Dossier
Molengraaff

P.M. van Welsenes

# BATCHLEACHING IN BAMBOO CREEK W.A.


Delft University DECEMBER '86

CONTENTS

# 1. INTRODUCTION

It is not uncommon for students at the Delft University for Mining and Petroleum Engineering to go abroad to investigate their final thesis topic.
In concert with professor P. van Leeuwen and professor P.J.M. Ypma it was agreed that I could go to Bamboo Creek (BBC) in Western Australia to carry out investigations for my final thesis.
Professor Ypma is working half the year for the Adelaide University as a lecturer in economic geology and therefore has many contacts within the Australian mining industry.
Professor Ypma was involved in the BBC gold project right from the beginning as a geological consultant. It was under his supervision that five dutch geology students carried out a geochemical and structural study on the geology in BBC in 1984. This also was part of their final thesis.
During the starting up period of the mine it became clear that the project did not develop according to plan. The planned investments were exceeded and the production targets were not realized in time. Overhead therefore was high and profitability dropped.
Professor Ypma tried to be of help by suggesting changes in the location of the tailingsdam and the open pit production schedule. Both alterations in his point of view would have helped to reduce the costs. Management however decided that the project had gone too far to realize these changes.
By now (1985) the Joint Venture future was not very promising and Kitchener Mining as one of the jointventurers decided to take over management to reorganize the mine which was in serious financial trouble.
The mine once more showed interest in the contributions of a student. It would help the mine if some cash could be generated on a short term without too much interference with normal proceedings. A student could make an investigation of possible projects. The first thought was to start a new open pit.
This report deals with the problems above. A proposal is made and worked out to some extend.
The studytrip was made possible by contributions of "Het Molengraafffonds" and "Het KIVIfonds".

## 2. GENERAL INFORMATION

### 2.1. LOCATION

Bamboo Creek (BBC) is situated in the Pilbara Shire. This is the Northwestern part of Western Australia. Marble Bar is the nearest village at 65 km southwest of BBC, and can be reached from the mine over a dirt track.
Marble Bar is the administrative centre of the East Pilbara Shire and has about 300 inhabitants. The Mines Department has an office in Marble bar. There is a small school and a state battery where it is possible to treat custom gold ore from small producers. Around Marble Bar several small scale mining operations are in progress.
The local airport has a daily connection with Port Hedland except on Sundays.
Twohundred Km west from the mine lies Port Hedland. This harbour is responsible for a large part of the iron ore transport from the Hammersley Range. With 10.000 inhabitants it is one of the larger cities in this area. The airport provides regular connections with the rest of Australia.

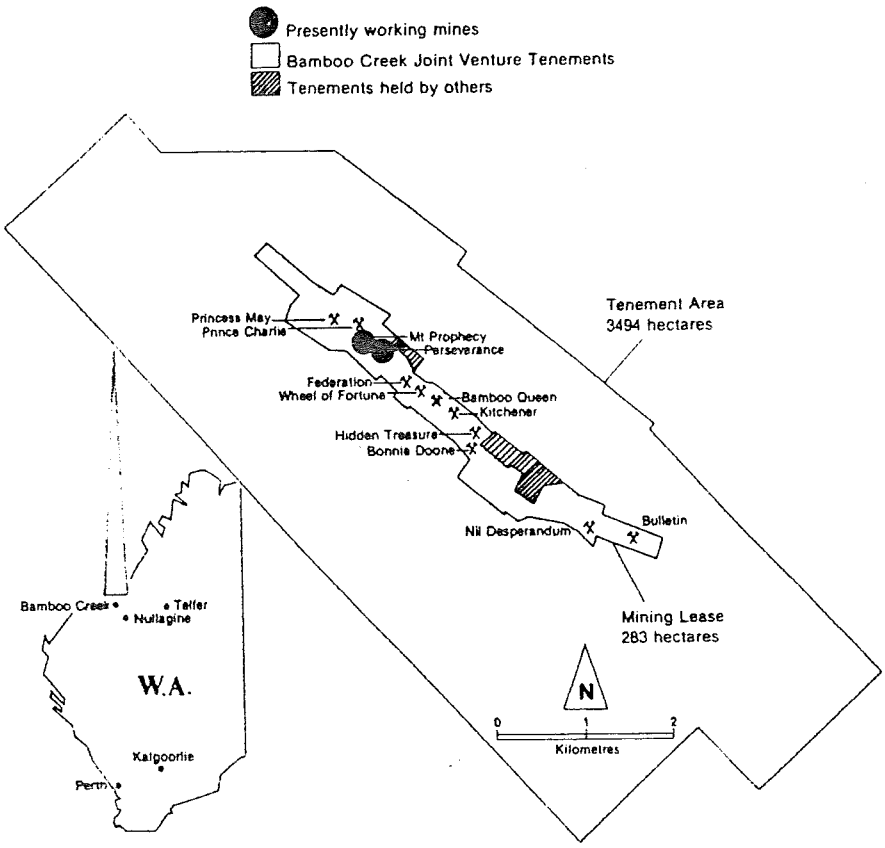



Fig. 1 Location of Bamboo Creek and overview of the leases.

The road to Port Hedland (about 250 Km long) is unsealed for its greater part.
The Pilbara is notorious for its hot and dry climate. In the summer temperatures reach 40 °C commonly. Winter normally shows temperatures around 30 °C. Marble Bar is known as the hottest town in Australia. Somewhere in the Twenties temperatures exceeded

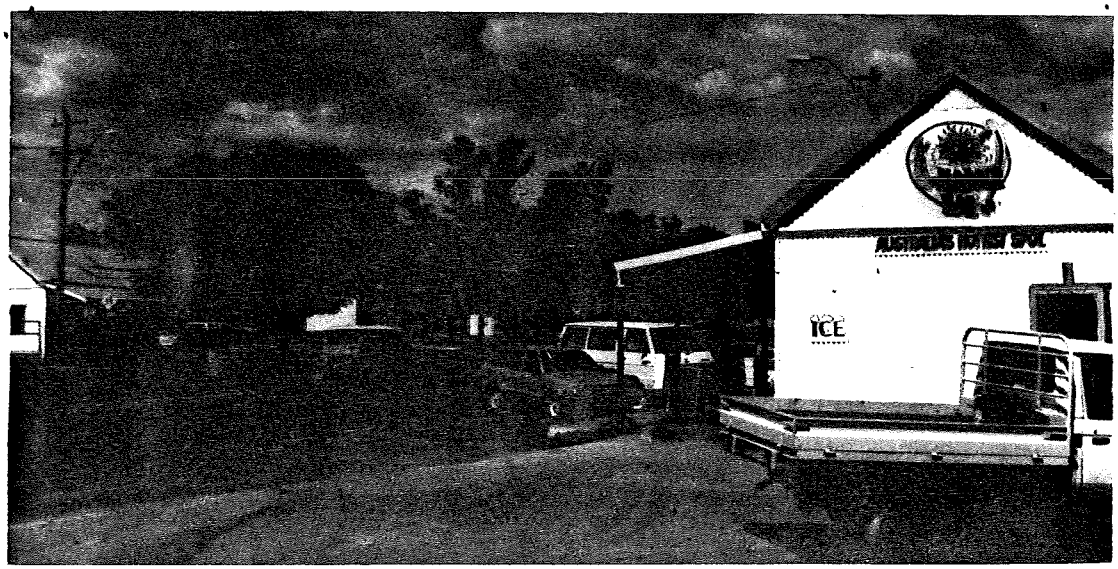



Fig. 2 Drugstore and main road of Australia's Hottest Spot: Marble Bar.

100 F (37.8 °C) for hundredsixty consecutive days (fig 2 & 3). The total yearly rainfall averages 378mm. Due to the high temperature evaporation is high and exceeds the rainfall by far (fig 4). Vegetation therefore is limited to spinifex and eucalypts such as the rough leafed gum and the snappy gumtree. Around the mine one can expect to find a variety of wild life. Kangeroos, wallabies, emus, donkeys, camels and snakes are common. With respect to the snakes one should be careful. Specially in the summer it is better not to take risks. During and shortly after heavy rainfall the dirt tracks become impassable. Rivers which are normally dry suddenly flood and cut the roads.

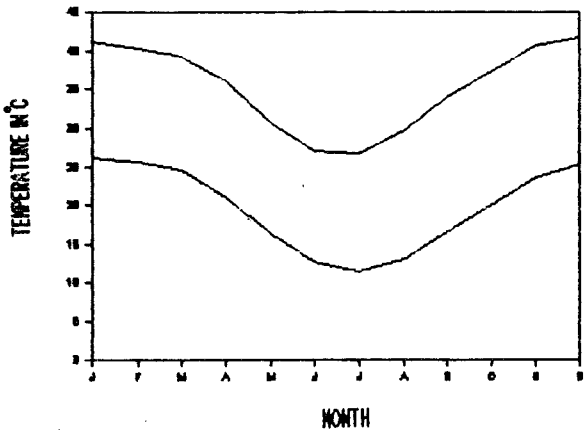



Fig. 3 Average max. and min. teperature in °C.

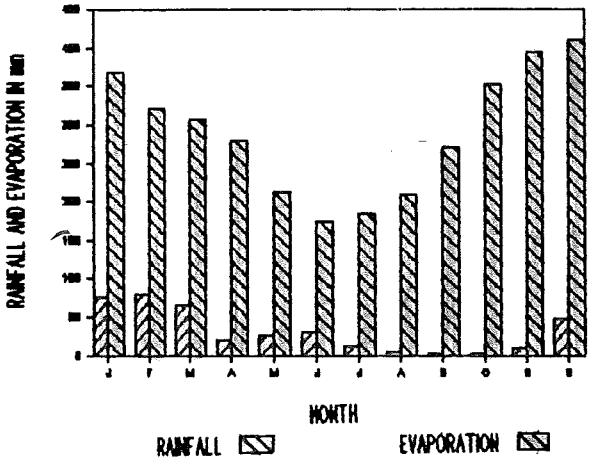



Fig. 4 Average rainfall and evaporation in mm per month.

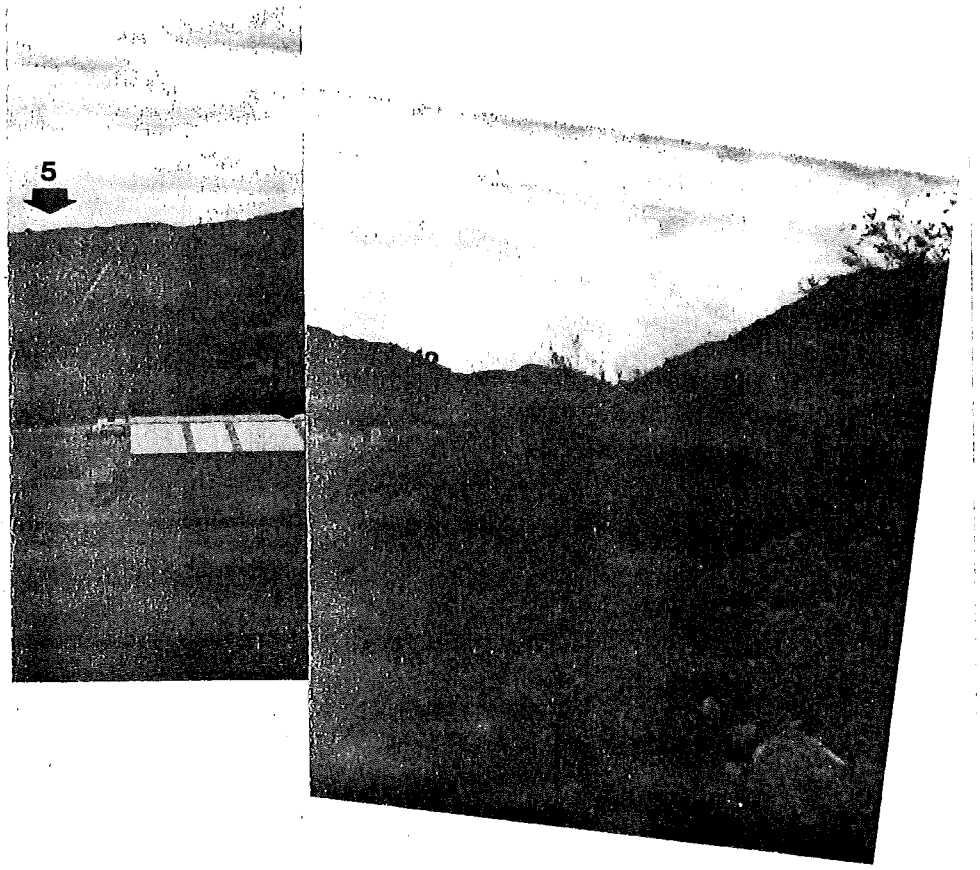
The mine owns an airstrip and leases a small six person aeroplane. Marble bar is visited twice a day to collect the mail and to drop the children at the school. Port Hedland is visited only once or twice every week for transport of people from and to the mine, or to transport the gold.

## 2.2 THE MINE

The mine area is fairly hilly. The minesite is situated in a valley with a width of about 200 meters oriented in a northwesterly direction.
The township was designed to accommodate about 150 people. There is a canteen, a small shop, a swimming pool, a first-aid post, a videoroom, and of course, a pub. Families live in normal houses, the miners have single rooms (fig 5).
With the current problems of reduced production not all accommodation is used. The total population numbers more or less 70 persons of which 50 form the mine workforce. The spare rooms are now used to accommodate tourists.

Fig. 5 Aerial photograph of the township at Bamboo Creek.

The mill, workshop, store and laboratory are situated one kilometre northwest of the township. In the laboratory one can find an AAS instrument. All sample analyses therefore can be carried out on the mine itself.
The mill is a carbon cyanide in pulp plant (CIP) with a recovery of 95 % and yearly capacity of 100.000 ton. The plant was designed and constructed by W.M.C. Engineering Services Pty. Limited.
Directly behind the mill on the hillside one can find the main water tank and a transport ramp to transport ore to the crushers. The mainoffice is situated on the opposite valleyside.
The underground office, workforce facilities and mine entrances are situated 500 m northwest of the mill. Following the main road even further will bring you at the Bamboo Queen (BBQ). The BBQ used to be an open pit but is now depleted (fig. 6).
A borefield parallel to the mine valley is responsible for the watersupply. The water is of drinking quality although the calcium content is a bit high. At the moment no treatment is required (fig. 7).
As one can see on the overview map (appendix A) all old mines are situated along the valley. Twelve of them are part of the BBC JV mining leases. Two mines are still owned by others.
The leases have an overall length of 11 km and they are about 500 m wide.




mine valle

Fig. 6 Mine valley overview.

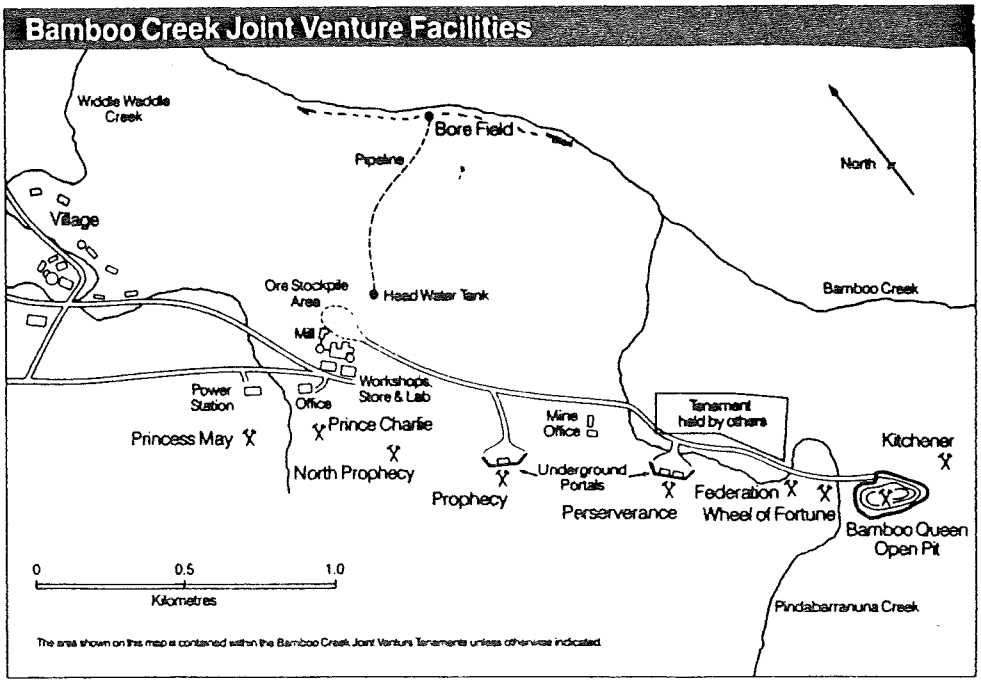



Fig. 7 Layout of the mine valley at Bamboo Creek.

## 2.2.1. THE MILL

The ore is transported from the mine to the mill with a 35 ton Euclid truck and dumped in the hopper above the primary crusher. The primary crusher crushes the ore to -90 mm afterwhich it is screened into +27 mm, +10 mm and -10 mm fractions.(fig. 8)
The +27 mm and +10 mm fractions are crushed respectively by the secondary and tertiary crusher afterwhich the material is recirculated to the screens. The -10 mm material is transported directly to the fine ore bin (850 ton capacity).
The material from the fine ore bin is transported to the 8'* 8' Marcy ball mill where it is ground to -75 micron. The discharge is screened and the oversize (50 %) is recirculated. The undersize is put over strake tables to recover free gold. A cyclone removes remaining coarse parts.
All -75 micron material is now transported to the thickener to reduce the water content of the pulp.
In the next part of the mill the gold is extracted from the ore. The underflow of the thickener passes through the five pachucas after the addition of lime (CaO) to increase the ph to 10.3. Sodiumcyanide (NaCN) is added directly to the pachucas. In the presence of oxygen and an alkaline environment NaCN is reduced to AuCN.
After this reduction the pulp passes through six absorption tanks that contain activated carbon. The carbon will absorb the gold. When the carbon from the first absorption tank is ready to be stripped all carbon is shifted one tank ahead. The reactivated carbon is added to the last (no 6) absorption tank. In this way a reversed circulation is created that will enable high recoveries. The pulp that has reached number six absorption tank goes out to the tails and should contain less than 0.5 gm/ton of gold.

In the last mill section the gold is recovered from the carbon. First contaminants are removed by washing the pulp with a 5% HCl solution. The carbon then is educted to the elution column (4000 litre capacity) where it is washed with water and eluded with a hot (120 °C) caustic cyanide solution to remove the gold from the carbon.

