-flotation.
3	0.46	0.42	0.40	0.44
6	1.07	0.29	0.44	0.54
9	0.96	0.91	0.64	0.87
12	1.20	0.87	0.58	0.95
15	0.87	0.84	0.58	0.83
24	1.00	0.87	0.56	0.81
Time for best recovery hours	12	9	9	12
% recovery in solution at that time.	74.1	59.0	52.1	59.4
Cyanide consumed to that time kg/t	8.7	5.0	5.0	4.7
Lime consumed to that time kg/t	3.1	1.8	1.9	1.2
% Au recovery in pre-flotation				16.8
% Mass pre-flotation concentrate				0.28
Au assay g/t pre-flotation concentrate				160

The recoveries shown above when applied to the calculated head assays of the samples from their sizing analyses allow cyanide residue assays to be calculated. They are as follows:-

	Sample		
	801579	801580	801581
Calculated head assay g/t	3.3	2.6	2.3
Recovery in cyanide solution %	74.1	59.0	52.1
Calculated residue assay g/t	0.85	1.07	1.10

These results show that the grade of gold in the tailings increases as you go down Middle Arm and that the gold is easier to recover.

Settling of Slimes from Tidal Flat Samples.

It was found that the slimes from the samples from the tidal flats areas did not settle on standing. However flocculation occurred and a clear water obtained by using sulphuric acid as a settling agent.

Flotation of Bulk Sample

The automatic sampler rejects from each of the three area samples were bulked and were screened on the 457 mm diam. Sweco screen fitted with a 75 µm screen cloth.

The screen oversize gravitated to a 305 mm diam X 305 mm Denver ball mill. The ball mill discharge was pumped back to the Sweco screen.

The Sweco screen undersize gravitated to one No. 5 Denver "Sub-A" pilot plant flotation cell where a carbon concentrate F 1 C was floated off. Fuel oil and the frother Teric 401 were added prior to flotation.

Potassium amyl xanthate and sulphuric acid were added to the carbon float tail followed by sulphide flotation in four No. 5 Denver flotation cells, to give a rougher concentrate F 2 C and a rougher tail.

At the completion of the grinding and flotation, a large split was found in the 75 µm screen cloth. A replacement screen cloth was obtained and the rougher tail was retreated through the circuit.

Reagent consumption was as follows:-

	g/t
Fuel Oil	909
Teric 401	1794
Sulphuric Acid	22507
Potassium amyl xanthate	218

The rougher tail F 2 T was sized with the following result:-

Aperture (µm)	% Mass	% Mass Cum.
+ 106	7.3	7.3
+ 90	2.2	9.5
+ 75	6.9	16.4
+ 63	7.2	23.6
+ 53	9.3	32.9
+ 45	9.8	42.7
+ 38	11.1	53.8
- 38	46.2	100.0

The presence of 16.4% of the material in the + 75 µm range is surprising and suggests that some leakage has occurred at the centre washer or around the edge of the screen cloth.

The rougher concentrate was subsequently cleaned in one No. 5 Denver flotation cell to produce a cleaner concentrate and a cleaner tail F 3 T. The cleaner concentrate was recleaned in one No. 5 Denver flotation cell to produce a final concentrate F 4 C and a recleaner tail F 4 T.

The mass balance and gold distribution for the bulk flotation test is as follows:-

Product	% Mass	Assay Au (g/t)	Au Distribution (%)
F 1 C	4.11	11.0	18.3
F 2 T	67.56	0.6	16.4
F 3 T	18.74	3.0	22.8
F 4 T	8.61	6.3	22.0
F 4 C	0.98	(51.7)	20.5
H	100.00	(2.47)	100.0

Cyanidation Tests on Flotation Products

Cyanidation tests were carried out on the rougher tail F 2 T, the cleaner tail F 3 T, and the recleaner tail F 4 T.

Cyanidation conditions were kept the same as the previous series of tests. In view of the optimum cyanidation times from the first series of tests, this series of tests was restricted to 15 hours.

