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A FEASIBILITY STUDY INTO THE MINING OF ALLUVIAL GOLD OVER AREA 1, LEFROY AND TASMANIA

PREPARED FOR EPOCH MINERAL EXPLORATION NL

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AMG REFERENCE POINTS ADDED

SUMMARY

1. The feasibility study has indicated that alluvial mining at Lefroy is potentially feasible using a relatively simple mobile plant and mining techniques.

2. The estimated cost of this plant, support vehicle and a loader/excavator to feed the plant is $225,000. It is suggested that other earthmoving at least initially be carried out by contractors.

3. The total reserves of higher grade wash indicated with Area 1, upstream on Sludge and Morning Star Creek, from their convergence is calculated to be 70,000 cubic metres at a grade of 0.6 gms/metre^3 subject to some minor confirmation exploration within certain parts of Area 1.

4. The estimated direct cost of mining and treatment, including payment of royalties is $3.88 per cubic metre. Addition provision for amortization of 1/3rd of the plant over these reserves, plus allowing for future exploration, increases the projected operating cost to $5.20 per cubic metre.

5. At a gold price of $A350 per ounce, the estimated profit is $1.57 per cubic metre. At $400 per ounce, the estimated profit increases to $2.55 per cubic metre. The breakeven gold price at a recovery of 0.6 gm/metre is $A 275. The breakeven recovery at a gold price of $A350 is 0.45 grams/cubic metres.

6. Including the potential reserves within Area 2 - 4, the Lefroy Alluvial Project could provide a total return of about $500,000 over a period of about 4 years, after total amortization of plant. At the end of 4 years the residual value of the plant could be in the order of $100,000. This is an overall return of about 67% (at this stage tax free) on money invested.

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7. An extensive exploration programme is recommended for the coming summer to confirm the actual reserves available in Areas 2 - 4 and to prospect further within unexplored sections of the other creeks. The construction and commissioning of a plant should be delayed until the results of this programme are known.

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1. INTRODUCTION

1.1 The evaluation of potential reserves of alluvial gold at Lefroy in Tasmania has to date involved 3 separately definable stages.

Stage I - Reconnaissance Exploration of alluvial areas to determine which of the creek systems are auriferous. From this work the Lefroy project area was divided into 4 separate areas for further evaluation (see Report 975).

Stage II - Detailed examination of one of these 4 areas (Area 1). Indicated reserves outlined within Area 1 are in the order of 70,000 bulk cubic metres at a grade of 0.6gms cubic metre.

Stage III - Trial mining and pilot plant operation. Two sites within Area I were selected for bulk testing using a small pilot plant. I00 cubic metres of wash was mined from each area and about I2 cubic metres from each area processed through the pilot plant.

I.2 The results of the pilot plant operation are described herein. Following completion of the pilot plant study a feasibility study was carried out, and is incorporated in this report.

I.3 The results of the feasibility study can now form the basis for the design, erection and commissioning of an alluvial production plant to treat the indicated reserves. However, prior to commencing such work, the company may prefer to complete a Stage II type programme over the other 3 areas, indicated during Stage I, to determine the total indicated reserves for the area. Many of the conclusions of the feasibility study, and its potential profitability, hinge on there being additional mineable reserves of alluvium in the area.

2. THE PILOT PLANT STUDY

2.1 Approximately one hundred loose cubic metres of wash was mined from Site 1. Eighty loose cubic metres was mined from Site 2. (see Plate 2 for location)

2.2 The mining was carried out using an hydraulic excavator supplied by G.A. Reid of Launceston and involved -

(a) The removal of about 0.3 metres of top soil and the stacking of this top soil aside for use in reclaiming the area after the trial mining was completed.

(b) The removal of the clay overburden (1 - 2 metres), which was later returned to the pit.

(c) The removal of the coarse gravelly wash which was transported to the pilot plant site by truck.

(d) The reclaiming of the area by returning overburden to the pit and the re-spreading of top soil over the pit area.

2.3 The pilot plant was established at Valken Mining Plant site near the Curries River in the Northern part of the Consolidated Mining Lease. The wash from Site 1 and 2 was stockpiled in separate piles at the plant.

2.4 A simple gravitation treatment process to treat the wash was established,as follows -

(a) The wash was loaded onto the back of an old tip truck using a very small loader, both of which are owned by Valken Mining.

(b) From the tip truck the wash was shovelled into an old mining skip,converted on site for use as a hopper. A static ¼" steel screen was fitted to the lower half of the hopper.

(c) The wash feed into the hopper was washed through the screen using a high velocity jet of water supplied by an ejector system developed by Mr. V. Rautner (the principal of Valken Mining).

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The hopper retained all + ¼ inch material, which was removed from time to time by turning the skip onto its side, using a mechanical arm.

(d) The minus ¼ inch material was collected in wheelbarrows and from there fed into a 6 inch diameter Knelson Concentrator using a second ejector system.

(e) The Knelson Concentrator works on a centrifugal basis with the heavier particles being thrown furthest away from the centre of the concentrator and the lighter particles being retained in the centre. The heavier particles are trapped and the lighter particles overflow from the concentrator and are removed as tailings.

(f) The concentrate obtained from the Concentrator was removed each day and placed in a large plastic garbage tin.

(g) The plant was capable of treating about 1 cubic metre per day. Material was taken from each heap for a period of about 10 days (about 12 cubic metres from each site).

(h) The concentrate from each site was kept separate and panned down to a second concentrate. The panned tailings from each site were combined and reprocessed through the Knelson Concentrator at the end of the pilot plant operation to form a third concentrate, which was again panned down.

(i) The three pan concentrates were screened through a 1/16 inch sieve to remove large particles of quartz and ironstone. They were then panned down to remove larger gold pieces. The pan tailings which would include the fine gold was retained for amalgamation tests.

(j) The gold removed from each concentrate was dried and weighed and evaluated by consultant metallurgists Robertson Research Australia. A copy of their report is attached as Appendix I.

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3. PILOT PLANT RESULTS

3.1 The material from site 1 was treated as three seperate sections.

(a) Sample P - Run of mine wash.

(b) Sample Pz - Run of mine wash. (smaller sample but probably more representative)

(c) Sample PU - Basal wash, including upper section of bedrock.

3.2 The material from site 2 was treated in two seperate sections.

(a) Sample 080A - Run of mine wash.

(b) Sample 080B - Run of mine wash (more representative sample).

3.3 The loose (disturbed) density of the wash at both sites was determined by filling a cut-down 200 litre drum to a height of 400mm with material (one full wheelbarrow). This material was then weighted on a local weigh bridge.

3.4 The disturbed density of material from Site 1 was found to be 1.05tonnes/loose cubic metre and from Site 2 0.95 tonnes/loose cubic metre. Therefore, the average disturbed density of material has been taken as 1.0 tonne/ loose cubic metre.

3.5 The wash is made up of 50% (at least) by volume larger quartz gravels cobbles and small boulders (in excess of + 6mm). The in-situ or bulk density, therefore shoud be in the order of 1.6 tonnes/loose cubic metres, suggesting a swell factor for the disturbed material of some 60%? The normal swell factor is 30%. A swell of 40% is not uncommon, but 60% or more is a little excessive as the clay content of the wash in this case is not excessively high. It will be necessary to determine the bulk density of the wash very accurately, during the next stage of exploration work, to accurately determine the grade in bulk cubic metres.

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3.6 However, for the purpose of this feasibility study, a swell factor of 30% has been used in calculating the grades in bulk cubic metres. We consider such a swell factor to be very conservative, but more realistic than using the indicated swell factor of 60%. If ultimately a bulk density higher than 1.3 is found to exist, then higher grades than those indicated herein, will exist,and a recalculation of reserves and potential profits will be necessary.

3.7 For sample P approximately 13.0 tonnes was fed through the plant. After clean-up a total of 7.8348 grams of coarse gold was recovered on site by panning of the concentrate and a further 0.103 grams of fine gold was recovered by mercury amalgamation carried out by the Consultant Metallurgents. The grade determined was:
0.62 grams/loose cubic metre or
0.80 grams/bulk cubic metres.

3.8 For Sample Pz approximately 8.5 tonnes was fed through the plant. After clean up a total of 3.72 grams of coarse gold was recovered and a further 0.089 of fine gold was recovered by amalgamation.
Grade: 0.45grams/loose cubic metre
or 0.60grams/loose bulk cubic metre

3.9 For Sample PU approximately 5.7 tonnes of amterial were fed through the plant. 2.227 grams of coarse gold and another 0.06 grams by amalgamatiion were recovered.
Grade: 0.40 grams/loose cubic metre
or 0.52 grams/bulk cubic metre

3.10 For Sample 080A approximately 4.7 tonnes were feed through the plant. After clean up a total of 0.710 grams of coarse gold and 0.009 grams by amalgamation were recovered.
Grade: 0.15grams/loose cubic metre
or 0.20 grams/bulk cubic metre

3.11 For Sample 080B approximately 9.5 tonnes were fed through the plant. After clean-up a total of 1.151

grams of coarse gold and a further 0.054 grams by amalgamation were recovered.