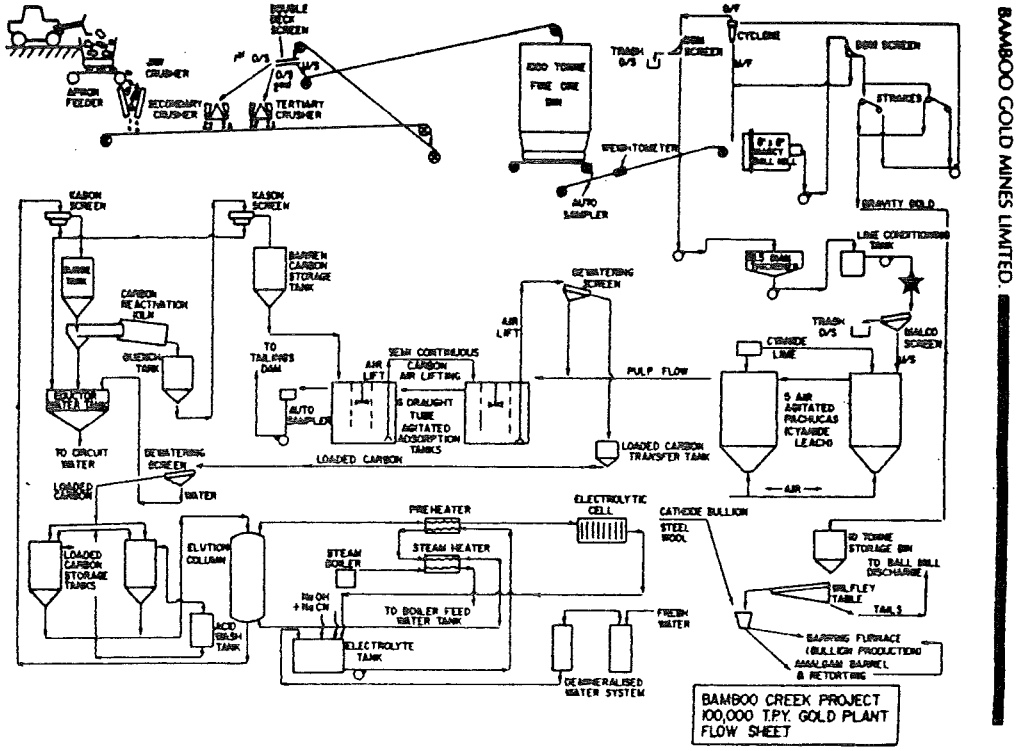



Fig. 8 Flow diagram of the mill. The introduction of the batterysands in the mill is pointed out. (★)

The carbon is returned to the milling circuit after reactivation. The gold bearing liquid is preheated and transported to the electrowinning tank. A 100 Amp 3.5 Volt current is placed through steel wool cathodes and the gold in solution is precipitated onto them. The remaining liquid is recirculated to the elution column. The steel wool cathodes are placed in concentrated HCl which dissolves the steel wool leaving the gold behind. Together with flux the gold is smelted at 1200 °C. The flux removes all the remaining sillicates and the gold is poured into a boullion.

To monitor the whole milling and mining operation samples taken on a regular basis are analysed in the laboratory. The results are important with respect to grade control and mill performances. A fire assay method is used to determine the gold content of the various samples.

First the samples are crushed and then pulverised in a C2000 ring. After being pulverised the samples are split into 300 gm samples for the fire assay.

Some mill samples are treated differently because they already are grinded to some extent and part of the sample is liquid. The liquids too have to be analysed and are seperated first. The dried solids are pulverised.

Together with a flux the samples are smelted in a fusion furnace at 1000 °C. The smelt is poured into a mould and allowed to cool down. At the bottom a lead button containing the gold and silver is formed.

The lead buttons are placed on a cuple in the cupelation oven to oxidize the the lead leaving the silver and gold prill behind. The gold and silver are dissolved seperately afterwhich the results can be read from the AAS machine using standard solutions for comparison.

## 2.2.2. UNDERGROUND MINING METHODS

The design of the original mining method was based on the following model of orebodies : Parallel steeply dipping orebodies of up to 14 metres wide, generally 10 to 20 metres apart and with reasonable extend. Ground conditions as evidenced in old workings were anticipated to be excellent.
An uphole benching retreat method was proposed to mine these orebodies. (fig. 9a & 9b). Small 1.5 cubic metre bucket loaders (TORO 150 LHD) were planned to operate in 3 * 3 m drives on lode. Access from the surface to each level of the mine was garantied by a 4.25 * 4 m decline at a grade of 1 in 8 that would be the roadway for all transport of equipment and haulage of waste and ore.

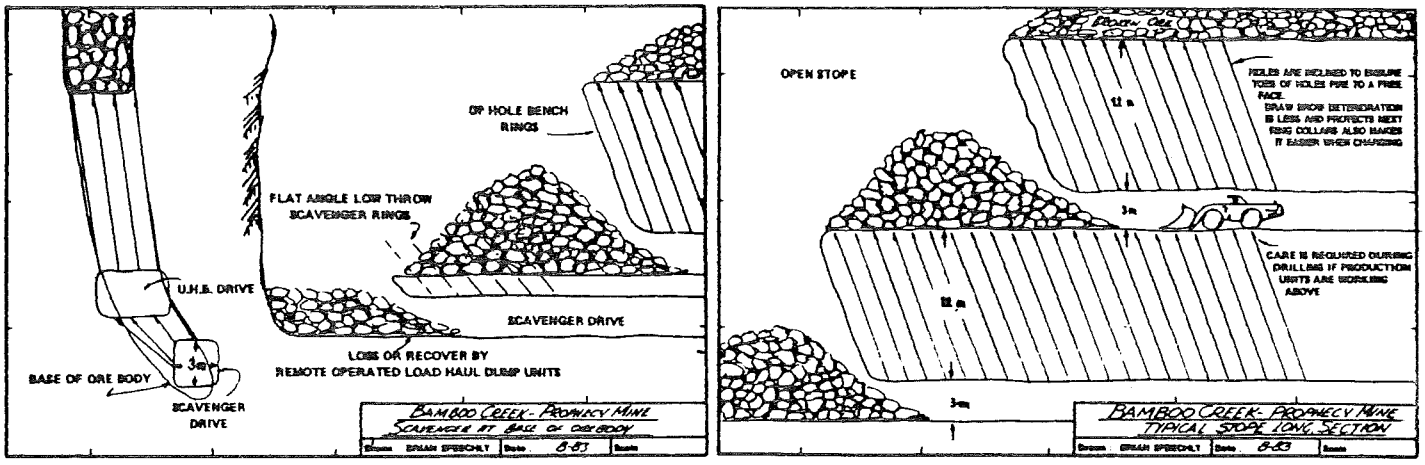



Fig. 9a & b Schematic representation of the uphole benching retreat mining method.

The decline also would provide a base from which each level could be developed.
Every 15 metres a new level would be developed. The remaining 12 metres to the level above (3 * 3 m drives) was considered far enough with respect to the accuracy of blasthole alignment.
Once two levels were driven to the extremities stoping could commence at either extrimity and retreat to the point of which access from the decline connects with the production level.
Stoping would commence at the top and progress down. This feature would minimize capital development and normal development required to bring the mine in production.
The orebodies in real turned out to be much smaller than anticipated. The actual lode width seldom exceeds 3m and varies all the time.
Driving on lode with 3 * 3 m drives necessary to allow the LHD to transport the ore out of the stopes caused enormous dilution.

Blasting the ore between the levels in one time increased dilution even more.
The introduction of scrapers to transport the ore out of the stopes made it possible to reduce the stope dimensions according to the actual lode width and reduced the dilution.
The ore between two levels is now fired in several separate slices to be able to follow the lode.
Capital development is reduced because only one level at the time has to be developed before toping can commence.

### 2.2.3. BOREFIELD

Despite pumping tests it is not exactly known how much water the aquifer can produce in the long run. Estimates range from 280 m^3 to 830 m^3 production capacity per day.
The aquifer presumingly is filled with water from an area of 29 km^2. If 4% of the annual rainfall is captured this would lead to a possible production capacity of 1000 m^3 per day.
However rainfall is not evenly spread over the year and may show large variations.
The primary permeability of the aquifer is low. Secondary permeability only reaches to a depth of 45 m. This limits the storage capacity of the aquifer. A shortfall in the dry season therefore is not inconceivable.
Nine of the twelve available boreholes are connected to the waterpipeline. The last three boreholes are a bit out of the way and not incorporated in the powersupply. This part of the aquifer therefore does not produce.
All boreholes are operated automatically 24 hours a day. Multistage submersible Grundfus pumps are used to pump the water. These pumps have a low flowrate but can pump a high head. By adding extra stages the head can be adjusted to suit the borefield layout.
The mill is designed to process 100.000 ton of ore per year. This would require 600 m^3 water per day. The watersupply therefore is rather critical, which implies that the aquifer production is kept for drinking water, shortfall and standby purposes. The milling operation uses mine water as much as possible.

## 2.3. HISTORY

Alluvial gold led to the discovery of the Bamboo Creek gold deposits in 1893. Shortly after de discovery two state batteries were erected and operational in 1894. With the increase of mining activities the population rose to a peak of 250.
Mining activities took place in a large number of seperate mines. A third batterystamp was erected (fig. 10) at the peak of activities.
Watersupply however was not sufficient to accommodate the high production rate and the large population. There was also a lack

Fig. 10 Old batterystamp near the Bulletin mine.

of working capital.
The surface deposits were depleted by 1900 and shaft sinking became necessary to continue mining. This was an expensive exercise during which production fell back. The state batteries also ceased to work.
The area lied quietly until 1914 when a new state battery was erected. From 1915 till 1928 the Kitchener mine was responsible for most of the production. Later on more mines started to produce again. A higher goldprice, as a result of the war, stimulated the mining industry even more.
The mining activities had their ups and downs until 1955. The BBQ had to close down because of the working conditions despite high grades. In the period 1955-1978 almost no gold was produced.
Kitchener Mining had obtained most of the leases by 1978 and first steps were taken for renewed mining. Appendix A shows the tenement plan.

BAMBOO CREEK EXPLORATION DRILLING 1929-1984

| DATE | ORGANISATION | HOLES | TYPE | LENGTH m | NUMBERS |
|---|---|---|---|---|---|
| 1929 | W.A. Dept of Mines | 3 | core | 444.1 | DDH1-3 |
| 1955-58 | W.A. Dept of Mines | 19 | core | 4070.0 | DDH4-22 |
| 1960 | Bamboo Creek Gold Mines N.L. | 19 | core | 744.3 | 1-19 |
| 1969-70 | Woodsreef Mines Ltd. | 21 | core | 3728.9 | B1-12 |
| | | | | | M1-9 |
| 1980 | CRAE | 5 | core | 1388.0 | KD1-3 |
| | | | | | KD3A-4 |
| 1981 | CRAE | 71 | core | 6157.0 | PS series |
| | | | | | PU series |
| 1981 | CRAE | 6 | core | 638.0 | D1-6 |
| 1981 | CRAE | 74 | percus. | 4359.0 | A1-59 |
| | | | | | M1-15 |
| 1983 | Bamboo Creek J.V. | 4 | core | 240.0 | BQ1-4 |
| 1984 | Bamboo Creek J.V. | 19 | percus. | 1175.0 | BQP1-13 |
| | | | | | WFP1-3 |
| | | | | | BDP1-2 |
| TOTAL | | 241 | | 22944.3 | |

Fig. 11 Overview of the exploration drilling at BBC.

Corporate policy was that large scale mining would make it possible to build an own mill instead of relying on a statebattery. This would increase profitibility by increase of scale.
The first Joint Venture was signed in 1980 between Kitchener and CRA-Exploration. Forrest Gold Company, a subsidiary of CRA, took care of the exploration programme (fig. 11). Disagreement as to the scale of exploration ended the Joint Venture.
+Kitchener Mining succeeded in forming a new Joint-Venture. This Joint-Venture had the following constitution: 50% Kitchener and 50% Bamboo Gold (Plenty river mining). It was not before long when Bamboo Gold decided to reduce its interest in the project to 25 % by selling the rest to Haoma N.W. N.L. The current constitution of the Joint Venture is 50% Kitchener Mining, 25% Bamboo Gold and 25% Haoma N.W.
Total production since the first gold was discovered according to the Mines Department records adds up to 84000 ton. In total 2294 Kg of gold were recovered. The average grade must have been round about 30 g/ton. The Kitchener mine and the Prophecy were responsible for over 50% of this production. Besides the gold 108 Kg Silver were recovered.
The exploration programme, together with the information of previous drillholes, resulted in an estimation of probable and possible reserves of approximately 1.2 million ton ore with an average grade of 11.3 g/t (fig. 12)

PRODUCTION OVERVIEW FROM 1893-1981 AND ESTIMATED RESERVES

| MINES (*) | PRODUCT. | RECOV. GOLD | AV.RECOV GRADE | PROB. RES. | GRADE | POS. RES. | GRADE |
|---|---|---|---|---|---|---|---|
| | t | g | g | t | g/t | t | g/t |
| MT. POPHECEY | 20363 | 546979.6 | 26.86 | 350720 | 11.00 | 350000 | 11.00 |
| MT PROPHECY NORTH | | | | 50800 | 9.90 | 76300 | 4.60 |
| KITCHENER | 15311 | 617561.2 | 40.33 | 12000 | 27.40 | 104000 | 20.00 |
| CHARLIE-MAY | 10769 | 224169.2 | 20.82 | | | 41400 | 20.00 |
| BULLITIN | 7367 | 312459.3 | 42.41 | 53000 | 12.20 | | |
| BONNIE DOONE | 4225 | 117801.3 | 27.88 | | | | |
| BAMBOO QUEEN | 3531 | 106336.6 | 30.12 | 64940 | 10.00 | 65000 | 10.00 |
| FEDERATION | 3074 | 68804.0 | 22.38 | | | | |
| WHEEL OF FORTUNE | | | | | | 43695 | 10.56 |
| TOTAL | 64640 | 1994111.2 | 30.85 | 531460 | 11.26 | 680395 | 11.40 |

Fig. 12 Production overview and reserves estimates.
*(All these mines are included in the J.V. areas.)

## 2.4. GEOLOGY

Bamboo Creek is part of a Precambriam shield. This shield contains large granite batholithes of circular shape. Between these intrusions belts of greenstone rock occur. Bamboo Creek lies in such a greenstone belt.
All rocks in the BBC area belong to one of the following three formations:

-Protorozoic Forescue Group with volcanics and sediments.
-Archean Warrawoona Group with basic and ultrabasic volcanics (now altered to greenstone).
-Mt Edgar Batholith.

The contact between Proterozoic and Archean is an unconformable one. The first one has hardly been deformed wereas the second one is highly foliated and strongly deformed.
The Forescue Group contains Mt. Roe Basalt, Hardy Sandstone and BBC porphyry. In the Warrawoona Group small intrusions of Mt Roe basalt and BBC porphyry can be found.
The contact between batholith and the Warrawoona Group is a tectonic one where lateral shortening resulted into nearly vertical folds.
The mafic and ultramafic rocks are the most important ones in this formation because they contain the gold ore.
As a result of the strong deformation the original layering has disappeared. Horizontal changes occur on a 10 to 100m scale, whereas vertical continuity, in general, does not exceed 50 to 200 m. This provides a lensoid structure with corresponding discontinuity of the lodes.
Gold mainly occurs in almost vertical shearzones and sometimes is associated with quartz and carbonate (pyrite galenae and fuchsite are good indicators too).
The gold may be occluded in pyrite or precipitated onto it but at least 50% constitutes free gold.
The lodes vary in width between 0.5 m and 2.0 m and the length seldom exceeds 120m. The lodes tend to plunge 45 to the west. Numerous shearzones and faults make it hard to follow the lodes. The erratic occurrence of gold makes it even harder. Recognizing gold bearing structures is more important than the gold grade during diamonddrilling.

## 3. THE POSITION OF THE MINE

The first exploration boreholes were drilled in 1929 by the W.A. Department of Mines. Until 1970 a total of 62 holes with a total length of 8987 m had been drilled.
CRAE started drilling in 1980 to gain additional information necessary for a reliable feasibility study. The exploration programme was focused on the BBQ, the Mt. Prophecy and the Perseverance (fig. 13).
Right from the start it was anticipated to make the BBQ an open pit and to develop the Prophecy and the Perseverance underground. The open pit would supply the mill with enough ore to use its full capacity until underground production could take over.
JACIA, a subsidiary of the Griffin Coal Mining Group, was contracted to take care of the management. Bellways as a contractor should carry out the open pit development and production.

Fig. 13 Entrance of the Perseverance decline.

The first problems occurred shortly after the start of the project. The mill was operational long before production had reached its planned volume. This shortfall originated in unexpected problems with the processing of the tailings.
The tailings had been scheduled to be reprocessed by the mill during the start up period. Mixed with the open pit production they had to pass through whole the milling circuit. Unfortunately the tailings did not settle in the thickener. To increase the settling rate a flocculant was added, however without success and the processing had to be stopped.
The open pit produced ore of a substantial lower grade than planned. The low grade was caused by a wrong production method used by Bellways. Bellways was payed for by the ton and not for grade. On top of this the old workings appeared to be more comprehensive than anticipated.
Apart from these design errors a lot of money was lost on unnecessarely expensive contracting work. Instead of employing their own geologist a consultant was hired. This caused a lack in continuity and expertise necessary to understand such a complicated mineralization as encountered in BBC.
Continuity of management was seriously hampered by the frequent

coming and going of people which was triggered by the disappointing results. In this way a lot of information was lost. This disasterous cource was arrested when Kitchener Mining took over the management from JACIA in 1985.
By that time debts had reached a peak and the mill only worked on half its capacity or less because underground production never reached the planned volume. When the BBQ was depleted production fell back from 1300 ton per week to 550 ton per week.
The underground mining methods used did not suit the type of deposits and it was difficult to control the head grade. The man who designed the mining programme no longer worked for the mine and the original plans therefore were not adjusted.
The introduction of scrapers improved things a little bit. Because of the enormous debts the mine had to reduce costs to an absolute minimum. Allmost all development was stopped and diamonddrilling was stopped completely by the new management.
With these drastic measurements one succeeded in stabilizing the finances and even realized a small profit. To secure the continuity of the mine and to pay back the loans it became essential to develop new levels. Sinking the decline further therefore is the only long term solution. Recently the Swiss Finance Corporation has approved an additional loan of 2 million dollar to finance this operation.
For the intermediate future it will be hard even to maintain the production on the low level as it is. The current stopes are almost mined out and the ore that is left is difficult to mine. Driving the decline will even further interfere temporarely with the production.
To overcome this temporary draw back in production and cashflow the mine is interested in projects which can be realized on short notice. Investment should be limited and if possible personnel and machinery on site should be used.

## 4. PRODUCTION ALTERNATIVES

In my opinion there are three ways to generate some cash in a short term that fulfil the restrictions described before :

-An open pit
-Production from old workings
-Processing of batterysands

During the feasibility study of the mine one was particularly interested in the BBQ, the Prophecy and the Perseverance. Most of the exploration drilling therefore was conducted in these areas. To start an open pit outside this area would require a new, expensive and time consuming exploration programme. Under the circumstances given this option therefore is not feasible.
Besides one may wonder whether it would be likely to find substantial deposits at the surface of a mined out area as it is. Modern technology has made it possible to drop the cut of grade considerably compared to a few decades ago. It is quite well possible that substantial ore reserves are still present in the old workings. It is not even unthinkable that part of it is already broken. In the old days it was custom to recover only the high grade pockets. The low grade ore was blasted into the stopes to form a muckpile on top of which the miners could stand to mine the next pocket.
Most of the old mines were developed by shafts. Sinking or renovating shafts or to start a decline is an expensive and time consuming excercise. This option therefore is not the answer to the current problem.
One could consider to develop the lode which is still present in the BBQ. With an increasing overburden ratio mining the open pit was stopped leaving a part of the ore behind.
One could start mining from the bottom of the open pit and drive on lode. Production most likely is limited to a few odd ton, costs however are restricted to mining costs only.
The third alternative, the processing of old tailings, is the most promising one. Sampling of the dump during the feasibility study of the mine resulted in an estimate of well over 30.000 ton of tailings with an average grade of 2.4 g/t.
According to tests it should be possible to process these tailings in the mill or by batchleaching. The advantages are, low investment costs and short term realization.
This option therefore is considered in more detail. By processing the tailings one is liable to pay royalties to the Mines Department and to act according to their regulations. These regulations concern the reporting of progress and the time in which the royalties have to be paid.
The excisting regulations were agreed at the first processing proposal to treat tailings in the mill. Should another method be chosen than these regulations should be reconsidered. The current regulations can be found in Appendix B.