The conditions were as follows:-

Time Hours	Condition		Sample		
			F2T	F3T	F4T
	Mass	kg	2.327	1.963	2.28
	Solution volume	ℓ	4.0	4.0	4.0
	Pulp density	% solids	36.8	32.9	36.3
3	Free cyanide	g/ℓ	1.52	0.92	1.14
	Protective alkalinity	g/ℓ	0.0	0.0	0.0
	NaCN added	g	2.0	6.32	3.44
	CaO added	g	1.2	1.2	1.2
6	Free cyanide	g/ℓ	1.84	1.26	1.43
	Protective alkalinity	g/ℓ	0.0	0.0	0.0
	NaCN added	g	0.65	0.65	0.65
	CaO added	g	3.5	3.5	3.5
9	Free cyanide	g/ℓ	1.80	1.23	0.49
	Protective alkalinity	g/ℓ	0.06	0.03	0.01
	NaCN added	g	0.8	3.0	6.0
	CaO added	g	1.4	1.5	1.6
12	Free cyanide	g/ℓ	1.92	1.23	0.71
	Protective alkalinity	g/ℓ	0.10	0.06	0.03
	NaCN added	g	0.3	3.0	5.3
	CaO added	g	1.2	1.4	1.5
15	Free cyanide	g/ℓ	1.96	1.50	0.84
	Protective alkalinity	g/ℓ	0.12	0.06	0.06
	NaCN needed	g	0.2	2.0	4.6
	CaO needed	g	1.1	1.4	1.4
Total cyanide consumption		kg/t	5.1	11.7	12.3
Total lime consumption		kg/t	4.3	5.4	4.7

The solution assays reported as mg/l of gold present in the solution were as follows:-

Time (Hours)	Sample F2T	 F3T	 F4T
3	0.18	0.14	0.17
6	0.20	0.15	0.18
9	0.21	0.46	0.25
12	0.24	0.63	0.43
15	0.26	0.64	0.63
Time for best recovery (hours)	15	15	15
% recovery in solution at that time	77.5	44.5	17.8

The above results show that cyanidation had stabilised in the case of F 2 T and F 3 T but may not yet have stabilised in the case of F 4 T.

The carbon concentrate F 1 C was calcined and the sulphide concentrate F 4 C was halved and one half was calcined. Cyanidation tests were then conducted on the calcined F 1 C, the calcined F 4 C, and the other half of the sulphide concentrate F 4 C.

Because of the small quantity of the sulphide concentrate and the calcined sulphide concentrate, the pulp densities for these tests were kept at 20% solids, instead of about 35% solids as was the case with all the other cyanidation tests.

An excessive amount of lime was required to be added to the calcined sulphide concentrate to get the desired alkalinity prior to commencement of cyanidation. Lime addition amounted to 298 kg/t of calcine.

The conditions for these three tests were as follows:-

Time Hours	Condition		Sample		
			F1C calcine	F4C calcine	F4C
	Mass	kg	1.922	0.577	0.65
	Solution volume	ℓ	3.84	2.31	2.53
	Pulp density	% solids	33.4	20.0	20.7
3	Free cyanide	g/ℓ	0.57	0.20	0.08
	Protective alkalinity	g/ℓ	0.001	0.64	0.20
	NaCN added	g	1.7	4.2	4.9
	CaO added	g	1.6	-	0.6
6	Free cyanide	g/ℓ	0.63	0.06	0.98
	Protective alkalinity	g/ℓ	0.001	0.51	0.26
	NaCN added	g	5.3	4.5	2.6
	CaO added	g	1.6	-	0.4
9	Free cyanide	g/ℓ	1.16	0.08	1.45
	Protective alkalinity	g/ℓ	0.02	0.60	0.28
	NaCN added	g	3.2	4.4	1.4
	CaO added	g	1.5	-	0.3
12	Free cyanide	g/ℓ	1.47	0.18	1.80
	Protective alkalinity	g/ℓ	0.03	0.60	0.28
	NaCN added	g	2.0	4.2	0.5
	CaO added	g	1.4	-	0.3
15	Free cyanide	g/ℓ	1.61	0.31	1.77
	Protective alkalinity	g/ℓ	0.03	0.46	0.25
	NaCN needed	g	1.5	3.9	0.6
	CaO needed	g	1.4	-	0.4
Total cyanide consumption		kg/t	11.1	44.7	22.9
Total lime consumption		kg/t	17.2	297.7	28.8