Grade: 0.I3 grams/loose cubic metre

or 0.I7 grams/bulk cubic metre

3.I2 These results indicates that the average grade per bulk cubic metre of wash at Site I is 0.70 grams/cubic metre. The average result from the excavator pit sampling was 0.68 grams/bulk cubic metre. The bulk test result is similar to that indicated by the test pit sampling.

3.I3 The average grade per bulk cubic metre at site 2 is 0.2 grams/bulk cubic metre. The average result from the excavater pit sampling was 0.4 grams/bulk cubic metre. The pilot plant results here are 50% lower than the test pitting.

3.I4 Although this sort of wide dispersity between test pits and bulk sampling, is not uncommon, it does place a question mark on the value of treating the potential reserves outlined within the Morning Star Creek Area. Admittedly the calculation of average grade here from the test pit results was made from only two samples. One sample was 0.49 grams/loose cubic metre , the other 0.11 grams/loose cubic metre. It appears that the 0.49 grams/loose cubic metre is perhaps less representative of this particular area, than the 0.11 grams/loose cubic metre result.

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4. RE-EVALUATION OF RESERVES

4.1 The bulk mining operation suggests that the potential reserves within Area I at Lefroy should be divided into higher and lower grade material. Site 1 tested on Sludge Creek might be regarded as higher grade and Site 2 along Morning Star Creek is lower grade. However, going on the test pitting results, it would not be unreasonable to include both the higher parts of the Morning Star Creek and the lower part around the M & M20 line in the higher grade reserves.

4.2 On Sludge Creek the sampling programme indicated that the higher grades commenced on Line P. Line P20, P40 etc. upstream were not so productive. However, areas downstream as far as the junction with Morning Star Creek are considered of higher grade.

4.3 Both line P & M20 (where higher grades on both creeks commence), are fairly close to major zone of structual weaknesses,postulated by ourselves from our reconstruction of the geology of the area and from limited geophysical work carried out to date. This increase in grade over and downstream from the zone,might suggest that these areas are prospective for hardrock mineralization. Exploration of these two areas in detail for hardrock mineralization, should occur once it has been cleared of vegetation for alluvial mining.

4.4 In calculating the probable reserves within Area I, the area from Line P downstream on Sludge Creek to its intersection with Morning Star Creek,and from the Beachford Rood on Morning Star Creek, downstream to its intersection with Sludge Creek has been regarded as high grade. Most of the area upstream from the Beachford Road on Morning Star Creek has been regarded as lower grade, largely because of the poorer results from the site 2 bulk test. (See Plate 2)

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4.5 In calculating the reserves an average thickness of wash of I.0 metres has been used and the area included in the reserve calculations is shown on Plate 2. The average mining width varies from 20-40 metres.

4.6 The higher grade reserves are calculated to be 70,000 cubic metres at a grade of 0.6 grams/cubic metres.
The lower grade reserves are calculated to be I0,000 bulk cubic metres at a grade of 0.2 grams/cubic metres.
In addition there are several small pockets of probably economical wash scattered throughout Area I, that could add another few thousand cubic metres to the above results.

4.7 In addition to the above reserves, the Areas 2,3 & 4 as defined by the Stage I programme, are yet to be tested in detail. The shallow portion of the deep lead and Sludge Creek downstream from Area I could also yield additional worthwhile reserves.

4.8 The general section to be mined occurs below 1-2 metres of clay overburden,with an average of 0.3 metres of soil at the surface. The soil and clay overburden have to be removed to get at the gold bearing wash. The quantity of overburden present has not been included in the above figures.

4.9 The pilot plant operation indicated that concentrators such as the Knelson Unit are probably no more effective at collecting fine gold at Lefroy, than the simple Wilfry table used in the testing programme. This being the case then it would be better to use a cheap and conventional jig system rather than expensive concentrators.
However, we are concerned that neither the table nor the concent-rater could recover much of the very fine gold. Obviously we will have to experiment further in this area and test work with spirals and gold saving sluice races,after the jigs, is suggested.

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5. THE ULTIMATE MINING PROCESS

5.1 The ultimate mining process if likely to involve -

(a) The winning of the gold bearing alluvial gravels (wash) using a hydraulic excavator.

(b) The processing of this wash through a trommel (scrubbing and screening section) and a concentrating section, (either a Knelson Concentrator, or more likely a conventional jigs, followed by spirals or a gold saving race, to collect as much fine gold as possible.)

(c) The cleaning up of the concentrate using a secondary concentration system or jigs.

(d) The reclaiming of the mined area to Department of the Environment and Mines Department standards.

5.2 The mining should be preceeded by the clearing of vegetation and construction of a by-pass drain during the summer preceeding mining. The area to be cleared should include the strip of mineable alluvium and a strip about 20 metres wide either side. The cleared vegetation should be stacked as far as possible to one side of the clearing.

5.3 The by-pass drain should then be constructed as close as possible to the stacked timber. The aim of the by-pass drain is to direct run-off away from mine workings.

5.4 The mining process should initially involve the removal of top soil probably by bulldozer and the stacking of this top soil in a wind-row, adjacent to the by-pass drain, outside the area to be mined.

5.5 The extraction of wash and overburden would involve the use of an hydraulic excavator. The clay overburden would be dumped by the excavator in the previously mined area. The wash would be stockpiled by the excavator in a wind-row outside the area to be mined, ready for treatment.

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5.6 The maximum reach of the excavator is likely to be only ? 20 metres. Where the mining width is more than 20 metres then either some double handling will be necessary, (perhaps with a small bulldozer) or alternatively mining could occur in seperate strips, each strip being about 20 metres width.

5.7 The wash from the wind-rows would be loaded into a hopper of a mobile treatment plant using a front-end loader or small excavator. The hopper would be fitted with a grizzly to prevent very large rocks entering the plant. The hopper should be lined with teflon sheets to prevent sticking of wash near to its base.

5.8 If a front-end loader was used, then the wash would be fed from the hopper onto a conveyer which would deliver it dry to the trommel. If a small excavator was used, then because of its superior reach, it could feed the material directly into the trommel bin, thus doing away with second bin and conveyor. Another advantage of a small excavator, that it has more power to move the plant from site to site.

5.9 Regardless of the feed system used, water, via a series of high pressure spray nozzles would be sprayed onto the feed in the trommel. The trommel would comprise a scrubbing section to pulp the alluvium and a screening section. Either a double or single screening system could be used. If a double system is used, the inner screen could be designed to remove all material plus 25mm and the outer screen all material plus 6mm. If a single screen was used then it is likely to have a 6mm aperature.

5.10 The minus 6mm feed would be fed into either a primary jig or similar concentrating device, capable of handling 40 bulk cubic metres per hour. The ultimate concentration system selected, will depend largely on price and performance tests and will require some shopping around to see what is on the market, when the decision to proceed with the project is made. Regardless of what system is ultimately used, it is likely that a simple spiral or sluice system will be required after

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the primary concentrator to collect as much as possible of the fine gold missed by the concentrator before the is released to tailings.

5.11 The minus 6mm tailings from the concentrating system, would be returned directly to the mined area, via closed pipe and placed on top of the clay overburden. The plus 6mm screenings dumped at the end of the trommel, would be used to gravel access ways.

5.12 The concentrate from the concentrating system would be collected and taken in drums to a central office/workshop and clean-up site to be selected in the Lefroy area. This site could be either Valken Minings facility in the north of the leases or perhaps a house in the town area. Here the concentrate would be feed through a secondary concentrator and clean-up section to remove the coarse gold. The fine tailings from the clean-up section could be amalgamated with mercury to recover fine gold.

5.13 After completion of mining in any given area the tailings would be levelled by the bulldozer and the top soil with native plant roots spread back over the reclaimed area and fertilized. The by-pass would be left to enhance the drainage of the area.

5.14 The ultimate mining process recommended is illustrated by a series of schematic diagrams which follow.

FIG 1 - shows the Stage I Premine Development to take place each summer, including the clearing, dam, and by-pass drain construction.

FIG 2 - shows the overall mine development schematic layout; with an excavator working a current pit; the area downstream that has been already mined, the position of water pumps and direction of water circulation within a closed system; and the mobile treatment plant.

FIG 3 - Show the proposed operations in the mining pit in more detail, with overburden dumped on the area already mined and the wash stockpiled to the right of the pit extremites.

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FIG 4 - Illustrates the treatment process in more detail, in particular the circulation of water within the system.