## 5. THE TAILINGS

Before a decision can be made regarding the way to treat the tailings, analyses have to reveal where and how the gold is present. The total amount of tailings and average grade have to be computed. Possible changes of grade within the dump have to be determined.
Most of these investigations were conducted in the beginning of the BBC gold project (1978). Analab, Micron Research and Associated Surveys were responsible for the sampling, testing and computing. The results have been verified randomly and when necessary additional tests were carried out.
The tailing dump is situated northwest from the mine between the Kitchener and the Bonnie Doone. The original dump has been eroded by rainfall and part of it is washed away. Now these washouts form a second and third dump. The original dump is the largest. The second dump is spread out over a fairly large area and covered with spinifex. The third dump however is so small in quantity and grade that it is not further considered (fig. 14).
In an early attempt to process the tailings 800 ton was transported to the mill. Since this first attempt to process the tailings failed these 800 tons still are available.

### 5.1. ESTIMATING VOLUME AND GRADE

The volume calculations were taken care of by Associated Surveys (Aust) pty ltd.. 144 samples, taken and analysed by Analabs, were used in the grade calculations. Information on the samples can be found in appendix C. To estimate the volume of the dump, aerial fotographs were used to calculate the area. The height of the dump was measured with the depth of the drillholes made for sampling. Both a reversed circulation rotary drill (80mm) and a hand auger (76mm) were used. The overall grades were computed from the Analabs data by assigning 12.5 m. grid cells over the entire pile surface for which a mean grade was interpolated cq. extrapolated from the adjacent drillholes using a weighted distance from each.
My own calculations, using the same samplesites but different samples, and taking cross sections, instead of a grid, showed more or less the same results with respect to the total amount. The washout dump in those calculations appeared to be considerably larger in volume, the main dump was smaller. This is partly because of variation in calculation and probably partly because of further washout.
A dozen samples randomly taken were in accordance with the Analab results. The computed results are shown in the table below. My own calculations are between brackets. The density of the sand was determined at 1.62 ton/m^3.

| | | |
|---|---|---|
| Tailings no1] | 35828 ton | [28119 ton] |
| | 2.4 g/ton | [2.7 g/ton] |
| Tailings no2] | 3157 ton | [5369 ton] |
| | 2.1 g/ton | [2.1 g/ton] |

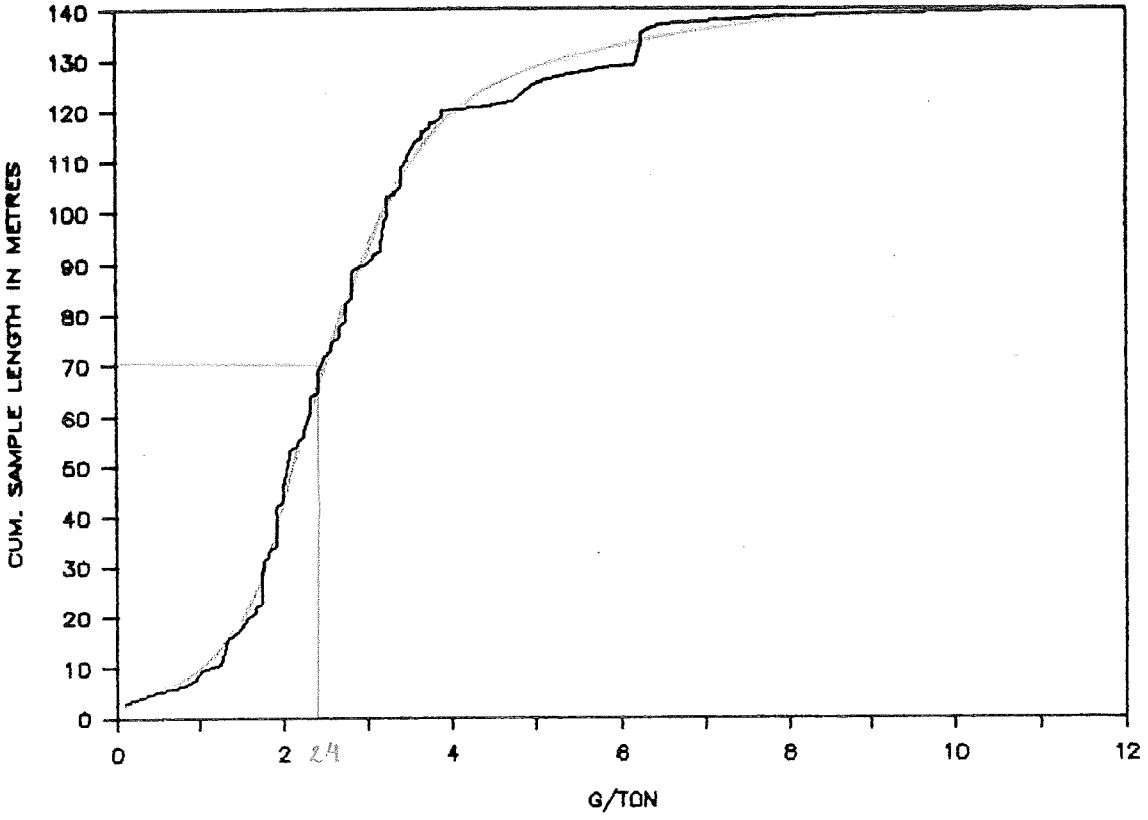



Fig. 15 Cumulative grade distribution tailingsdump 1.

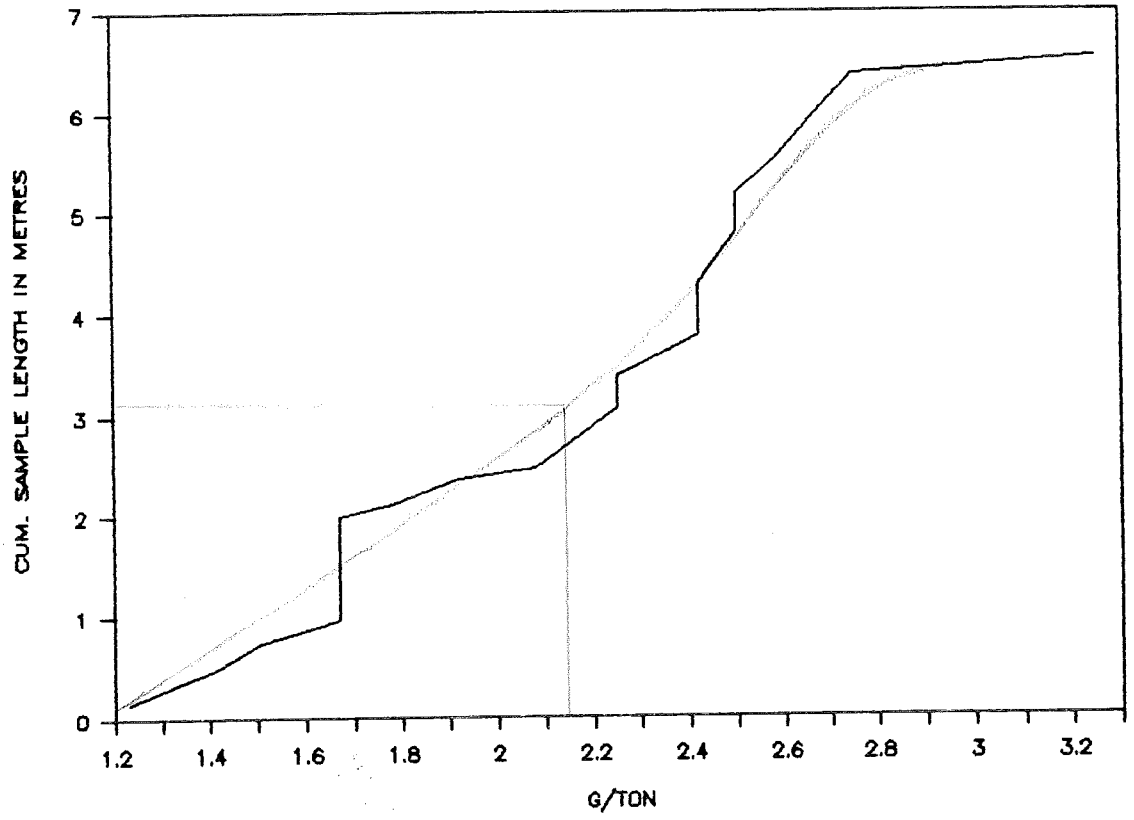
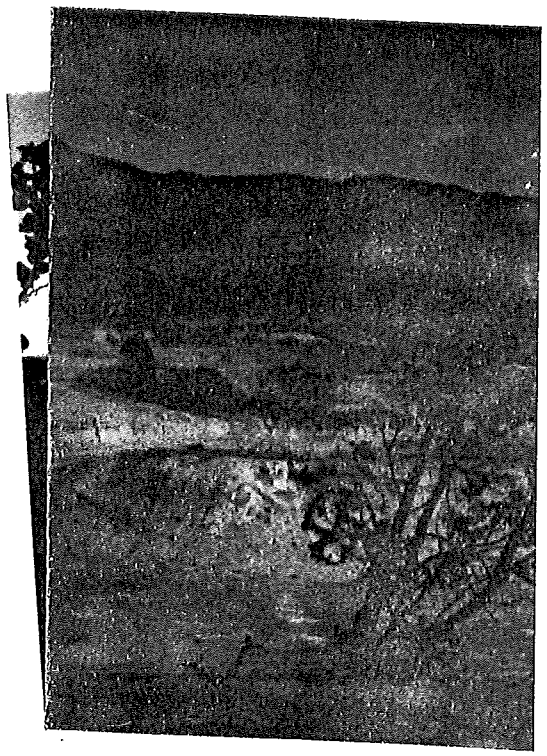



Fig. 16 Cumulative grade distribution tailingsdump 2.



mp .

Fig. 14 Batterysands overview.

When the sample grade is plotted against the cumulative borehole length the average grade follows for half of the total borehole length. This value agrees well with the computed results (fig. 15,16).
Tailings between the road and the old batterystamp are the highest in grade. The first batch therefore should be filled with these sands. Part of these sands were transported to the mill at the first attempt to process the sands. As mentioned before these sands are still available.

## 5.2. PROCESSABILITY

The tailings are very fine as they are and this would suggest that no further grinding is required to treat the sands. Sizing analyses show the gold distribution over the various fractions and how this distribution is effected by cyanidation (fig. 18).
Cyanidation was carried out for samples without further grinding and for samples grinded to respectively -106 micron and -45 micron.

SIZING ANALYSES -tailings

| SIZE | SOLN | MASS | MASS | GOLD ASSAY | GOLD TOTAL |
|---|---|---|---|---|---|
| um | ml | g | % | ug/g | ug | % |
| +500 | 11 | 1.1 | 4.61 | 50.71 | 2.15 |
| +250 | 231 | 23.1 | 3.13 | 723.03 | 30.66 |
| +125 | 219 | 21.9 | 2.13 | 466.47 | 19.78 |
| +45 | 186 | 18.6 | 1.40 | 260.40 | 11.04 |
| -45 | 353 | 35.3 | 2.43 | 857.79 | 36.37 |
| TOTAL | 1000 | 100 | 2.36 | 2358.4 | 100 |

MICRON RESEARCH TAILINGS DUMP SUMMARY RECOV.

1) Cyanidation at size 48.9 %
2) Cyanidation at 100% -106 micron 70.6 %
3) Cyanidation at 100% -45 micron 75.6 %
4) Long term static leach 60.3 %

Fig. 17 † Summary of the Micron Research results.

Fig. 18 Sizing analyses of the battery sands

Recovery is considerably increased by grinding. Treatment without grinding only gives a recovery of 48.9%. This can be increased to 70.6% or even 76% by grinding to -106 cq -45 micron (fig. 17).
Our tests are in accordance with these findings. The ball mill capacity however is limited and a size reduction to -45 micron would decrease its capacity from a normal 15t/h to 10 t/h or less. Grindingcosts involved are considerable and will only be marginally covered by the increase of recovery.
The original idea was to treat the sands in the mill to fill up spare capacity until normal production would take over.
Unfortunately it turned out that the battery sands did not settle quickly enough in the thickener. This reduced the mill capacity considerably and the processing of the tailings was stopped. Using the whole milling circuit therefore would defy the purpose.
Apart from the 24 hours cyanidation tests a long static leach was conducted without further grinding. Although results were not spectecular a recovery of 60.3% was realized within 36 days.
Of course it will be a great advantage if it is possible to seperate a large quantity of waste material before grinding. Gravitational seperation using a Wilfley table was not a great success. Although a reasonable concentrate was realized, only

16% of the gold was recovered. This of course is not enough to be succesful. The Micron research results can be found in appendix D. Summarizing the Micron research results the following conclusions can be drawn :

- -Gold in the tailings is very fine and part of it is distributed in the rock matrix.
- -This rock matrix is fairly cyanide resistent.
- -To increase the recovery extremely fine grinding is required.
- -It is not very likely that the extra costs of grinding will be made up for by the extra gold-recovery.
- -A long static leach gives a satisfactory recovery without the need of further treatment

## 5.3. RECOVERY

Micron research tests show that a 60.3% recovery is possible in a static leach. This result was realized in 36 days at 15% water content. At the end of this test the material was repulped to 50% pulp density with water and agitated for an hour.
Not only the final result is important. To know how this recovery develops in time during leaching is equally important. Once this relation is known one can estimate how long it will take to reach break-even and optimize the process.

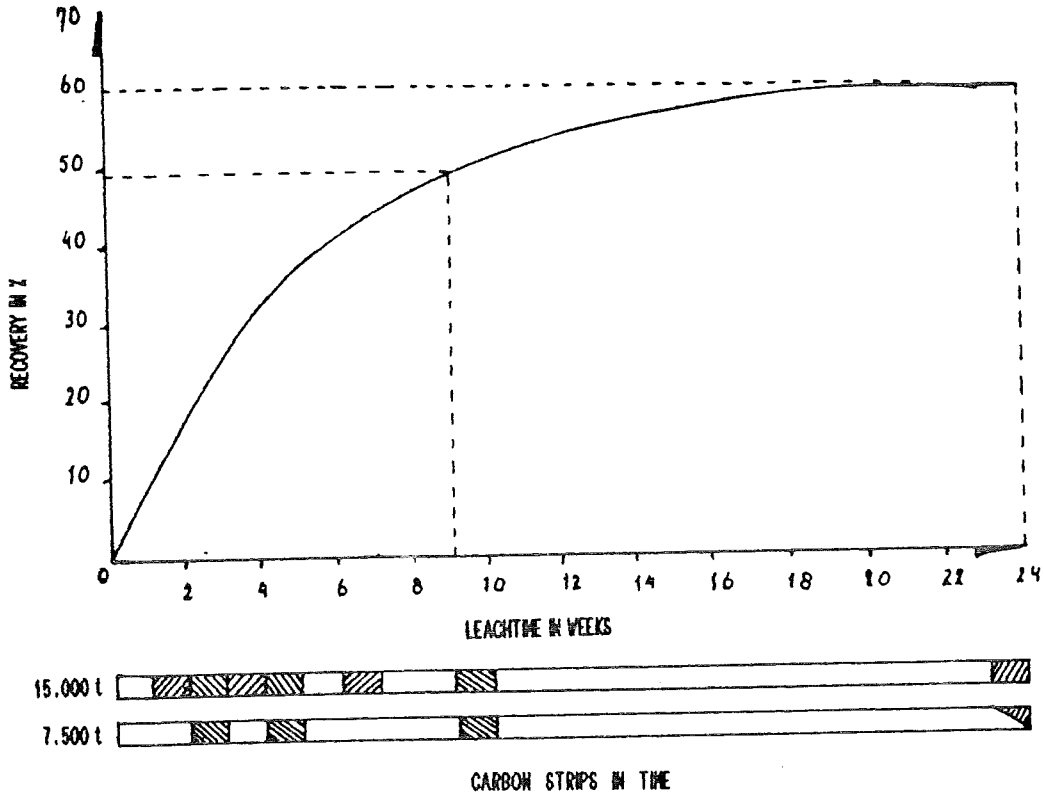



Fig. 19 Estimated recovery versus time relation for batchleaching and stripping frequency.

With normal cyanidation a recovery of 48.9% is realized within 24 hours. It is likely that this recovery can be realized relatively quick during leaching. Main factor is the waterflow through the batch.
To estimate this waterflow the following experiment was carried out : A plastic pipe with a diameter of 31 cm and a length of 2.5 meter was filled with tailings over a length of two meters. At the top water was added. After the water had reached the bottom the flowrate was measured by measuring the drop in waterlevel against time.
From this data the flowrate was computed at 9.6 l/hour m^2. The water in the batch has to travel twice the distance. To compensate this, half the calculated flowrate is used in calculations concerning the batches.
If we look at the proposed time recovery curve it follows that the 48.9% recovery is reached after 9 weeks (fig. 19). In this period of time the water has circulated 13 times according to the estimated flowrate. This should be enough to realize the 48.9% recovery. After all this part is easy to recover.
Although the recovery versus time relation is of no concern for the end result, it has a great influence on the cashflow. The quicker the gold is recovered the quicker the investments can be payed back.
The first batch in this respect is a test case and the recovery time curve should be followed closely. If necessary the original curve has to be adjusted, so next batches can be planned according to the real curve.
Once the batch size is known together with the estimated grade and recovery one can calculate how much gold approximately will be extracted from the batch by leaching. The number of strips to extract the calculated amount of gold depend on the amount of carbon used. Once the amount of carbon has been chosen stripping is directly related to the recovery time curve.
One has the possibility to spread the earnings by changing the amount of carbon used in the carbon columns. Costs will increase slightly with smaller portions because of additional labour.

## 6. PROCESSING THE TAILINGS

The Micron Research tests clearly reveal that no high recovery can be expected without further grinding. The only way to make processing feasible is to restrict the costs. In the light of the limited tonnage investment therefore is limited. The following alternatives will be evaluated :

-Processing in a part of the mill
-Processing by batchleaching

### 6.1. PROCESSING IN THE MILL

As mentioned before the mill can only be used when the thickener is bypassed. This can be realized by feeding the tailings straight into the pachucas without grinding. A small adjustment of the mill would be necessary (fig. 8).
It would require a conveyerbelt and a feeder hopper. Also a scrubber drum to remove organic material should be part of the installation. All these equipments are available for hire in Marble Bar from a former tin mining operation.
Treating normal mine output and tailings at the same time would not be much of a problem. Monitoring the results of the two different ores seperately would be hard to realize. It is important however to know how efficient the process is. The best solution would be to treat the tailings on their own and stockpile he normal mine output. In this way equipment hire also is restricted.
There is no space problem because the planned stockpiles are almost empty due to a lower production than planned. If the treatment of the tailings takes about three months, in total about 7500 ton has to be stockpiled. If 3 m is taken as stockpile height and 2 t/m^3 for density only a 35 metre square area is needed.
Afterwards the stockpiled ore can be treated at an increased rate. With the current overcapacity of the mill this should not be a problem. It is likely that production will fall back even more when the sinking of the decline is in full progress. At the moment production is down to 600 m^3 a week.
With the low grade and limited recovery, processing of the tailings should be conducted at a high rate to keep earnings at a satisfactory level. With two twelve hour shifts this could be realized quite easily, because both the ball mill and crusher are not used during this operation. Their operators can be allocated to the night shift. Of course this will increase labour costs due to penalty rates but then again the equipment hire will be reduced.

## 6.2. BATCHLEACHING

Batchleaching is a low cost way of treating gold ore. First a basin is built with dirtwalls and plastic. Next the basin is filled with the tailings together with some of the chemicals. Cyanide is put on top of the batch and a substantial amount of water is added, enough to start a circulation. Water with dissolved cyanide slowly penetrates through the sand and is collected by a drainage system (fig. 20).