The solution assays reported as mg/ℓ of gold present in the solution were as follows:-

Time (Hours)	Sample F1C Calcine	Sample F4C Calcine	Sample F4C
3	-	-	-
6	3.25	3.00	1.31
9	3.33	3.55	2.28
12	3.46	3.55	2.95
15	4.01	4.05	3.25
Time for best recovery (hours)	15	15	15
% recovery in solution at that time	72.3	33.6	25.1

The solutions at 3 hours were returned to the tests and were not available for Au determinations.

Examination of these results show that the optimum cyanidation may not have yet been reached. However, the gold recoveries in solution of the two samples derived from the sulphide concentrate are low, and could not be expected to improve greatly.

Solution recoveries from F 1 C calcine and F 4 C calcine and from the tailings F 2 T, F 3 T, and F 4 T when applied to the mass balance for the flotation test suggest that the overall gold recovery would be 44.7%.

A weighted overall recovery cannot be derived from the cyanidation tests on the individual samples because the dry masses of the samples are not known, but would be better than the lowest recovery obtained, viz. 52.1%.

Conclusions.

There is no advantage in including flotation in a cyanidation plant for the treatment of Beaconsfield tailings from Middle Arm.

The presence of charcoal does not appear to have a marked effect to gold recovery in the cyanide solution.

The gold in the stockpile area is most difficult to recover and the gold from the fine area is the easiest to recover.

Senior Metallurgist...........

(H. K. Wellington)
Chief Chemist & Metallurgist.

Mitchell Cotts Projects

MITCHELL COTTS PROJECTS (AUSTRALIA) PTY. LTD.

REPORT ON LABORATORY TESTWORK
AND
RECOMMENDATIONS FOR TREATMENT
OF
GOLD TREATMENT PLANT TAILINGS
AT
BEACONSFIELD, TASMANIA
FOR
B.M.I. MINING PTY. LIMITED

Submitted by: C. HARRIS
Date: 31st October, 1980

J207

1.0 INTRODUCTION

Following the appraisal of earlier testwork and the acceptance of the test programme formulated by Mitchell Cotts Projects (Aust.) Pty. Ltd., the collection of samples by B.M.I. Mining Pty. Limited was completed in June, 1980.

Laboratory testwork proceeded during the final stages of sampling and was finalised in October, 1980.

The Department of Mines Report No. R792 dated 15th October, 1980 details the methods used and the results obtained. A copy of this report is included herein. Testwork proceeded over a longer period than expected because of equipment breakdown in the laboratory and the subsequent interference with the completion by other pressures on the laboratory staff.

2.0 DISCUSSION

2.1 Testwork

The programme of testing was aimed at selecting a preferred treatment route, at confirming other testwork previously carried out and to compare recoveries of gold in simple and more complex methods of treatment.

At an early stage it was recognised that poor recoveries obtained in testwork were related to the re-precipitation of dissolved gold at some stage or stages in the contact time. Such re-precipitation commonly occurs if some form of carbon is present or if some types of sulphide minerals are present.

As a result of this possibility an attempt was made to separate the detrimental material(s) by a preflotation stage. A parallel test without pretreatment monitored the effect on cyanidation.

The effect of this pretreatment was reported in R792 to be "not marked". This aspect of the report has been disputed and calculations show that a significant advantage in recovery of gold occurs. Further, savings in cyanide and lime are shown.

In the initial tests on raw tailings the dissolution of gold progressed to between nine and twelve hours and thereafter re-precipitation of gold progressed to the end of the tests. There is some evidence that two of the samples may have begun a re-dissolution cycle at twenty four hours. It has been concluded that the maximum recovery of gold takes place between nine and twelve hours of contact with lime-cyanide solution.