FIG 5 - Shows the proposed restoration details within the area being levelled; top soil pushed back; replanted with native trees and the construction of a secondary drain.

Schematic Mine Plan – Lefroy, Tasmania
STAGE I – Premine Development

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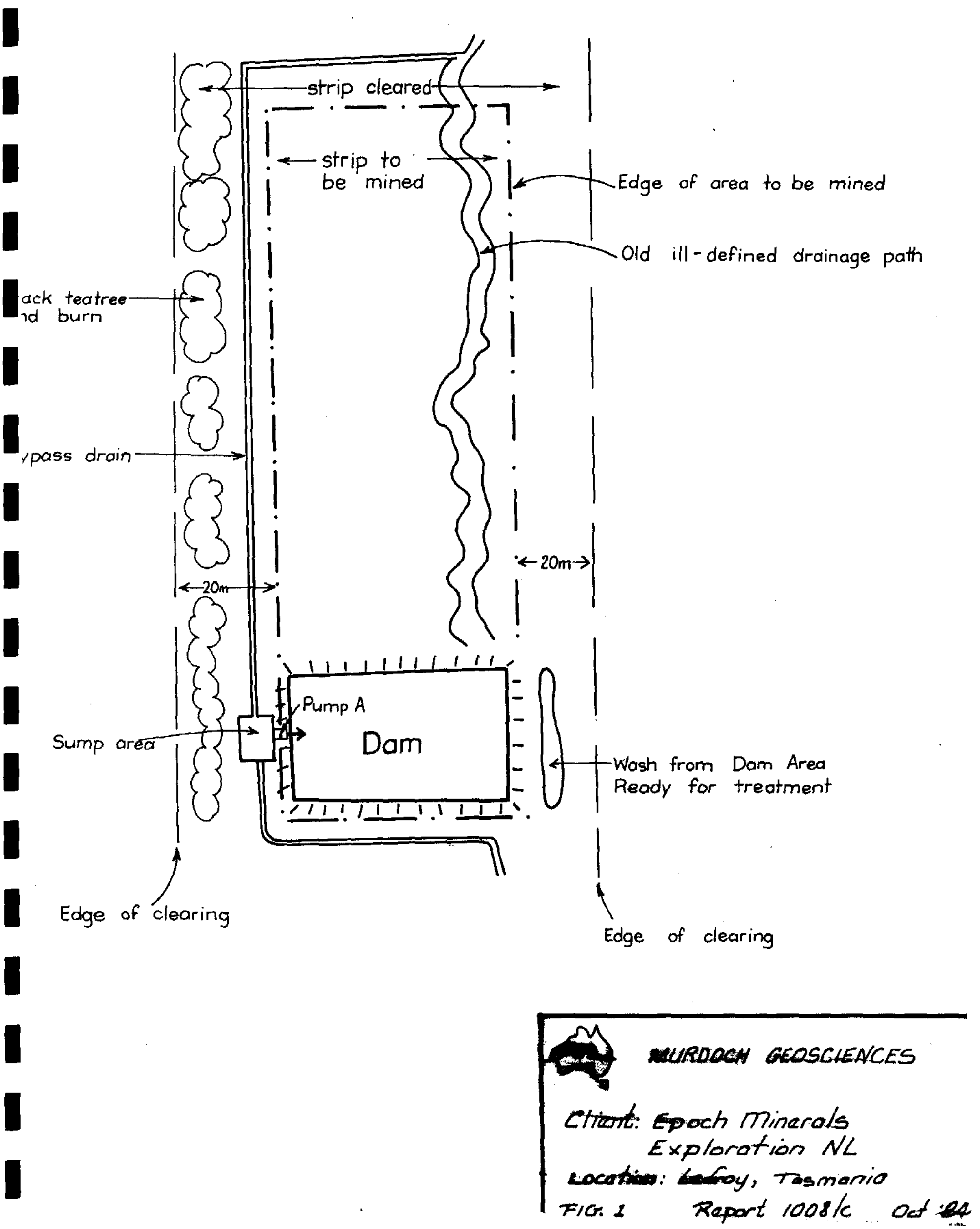

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Schematic Mine Plan - Lefroy, Tasmania
STAGE II - Overall Mining Development

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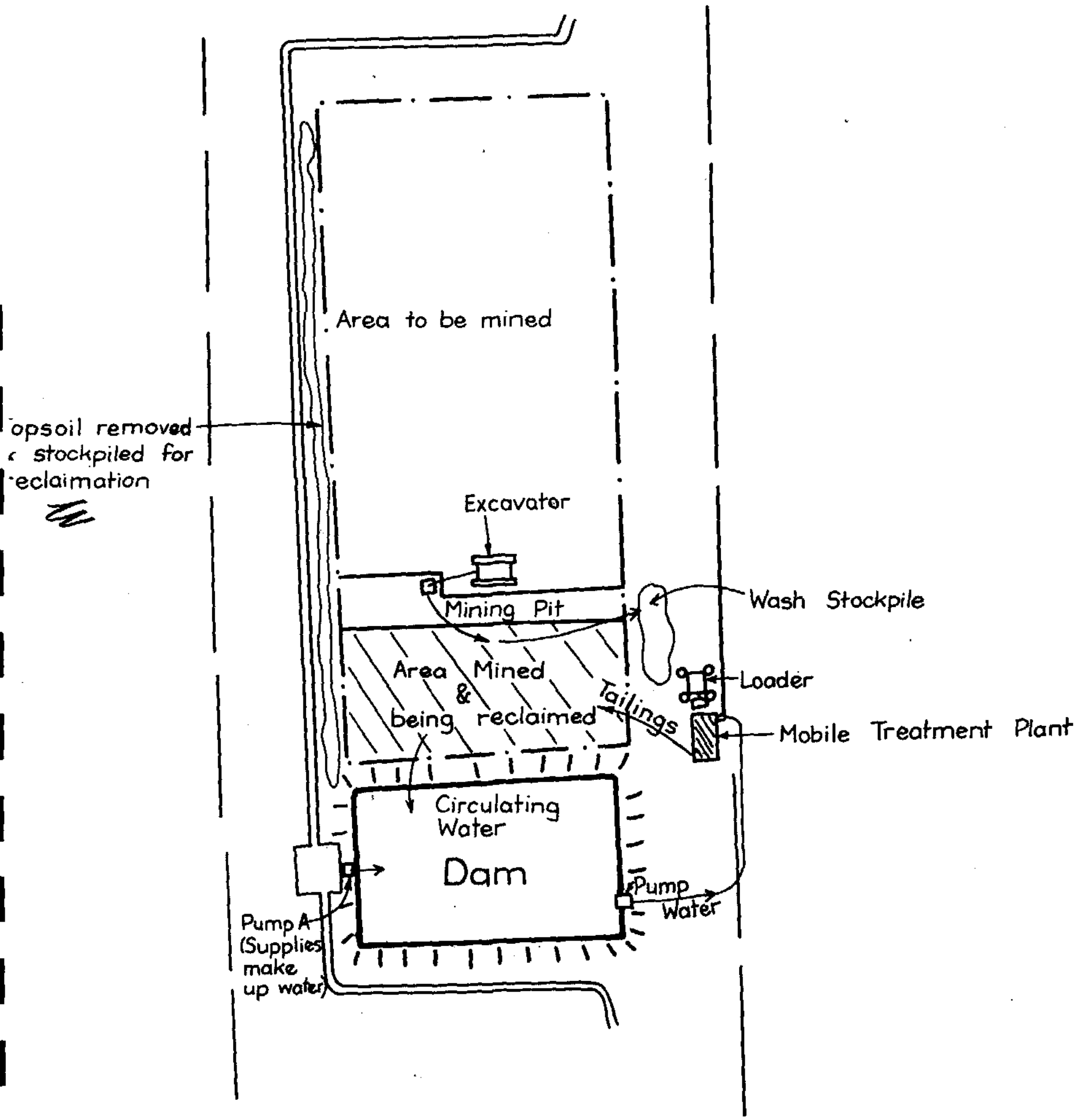

MURDOCH GEOSCIENCES

Client: Epoch Minerals Exploration Nl

Location: Lefroy, Tasmania

FIG. 2 Report 1008/c Oct.