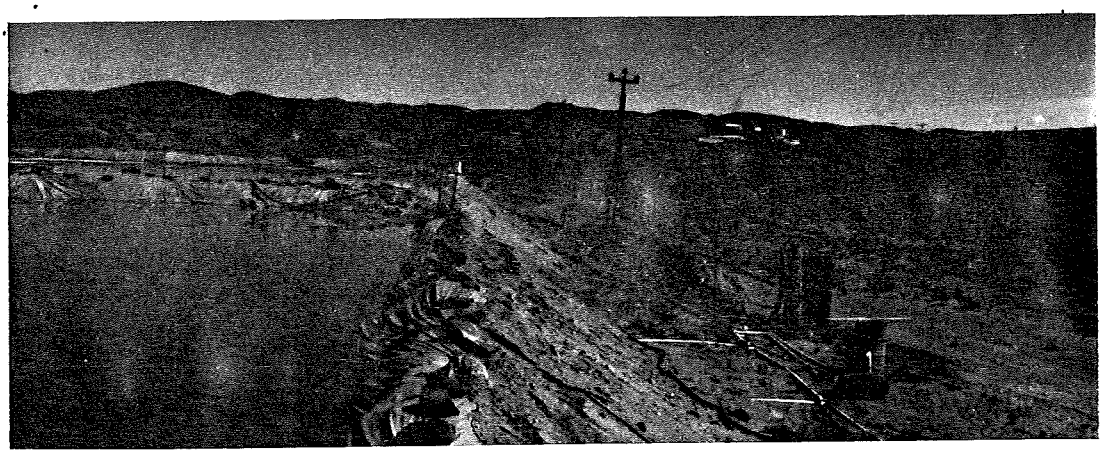


Fig. 20 Batchleach example in Marble Bar. Piping, pump and carbon column can been seen on the right.

The water is pumped on top of the tailings again and in this way a circulation is created. A carbon column is placed in this pumping circuit and the water passes through it before it flows back in the batch. The cyanide-gold complex is collected in the carbon.
When the carbon is saturated with gold it is removed and stripped at the mill. After reactivation of the carbon in the kiln the carbon is put back in the circuit.
The leaching process can be continued for months at low costs. Sampling of the waterflow and the carbon on a regular basis will tell if continuation is useful.
The whole batchleach project can be executed by the mine itself. Another possibility is to get a contractor to do the job.
There are many contractors available in Western Australia who carry out small mining projects. They accept all responsibility and pay royalties to the mine. In this way earnings are guaranteed without any risk. On the other hand these earnings will be significantly lower than when the whole project is executed by the mine.
All required machinery is available on site as well as the workforce. By taking care of all the work by themselves equipment and labour can be better utilized. Contracting a consultant will help to overcome the lack of experience and reduce the risk of failure.
One is free to choose the amount of tailings treated at one time. However the larger the size of the batch the more time is needed to construct and fill one batch. It will take longer to reach break-even and the amount money upfront will increase.

Small batches on the other hand will be to expensive. Every batch needs its own "infrastructure" like a pump and sump. Small batches produce not enough gold to pay for this.
Sizes of 7500 ton and 15000 ton are commonly used. These are realistic with respect to the amount of tailings available. Plastic sheeting for these size batches are available from stock.

## 7. TAXATION

At the moment gold is the only untaxed commodity in Australia, however the Government is considering the introduction of a goldtax.
Gold taxation is a hot issue in Australian politics. Because of the many debts it would not effect BBC in a direct way, but it would effect the gold mining industry.
In a paper produced by Savory Milln Ltd. (members of the London stock exchange) this subject was discussed in the light of investment in Australian Gold. The following text originates from this report:
"No gold mines in Australia currently pay tax. This has been defended by the miners. Particularly those in Western Australia, for many years. The argument stated for the non taxation of gold mining companies was, and remains, that:

- The Government benifits from the foreign exchange earned on gold sales.
- Gold mining provides employment in areas of relatively high unemployment.

The Hawke governments' Treasurer, Paul Keating, proposed reforming of the tax system, including the taxation of gold companies.
Following vociferous opposition, it was not included in the last budget.
The introduction of a goldmining tax on companies has invoked a strong reaction. It has been estimated that it will cause so much loss of employment that it will cost the government 60 million A$ in lost tax revenues. There is, however, no logical reason to tax base metal and other companies and to exclude gold mines. Providing capital expenditure is allowable to offset, and there is a capital allowance at rate commensurate with the level of inflation, the burden would not be intolerable, and there should be a decrease in mining activity. If exploration is also allowable it could be benificial for it would encourage the finding and developing of other orebodies, thus extending the life of the company.
At the moment however the whole question of gold mining tax has been shelved for the time being, ostensible to allow discussion between the industry and government on this issue, but also perhaps until after the Western Australia elections in March. A commission is due to report back to the government in 1986 but no moves should be expected until after the next Federal elections due at the latest at the end of 1987. Political compromise and expediency may make the question of a gold tax too much a political hot potato even after the next federal elections. We do not expect any taxation before 1990, and then only in a very muted form."

## 8. CASHFLOW

In order to evaluate the previously discussed projects one has to look not only at the final result, but also at the additional costs on top of the running costs, and the cashflow fluctuations. The project has to fit in with normal work. Sensitivity for changes in the parameters is very important.
In the section below, several parameters affecting the cashflow are explained in some detail and the end result is discussed with respect to the remarks mentioned before.
Stripping costs for batchleaching are higher than those for milling. The carbon in the mill is better charged because one uses 6 adsorbtion tanks in a reversed circulation flow instead of only one or two in batchleaching. Also less labour is involved in stripping the mill carbon because the carbon does not have to be transported.
Chemical consumption is based upon the mill figures. For lime this will be realistic. For cyanide the mill figures most probably are too high because less gold is available in the tailings, and as a consequence less cyanide will be consumed.
One has to differentiate between cyanide consumption and the overdosis needed. Tests from Micron research show that 0.1 kg of cyanide is consumed per ton of tailings. For the mill this normally is 0.75 kg per ton. The total amount of cyanide used in the mill is 1.15 kg per ton.
If the same overdosis of cyanide is used for batchleaching as in the mill 0.5 to 0.6 kg of NaCN per ton should be used. This concentration is realized with the first cyanide addition. The second cyanide addition will bring this to 1.15 kg per ton.
Analyses have to reveal how much cyanide should be used. The cashflows of the various options can be found in appendix E.

## 9. CHOICE

A choice between the alternative options is not only a matter of finance. Possible risk, spreading work and optimalization of machinery and labour are important factors too. Normal production continues and should not be interfered with.
The critical financial status of the mine does not allow high additional costs on top of the running costs. Investments therefore should be kept to a minimum and be paid back as quick as possible. Even if the chosen option would not be the optimal one with respect to the earnings.
The mine uses a four week financial account. A regular cashflow is important not to upset this periodical account. With a heavily fluctuating cashflow, problems might occur with the creditors.
Treatment in the mill provides a very regular cashflow and the money upfront is the lowest. Unfortunately the final result of this option is rather disappointing. In the base case total profit is only half the profit that can be realized with batchleaching.
During the processing of the tailings normal income will stop. This is a very big disadvantage not withstanding the fact that increased production afterwards will catch up.
Adjustment of the mill is a rather complex operation. Delay is not unlikely and has a tremendous effect on the output. Equipment hire expenses will continue and normal production will be delayed even further.
Despite the positive aspects of treating the tailings in the mill the 7500 ton batchleaching process is more promising. The upfront investment is only slightly higher, however compared with the mill option the final result is much better. Prolems that might occur will not interfere with normal mine production.
The total cashflow for batchleaching is obtained by adding the cashflows of all the batches. They are not all alike because some of the costs occur only once, for example the costs for a consultant.
The total cashflow can be adjusted to suit the mine by changing the interval between the start of the different batches. Specially in case of the 7500 ton batches there is enough room for manoeuvring. For instance one could finance the last three batches with the income of the first one.
The smaller batches have the advantage of greater flexibility with respect to the water needed.
There are not many advantages in building 15.000 instead of 7500 ton batches. The final result is only slightly higher in comparison with the 7500 ton batches. Investment capital however has to be larger. The starting up period is longer and as a consequence it takes longer to reach a break-even point.
For the larger size batches more water at once is needed. There is less opportunity to check the water reserve estimations. With the small batches one has the possibility to check water reserves and water consumption. If needed the following batches can be postponed.
Altogether the 7500 ton batchleaching is the most promising project. It has a great flexibility with acceptable risks and good profit. The money upfront is only half of the weekly running costs and break-even is realized within five weeks.

In the following part of this paper it is discussed how to go about this project.

## 10. THE EXECUTION

### 10.1. BATCH SIZE

One is more or less free to choose a suitable size. It is easy to adjust the length and width in accordance to the location. The plastic commonly used is available in rolls of 100 m length and 6 m. width. Special tape goes with the plastic to connect the seperate sheets. It is essential that these welds are absolutely watertight. Every weld is an extra weak spot and increases the chance of leakage.
Apart from these rolls is it possible to get standard size sheets which do not have the seal problems. These sheets are relatively cheap. Standard sizes available are 50*24 m and 76*36 m. If it suits the batch size the large sheets are preferred.
There is not a big price difference between the rolls and large sheets, although the large sheets are slightly cheaper. It is more the quality of the plastic that dictates the price. The better the quality the thicker the plastic the higher the price.
With a tailings density of 1.62 ton/m^3 and a batch height of 2.5 m a batch of 7500 ton would require a size of 1850 m^2. Taking in account the dirtwalls of three metres height and slopes of 60, a batch size with outside measures of 50*50 m will do (fig. 21).

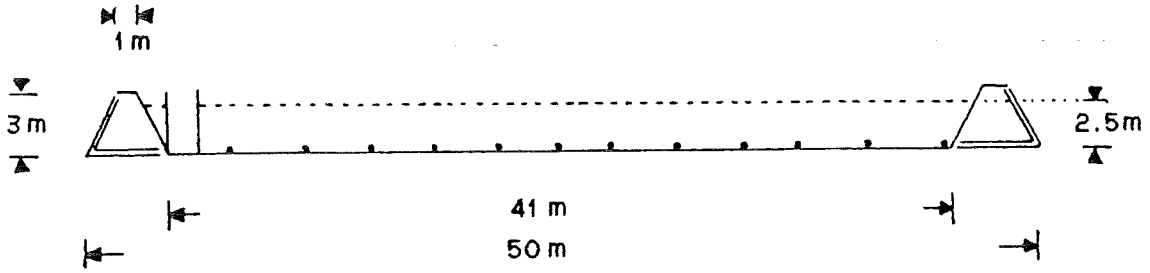



Fig. 21 Dimensions of a 7500 ton batch.

Two standard sheets of 50*24 m will do nicely. If the slope angle differs and a slightly larger size sheet is needed it is allways possible to put in one or more strokes of 6 m from a roll.
Jaylon industries Pty Ltd. sell the 50*24 m sheets from stock.
For drainage 50 mm perforated piping is used covered in filter sock. The distance between the pipes is about 3 m. Total piping length for the proposed batch would be 660m.

### 10.2. WATER

Rather large quantities of water at once are needed for batchleaching. In the Micron Research test 15 wt% of water was used. All calculations are based on this water consumption figure but changes are quite well possible.
It is important to realize water circulation as quickly as possible for only then gold can be recovered. During the leaching process water lost by evaporation must be replaced.

The watersupply as it exists can not cope with these large demands. Fortunately, there are enough alternatives to compensate for this lack of water. The following alternatives can be used to meet the extra demand for water :

-True Blue
-Prince Charlie
-Kitchener Mine

Early investigations, to find a location for the borefield, did not reveal any aquifers near at hand other than the one currently used.

## 10.2.1. TRUE BLUE

The True Blue is not part of the BBC JV leases. Nevertheless the owner has allowed the JV. to pump water out of the mine. At the moment the True Blue is used as an intermediate pump-accumulationstation. Minewater of the Perseverance is pumped to the True Blue before it is pumped to the mill. This detour was necessary to overcome problems with pumpcapacities.
The capacity of the True Blue is about 750 m^3 water. It is quite well possible that more water is available but this can not be reached due to the position of the pump. Because the True Blue is not part af the leases the plans to verify this were not available. At the moment the pump is located in an old decline instead of in the main shaft.
Even for normal watersupply it would be an advantage to look into this possibility. When the new decline of the Prophecy Mine has reached the new levels current mine production will increase. Water consumption therefore will rise as well. A larger buffer capacity would be advisable.

## 10.2.2. PRINCE CHARLIE

The Prince Charlie is located westerly opposite the mill and more or less under the current tailingsdam. In the early days water of the Prince Charlie was used in the mill. At the moment mine water of the Prince Charlie has not been used for no obvious reason. Just like in case of the True Blue this should change regardless the leachingproject.
First of all one can not permit not to use an easily accessible water source. Secondly almost all infrastructure is available. Costs to utilize this source negligible therefore.
Thirdly it is quite certain that water in the Prince Charlie originates partly from the current tailingsdam. This would mean indirect recycling.
Fourthly there is the danger of water from the tailings finding its way to penetrate the aquifer of the borefield. Pollution with cyanide would be catastrophic for there are no other drinkingwater alternatives.

Draining the tailingsdam by pumping the Prince Charlie would slow down or even stop this process. Cyanide in water used in the mill would only benifit the mill operation.
All these statements regarding the tailingswater are not proven and based on assumptions only. Nevertheless one can not permit to neglect them looking at the costs and risks involved. Pumping the Charlie therefore should start at once. When a geologist is available he should look into these assumptions and verify them.
Measurements of the waterlevels in the shafts of the Prince Charlie showed that the mine was partly filled with water. The water content was computed using a map of the underground workings of the Prince Charlie.
For the drive dimensions 2*1.5 m. was used. The stope width was taken to be 1 m. This resulted in a total waterreserve of around 1300 m^3.
I did not detect any cyanide in the samples I took in the various shafts of the Prince Charlie. The map of the underground workings can be found in appendix F.

### 10.2.3. THE KITCHENER

The Kitchener mine is the most important watersource regarding the batchleaching project. This mine presently is partly filled with water and the old workings are quite comprehensive. The watersupply therefore is in proportion around 5100 m^3.
Just like the Prince Charlie a map of the underground workings was used to compute the waterreserves. For the drive dimensions 2*1.5 m was used. The stope width was taken to be 1m.
The map of the underground workings only show one plane. An underground visit to the Kitchener learned that these workings are more extensive and that it is likely that the waterreserves are substantially bigger.
A small part of the reserves can not be reached due to the pump position. To be able to move the pump the waterlevel has to be lowered first. The map of the underground workings can be found in appendix G.
The water is well accessible via the main shaft. The distance to the proposed leachsite is only 150 m. Adventitious is the fact

Fig. 22 Catterpillar bulldozer (D9) at work to build a batch.

that pumping will make it possible to investigate the Kitchener Mine and sample the old workings. Since one is thinking of leasing the mine to contractors the knife cuts on both sides.
The distance between tailingssite and mine is 1500 m. No electricity is available and a transportable generator would be needed.
At the moment the True Blue is operating with such a generator but it will not take long till the True Blue is hooked up with the main powersupply. The powerline from the Prophecy decline entrance will be extended. The spare generator then could be used at the Kitchener.

## 10.3. LOCATION OF THE BATCHES

One has to take several factors in consideration when a location for the batches is chosen. It is essential that the site is horizontal or gently sloping. The ground should be suitable to handle, in this way a basin of dirtwalls can be built (fig. 22).
If the situation is suitable one could connect several batches with eachother by using one sump. Of course this only is possible if height differences can be used to get a natural waterflow from one batch to another. In this way one could save a lot on pumping installations. On the other hand it will make the operation more complicated and it will be harder to realize changes.
The distance to the watersupply should be kept to a minimum. Long distances will reduce the pump performances and then it will take too long to fill the batches with water. Of the same importance is the distance from batches to tailings. All transport done by loaders. Long travelling time is expensive and will slow down the batchfilling.
A suitable location was found between the Kitchener mine and the tailingsdump. The consumption of tailings for the first batches will create space for the later ones. The distance from leachsite to both tailingsite and Kitchener Mine is 150 m. The chosen area is not very suitable to connect several batches with eachother.

## 10.4. BATCH CONSTRUCTION

To stimulate the flow in the pipes the batch should gently slope towards the corner where the sump is planned. This sump can be situated in or outside the batch.
When the collection pond lies outside the batch, this has the advantage that valves can be used in the piping to regulate the flow. Part of the batch can be shut off temporarely by closing some valves in order to stimulate the flow in another area.
A disadvantage is that the piping has to perforate the plastic which increases the risk of leakage. One may wonder if the regulation possibility is worth to take this risk. It will be hard to judge if a part of the batch has flow problems or not.
Situating the sump inside the batch the problem of penetrating the plastic does not occur. There is also no risk of losing any

pregnant liquid if something goes wrong. Overflow is impossible. The pond can be used to add chemicals without any problem and to pump up the water.
A piece of large diameter plastic piping can be used to make this sump. During installation one should take care not to perforate the plastic.

## 10.5. CARBON COLUMNS

The carbon columns are just tanks that contain activated carbon. The pregnant leachwater circulates through these tanks and the dissolved cyanide-gold-complex is absorbed by the carbon.
In the mill the same process takes place in the absorbtion tanks. Then 6 tanks are used to improve the recovery with reversed circulation. The pregnant liquid flows from the carbon with the highest gold content to the carbon with the lowest gold content.
It is also possible to arrange some reversed circulation in case of batchleaching by using two columns instead of one.
The carbon used is rather expensive. Often only one tank is used. Because the carbon can be reused in the mill afterwards, costs are confined to the carbon loss. The carbon can only be used for a limited number of stripping cycles. This too has to be payed for. Altogether carbon costs in this case are minimal.
Because reversed circulation possibilities are limited in the batchleaching process the carbon will be charged less efficiently in comparison to the mill. Stripping costs therefore will be slightly higher for every ounce of gold recovered. There are also additional labour costs for the transport of the carbon to the mill.

## 10.6. FILLING OF THE BATCHES

Filling the batches with the tailings should be organized in such a way that compaction of the tailings is avoided. Compaction could reduse the flowrate considerably and even make leaching impossible. It is equally important not to damage the plastic.
All these conditions can be realized by backfilling the batches. First a batch is constructed except for the last wall, the plastic and piping. Filling is started at the far end of the batch together with the unrolling of the plastic, and installing of the piping. The principle is shown in the figure 23.
While leaching water is pumped on top of the batch the surface therefore preferably should be horizontal or even. The dirtwalls are only 50 cm higher than the filling. Filling the batch with water one could sprout the surface level.
The Lime (CaO) should be mixed simultaneously with the sand at the same time the batch is filled. Cyanide can be spread out on top of the batch. The cyanide tablets, will gradually dissolve in the water.
Cyanide is only effective in an alkaline environment. This is the reason for premixing the lime with the tailings. If cyanide

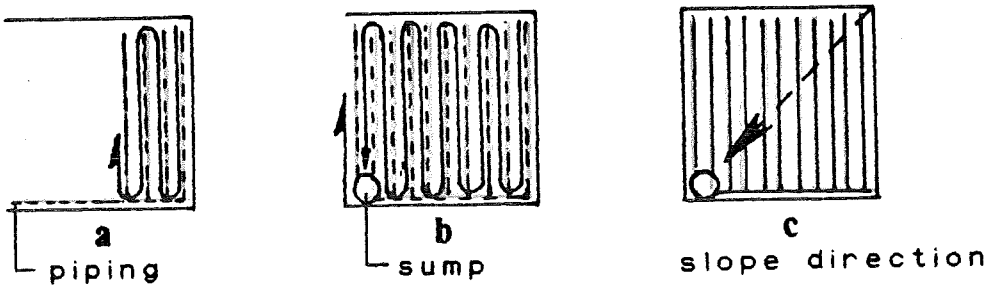



Fig. 23 Principle of batch filling.

encounters an acidic environment poisonous gases can develop.
When the batch is filled and the last dirtwall is erected the water filling can start. With a watercontent of 15% , 1125 m^3 water is needed for every 7500 ton batch. During the process evaporation has to be compensated for.
The total average evaporation per annum is 3388 mm. Specially in the summermonths November, December, and January, when evaporation reaches a peak. A batch with a surface of 45*45 will require an additional amount of 3260 m^3 in a 24 weeks period of time. This is 19.4 m^3 per day per batch. Depending on the watersupply situation one could consider to cover the batches with colourless plastic, or to reduce the leachtime.