A pilot plant test was carried out on the bulk of samples available to obtain, by grinding and flotation, a sulphide concentrate with a high gold content and a low grade tailing. The object of this was to examine the possibility of discarding the flotation tailing, roasting the concentrate before cyanidation or, alternatively, dissolve the gold from the sulphide concentrate in its raw state.

It was found, first, that the flotation tailing yielded a high percentage of its gold to cyanidation, the sulphide concentrate raw was low in recovery of gold and the sulphide calcine showed high consumption of both lime and cyanide with a disappointing 72.3% recovery of gold.

A tabulation follows which compares the calculated recoveries under conditions of testing.

The report R792 quotes no advantage in separating carbon or re-precipitants in preflotation.

However, a possible advantage ranging between 7.8% and 16.6% is indicated when the gold contained in the pre-flotation concentrate is calculated back to the head value of the cyanidation feed in the three cases.

The range of recovery depends upon the use of 76.34% recovery (sample 801580 pre-treated) or 25.1% representing the recovery on sample F4C (see page 12 of R792) for the purpose of assessing the gold recoverable from that pre-flotation concentrate.

Using the lower value 7.8% the comparison is:-

Table 1

Sample	801579	801580	801581	Arith. Average
D of M	74.1	59.0	52.1	61.7
M.C.P.	82.2	67.5	60.0	69.9

Calculated recoveries without the addition of any advantage from the pre-flotation technique follow and show disagreement between the Department of Mines and Mitchell Cotts Projects in detail but average a discrepancy of .4% in recovery.

Table 2

Sample	801579	801580	801581	Arith. Average
D of M	74.1	59.0	52.1	61.7
M.C.P.	74.4	59.7	52.2	62.1

2.2 Flow Sheet Development

Two aspects of the ore affecting development of a flow route for the plant feed were the very viscous nature of the plant feed and the unexpectedly high silica content of the coarse fractions.

The viscous nature of the material is related to the dispersion of fines which would make the preparation of a suitable density for cyanidation difficult.

It was found qualitatively that suphuric acid in very small quantities gave excellent settling characteristics whilst polymers were ineffective. This aspect of testing was not pursued quantitatively because of the shortage of sample respective to the low sulphide content. This sulphide was required for later tests and the effect of reagents in these tests not known.

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The Table 3 at the end of this item shows a calculated plant feed sizing using equal tonnages from each of the three areas. The plus 75 micron material was assayed for insoluble and carbonate. These were calculated as silica and ankerite respectively. Sulphur shown in the distribution tables was expressed as pyrite. This approach was adopted after receipt of B.M.I. telex 5590 dated 17th July, 1980.

The mass and content of the average plus 75 micron material dictates a ball mill for grinding 24 t.p.h. of material having 88.4% silica. This indicates a ball mill of capacity to absorb about 120 kW.

Table 3

Fraction Aperture micron	Size % Mass		% (S)'	% SiO_2	% (CO_3)"
+ 600	4.1)	69.9	1.3	+200 # Only 88.4	10.3
+ 500	4.2)				
+ 425	7.9)				
+ 355	3.7)				
+ 300	9.7)				
+ 250	3.3)				
+ 212	10.3)				
+ 180	2.6)				
+ 150	9.9)				
+ 106	7.8)				
+ 75	6.4)				
- 75)	(1.9)	4.1	3.5		
+ 53)	(2.2)				
+ 38	4.7				
- 38	21.3		3.2		
	100.0		1.9		

The feed pulp density coming from the dredge is unknown, but thickening will be needed to obtain constant conditions of feed density to the plant.

It is anticipated that shells, and other trash will be removed on the dredge. Although for study requirements a thickener of about 20 metres diameter has been allowed for, there could be savings in space and expenditure with the use of a lamella thickener by SALA.