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Schematic Mine Plan - Lefroy, Tasmania
STAGE II Mining Detail

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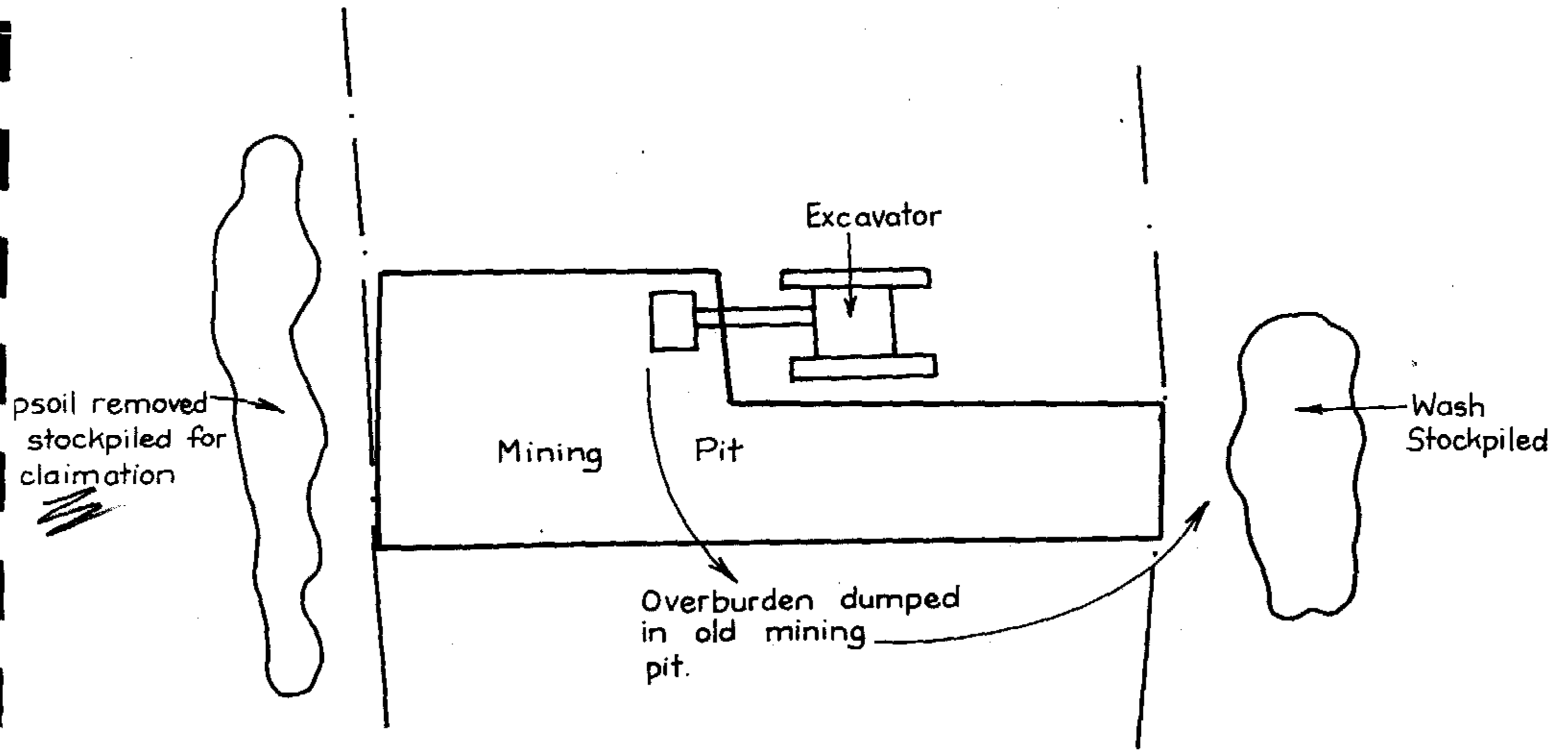

MURDOCH GEOSCIENCES

Client: Epoch Minerals Exploration N

Location: Lefroy. Tasmania

FIG 3 Report 1008/C Oct '8

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Schematic Mine Plan – Lefroy, Tasmania

STAGE II – Treatment Details

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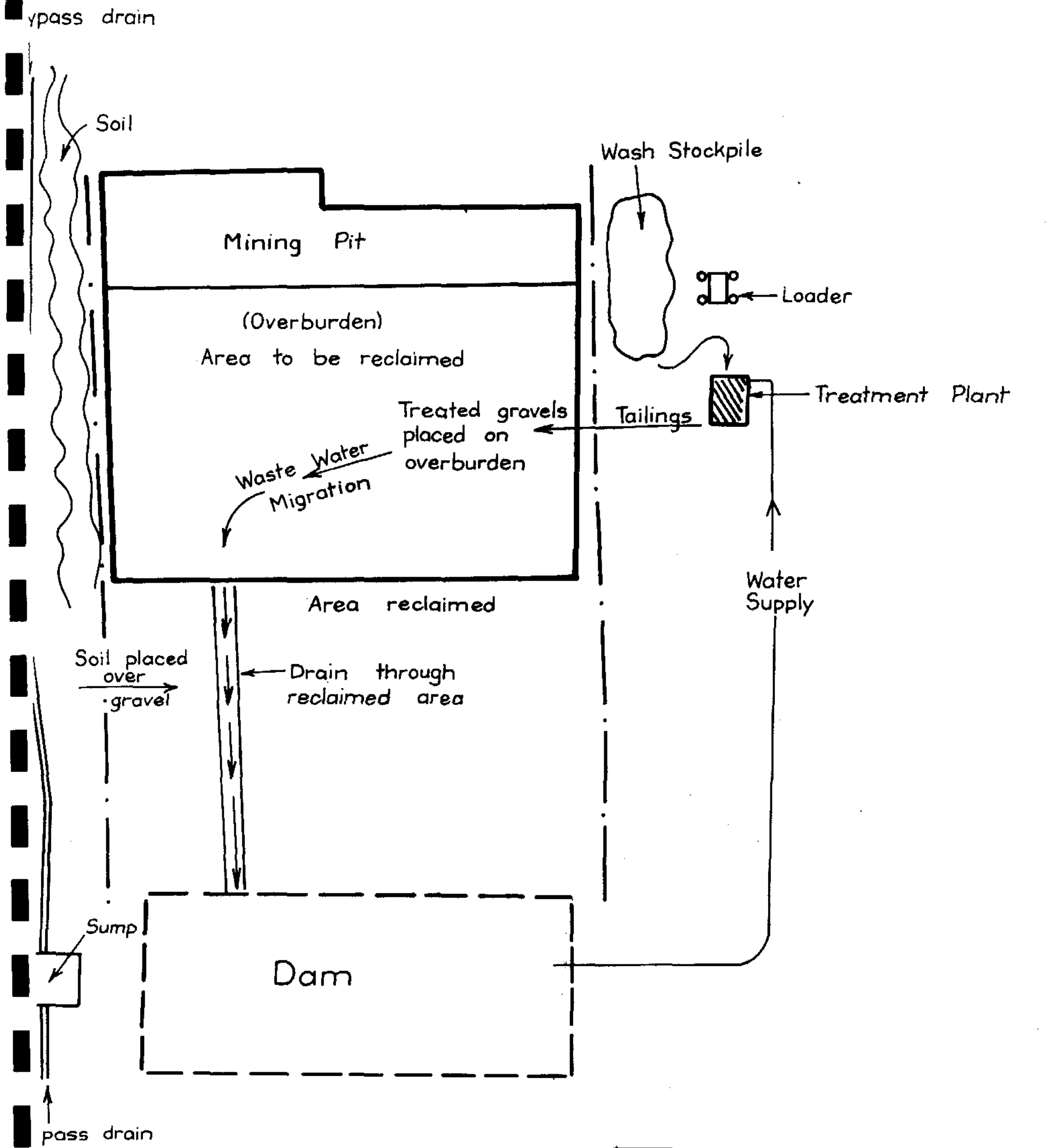

MURDOCH GEOSCIENCES

Client: Epoch Minerals Exploration NL

Location: Lefroy, Tasmania

FIG.4 Report 1008/C Oct '84

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Schematic Mine Plan – Lefroy, Tasmania
STAGE III – Restoration Details