## 11. LEACHTIME OPTIMIZATION

Although the proposed time-recovery curve may differ from reality it is certain that the recovery-rate will drop in time. The last few percentage points of recovery take the longest to be achieved. Only 11% is recovered in the last 16 weeks (fig. 24). In this period of time a lot of water will evaporate and one could consider to close down a batch within the planned leachtime to save on water and accept a lower recovery. This decision depends largely on the water availability for almost no extra costs are involved.
Another aspect is the fluctuation of the goldprice. It might be advisable to close down a batch if a price drop is anticipated. The new price might give a lower revenue although more gold is recovered.
In the first nine weeks of leaching this situation is not very likely to arise. So much gold is recovered in such a short period of time that small price falls are not of influence on the leachtime.
In the last 16 weeks relatively small price changes could effect the optimal leachtime. To help making this decision a graph has been drawn with revenues plotted in time for different goldprices. If for instance the current gold price would be 460 A$ and a 60 A$ drop is anticipated, the batch should be stopped if it is in production for over 18.5 weeks (fig. 25).
This principle, of course, will only work if management keeps an eye on the goldmarket and predicts the goldprice.
Changes in royalty rates due to higher goldprices were not taken into account when this graph was computed. At the moment only the old regulations exist concerning the royalty rates. It is therefore uncertain at which moment the royalties have to be paid.
Together with the mines department new regulations have to be agreed. Payment after a fixed period of time, related to the time recovery curve, should be the basis of these discussions.
In the calculations the old regulations were used. The royalties therefore were paid before the last 16 weeks. The graph for leachtime optimization therefore is not influenced by the royalty rates.

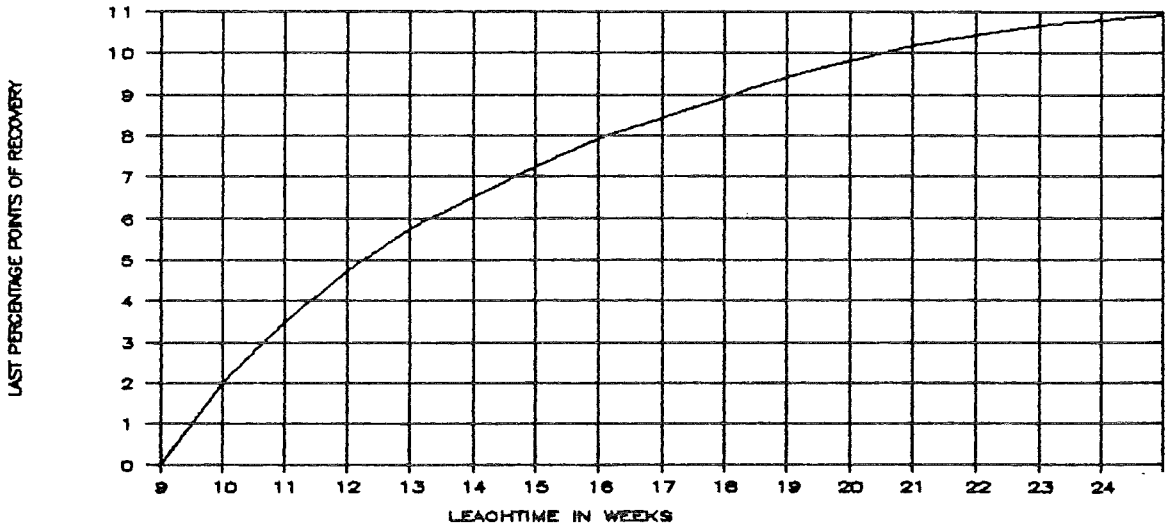



Fig. 24 Time recovery curve of the last 16 weeks.

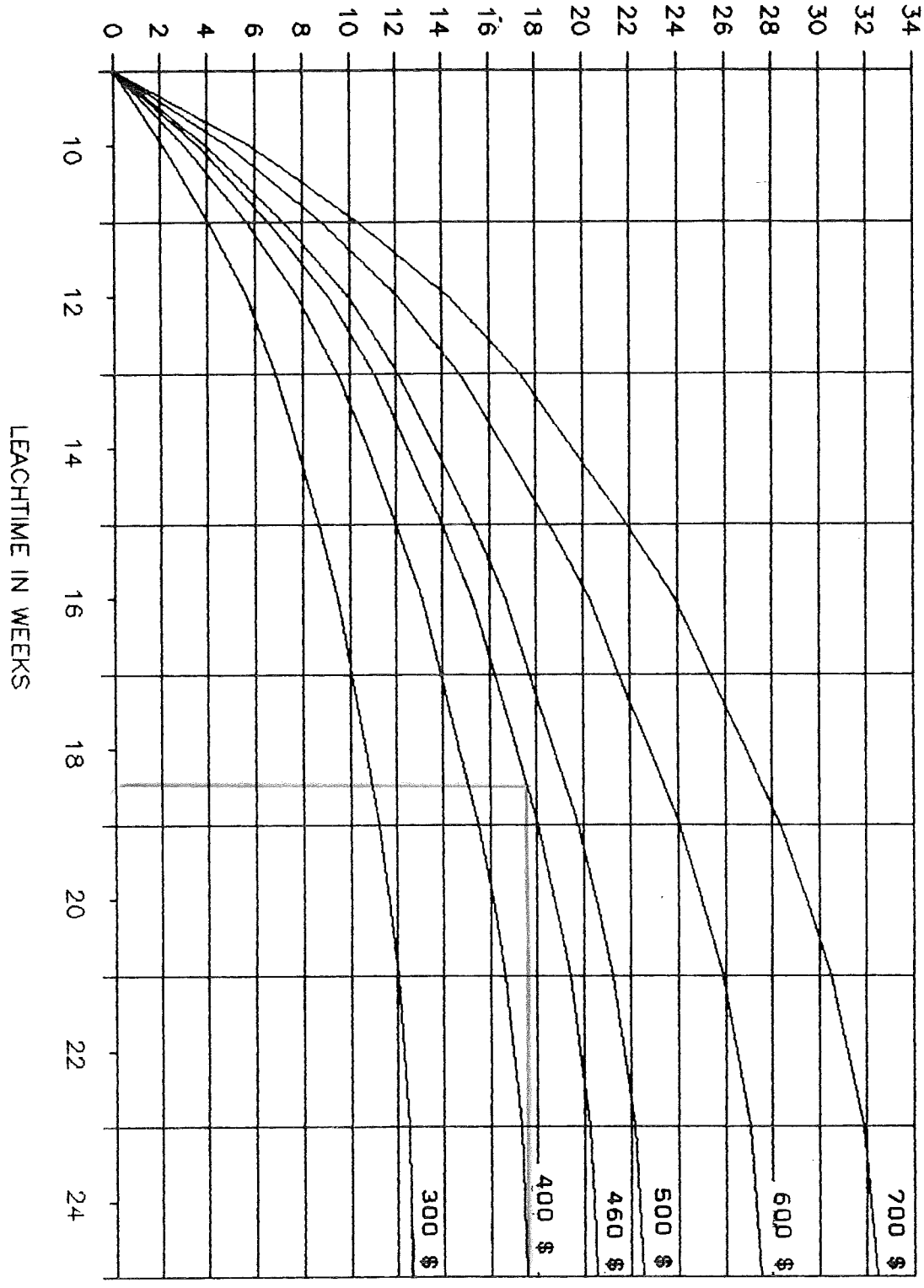



Fig. 25 Leachtime optimization graph with curves for several goldprices.

## 12. CONCLUSIONS

In the text it is shown that treating the batterysands can be very profitable. Even with large fluctuations in goldprice or costs the endresult still is positive.
Notwithstanding these promising figures one has to proceed very careful. Leaching is a new line of work at the mine and no expertise is available. In principle leaching is a simple exercise however mistakes can cause serious trouble and will increase the costs and reduce profitability.
Once a batch is filled it is hard to make changes. The advice of a consultant in an early stage will help to overcome starting problems and reduce the risk of failure. It is essential to be sure that the whole principle of leaching is fully understood the first time. Than the next batches can be built without outside help. The first batch therefore should be looked at as a test case.
As far as I can see there are two possible problems. The first one is the waterflow:
If the waterflow is lower as anticipated it probably takes longer for the gold to be recovered. Therefore it is important to check the waterflowrate and to keep an eye on the recovery time curve. The regulations to be negociated with the Mines Department for treating the tailings should be based on this relation. Royalties should not be paid before a significant part of the recoverable gold is recovered.
A second problem could be the water reserves:
The reserves are only estimates and based on the limited data available. By monitoring the waterlevel of the Kitchenermine and the amount of water pumped, one can verify the data used.
At least enough water is available to fill one batch and to provide the water needed to compensate for evaporation during the leaching process. This additional waterconsumption should also be recorded. Once one is confident about the real figures concerning the recovery time curve and waterconsumption, the following batches can be adjusted accordingly and started.
It is not only the extra income which makes it an interesting project. When the mine production is increased after the development of the the new levels, an increase in waterconsumption can be compensated by the new knowledge about additional waterresources.
Also sampling of the Kitchener Mine and possible negotiations about leasing the mine to contractors is now within reach when the Kitchener is pumped dry. At least one will be able to have a look in the old workings.
After the leaching of the batterysands one could think of starting to process other lowgrade ores the same way. The crusher capacity is large enough for the time being to crush a little additional ore. When enough ore is available this could be heapleached or batchleached. With the experience gained in the batchleachingprocess this should be easy to realize.

ACKNOWLEDGEMENTS

I wish to acknowledge Het KIVIfonds and Het Molengraafffonds for their financial support. In addition I would like to express my gratitude for the contributions of Ray Millard, Prof. Ypma, Alan Robertson, Stuart Stubbs, All personnel on site, Joke Hopman and Wilma Vierling in realizing this thesis.

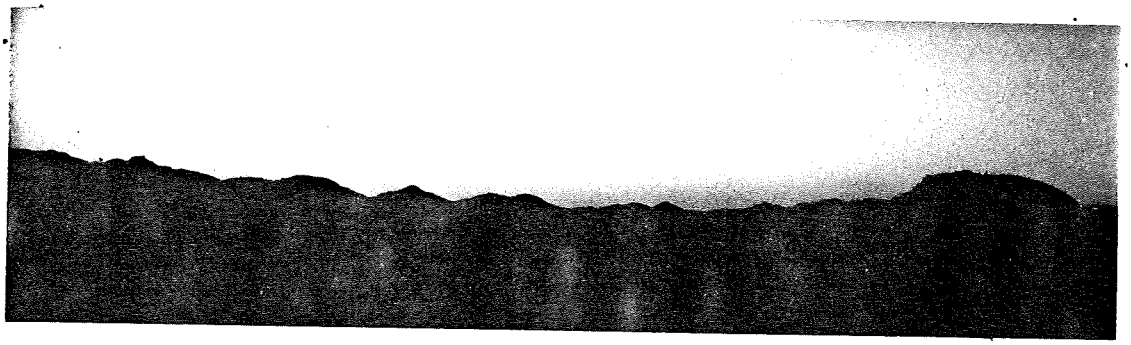
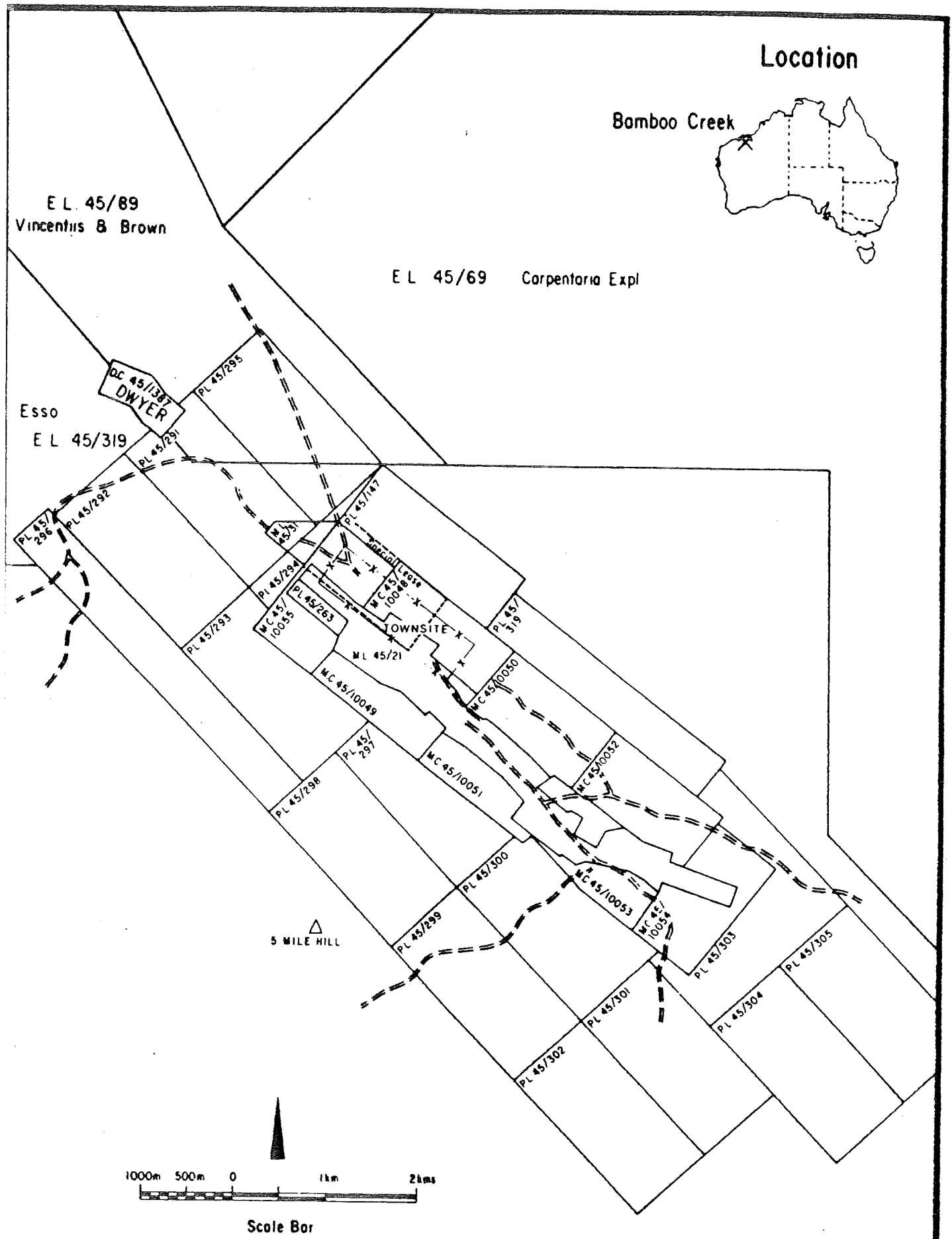


Fig. 26 Golden sunset in Bamboo Creek.

Appendix A

[ Tenement plan ]




# BAMBOO CREEK J.V.

# TENEMENT PLAN

# Appendix B

[ Mines Dep. Regulations ]

DEPARTMENT OF MINES

REGULATIONS CONCERNING TREATMENT OF TAILINGS [1]

1) Treatment being carried out, on a regular basis and completed within three months from notification.

2) Leaving the site from which the tailings are removed in such a condition as will not constitute a danger,and all plant equipment being removed from the site within six months of the completion or termination of treatment rights.

3) Submitting to the Under Secretary for Mines every four weeks during the term a statement on Department of Mines Form 27 of the tailings treated at the plant during the preceding four week period (the statement to be submitted no later than seven days from the completion of each four week period)

4) Payment being made within four weeks from the submission of such statement to the under Secretarty for Mines on behalf of the Government at the following royalty rates.

| GOLD PRICE | % of Gold price per ounce /ton treated |
|---|---|
| less than | $400.- flat rate $1.00/t |
| greater than | $400.- to $450.- 0.5 |
| greater than | $450.- to $500.- 0.6 |
| greater than | $500.- to $600.- 0.7 |
| greater than | $600.- to $700.- 0.9 |
| greater than | $700.- to $800.- 1.0 |
| greater than | $800.- to $900.- 1.1 |
| greater than | $900.- to be negociated |

The department reserves the right to terminate the arangements if royalties are not paid on time and if any other conditions are not complied with.

5) All operations of shifting the tailings to be the responsability of Kitchener mining NL and to be operated in accordance with the Mines Regulations Act and Regulations

6) The entering into a formal agreement if called upon by the Hon. Minister for Mines containing these conditions and such other conditions may be agreed by the parties.

[1] These regulations where agreed at the first attempt to treat the battery sands

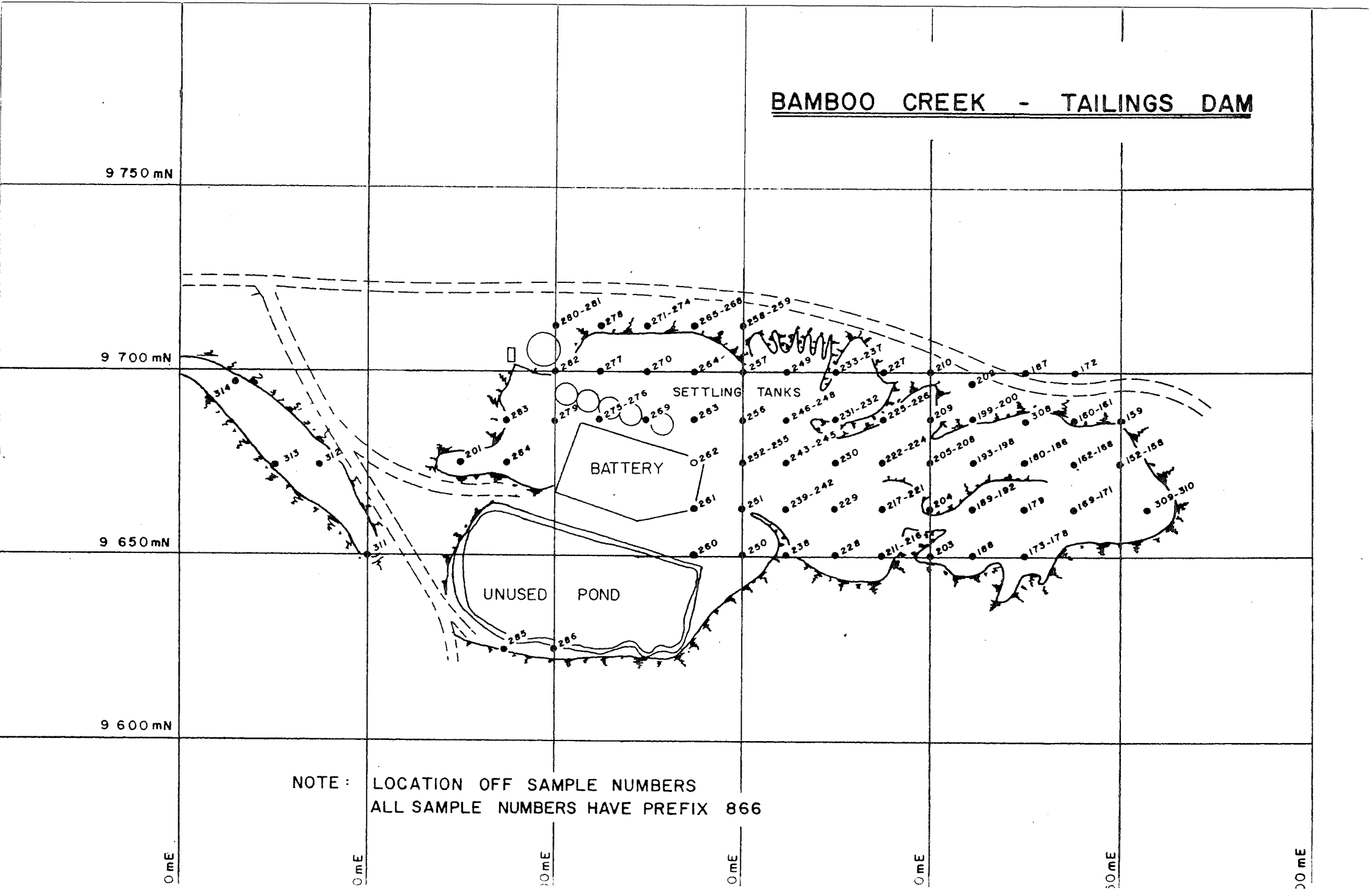
# Appendix C

[ Overview samples and map ]