The conceptual flow sheet which has been included in this report is conventional and in principle follows the fine grinding, batch cyanidation for control of contact time, recovery of solutions by filtration in two stages because of the washing difficulties expected, zinc dust precipitation

..../4

because of the possible presence of mercury and because this is a well proven and reliable process.

Gold will be recovered by acidifying the precipitate to dissolve copper, lead, zinc followed by fluxing the damp filter cake and smelting to a high grade bar.

3.0 OPERATING COSTS

		($ per t of ore treated)
3.1	Reagents	10.58
3.2	Power	5.90
3.3	Labour and Supervision	0.77
3.4	Smelting	0.01
	Total:	17.26

4.0 CAPITAL EXPENDITURE - Order of Magnitude

		$\$ \times 10^6$
4.1	Machines	0.487
4.2	Tanks and Plates	0.130
4.3	Transport	0.019
4.4	Civil	0.051
4.5	Electrical & Instruments	0.154
4.6	Mechanical and Piping	0.206
	Total:	1.047

5.0 ENGINEERING

Additional to Item 4.0 capital expenditure, it is estimated that final process engineering, detail design, procurement and project management would be of the order of: $\$0.200 \times 10^6$

6.0 RECOVERY OF GOLD

Treatment Rate	750 t.p.d.
Ore Grade	2.6 g/t
Metal Recovery	69.9%
Gold Recovered Rate	1,36 kg/d
Operating Time per Year (90% x 360)	324 days
Gold Recovered per Year	440 kg

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7.0 CONCLUSIONS

The Middle Arm tailings will yield 69.9% of the gold value by a cyanide process designed to give strict control of contact time and having provision for de-activation of gold precipitants before cyanide contact.

The maximum recovery of gold occurs between nine (9) and twelve (12) hours and that unstable dissolution conditions obtain thereafter.

The cyanidation reagents cannot be added to the grinding circuit.

A treatment circuit consisting of sulphide flotation, roasting of the concentrate followed by cyanidation of the calcine would give a low yield relative to control difficulties, capital cost and relatively low gold recovery with high operating costs.

Cyanidation of raw sulphides gives very low recovery of gold.

8.0 RECOMMENDATIONS

That a batch agitation plant with vacuum filters and zinc dust precipitation be considered the best treatment route.

The plant be designed without extensive buildings.

Cover for control instruments and a minimal workshop to be provided for operators and tradesmen.

C. HARRIS

31st October, 1980

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APPENDIX III

Pontifex & Associates Pty. Ltd.

976032

TEL. 332 6744
A.H. 31 3816

26 KENSINGTON ROAD, ROSE PARK
SOUTH AUSTRALIA

P.O. BOX 91, NORWOOD
SOUTH AUSTRALIA 5067

MINERALOGICAL REPORT NO. 2949

15th July, 1980

TO:	Mr. Colin Harris, Mitchell Cotts Projects (Aust) Pty. Ltd., 246 St Georges Terrace, PERTH, W.A. 6000
YOUR REFERENCE:	Sample personally delivered 11/7/80
MATERIAL:	Metallurgical test, sulphide concentrate
IDENTIFICATION:	R792 (BMI) Sample No. 2 Carbon Float Concentrate
WORK REQUESTED:	Mineralogical examination, particularly to report on possible carbon content
SAMPLE & SECTION:	Retained

PONTIFEX & ASSOCIATES PTY. LTD.

Sample R792 (BMI)
Sample No. 2
Carbon Float Concentrate

This sample was submitted mainly to determine the presence (or otherwise) of carbon as charcoal or graphite, but also for a qualitative and semiquantitative analysis of the total mineralogical composition.

Accordingly the sample was examined initially in polished-thin section. One thousand grains were counted to obtain an estimate of abundance of the components, and the results are as follows:-

Grain size : the grains examined in the polished-thin section are generally monomineralic; they are all quite angular and commonly microfractured. They range in size from 0.002 mm to a maximum size of about 0.075 mm x 0.1 mm; average maximum dimension is about 0.02 mm.