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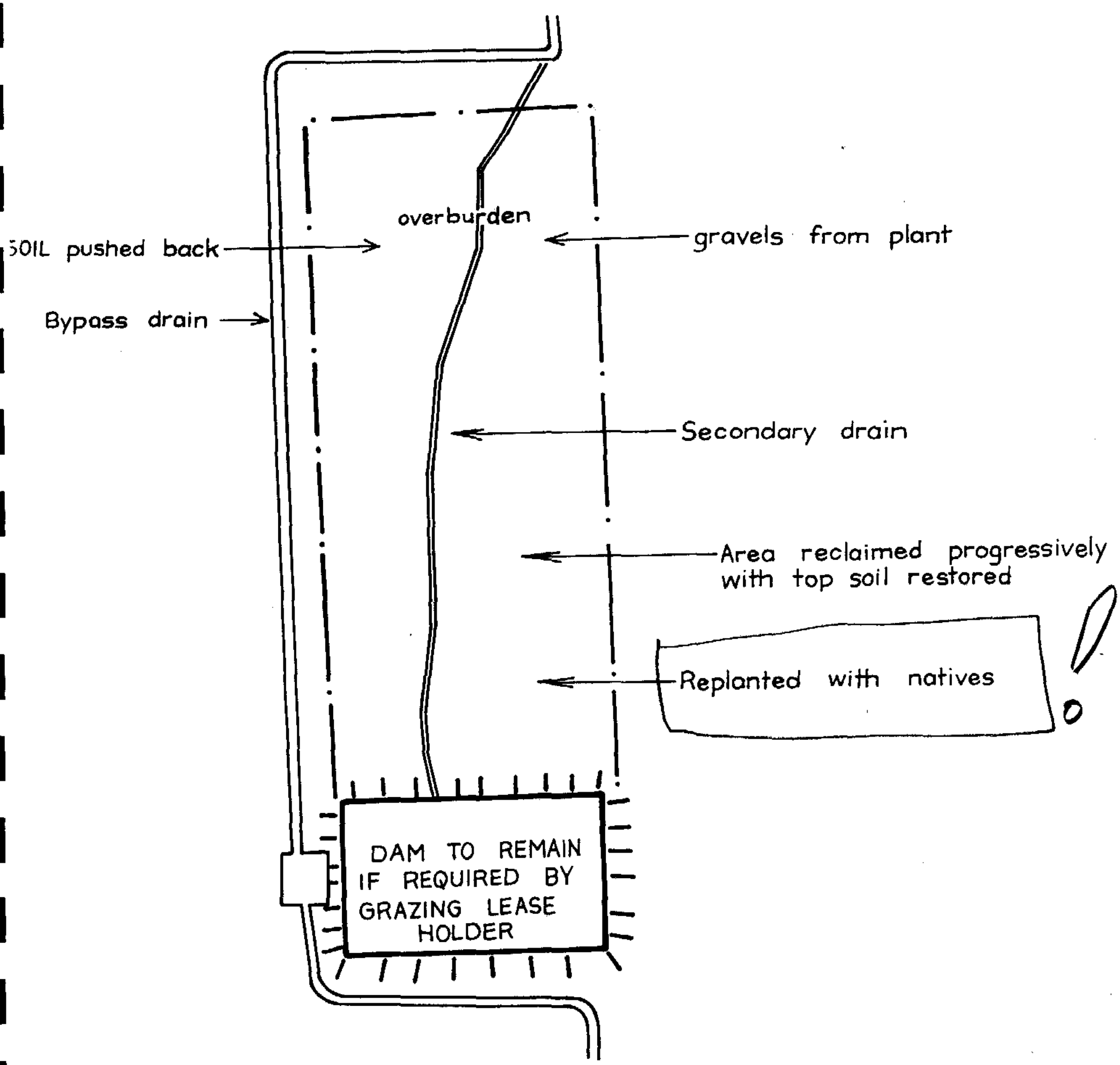

MURDOCH GEOSCIENCES

Client: Epoch Minerals Exploration N.

Location: Lefroy, Tasmania.

FIG. 5 Report 1008/C Oct. '84

6. WATER SUPPLY AND WATER QUALITY

6.1 It is proposed that the water supply will come from dams fed by run-off. The necessary dams will be constructed each summer on either slightly higher ground or terraces, that are to be mined, at the downstream end of each mining section. The clay overburden from the dam area will be used to construct impervious walls, whilst the wash would be removed for ultimate treatment through the plant.

6.2 A sump would be constructed within the by-pass drain, adjacent to the dams, to allow make-up water to be pumped into the dams as required during the mining process. At times when make-up water is not required, the by-pass drain will be allowed to ultimately discharge onto the creek flat,downstream from the area to be mined.

6.3 From the holdings dams, water will be pumped to the plant for use in the treatment process. Excess water from the tailings dams will gravitate via a second drain back into the holding dam, to create a close water system. Flocculants will be added to the dam water supply to assist in the settlement of suspended clays, as required.

6.4 Only at times of extended and extremely wet periods would the dams be likely to overtop and discharge water into the by-pass channel.

6.5 At present on all creeks, the water channel is ill-defined within tea-tree areas, which because of the presence of impervious clays close to the surface, tend to become swampy. Mining and construction of defined drains will substantially improve the drainage of the area. The swamp areas downstream from the areas to be mined are likely to provide a good filtration buffer for water over-topping from the treatment dams, assisting in the removal of suspended clays prior to these waters entering agricultural land north towards Beachford.

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6.6 No chemicals, apart from flocculants used where necessary to settle suspended solids, will be used in the mining or treatment process.

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7. THE SOCIAL IMPACT OF MINING

7.1 The area proposed to be mined within Area 1 at Lefroy is set out on Plate 2 accompanying this report. The area to be mined includes that portion of Sludge Creek that passes through the township of Lefroy.

7.2 It is not proposed that any private land be mined. The areas to be mined are essentially held under grazing leases. The bulk of the grazing leases are unimproved and the prospective allivial areas covered with dense tea-tree scrub. The ultimate restoration process will only enhance these areas, leaving them well drained. Whilst we propose only to replace natives in these areas, any energetic grazing lease holder would have a valuable area after restoration for cropping.

7.3 Included in our restoration work will have to be the replacement of any fencing removed during mining and the fencing of areas to be mined, from the general public and stock. As most of the fencing in the area is currently in a poor state, this will again enhance the ultimate value of the lease-holds.

7.4 The lower section of Sludge Creek and Morning Star Creek occur within the only improved grazing lease effected . This area is held by N & P Fox. The Fox's have cleared the area; effectively drained it and planted it down with pasture and with heavy annual applications of fertilizer. Some form of compensation ~~may~~ will be necessary for use of this pastured land; we will need to fence off the area to be mined and after restoration we will have to replace the pasture and fertilize it for some period to be agreed.

7.5 Mining in the area will only employ about 3 people and is unlikely to have a major impact on the Lefroy economy, as possibly all persons employed will be from out of town.

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7.6 We are unlikely to have much co-operation from towns people. I anticipate that vandalism and petty stealing and complaints to authorities will require a constant and effective public relations effort.

7.7 Enviromentally concience people from surrounding areas may also see the minig action as being deteriormental to the to the environment.

7.8 Residents of the village of Beachford to the north, and downstream from the mining operation, may also see the operation as being deteriormental to the water supply in their area.

7.9 We have applied for an environmental licence to treat up to 100,000 tonnes per annum. Our application has been advised twice in the Launceston Examiner and objections close 15th October, 1984. We anticipate several objections being lodged and an ultimate answer in December 1984.

7.10 To date we have had excellent co-operation and assistance from the Dept. of Mines; encouraging support from the Department of the Environment and enthusiastic support from the local media. It appears that even the creation of 2-3 jobs is important to the Tasmanian economy.

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8. LABOUR AND CONTRACTS

8.1 The proposed mining process is not labour intensive. One operator is required to drive the excavator. It has been estimated by consultant Mining Engineer Mr. S. Griffith, that a machine with a 1½ yard bucket would be working to about 75% capacity to mine and stockpile the 40 cubic metres of wash required per hour.

8.2 A second small excavator is suggested for feeding the treatment plant. This machine would work intermittently. The operator would also oversee the general running of the treatment plant.

8.3 A third person (foreman) would be involved in maintainance, site administration and clean up.

8.4 From time to time a bulldozer would be required, say on a contract basis, for clearing; double handling wash and for levelling areas to be reclaimed and replacing soil.

8.5 In the capital costings we have included the cost of only one small excavator or loader. It is proposed that other plant be supplied by contractors.

8.6 It has also been suggested by the Consultant Engineer that fuel and not electrical motors be used, because of the number of movements we shall make. Alluvial treatment plant are not large uses of electricity, so that the diesel costs are not high anyway.

8.7 Therefore in the costing, it is proposed that Epoch employ only 2 people, with contractor supplying 1-2 people, making a total employment of 3-4 people on site.

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9. LIKELY OPERATING COSTS

9.1 The pre-mining costs of land clearing, construction of by-pass drain and dam storage is estimated to cost a total of $0.10 per bulk cubic metres mined, using contractors.

9.2 The cost of removal of soil, removal and reclaiming overburden and mining and stockpiling of wash is estimated to be $1.25 per bulk cubic metre.

9.3 The cost of treatment of the wash through the plant is estimated to be $1 per bulk cubic metre.

9.4 Overheads, such as cleaning up the gold, restoring the site, management and accounting is expected to add another $0.80 per cubic metre to treatment costs.