## SAMPLES TAKEN AT THE TAILINGS DUMP

| SAMPLE No | GOLD g/t | DEPTH | COORD-X | COORD-Y | SAMPLE No | GOLD g/t | DEPTH | COORD-X | COORD-Y | SAMPLE No | GOLD g/t | DEPTH | COORD-X | COORD-Y |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | | | | | | | | | TAILINGS WASHOUT 1 | | | |
| 866152 | 1.540 | 0-1 | 11650.0 | 9675.0 | 866269 | 3.670 | 0-1.5 | 11525.0 | 9687.5 | 866287 | 1.920 | 0-0.25 | 11875.0 | 9650.0 |
| 866153 | 1.920 | 1-2 | 11650.0 | 9675.0 | 866270 | 0.850 | 0-1.5 | 11525.0 | 9700.0 | 866288 | 1.780 | 0-0.12 | 11875.0 | 9675.0 |
| 866155 | 1.030 | 2-3 | 11650.0 | 9675.0 | 866271 | 6.250 | 0-4 | 11525.0 | 9712.5 | 866289 | 1.670 | 0-0.12 | 11875.0 | 9713.0 |
| 866156 | 0.950 | 3-4 | 11650.0 | 9675.0 | 866275 | 3.920 | 0-1 | 11512.5 | 9687.5 | 866290 | 2.250 | 0-0.59 | 11775.0 | 9631.0 |
| 866157 | 0.992 | 4-5 | 11650.0 | 9675.0 | 866276 | 7.500 | 1-1.3 | 11512.5 | 9687.5 | 866291 | 2.420 | 0-0.5 | 11825.0 | 9625.0 |
| 866158 | 1.580 | 5-6 | 11650.0 | 9675.0 | 866277 | 6.250 | 0-2 | 11512.5 | 9700.0 | 866292 | 1.670 | 0-0.22 | 11825.0 | 9675.0 |
| 866159 | 3.920 | 0-0.23 | 11650.0 | 9687.5 | 866278 | 10.670 | 0-1 | 11512.5 | 9687.5 | 866293 | 1.420 | 0-0.34 | 11850.0 | 9625.0 |
| 866160 | 2.000 | 0-1 | 11637.5 | 9687.5 | 866279 | 0.100 | 0-3 | 11500.0 | 9687.5 | 866294 | 2.500 | 0-0.4 | 11875.0 | 9625.0 |
| 866161 | 2.420 | 1-1.45 | 11637.5 | 9687.5 | 866280 | 3.420 | 0-1 | 11500.0 | 9712.5 | 866295 | 1.230 | 0-0.15 | 11850.0 | 9675.0 |
| 866162 | 1.750 | 0-1 | 11637.5 | 9675.0 | 866281 | 2.170 | 1-1.5 | 11500.0 | 9712.5 | 866296 | 2.580 | 0-0.32 | 11725.0 | 9675.0 |
| 866163 | 1.670 | 1-2 | 11637.5 | 9675.0 | 866282 | 5.250 | 0-0.3 | 11500.0 | 9700.0 | 866297 | 2.420 | 0-0.4 | 11775.0 | 9655.0 |
| 866164 | 1.830 | 2-3 | 11637.5 | 9675.0 | 866283 | 6.170 | 0-0.3 | 11487.5 | 9678.5 | 866298 | 1.670 | 0-0.22 | 11750.0 | 9675.0 |
| 866165 | 1.830 | 3-4 | 11637.5 | 9675.0 | 866284 | 3.830 | 0-0.15 | 11487.5 | 9625.0 | 866299 | 3.250 | 0-0.17 | 11850.0 | 9687.5 |
| 866166 | 2.420 | 4-5 | 11637.5 | 9675.0 | 866285 | 7.600 | 0-0.1 | 11487.5 | 9625.0 | 866300 | 2.250 | 0-0.33 | 11850.0 | 9650.0 |
| 866167 | 2.750 | 5-6 | 11637.5 | 9675.0 | 866286 | 11.160 | 0-0.27 | 11500.0 | 9625.0 | 866301 | 1.670 | 0-0.46 | 11725.0 | 9655.0 |
| 866168 | 6.330 | 6-7 | 11637.5 | 9675.0 | 866308 | 3.580 | 0-2.74 | 11625.0 | 9687.5 | 866302 | 2.750 | 0-0.38 | 11800.0 | 9625.0 |
| 866169 | 2.920 | 0-1 | 11637.5 | 9662.5 | 866309 | 2.080 | 0-5 | 11656.5 | 9662.5 | 866303 | 2.080 | 0-0.11 | 11875.0 | 9695.0 |
| 866170 | 1.920 | 1-2 | 11637.5 | 9662.5 | 866310 | 0.470 | 5-7 | 11656.5 | 9662.5 | 866304 | 2.670 | 0-0.47 | 11750.0 | 9653.0 |
| 866171 | 1.750 | 2-2.4 | 11637.5 | 9662.5 | 866216 | 3.000 | 5-5.5 | 11587.5 | 9650.0 | 866305 | 1.670 | 0-0.25 | 11800.0 | 9650.0 |
| 866172 | 3.330 | 0-0.38 | 11637.5 | 9700.0 | 866217 | 2.670 | 0-1 | 11587.5 | 9662.5 | 866306 | 1.500 | 0-0.23 | 11825.0 | 9650.0 |
| 866173 | 2.330 | 0-3.36 | 11625.0 | 9650.0 | 866218 | 2.250 | 1-2 | 11587.5 | 9662.5 | 866307 | 2.500 | 0-0.51 | 11875.0 | 9611.0 |
| 866179 | 1.330 | 0-5 | 11625.0 | 9662.5 | 866219 | 1.750 | 2-3 | 11587.5 | 9662.5 | | | | | |
| 866180 | 2.030 | 0-1 | 11625.0 | 9675.0 | 866220 | 2.750 | 3-4 | 11587.5 | 9662.5 | | | | | |
| 866181 | 1.670 | 1-2 | 11625.0 | 9675.0 | 866221 | 8.000 | 4-4.4 | 11587.5 | 9662.5 | | TAILINGS WASHOUT 2 | | | |
| 866182 | 2.170 | 2-3 | 11625.0 | 9675.0 | 866222 | 3.920 | 0-1 | 11587.5 | 9675.0 | 866311 | 2.150 | 0-0.85 | 11450.0 | 9650.0 |
| 866183 | 3.500 | 3-4 | 11625.0 | 9675.0 | 866223 | 2.670 | 1-2 | 11587.5 | 9675.0 | 866312 | 3.920 | 0-0.4 | 11437.5 | 9675.0 |
| 866184 | 2.750 | 4-5 | 11625.0 | 9675.0 | 866224 | 4.750 | 2-2.5 | 11587.5 | 9675.0 | 866313 | 5.000 | 0-0.8 | 11425.0 | 9675.0 |
| 866185 | 2.420 | 5-6 | 11625.0 | 9675.0 | 866225 | 3.170 | 0-1 | 11587.5 | 9687.5 | 866314 | 1.850 | 0-0.15 | 11415.0 | 9697.0 |
| 866186 | 3.750 | 6-7 | 11625.0 | 9675.0 | 866226 | 3.500 | 1-1.4 | 11587.5 | 9687.5 | | | | | |
| 866187 | 3.330 | 0-0.16 | 11625.0 | 9700.0 | 866227 | 5.580 | 0-1.32 | 11587.5 | 9700.0 | | | | | |
| 866189 | 1.920 | 0-1 | 11612.5 | 9662.5 | 866228 | 2.500 | 0-3 | 11575.0 | 9650.0 | | | | | |
| 866190 | 1.920 | 1-2 | 11612.5 | 9662.5 | 866229 | 3.210 | 0-4.5 | 11575.0 | 9662.5 | | | | | |
| 866191 | 2.830 | 2-3 | 11612.5 | 9662.5 | 866231 | 1.500 | 0-1.3 | 11575.0 | 9687.5 | | | | | |
| 866192 | 2.750 | 3-3.6 | 11612.5 | 9662.5 | 866233 | 5.000 | 0-3.35 | 11575.0 | 9700.0 | | | | | |
| 866193 | 2.580 | 0-1 | 11612.5 | 9675.0 | 866238 | 3.330 | 0-0.43 | 11562.5 | 9650.0 | | | | | |
| 866194 | 1.250 | 1-2 | 11612.5 | 9675.0 | 866239 | 3.250 | 0-1 | 11562.5 | 9662.5 | | | | | |
| 866195 | 1.750 | 2-3 | 11612.5 | 9675.0 | 866240 | 5.920 | 1-2 | 11562.5 | 9662.5 | | | | | |
| 866196 | 2.420 | 3-4 | 11612.5 | 9675.0 | 866241 | 6.500 | 2-3 | 11562.5 | 9662.5 | | | | | |
| 866197 | 2.420 | 4-5 | 11612.5 | 9675.0 | 866242 | 3.750 | 3-3.7 | 11562.5 | 9662.5 | | | | | |
| 866198 | 2.420 | 5-5.5 | 11612.5 | 9675.0 | 866243 | 2.670 | 0-1 | 11562.5 | 9675.0 | | | | | |
| 866199 | 1.750 | 0-1 | 11612.5 | 9687.5 | 866244 | 1.420 | 1-2 | 11562.5 | 9675.0 | | | | | |
| 866200 | 2.250 | 1-2 | 11612.5 | 9687.5 | 866245 | 2.580 | 2-2.4 | 11562.5 | 9675.0 | | | | | |
| 866201 | 3.670 | 0-0.6 | 11475.0 | 9675.0 | 866246 | 2.830 | 0-1.1 | 11562.5 | 9687.5 | | | | | |
| 866202 | 2.580 | 0-1.39 | 11612.5 | 9675.0 | 866247 | 4.500 | 1.1-2 | 11562.5 | 9687.5 | | | | | |
| 866203 | 1.780 | 0-3.7 | 11600.0 | 9650.0 | 866248 | 4.500 | 2-2.4 | 11562.5 | 9687.5 | | | | | |
| 866204 | 3.250 | 0-2.7 | 11600.0 | 9662.5 | 866249 | 3.250 | 0-1.4 | 11562.5 | 9700.0 | | | | | |
| 866205 | 1.920 | 0-3.75 | 11600.0 | 9675.0 | 866250 | 2.170 | 0-0.58 | 11550.0 | 9650.0 | | | | | |
| 866209 | 2.830 | 0-3.14 | 11600.0 | 9687.5 | 866251 | 3.080 | 0-0.47 | 11550.0 | 9662.5 | | | | | |
| 866210 | 2.670 | 0-0.25 | 11600.0 | 9700.0 | 866252 | 5.170 | 0-1 | 11550.0 | 9675.0 | | | | | |
| 866211 | 3.170 | 0-1 | 11587.5 | 9650.0 | 866253 | 2.080 | 1-2 | 11550.0 | 9675.0 | | | | | |
| 866212 | 3.080 | 1-2 | 11587.5 | 9650.0 | 866254 | 2.750 | 2-3 | 11550.0 | 9675.0 | | | | | |
| 866213 | 1.920 | 2-3 | 11587.5 | 9650.0 | 866263 | 7.170 | 0-0.8 | 11537.5 | 9687.5 | | | | | |
| 866214 | 2.330 | 3-4 | 11587.5 | 9650.0 | 866264 | 2.000 | 0-3 | 11537.5 | 9700.0 | | | | | |
| 866215 | 2.830 | 4-5 | 11587.5 | 9650.0 | 866265 | 3.420 | 0-3.8 | 11537.5 | 9712.5 | | | | | |

# BAMBOO CREEK - TAILINGS DAM




NOTE: LOCATION OFF SAMPLE NUMBERS
ALL SAMPLE NUMBERS HAVE PREFIX 866

# Appendix D

[ Micron Research results ]

SIZING ANALYSES - [illegible]

| SIZE um | SOLN ml | MASS g | MASS % | GOLD ASSAY ug/g | GOLD TOTAL ug | % |
|---|---|---|---|---|---|---|
| | | | | | | |
| +600 | | 11 | 1.1 | 4.61 | 50.71 | 2.15 |
| +250 | | 231 | 23.1 | 3.13 | 723.03 | 30.66 |
| +125 | | 219 | 21.9 | 2.13 | 466.47 | 19.78 |
| +45 | | 186 | 18.6 | 1.40 | 260.40 | 11.04 |
| -45 | | 353 | 35.3 | 2.43 | 857.79 | 36.37 |
| TOTAL | | 1000 | 100 | 2.36 | 2358.4 | 100 |

CYANIDATION DATA - [illegible] 24 hours

| SIZE um | SOLN ml | MASS g | MASS % | GOLD ASSAY ug/g | GOLD TOTAL ug | % |
|---|---|---|---|---|---|---|
| | 1060 | | | 1.33 | 1409.80 | 48.94 |
| +500 | | 11 | 1.1 | 5.45 | 59.95 | 2.08 |
| +250 | | 224 | 22.4 | 2.90 | 649.60 | 22.55 |
| +125 | | 216 | 21.6 | 1.68 | 362.88 | 12.60 |
| +45 | | 191 | 19.1 | 0.85 | 162.35 | 5.64 |
| -45 | | 358 | 35.8 | 0.66 | 236.28 | 8.20 |
| TOTAL | | 1000 | 100 | 2.88 | 2880.9 | 100 |

CYANIDATION DATA -106 [illegible] 24 hours

| SIZE um | SOLN ml | MASS g | MASS % | GOLD ASSAY ug/g | GOLD TOTAL ug | % |
|---|---|---|---|---|---|---|
| | 762 | | | 1.52 | 1158.2 | 70.6 |
| +75 | | 141.1 | 19.9 | 1.27 | 179.2 | 10.9 |
| +45 | | 96.3 | 13.6 | 0.80 | 77.0 | 4.7 |
| -45 | | 471.0 | 66.5 | 0.48 | 226.1 | 13.8 |
| TOTAL | | 708.4 | 100 | 2.32 | 1640.6 | 100 |

CYANIDATION DATA -45 [illegible] 24 hours

| SIZE um | SOLN ml | MASS g | MASS % | GOLD ASSAY ug/g | GOLD TOTAL ug | % |
|---|---|---|---|---|---|---|
| | 922 | | | 1.62 | 1493.6 | 76.7 |
| -45 | | 888 | 100 | 0.51 | 452.9 | 23.3 |
| TOTAL | | 888 | 100 | 2.19 | 1946.5 | 100 |

LONG TERM STATIC LEACH

| SIZE um | SOLN ml | MASS g | MASS % | GOLD ASSAY ug/g | GOLD TOTAL ug | % |
|---|---|---|---|---|---|---|
| | 2000 | | | 1.56 | 3120.0 | 60.3 |
| +500 | | 17.8 | 0.9 | 4.18 | 74.4 | 1.4 |
| +250 | | 468.9 | 24.8 | 2.06 | 965.9 | 18.7 |
| +125 | | 401.2 | 21.2 | 1.18 | 473.4 | 9.1 |
| +45 | | 326.2 | 17.2 | 0.72 | 234.9 | 4.5 |
| -45 | | 679.5 | 35.9 | 0.45 | 305.8 | 5.9 |
| TOTAL | | 1893.6 | 100 | 2.73 | 5174.4 | 100 |

MICRON RESEARCH TAILINGS DUMP SUMMARY RECOV.

| | |
|---|---|
| 1) Cyanidation at size | 48.9 % |
| 2) Cyanidation at 100% -106 micron | 70.6 % |
| 3) Cyanidation at 100% -45 micron | 75.6 % |
| 4) Long term static leach | 60.3 % |

## Appendix E

[ All cashflows and graphs ]

CASHFLOW FOR THE FIRST 7500 ton BATCH IN AUST $. BASE CASE

| | | |
|---|---|---|
| LIME CONS. | 0.0020 | t/ton |
| CYANIDE CONS. | 0.0001 | t/ton |
| LOADER | 16.00 | $/h |
| DUMP TRUCK | 26.00 | $/h |
| LABOUR DAY | 18.00 | $/h |
| TRANSPORT RATE | 120.00 | t/h |
| GOLD PRICE | $460.00 | |
| TONNAGE | 7500.00 | t |
| GRADE | 2.40 | g/t |
| RECOVERY | 60.00 | % |
| CONTINGENCIES | 5.00 | % |
| ROYALTIES | 0.60 | % |

| CONSTRUCTION | WEEK | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 |
|---|---|---|---|---|---|---|---|---|---|---|---|
| BUILDING DAM | 1,500 | 1,500 | | | | | | | | | |
| WATER SUPPLY | 3,000 | 3,000 | | | | | | | | | |
| PLASTIC LINER | 4,000 | 4,000 | | | | | | | | | |
| PIPING | 1,400 | | 1,400 | | | | | | | | |
| PUMPS | 2,500 | | 2,500 | | | | | | | | |
| CARBONCOLUMNS | 2,000 | | 2,000 | | | | | | | | |
| CONSULTING | 3,000 | 1,500 | 1,500 | | | | | | | | |
| INSTALLATION | 2,000 | 750 | 1,250 | | | | | | | | |
| MINING | | | | | | | | | | | |
| MINING | 4,875 | | 4,680 | 195 | | | | | | | |
| WATER FILLING | 1,000 | | | 1,000 | | | | | | | |
| LEACHING | | | | | | | | | | | |
| LIME | 3,000 | | 2,920 | 80 | | | | | | | |
| CYANIDE | 12,000 | | | 6,000 | | | | 6,000 | | | |
| STRIPPING | 3,500 | | | | | | 1,000 | | 1,000 | | |
| FUEL | 1,000 | | | 42 | 42 | 42 | 42 | 42 | 42 | 42 | 42 |
| MAINTENANCE | 1,500 | | | 63 | 63 | 63 | 63 | 63 | 63 | 63 | 63 |
| DISMANTLE | 2,000 | | | | | | | | | | |
| CONTINGENCIES | 2,414 | 538 | 813 | 369 | 5 | 5 | 55 | 305 | 55 | 5 | 5 |
| ROYALTIES | 20,700 | | | | | | | | | | |
| REVENUES | 159,730 | | | | | | 45,637 | | 45,637 | | |
| CASHFLOW | 88,341 | (11,288) | (17,063) | (7,748) | (110) | (110) | 44,477 | (6,410) | 44,477 | (110) | (110) |
| CUMULATIVE CASHFLOW | | (11,288) | (28,350) | (36,098) | (36,208) | (36,318) | 8,159 | 1,750 | 46,227 | 46,117 | 46,008 |