The mineral species and their abundance based on the count of 1,000 grains are listed below. Comments on the identity and specific characteristics of these species follow this list. Note that the abundances are based on number of grains counted, however since the size range is considerable, this number of grains is not necessarily an accurate assessment of the true volume percent.

	approx % abundance
pyrite	53.2
chalcopyrite	26.6
marcasite (?or possible arsenopyrite)	5.5
sphalerite	7.8
limonite (including goethite)	3.2
graphite	3.4
? possible 'charcoal' grains	? 2.4
gangue	7.8

Sample R792 continued :

Pyrite rarely shows alteration to marcasite, and there was sufficient, apparent single grains of marcasite present to count it as a separate species. (However, at this fine size the optical properties whereby the single marcasite grains were identified is essentially the same as arsenopyrite and related sulphosalts, and whereas these minerals were not positively distinguished a check analysis for As, Sb and Bi is probably justified as a double-check.) Rare framboidal pyrite grains are present.

Sphalerite is fairly distinctive, but may be optically confused with goethite (which at this size range has similar optical properties). In the absence of other Zn minerals, a Zn assay is the most precise means to determine the abundance of sphalerite.

The chalcopyrite is rarely accompanied by trace chalcocite.

Graphite is optically reasonably distinctive, however in this sample is is generally smaller than average size range, i.e. <20 microns however some grains to 0.08 mm are present. It is unusually cryptocrystalline (rather than distinctly flaky) which throws minor doubt on its precise identification. (The remote possibility exists that this 'graphite' is optically confused with mackinawite (FeNiCo) sulphide or tochilinite (an interlayered nickeliferous, Fe sulphide and MgOH phase).)

The identification of amorphous carbon (charcoal) in polished section is highly inconclusive. However in this sample there are about 2.4% opaque grains, all <20 microns, which cannot be positively identified by reflected light microscopy as a sulphide or oxide phase, or as graphite. These may? be sphalerite (without internal reflection), however in view of the immediate problem of this concentrate they must be regarded as "possible" charcoal.

Gangue mineral contaminants appear to be mainly quartz, although trace carbonate may be present.

Sample R792 continued :

Conclusions and Recommendations

The bulk of this sample consists of pyrite (some gradational to marcasite), with subordinate chalcopyrite and minor sphalerite.

Graphite, with an anomalous cryptocrystalline form, and possible amorphous carbon (charcoal ?) together form a maximum of 6.8% of the number of grains in the whole sample, but because of their ubiquitous smaller size than the majority of the components, they almost certainly form <5% and probably <3% volume of the whole sample.

The identity of these components should be confirmed by a chemical analysis for non-carbonate carbon; (and at the same time an assay for Cu, Zn and Au is recommended) to provide maximum information on this concentrate, also checks for As, Bi, Sb, and for Ni, to help resolve any doubt in optical identification of such small grains, as noted in the above description.