9.5 Royalties to Valken Mining (7%) and Tasmanian Department of Mines (2½%) are expected to add another $0.73 per metre.

9.6 Total mining costs is therefore likely to be in the order of $3.88 per cubic metre.

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10. LIKELY CAPITAL COSTS

10.1 The capital costs for a new treatment plant capable of treating 40 cubic metres per hour, has been costed as follows;

Trommel Scrubber	$50,000
Concentratin Section	$30,000
Pumps and pipelines	$20,000
Generator 75 KVA	$15,000
Secondary Treatment	$12,000
Stores and Miscellenous	$20,000
Site Vehicle	$13,000
Loader	$25,000
Design, Management and Contingencies	$40,000
TOTAL	$225,000

10.2 A small operator and plant builder in Central Queensland has given a preliminary price of $100,000 for a mobile plant, which with contingencies and vehicles reduces our total capital estimate by about $50,000.

10.3 The 1½ yards excavator required for mining purposes would cost in the vicinity of $60-70,000. This unit could either be purchased or a contractor engaged. In this study it has been assumed that a contractor would be used. A bulldozer would only be used intermittently and hence its purchase could not be recommended.

10.4 For the purpose of amortization we have assumed a total capital cost of $225,000. The reserves of higher grade wash within Area 1 are 70,000 bulk cubic metres at 0.6 grams/cubic metre. Adding areas 2 -4 then the ultimate reserve could be as high as 200,000 cubic metre? Amortization of the plant over these reserves would add $1.12 per cubic metre.

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10.5 Based on results to date the cost of further exploration of Areas 2, 3 and 4 is estimated to be $0.20 per cubic metre of higher grade wash proven.

10.6 Adding the cost of further exploration and also amortization, then the total cost of the operation is estimated to be $5.20 per bulk cubic metre.

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11. POSSIBLE INCOME

11.1 It is proposed to use a treatment plant capable of processing 40 bulk metre per hour. If this plant runs for 8 hours per day, 5 days per week, 46 weeks per year for 90% of its operating time, the annual throughput is likely to be in the order of 60,000 bulk cubic metres.

11.2 At an annual treatment rate of 60,000 cubic metres, the higher grade reserves within Area 1 would last only 1 year and 2 months. This means that reserves within Areas 2, 3 and 4 would have to be proven well ahead of that time.

11.3 At a recoverable grade of 0.6 gms/tonne, the annual productiion of gold is likely to be 36,000 grams or 1160 ounces.

11.4 The projected annual incomes for 1160 ounces using different prices for gold are as follows:

Price (W. possibly less)	Total Income	Income per metre
$A300	$350,000	($5.83)
$A350	$406,000	($6.67)
$A400	$465,000	($7.75)
$A500	$580,000	($9.67)

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12. PROJECTED PROFITS ????

12.1 At the most likely situation of a gold price of $A400 per ounce and recovery of 0.60 grams/cubic metre the operation would record a profit of about $2.55 per cubic metre on $153,000 per annum. (say $150,000) The total estimated profit for the higher grade reserves outlined within Area 1 is $180,000 and this is after allowing for the amortization of 1/3 of the plant and the total exploration of Area 1. If the entire cost of the plant was amortized against Area 1 reserves only, then the net income after total write-off would be about $30,000

12.2 In the event of the gold price reduced to around $A350 then the estimated profit after amortization of plant and exploration write-off would be $1.57 per cubic metre, or about $110,000 for the reserves outlined within Area 1. If the entire cost of the plant was amortized against Area 1 reserves only, the residual owing after completion of Area 1 mining would be about $35,000.

12.3 The breakeven gold price for the operation assuming recovery of 0.6 grams/cubic metre and amortization over 200,000 metres would be $A270 per ounce. The breakeven recovery assuming a gold price of $A350 per ounce would be 0.47 grams/cubic metre.

where does this figure suddenly come from?

12.4 The above profit projectives suggests that the reserves of higher grade wash available within Area 1 are only probably sufficient to payoff a mobile alluvial plant.

12.5 However, if say another 130,000 cubic metres of wash of a similar grade are outlined in Areas 2, 3 and 4, then the net profit projected over the total available wash at Lefroy could be nearly $500,000. (assuming gold price $A400)

ASSUME WORST CONDITIONS.

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12.6 Once the plant and exploration costs has been totally amortized, then if the gold price is $A400 per ounce, the breakeven grade for mining would be reduced to 0.3 grams/cubic metre. If over the next 2 - 3 years the gold price rises significantly , relative to inflation then the lower grade areas within Area 1 and elsewhere may become economical to mine.

Too many 'if's 'may' "could" etc

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13. RECOMMENDATIONS

13.1 The feasibility study and pilot plant operation indicates that a worthwhile return can be obtained from an alluvial operation at Lefroy. // NOT TRUE

13.2 However, in our opinion there are some outstanding queries with regard to available reserves and bulk densities that should be answered prior to the company actually committing itself to a mining operation.

13.3 Within Area 1,.the detailed exploration needs to be extended down onto Fox's property,to and beyond the intersection of Sludge Creek and Morning Star Creek. Worthwhile additional reserves could occur downstream of this intersection and also within shallow leads on Fox's property.

13.4 Elsewhere within Area 1, there are some areas where we feel worthwhile additional reserves could occur. However, this requires confirmation.

13.5 The detailed exploration needs to be extended to Areas 2, 3 and 4. Lines of excavator test pits 200 metres apart need to be dug in these areas.

13.6 The reconnaissance exploration needs to be extended to the Curries River tributories in the northern part of the leases, which are yet to be tested ,and also onto the Exploration Licence, where some worthwhile reserves could occur?

13.7 It is recommended that an extensive testing programme be carried out during the coming summer, coincident with the final design and detailed costing of the treatment plant. The objectives should be to have the total reserves available, proven and the plant design ready for construction by Easter 1985. It would take about 3 months to set up the plant and production could then commence in mid winter 1985. Land clearing and dam sinking could commence in April. 1985.

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13.8 The estimated cost of the proposed exploration programme is:

FIELD WORK:	Excavator 280 test pits	$5500
	Excavator Superviser (1 man x 4 weeks)	$4000
	Sample taking and carting of wash (2 men x 4 weeks)	$3000
	Sample washing (2 men x 4 weeks)	$3000
	Plant Costs, power vehicle and incidentals	$2000
	Senior Geologist and mapping	$5500
	Vehicle Hire, expenses & Air travel	$3000
	SUB	$26000
	Final sample preparation (lman x 2 weeks)	$2000
	Evaluation - Geologist	$1700
	Assaying of selected samples only	$2000
	SUB	$5700
REPORT AND EVALUATION:	Geologist	$2000
	Drafting	$1000
		$3000
	Estimated Cost	$35,000

13.9 If desired the programme could be divided into 2 parts, one during the coming summer over areas 1 - 4, which would cost about $25,000, with the reconnaissance exploration of other areas being left until a later date.

13.10 The other alternative is to proceed only with a limited programme to complete Area 1 (Estimated $10,000); the design and evaluation of plant ($10,000) and then the construction of a plant ($215,000) to establish a cashflow by Easter 1985. This latter cause of action is the higher risk alternative, but the remaining exploration can be financed from cashflow.

This is an Irish company

13.12 Either of these three alternatives are acceptable to the author. In some way the ultimate decision will be guided by the results of our application for an extended environmental licence. Later the reef areas have to be explored also. However, the most prospective reef area also coincide with the alluvial areas, and hence exploration of these areas is better deferred until such areas are cleared ready for alluvial mining.

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ROBERTSON RESEARCH (AUSTRALIA) PTY. LTD.

PROJECT NO. 2381

MEMORANDUM NO. 1394

'TREATMENT OF ALLUVIAL GOLD CONCENTRATES FROM LEFROY, TASMANIA'

by

K.A. Andrews, B.Sc., B. Econ, M.A.I.M.M.

OCTOBER, 1984

PREPARED FOR:
Murdoch Geosciences,
203 Main Road,
MAROOCHYDORE. Q. 4558

INTRODUCTION:-

Three samples were received in early August, 7 received in late August, and final samples received on 11th September, 1984.

Samples were damp alluvial concentrates from Lefroy, Tasmania. The recoverable gold content of each sample was required. Other information required was the non-recoverable gold content of each sample, a general indication of the nature and size of the gold grains and some details on the silver and mercury content of the gold.

TESTWORK:-

The two larger damp samples were 'fine concentrates', obtained by retreating tailings from previous stage (which yielded 'coarse concentrates')and accordingly no coarse gold was expected. These two samples 'P fine con' and 'Fines 0-80' (original) were amalgamated by bottle rolling 24 hours as dense slurries with NaOH (17g and 5g respectively) and mercury (10.64g and 10.04g respectively). In each case the amalgam was recovered by elutriation, washed, dried, weighed (10.87g and 10.2g respectively), then treated with 1:1 nitric acid to yield a spongy mass of gold which was weighed, then cupelled and reweighed.