| | | |
|---|---|---|
| RECOVERED GOLD | 10.80 | KG |
| COSTS/TON | 9.52 | $/ton |
| REVENUES/TON | 21.30 | $/ton |

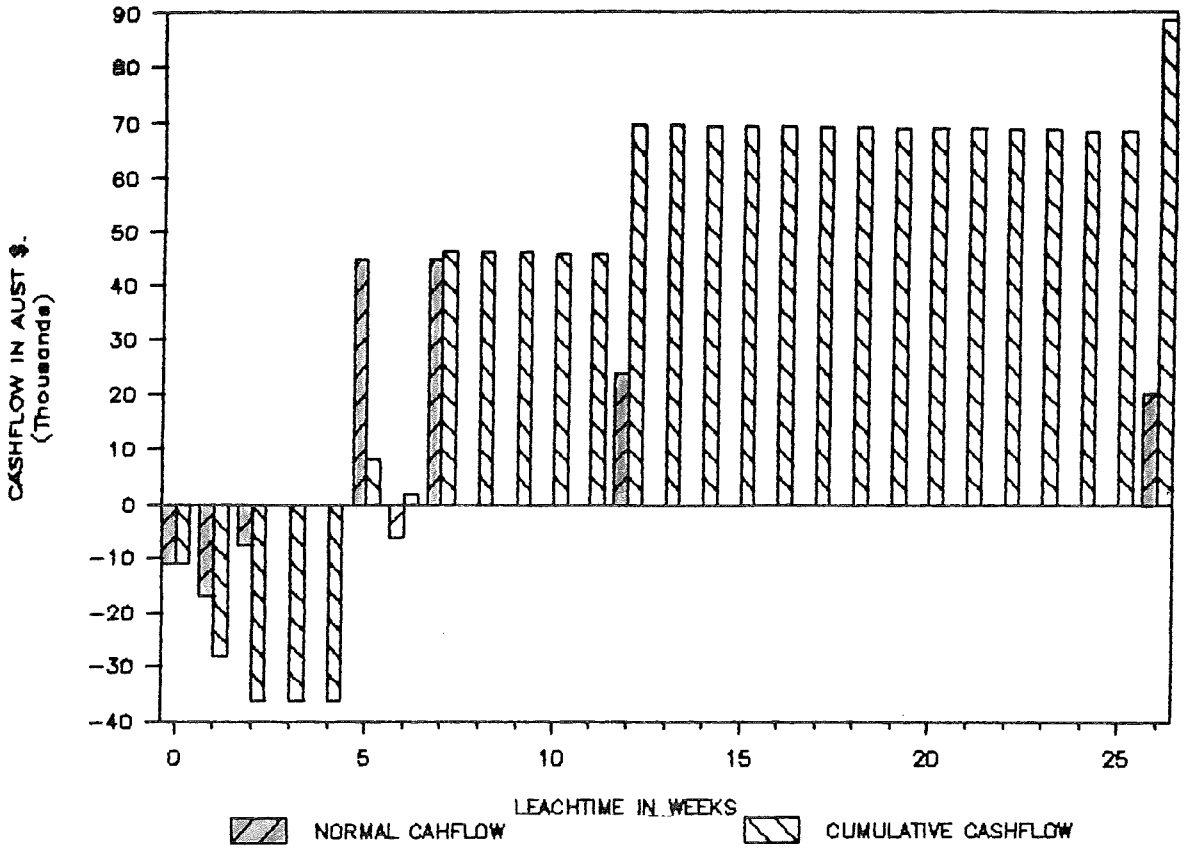
SHFLOW FOR THE FIRST 7500 ton BATCH IN AUST $. BASE CASE

| STRUCTION | 11 | 12 | 13 | 14 | 15 | 16 | 17 | 18 | 19 | 20 | 21 | 22 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| LDING DAM | | | | | | | | | | | | |
| ER SUPPLY | | | | | | | | | | | | |
| STIC LINER | | | | | | | | | | | | |
| ING | | | | | | | | | | | | |
| PS | | | | | | | | | | | | |
| BONCOLUMNS | | | | | | | | | | | | |
| SULTING | | | | | | | | | | | | |
| TALLATION | | | | | | | | | | | | |
| ING | | | | | | | | | | | | |
| ING | | | | | | | | | | | | |
| ER FILLING | | | | | | | | | | | | |
| CHING | | | | | | | | | | | | |
| E | | | | | | | | | | | | |
| NIDE | | | | | | | | | | | | |
| IPPING | | | 1,000 | | | | | | | | | |
| L | 42 | 42 | 42 | 42 | 42 | 42 | 42 | 42 | 42 | 42 | 42 | 42 |
| NTENANCE | 63 | 63 | 63 | 63 | 63 | 63 | 63 | 63 | 63 | 63 | 63 | 63 |
| MANTLE | | | | | | | | | | | | |
| TINGENCIES | 5 | 5 | 55 | 5 | 5 | 5 | 5 | 5 | 5 | 5 | 5 | 5 |
| ALTIES | | | 20,700 | | | | | | | | | |
| ENUES | | | 45,637 | | | | | | | | | |
| HFLOW | (110) | (110) | 23,777 | (110) | (110) | (110) | (110) | (110) | (110) | (110) | (110) | (110) |
| ULATIVE CASHFLOW | 45,898 | 45,788 | 69,566 | 69,456 | 69,346 | 69,236 | 69,127 | 69,017 | 68,907 | 68,798 | 68,688 | 68,578 |

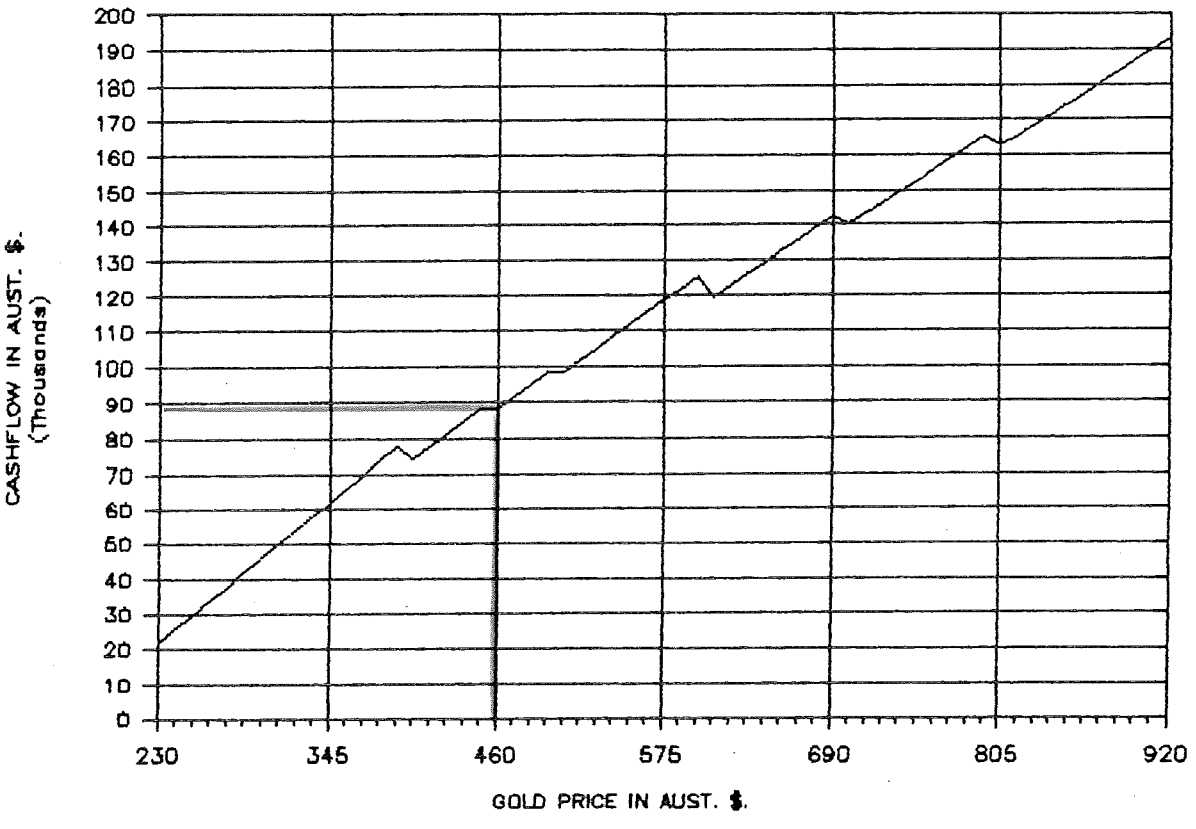
CASHFLOW FOR THE FIRST 7500 ton BATCH IN AUST $. BASE CASE

| CONSTRUCTION | 23 | 24 | 25 | 26 | 27 | |
|---|---|---|---|---|---|---|
| BUILDING DAM | | | | | | 1,500 |
| WATER SUPPLY | | | | | | 3,000 |
| PLASTIC LINER | | | | | | 4,000 |
| PIPING | | | | | | 1,400 |
| PUMPS | | | | | | 2,500 |
| CARBONCOLUMNS | | | | | | 2,000 |
| CONSULTING | | | | | | 3,000 |
| INSTALLATION | | | | | | 2,000 |
| MINING | | | | | | |
| MINING | | | | | | 4,875 |
| WATER FILLING | | | | | | 1,000 |
| LEACHING | | | | | | |
| LIME | | | | | | 3,000 |
| CYANIDE | | | | | | 12,000 |
| STRIPPING | | | | | 500 | 3,500 |
| FUEL | 42 | 42 | 42 | 42 | | 1,008 |
| MAINTENANCE | 63 | 63 | 63 | 63 | | 1,500 |
| DISMANTLE | | | | | 2,000 | 2,000 |
| CONTINGENCIES | 5 | 5 | 5 | 5 | 125 | 2,414 |
| ROYALTIES | | | | | | 20,700 |
| REVENUES | | | | | 22,819 | 159,730 |
| CASHFLOW | (110) | (110) | (110) | (110) | 20,194 | 88,333 |
| CUMULATIVE CASHFLOW | 68,468 | 68,359 | 68,249 | 68,139 | 88,333 | |

## CASHFLOW FOR ONE 7500 t BATCH




## INFLUENCE OF GOLD PRICE VARIATION



CASHFLOW FOR THE FIRST 15000 ton BATCH IN AUST $. BASE CASE

| | | |
|---|---|---|
| LIME CONS. | 0.002 | t/t.ore |
| CYANIDE CONS. | 0.0001 | t/t.ore |
| LOADER | 16 | $/h |
| DUMP TRUCK | 26 | $/h |
| LABOUR DAY | 18 | $/h |
| TRANSPORT RATE | 120 | t/h |
| GOLD PRICE | $460.00 | |
| TONNAGE | 15000 | t |
| GRADE | 2.4 | g/t |
| RECOVERY | 60 | % |
| CONTINGENCIES | 5 | % |
| ROYALTIES | 0.6 | % |

| CONSTRUCTION | WEEKS | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 |
|---|---|---|---|---|---|---|---|---|---|---|---|
| BUILDING DAM | 5,000 | 2,500 | | | | | | | | | |
| WATER SUPPLY | 3,000 | 3,000 | | | | | | | | | |
| PLASTIC LINER | 7,000 | 7,000 | | | | | | | | | |
| PIPING | 2,605 | | 2,605 | | | | | | | | |
| PUMPS | 5,000 | | 5,000 | | | | | | | | |
| CARBONCOLUMNS | 2,000 | | 2,000 | | | | | | | | |
| CONSULTING | 3,000 | 1,500 | 1,500 | | | | | | | | |
| INSTALLATION | 4,000 | 2,000 | 2,000 | | | | | | | | |
| MINING | | | | | | | | | | | |
| MINING | 9,750 | | 2,560 | 3,840 | 1,600 | | | | | | |
| WATER FILLING | 2,000 | | | | 2,000 | | | | | | |
| LEACHING | | | | | | | | | | | |
| LIME | 6,000 | | 1,920 | 2,880 | 1,200 | | | | | | |
| CYANIDE | 24,000 | | | | 24,000 | | | | | | |
| STRIPPING | 7,000 | | | | | | | 1,000 | 1,000 | 1,000 | 1,000 |
| FUEL | 2,000 | | | | 75 | 75 | 75 | 75 | 75 | 75 | 75 |
| MAINTENANCE | 3,000 | | | | 110 | 110 | 110 | 110 | 110 | 110 | 110 |
| DISMANTLE | 4,000 | | | | | | | | | | |
| CONTINGENCIES | 4,468 | 800 | 879 | 336 | 1,449 | 9 | 9 | 59 | 59 | 59 | 59 |
| ROYALTIES | 41,400 | | | | | | | | | | |
| REVENUES | 319,460 | | | | | | | 45,637 | 45,637 | 45,637 | 45,637 |
| CASHFLOW 1e BATCH | 184,237 | (16,800) | (18,464) | (7,056) | (30,434) | (194) | (194) | 44,393 | 44,393 | 44,393 | 44,393 |
| CUMULATIVE CASHFLOW | | (16,800) | (35,264) | (42,320) | (72,755) | (72,949) | (73,143) | (28,750) | 15,643 | 60,036 | 104,428 |

| | | |
|---|---|---|
| RECOVERED GOLD | 21.60 | KG |
| COSTS/TON | 9.01 | $/ton |
| REVENUES/TON | 21.30 | $/ton |

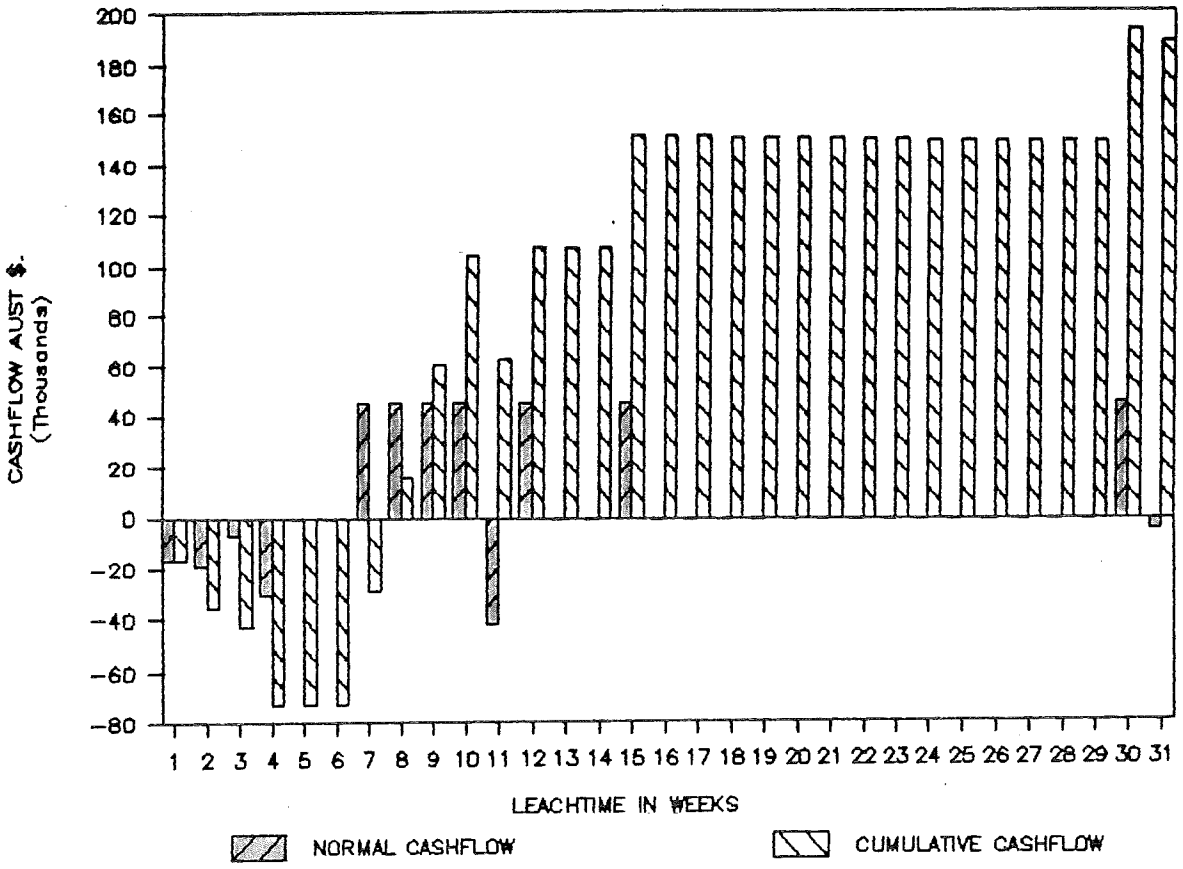
SHFLOW FOR THE FIRST 15000 ton BATCH IN AUST $. BASE CASE

| NSTRUCTION | 11 | 12 | 13 | 14 | 15 | 16 | 17 | 18 | 19 | 20 | 21 |
|---|---|---|---|---|---|---|---|---|---|---|---|
| ILDING DAM | | | | | | | | | | | |
| TER SUPPLY | | | | | | | | | | | |
| ASTIC LINER | | | | | | | | | | | |
| FING | | | | | | | | | | | |
| MPS | | | | | | | | | | | |
| RBONCOLUMNS | | | | | | | | | | | |
| NSULTING | | | | | | | | | | | |
| STALLATION | | | | | | | | | | | |
| NING | | | | | | | | | | | |
| NING | | | | | | | | | | | |
| TER FILLING | | | | | | | | | | | |
| ACHING | | | | | | | | | | | |
| ME | | | | | | | | | | | |
| ANIDE | | | | | | | | | | | |
| RIPPING | | 1,000 | | | 1,000 | | | | | | |
| EL | 75 | 75 | 75 | 75 | 75 | 75 | 75 | 75 | 75 | 75 | 75 |
| INTENANCE | 110 | 110 | 110 | 110 | 110 | 110 | 110 | 110 | 110 | 110 | 110 |
| SMANTLE | | | | | | | | | | | |
| NTINGENCIES | 9 | 59 | 9 | 9 | 59 | 9 | 9 | 9 | 9 | 9 | 9 |
| YALTIES | 41,400 | | | | | | | | | | |
| VENUES | | 45,637 | | | 45,637 | | | | | | |
| SHFLOW 1e BATCH | (41,594) | 44,393 | (194) | (194) | 44,393 | (194) | (194) | (194) | (194) | (194) | (194) |
| MULATIVE CASHFLOW | 62,834 | 107,227 | 107,033 | 106,839 | 151,231 | 151,037 | 150,843 | 150,649 | 150,454 | 150,260 | 150,066 |

CASHFLOW FOR THE FIRST 15000 ton BATCH IN AUST $. BASE CASE

| CONSTRUCTION | 22 | 23 | 24 | 25 | 26 | 27 | 28 | 29 | 30 | 31 | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| BUILDING DAM | | | | | | | | | | | 2,500 |
| WATER SUPPLY | | | | | | | | | | | 3,000 |
| PLASTIC LINER | | | | | | | | | | | 7,000 |
| PIPING | | | | | | | | | | | 2,605 |
| PUMPS | | | | | | | | | | | 5,000 |
| CARBONCOLUMNS | | | | | | | | | | | 2,000 |
| CONSULTING | | | | | | | | | | | 3,000 |
| INSTALLATION | | | | | | | | | | | 4,000 |
| MINING | | | | | | | | | | | |
| MINING | | | | | | | | | | | 8,000 |
| WATER FILLING | | | | | | | | | | | 2,000 |
| LEACHING | | | | | | | | | | | |
| LIME | | | | | | | | | | | 6,000 |
| CYANIDE | | | | | | | | | | | 24,000 |
| STRIPPING | | | | | | | | | 1,000 | | 7,000 |
| FUEL | 75 | 75 | 75 | 75 | 75 | 75 | 75 | 75 | 75 | | 2,025 |
| MAINTENANCE | 110 | 110 | 110 | 110 | 110 | 110 | 110 | 110 | 110 | | 2,970 |
| DISMANTLE | | | | | | | | | | 4,000 | 4,000 |
| CONTINGENCIES | 9 | 9 | 9 | 9 | 9 | 9 | 9 | 9 | 59 | 200 | 4,255 |
| ROYALTIES | | | | | | | | | | | 41,400 |
| REVENUES | | | | | | | | | 45,637 | | 319,460 |
| CASHFLOW 1e BATCH | (194) | (194) | (194) | (194) | (194) | (194) | (194) | (194) | 44,393 | (4,200) | 188,705 |
| CUMULATIVE CASHFLOW | 149,872 | 149,677 | 149,483 | 149,289 | 149,095 | 148,900 | 148,706 | 148,512 | 192,905 | 188,705 | |