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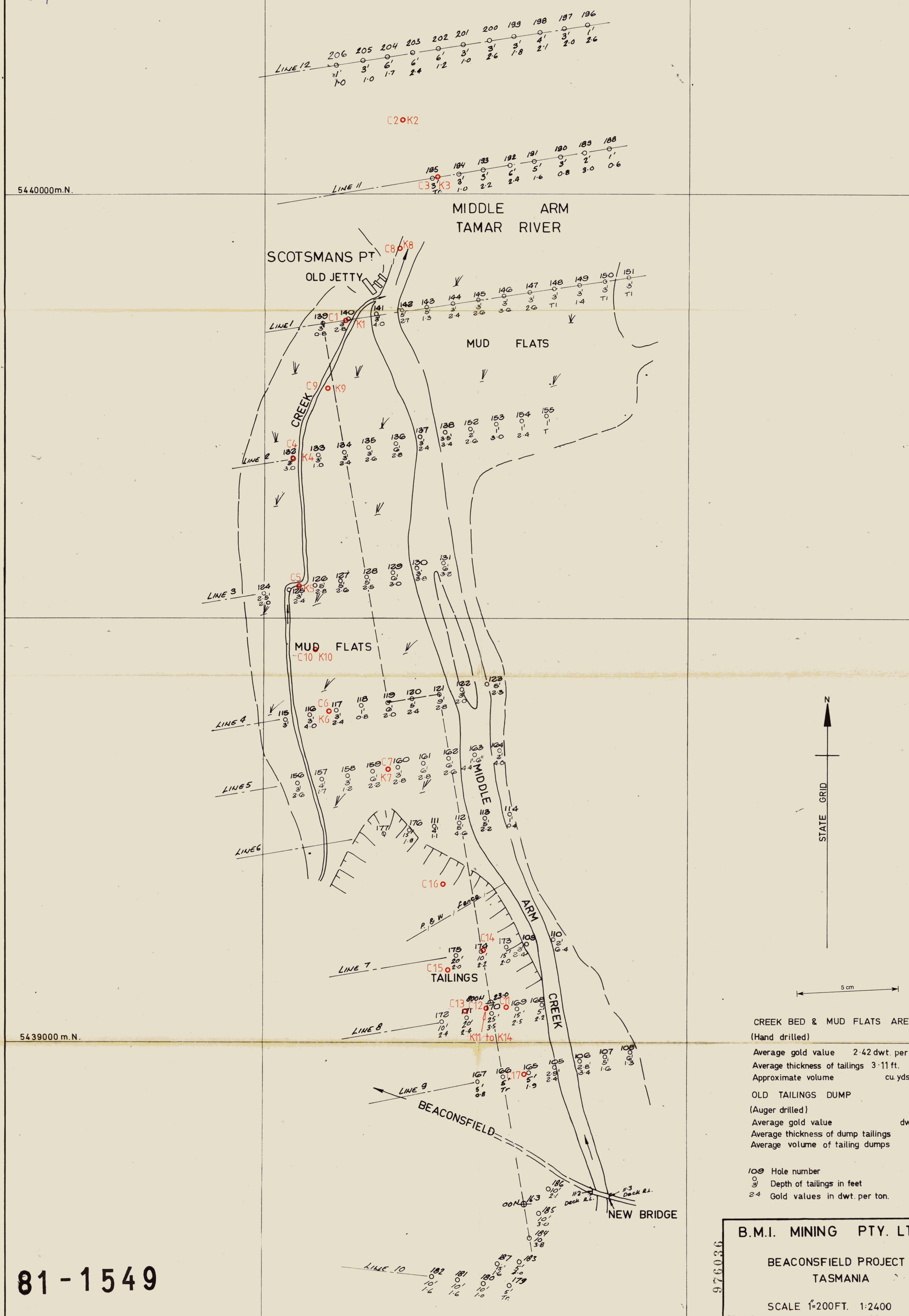

CREEK BED & MUD FLATS AREAS
(Hand drilled)

Average gold value 2·42 dwt. per ton
Average thickness of tailings 3·11 ft.
Approximate volume cu. yds.

OLD TAILINGS DUMP
(Auger drilled)

Average gold value dwt. per ton
Average thickness of dump tailings ft.
Average volume of tailing dumps cu. yds

109 Hole number
3' Depth of tailings in feet
2·4 Gold values in dwt. per ton.

81 - 1549

REVISIONS		
A. R.L.C. 7.2.74 DRG. No WAS BMI-M4-1	B) LINE Nos ADDED. HOLES & VALUES FOR LINES 10 11 & 12 ADDED K.A.L. 10.2.77	C) LINE 10 RELOCATED & BASE LINE ADDED. 7.7.77 K.A.L.

976036

B.M.I. MINING PTY. LTD.

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BEACONSFIELD PROJECT
TASMANIA

SCALE 1"=200FT. 1:2400

C2●K2 Sample sites for Mitchell Cotts (C) and Miller-Kappes (K) metallurgical testwork.

JULY 1973

25-5-81
D.J.S.

ASSAY PLAN

B.M.I.-X32-1/c.