Each of the remaining samples contained rusted iron adhering to the sample containers. Accordingly it was necessary to digest each of these samples and the sample containers (mostly bottles) with hot dilute sulphuric acid. The samples were then filtered and washed. The solutions would not contain any significant amount of gold but could contain silver and mercury (globules of mercury and amalgam were noted in several samples) and accordingly submitted for analysis for Ag and Hg.

Each of the acid leached residues were then treated on a macropanner to concentrate the free gold, with negligible loss of 'recoverable' gold to tailings. The tailings from the 'coarse cons' were combined with the tailings from the corresponding 'fine cons' for samples 0-80, PU and PZ. Very little fine gold (i.e. less than 200 mesh size particles) was evident in any of the samples treated on the macropanner. The macropanner concentrates and tailings were dried and weighed. The macropanner concentrates from P Coarse Con and PU Coarse Con had a pale silvery appearance, possibly due to mercury. Accordingly these samples were heated to 550°C and reweighed. The appearance changed to the normal yellow gold colour, suggesting that most of the weight loss was in fact mercury. All of the products were submitted for fire assay. Addition of sufficient silver to permit parting of all prills was impractical so gold content was determined from the prill weights and purity of selected prills.

Additional details of samples observed during testwork were as follow:-

Sample P Coarse contained a considerable proportion of coarse gold up to 4mm. Gold was irregular in shape, showed very little rounding due to stream transport and included 3 composite gold-quartz grains.

Sample 0-80 Coarse (original sample) contained both yellow and silvery gold grains. The gangue consisted mainly of fine zircon, a little rutile and a trace of unidentified rounded black non-magnetic grains.

The remaining samples treated on the macropanner showed fine zircon, rutile, a little monazite, 'ironstone' pebbles and a few black non-magnetic grains. Fragments of weld slag, wear steel, brass or bronze, copper wire and lead fragments were also present. The brass/bronze and wear steel fragments frequently resembled stained gold. The 'fine overall clean-up concentrate tails panning' sample also contained approximately 0.3g of mercury-brass-copper wire-wear steel amalgam mixture.

RESULTS:-

Full details of sample weights, product weights and gold assays weights etc. are given in Table 1.

CONCLUSIONS:-

The native gold in the samples is extremely low in silver based on purities of bullion prills and the very low silver assays on the acid leach solution.

In the acid leach step a little mercury was dissolved from some of the samples at approximately 1.4mg from 0-80 Coarse (original) sample, approximately 0.2mg from PZ samples and much less from all other samples. Much of the mercury however remained undissolved, probably due to the strongly reducing conditions in the acid leaches. The weight loss on heating of the P coarse concentrate sample together with the change in colour of the sample indicates approximately 950mg of Hg in this sample.

For the testwork on the coarse and fine concentrate samples the distribution of gold in the various test products shows from 95.5 to 98.8 percent of the gold reporting to the coarse, clean concentrates with a further 0.81 to 4.25 percent reporting to the fine high grade concentrates. Losses to the combined test product tailings are very low, ranging from 0.07 to 0.39 percent.

For the clean-up concentrate the gold losses are higher at around 4.9 percent but the actual grade of the concentrate is so low that the separation is still very good.

Two of the larger samples were treated by amalgamation and these yielded tailings much lower in grade than the tailings from treatment of the samples on the macropanner. Accordingly barrel amalgamation appears to be the most suitable means for final upgrading of gravity plant products. The presence of considerable mercury in the samples, derived from previous workings, means that using an amalgamation step will not cause any additional pollution problems. In any event retorting of final upgraded concentrates to control mercury fumes and/or recover mercury prior to smelting to bullion will be required.

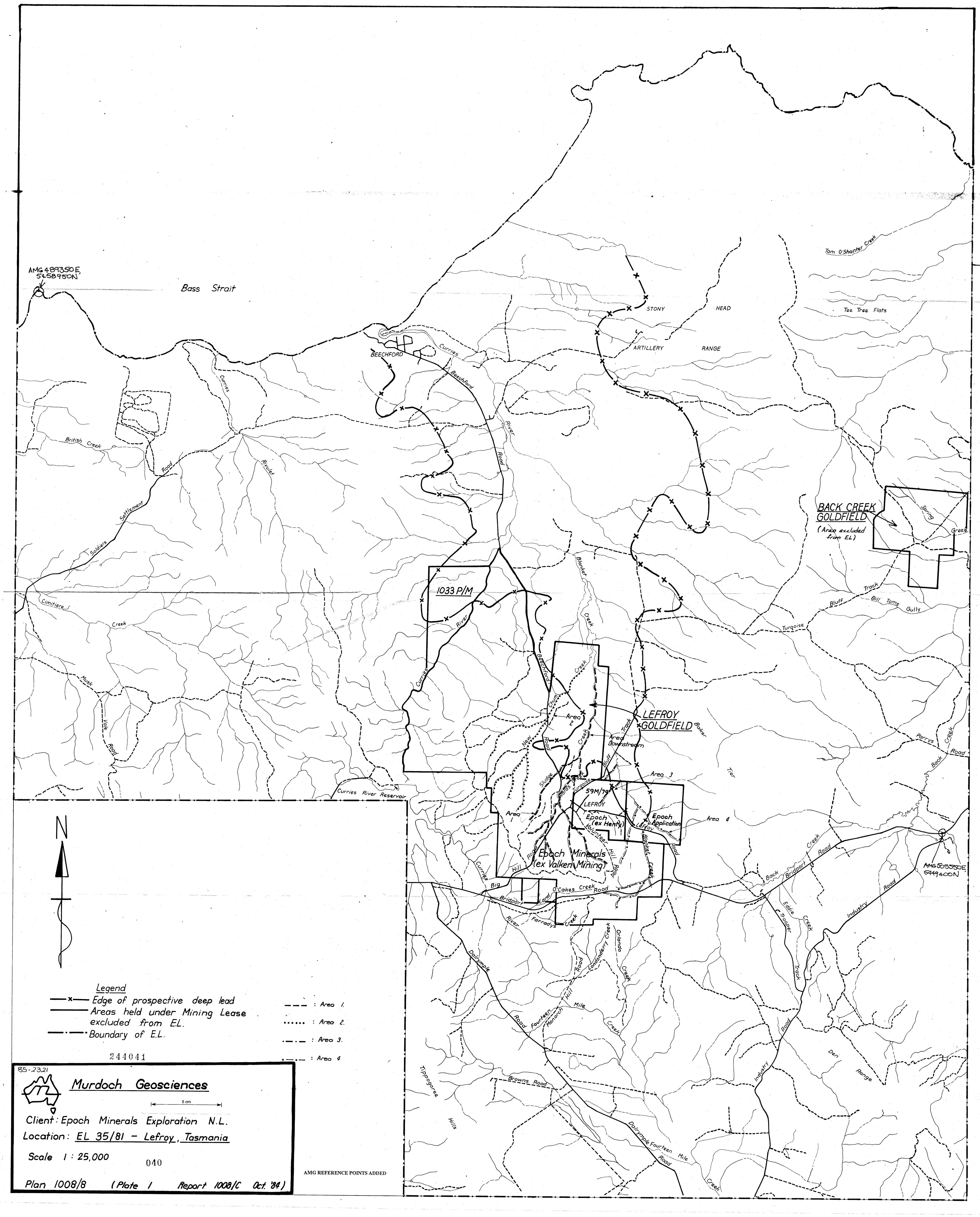
TABLE 1

RESULTS OF TREATMENT OF ALLUVIAL GOLD CONCENTRATE SAMPLES FROM LEFROY, TASMANIA.