# CASHFLOW FOR ONE 15000 t BATCH



CASHFLOW FOR THE MILL OPTION IN AUST. $

| GENERAL | | CHEMICALS | | | | |
|---|---|---|---|---|---|---|
| TONNAGE | 30000 ton | LIME CONS. | 0.002 t/t.ore | RECOVERED | 36.00 KG | |
| PROD. RATE | 20 t/hr | CYANIDE CONS. | 0.001 t/t.ore | COSTS/TON | 12.79 $/ton | |
| HEAD GRADE | 2.4 g/ton | CAUSTIC SODA | 0.055 kg/t.ore | REVENUES/T | 17.75 $/ton | |
| RECOVERY | 50 % | STEEL WOOL | 0.004 $/t.ore | | | |
| AVAIL. MILL | 95 % | CARBON | 0.037 $/t.ore | | | |
| ROYALTIES | 0.6 % | HCl | 0.020 $/t.ore | | | |
| GOLD PRICE | 460 $/oz | | | | | |
| MILLING TIME | 24 hr/day | | | | | |
| CONTINGENCIES | 5 % | | | | | |

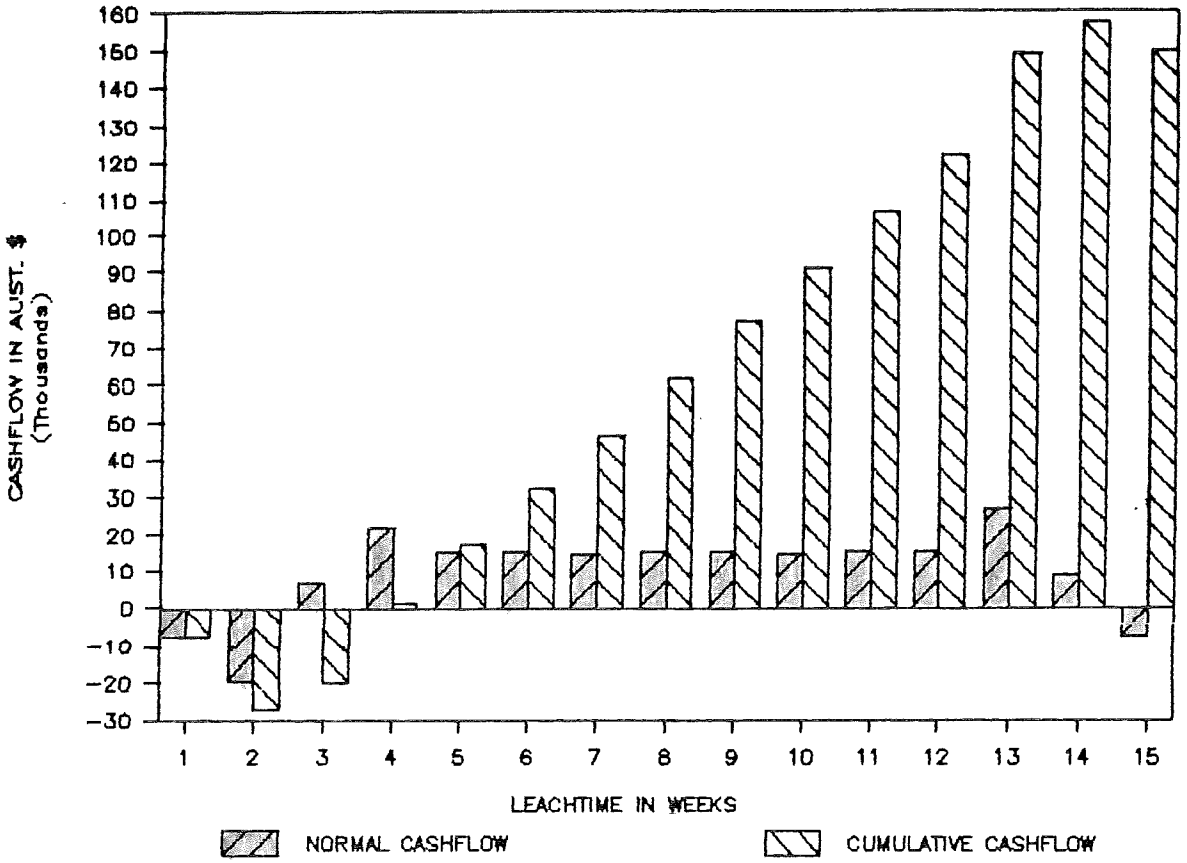
| MACHINERY & LABOUR | | OPERATION TIME | |
|---|---|---|---|
| CRANE | 80 $/hr | EQUIP. TRANS | 2 days |
| TRUCK | 40 $/hr | EQUIP. EREC. | 6 days |
| LOADER | 32 $/hr | EQUIP. DISM. | 3 days |
| DUMP TRUCK | 52 $/hr | BUILDING RAMP ? | days |
| LABOUR DAY | 18 $/hr | MAINTENANCE | 6 week |
| LABOUR NIGHT | 22 $/hr | TREATMENT | 65.79 days |
| ELECTRICITY | $/Kw | EQUIP. HIRE | 6500 $/month |
| SAND TRANSPORT | 120 t/hr | ADDITIONAL MAT | 3000 $ |

| | | 1 | 2 | 3 | 4 | 5 | 6 |
|---|---|---|---|---|---|---|---|
| MILL SETUP : | | | | | | | |
| TRANSPORT NEW EQUIPMENT | 3,120 | 3,120 | | | | | |
| ERECTION NEW EQUIPMENT | 3,240 | 2,160 | 1,080 | | | | |
| DISMANTLE + TRANSPORT | 4,740 | | | | | | |
| MINING : | | | | | | | |
| BUILDING LOADING RAMP | 500 | | 500 | | | | |
| ACTUAL MINING | 25,500 | | 2,040 | 2,040 | 3,060 | 2,040 | 2,040 |
| PROCESSING : | | | | | | | |
| EQUIPEMENT HIRE | 20,797 | 1,625 | 1,625 | 1,625 | 1,625 | 1,625 | 1,625 |
| LOADER | 50,526 | | 3,072 | 4,608 | 4,608 | 4,608 | 4,608 |
| C.I.P. OPERATOR | 63,158 | | 3,840 | 5,760 | 5,760 | 5,760 | 5,760 |
| C.I.P. CHEMICALS | 60,000 | | 3,648 | 5,472 | 5,472 | 5,472 | 5,472 |
| ELUTION & FIRE ASSAY | 15,132 | | 920 | 1,380 | 1,380 | 1,380 | 1,380 |
| ELUTION CHEMICALS | 5,091 | | | 310 | 464 | 464 | 464 |
| ELECTRICITY | 11,000 | | 669 | 1,003 | 1,003 | 1,003 | 1,003 |
| MAINTENANCE + MATERIALS | 23,684 | | 1,440 | 2,160 | 2,160 | 2,160 | 2,160 |
| CONTINGENCIES | 14,324 | 345 | 942 | 1,218 | 1,277 | 1,226 | 1,226 |
| ROYALTIES | 82,800 | | | | | 7,527 | 7,527 |
| REVENUES | 532,433 | | 0 | 32,372 | 48,558 | 48,558 | 48,558 |
| CASHFLOW | 148,821 | (7,250) | (19,775) | 6,796 | 21,749 | 15,293 | 15,293 |
| CUMULATIVE CASHFLOW FLOW | | (7,250) | (27,026) | (20,229) | 1,519 | 16,812 | 32,104 |

ASHFLOW FOR THE MILL OPTION IN AUST. $

| | 7 | 8 | 9 | 10 | 11 | 12 | 13 | 14 | 15 | |
|---|---|---|---|---|---|---|---|---|---|---|
| ILL SETUP : | | | | | | | | | | |
| ---------- | | | | | | | | | | |
| RANSPORT NEW EQUIPMENT | | | | | | | | | | 3,120 |
| RECTION NEW EQUIPMENT | | | | | | | | | | 3,240 |
| ISMANTLE + TRANSPORT | | | | | | | 4,740 | | | 4,740 |
| INING : | | | | | | | | | | |
| ------- | | | | | | | | | | |
| UILDING LOADING RAMP | | | | | | | | | | 500 |
| CTUAL MINING | 3,060 | 2,040 | 2,040 | 3,060 | 2,040 | 2,040 | | | | 25,500 |
| ROCESSING : | | | | | | | | | | |
| ---------- | | | | | | | | | | |
| QUIPEMENT HIRE | 1,625 | 1,625 | 1,625 | 1,625 | 1,625 | 1,625 | 1,625 | | | 21,125 |
| OADER | 4,608 | 4,608 | 4,608 | 4,608 | 4,608 | 4,608 | 1,536 | | | 50,688 |
| .I.P. OPERATOR | 5,760 | 5,760 | 5,760 | 5,760 | 5,760 | 5,760 | 1,920 | | | 63,360 |
| .I.P. CHEMICALS | 5,472 | 5,472 | 5,472 | 5,472 | 5,472 | 5,472 | 1,824 | | | 60,192 |
| LUTION & FIRE ASSAY | 1,380 | 1,380 | 1,380 | 1,380 | 1,380 | 1,380 | 460 | | | 15,180 |
| LUTION CHEMICALS | 464 | 464 | 464 | 464 | 464 | 464 | 464 | 155 | | 5,107 |
| LECTRICITY | 1,003 | 1,003 | 1,003 | 1,003 | 1,003 | 1,003 | 334 | | | 11,035 |
| AINTENANCE + MATERIALS | 2,160 | 2,160 | 2,160 | 2,160 | 2,160 | 2,160 | 720 | | | 23,760 |
| ONTINGENCIES | 1,277 | 1,226 | 1,226 | 1,277 | 1,226 | 1,226 | 681 | 8 | 0 | 14,377 |
| OYALTIES | 7,527 | 7,527 | 7,527 | 7,527 | 7,527 | 7,527 | 7,527 | 7,527 | 7,527 | 82,800 |
| EVENUES | 48,558 | 48,558 | 48,558 | 48,558 | 48,558 | 48,558 | 48,558 | 16,186 | | 534,137 |
| ASHFLOW | 14,222 | 15,293 | 15,293 | 14,222 | 15,293 | 15,293 | 26,726 | 8,496 | (7,527) | 149,412 |
| UMULATIVE CASHFLOW | 46,326 | 61,618 | 76,911 | 91,132 | 106,425 | 121,717 | 148,443 | 156,939 | 149,412 | |

# CASHFLOW FOR MILL OPTION



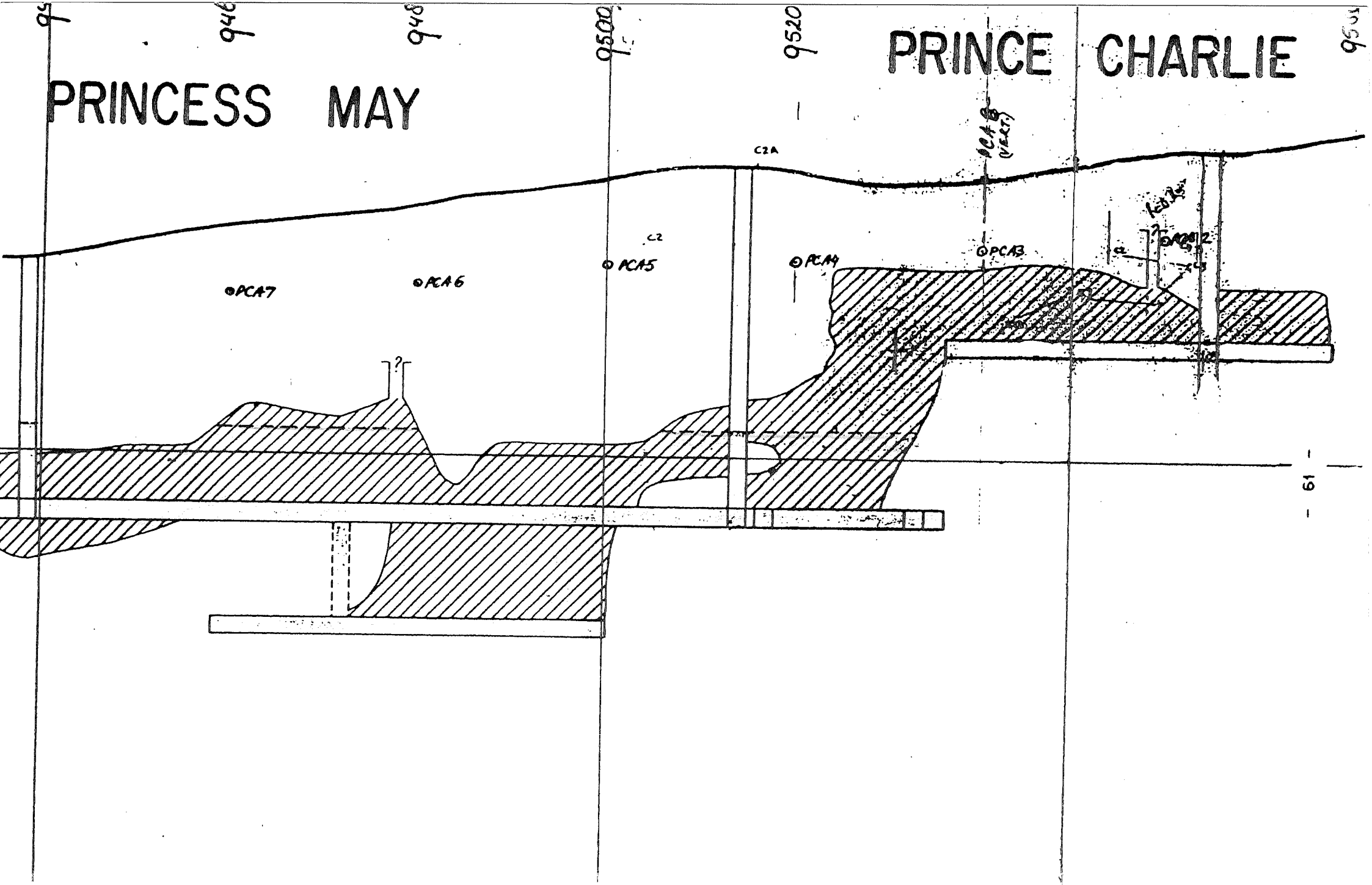

OVERVIEW OF TOTAL REVENUS FOR THE THE THREE OPTIONS

| | |
|---|---|
| PROCESSING IN THE MILL | $149,412 |
| BATCHLEACHING: TWO 15.000 TON BATCHES (*) | $335,332 |
| BATCHLEACHING: FOUR 7.500 TON BATCHES (*) | $294,326 |

(*) Consulting costs and cost to secure the water supply only occur once.

Appendix F

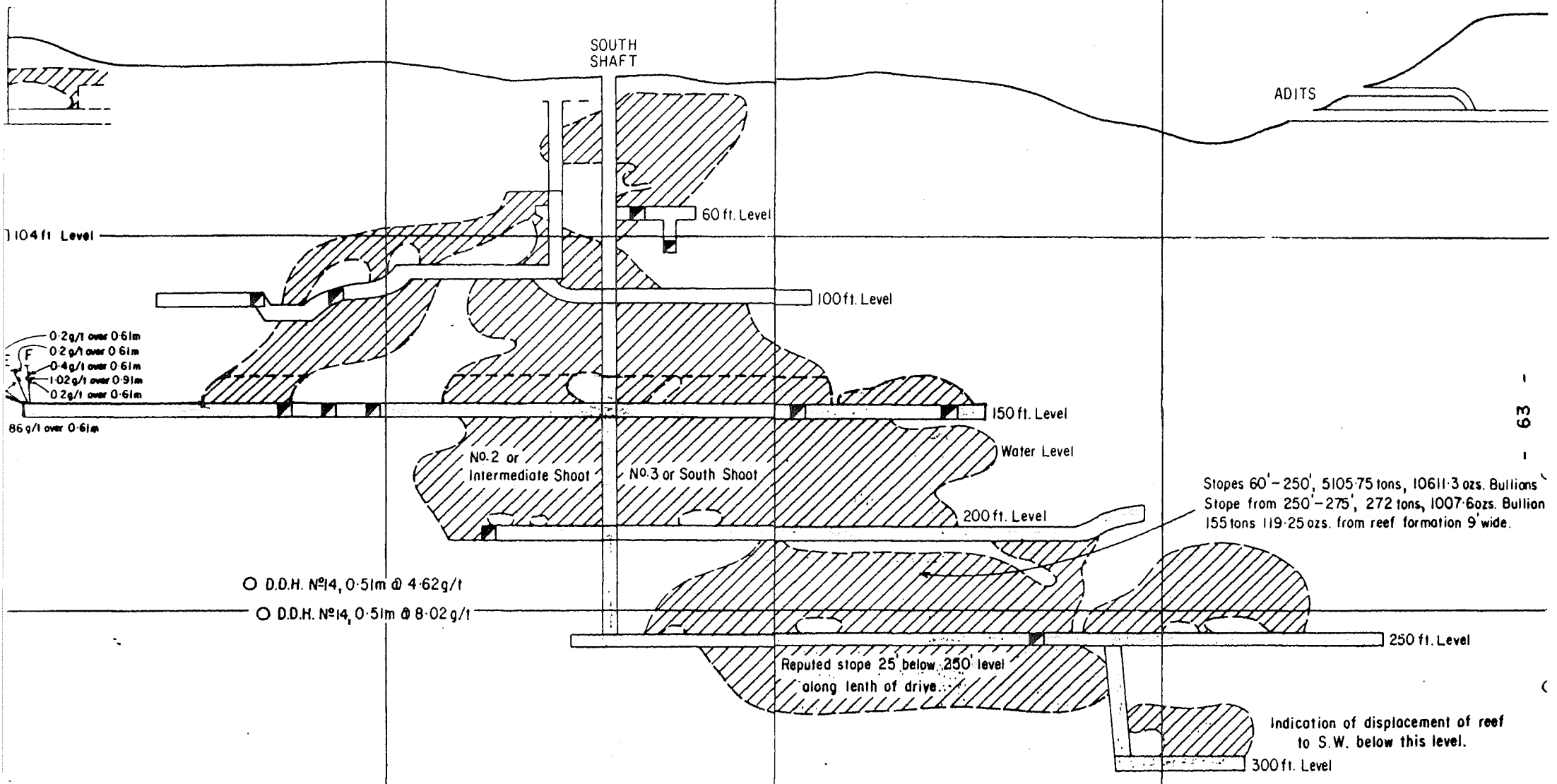
[ Map of underground workings of the Prince Charlie ]


PRINCESS MAY
PRINCE CHARLIE
9480
9500
9520
C2A
C2
PCA7
PCA6
PCA5
PCA4
PCA3
PCA2
PCA1
?
- 61 -

Appendix G

[ Map of underground workings of the Kitchener ]

# KITCHENER


SOUTH SHAFT
ADITS
60 ft. Level
104 ft Level
100 ft. Level
0·2g/t over 0·6lm
0·2g/t over 0·6lm
0·4g/t over 0·6lm
1·02g/t over 0·9lm
0·2g/t over 0·6lm
86 g/t over 0·6lm
150 ft. Level
Water Level
No.2 or Intermediate Shoot
No.3 or South Shoot
200 ft. Level
Stopes 60'–250', 5105·75 tons, 10611·3 ozs. Bullions
Stope from 250'–275', 272 tons, 1007·6ozs. Bullion
155 tons 119·25 ozs. from reef formation 9' wide.
O D.D.H. Nº14, 0·5lm @ 4·62g/t
O D.D.H. Nº14, 0·5lm @ 8·02g/t
250 ft. Level
Reputed stope 25' below 250' level along lenth of drive.
Indication of displacement of reef to S.W. below this level.
300 ft. Level
63