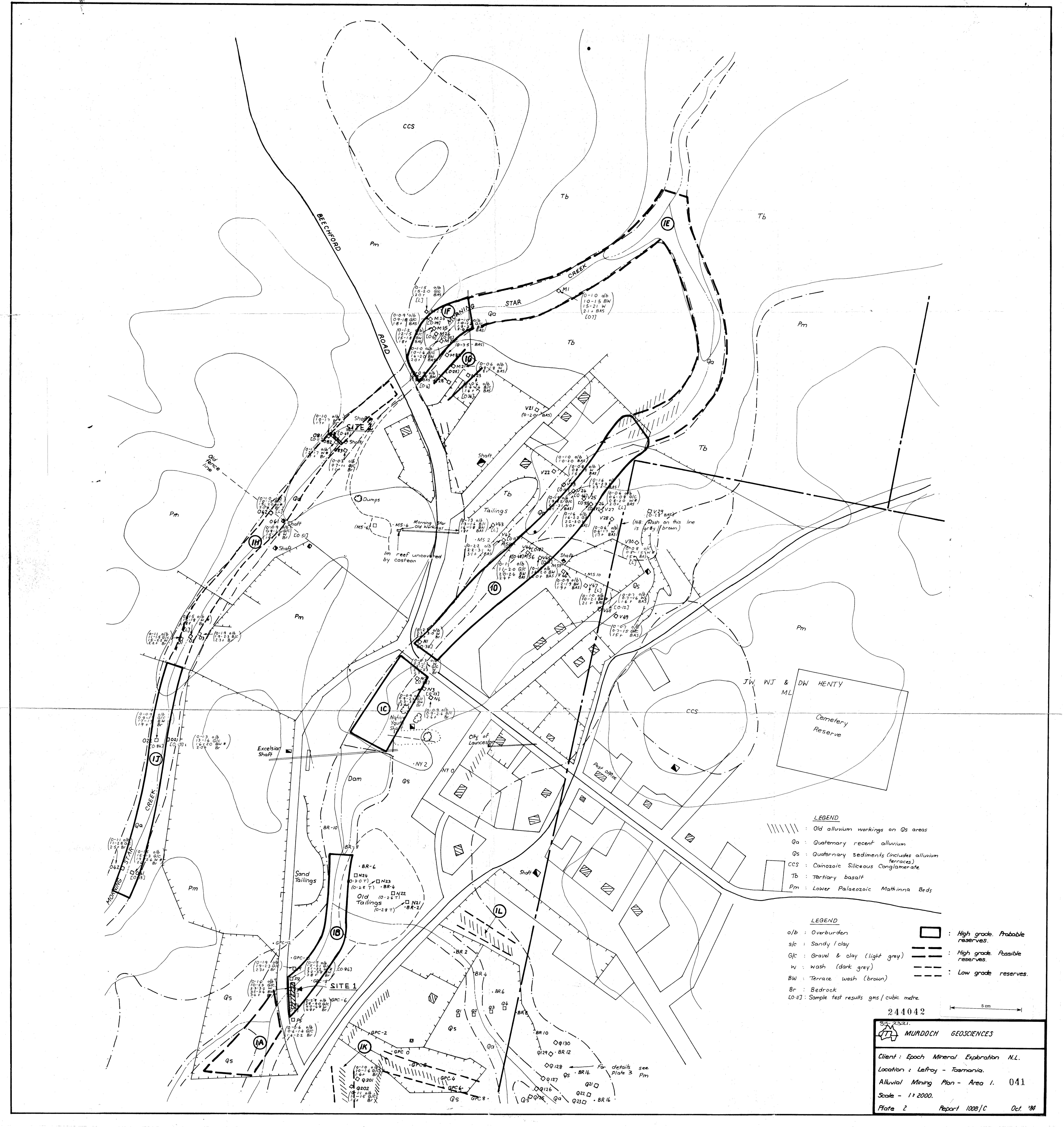
Sample Details	Test Products	Product Weights/Volumes	Au Assay g/t	Bullion Prills: Prill Weight g	Bullion Prills: Purity# Assayed (Estimates) % Au	Gold Content of Test Product, g.	Distribution of Gold to Test Cons and Tails, percent	Solution Assays
P Coarse	Acid Leach Soln	100mls						<0.01ppm Ag, 55ppb Hg
	Macropanner Con	8.3364g (8.2418g*)		7.8477	99.83	7.8340	98.63	
	Macropanner Tail	2.6973g	1465			0.0040	0.05	
P Fine	Au Sponge from Amalgam	0.1211g		0.1031	(100)	0.1031	1.30	
	Amalgamation Tail	1816.1g	1.16			0.0021	0.02	
0-80 (original) Coarse	Acid Leach Soln	100mls						0.09ppm Ag, 14ppm Hg
	Macropanner Con	0.7283g		0.7110	(98.8)	0.7096	98.80	
	Macropanner Tail	0.7503g	3665			0.0027	0.38	
0-80 (original) Fines	Au Sponge from Amalgam	0.0070g		0.0058	(100)	0.0058	0.81	
	Amalgamation Tail	503.25g	0.21			0.0001	0.01	
0-80 Coarse and Fine	Comb. Acid Leach Solns	340mls						<0.01ppm Ag, 8ppb Hg
	Coarse Macropanner Con	1.1829g		1.1522	99.88	1.1508	95.50	
	Fine Macropanner Con	0.0582g		0.0513	(99.9)	0.0512	4.25	
	Comb. Macropanner Tails	11.49g	261			0.0030	0.25	
PU Coarse and Fine	Comb. Acid Leach Solns	400mls						<0.01ppm Ag, 6ppb Hg
	Coarse Macropanner Con	2.5160g (2.3530g*)		2.2733	98.04	2.2287	97.04	
	Fine Macropanner Con	0.1697g		0.0609	(98.0)	0.0597	2.60	
	Comb. Macropanner Tails	181.2g	45.2			0.0082	0.36	
PZ Coarse and Fine	Comb. Acid Leach Soln	395mls						0.04ppm Ag, 450ppb Hg
	Coarse Macropanner Con	3.9098g		3.649	99.72	3.6388	97.24	
	Fine Macropanner Con	0.3059g		0.0895	(99.7)	0.0892	2.38	
	Comb. Macropanner Tails	214.4g	66.5			0.0143	0.38	
Fine Overall Clean-up Concentrate Tails Panning	Acid Leach Soln	360mls						<0.01ppm Ag, 7ppb Hg
	Amalgam plus Macropanner Con	0.4905g		0.0393	(98)	0.0385	95.1	
	Macropanner Tail	102.6g	19.6			0.0020	4.9	

* Product weights in brackets are weights after heating at 550°C for 30 mins. The weight loss represents mainly mercury.
# Bullion Prill Purities in brackets are estimated values. Only 4 prills were assayed for purity.

← How do you estimate purity???



Bass Strait
AMG 489350 E, 5458950 N
AMG 505550 E, 5449400 N
BEECHFORD
Curries River
Beechford Road
British Creek
Settlement Road
Soldiers Settlement Road
Rivulet
Cimitiere Creek
Musk Vale Road
STONY HEAD
ARTILLERY RANGE
Tom O'Shanter Creek
Tee Tree Flats
BACK CREEK GOLDFIELD
(Area excluded from EL)
Spring Creek
Bluff Track
Bill Toms Gully
Turgoise
1033 P/M
Blanket Creek
LEFROY GOLDFIELD
Baker Tier
Parrys Road
Back Creek
Area 1
Area 2
Area 3
Area 4
Area Downstream
New Chum Road
Chum Creek
Sludge
59M/79
LEFROY
Epoch (ex Henty)
Epoch Application
Epoch Minerals (ex Valken Mining)
Volunteer Hill Road
Curries River Reservoir
Big Hill
O'Cakes Creek Road
Bridport Road
Back Bridport Road
Farradys Creek
Londonderry Creek
Orlando Creek
Monarch Hill Road
Dalrymple Road
Fourteen Mile Creek
Eddie Creek
Cooper Track
Industry Road
Den Range
Browns Road
Tippogoree Hills
Dalrymple Fourteen Mile Road
Creek
N
Legend
Edge of prospective deep lead
Areas held under Mining Lease excluded from E.L.
Boundary of E.L.
Area 1.
Area 2.
Area 3.
Area 4
244041
85-2321
Murdoch Geosciences
5 cm
Client: Epoch Minerals Exploration N.L.
Location: EL 35/81 - Lefroy, Tasmania
Scale 1 : 25,000
040
AMG REFERENCE POINTS ADDED
Plan 1008/8 (Plate 1 Report 1008/C Oct. '84)


CCS
Tb
Pm
Qa
Qs
BEECHFORD
ROAD
MORNING
STAR
CREEK
SITE 2
SITE 1
1E
1F
1G
1H
1J
1C
1D
1B
1A
1K
1L
Old fence line
Dumps
Shaft
Excelsior Shaft
Morning Star Old Workings
1m reef uncovered by costean
Tailings
Dam
Sand Tailings
Old Tailings
City of Launceston
Post office
JW WJ & DW HENTY ML
Cemetery Reserve
(N8 Wash on this line is grey (brown)
For details see plate 3
LEGEND
Old alluvium workings on Qs areas
Qa : Quaternary recent alluvium
Qs : Quaternary sediments (includes alluvium terraces)
CCS : Cainozoic Siliceous Conglomerate
Tb : Tertiary basalt
Pm : Lower Palaeozoic Mathinna Beds
LEGEND
o/b : Overburden
s/c : Sandy / clay
G/c : Gravel & clay (light grey)
W : wash (dark grey)
BW : Terrace wash (brown)
Br : Bedrock
[0·11] : Sample test results gms / cubic metre
: High grade. Probable reserves.
: High grade. Possible reserves.
: Low grade reserves.
5 cm
244042
MURDOCH GEOSCIENCES
Client : Epoch Mineral Exploration N.L.
Location : Lefroy - Tasmania.
Alluvial Mining Plan - Area 1.
041
Scale - 1 : 2000.
Plate 2
Report 1008/C
Oct. '84