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75-1108 Vol. I

E.L. 17/73

ALLSTATE EXPLORATIONS N.L.

OPEN FILE

REPORT

on the

FEASIBILITY OF RE-OPENING and OPERATING

THE TASMANIA GOLD MINE

at

BEACONSFIELD, TASMANIA

PART A.

INITIAL REPORT

AUGUST 1975

75-1108 Vol 1/2

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ALLSTATE EXPLORATIONS N.L.

BEACONSFIELD GOLDMINE PROJECT.

FEASIBILITY REPORT - JULY, 1975.

This report is presented in two parts:

PART "A" - THE PROJECT

Background information on past mining operations and recent diamond drilling; details of Ore Reserves, mine dewatering and development, mine operation, ore processing, capital and operating cost estimates, infrastructure and timetables; financial analyses for the project; environmental and other Government considerations; conclusions, and current position on the Project.

Maps, plans, projections and other brief data relevant to the text are included in Part "A" of the Report.

PART "B" - COPIES OF RELEVANT DOCUMENTS

Copies of relevant reports (1891 to 1974), Title Document, and miscellaneous plans.

The report has been compiled by the staff of Allstate Explorations N.L.

Its description of, and data relating to past working of the Tasmania reef orebody are taken from numerous plans and reports made available by the Tasmanian Department of Mines, the majority of which are included in the attachments to Part "A" and in Part "B" of the Report.

Its data relating to drilling from the surface of the orebody extension is based on drill logs and core assays from the Tasmanian Department of Mines and from Allstate records.

Its projections of future activity are based largely on recommendations of, reports by, or data supplied by external consulting or specialist bodies, and partly by Allstate qualified staff, including the parties listed below:

Ore Reserves.

- Mr. T.W. Willsteed, a qualified and experienced Mining Engineer; in conjunction with Watts, Griffis & McOuat, Consulting Geologists and Mining Engineers.

- Mr. Albert Silver, General Manager of Allstate, a qualified and experienced mining Engineer.

Mine Dewatering and Confirmatory Drilling - Methods and Estimates.

- Messrs. T.W. Willsteed and A. Silver.

- Mr. P. Arnold, Consultant.

- Pearson Bridge Pty. Ltd.

- E.A. Marr & Sons (Sales) Pty. Limited.

- Pullen Fluid Dynamics Pty. Limited.

- Associated Diamond Drillers Pty. Ltd.

- Pan Pacific Pumps (agents for Pleuger).

Mine Development and Operating - Methods and Estimates.

- Messrs. T.W. Willsteed and A. Silver.

- Mr. P. Arnold, Consultant.

- Mr. A. Goninan, Consultant.

- Supporting cost data from Central Norseman N.L., Scotia Mining Operations, Civil & Civic Pty. Limited, C.G.F.A., and Austin Anderson.

- Hydro Electric Commission of Tasmania.

- Quotations from Allied Constructions; Atlas Copco; Joy Manufacturing; Fox Manufacturing; Eimco Machinery, and others.

Metallurgical Studies Affecting Ore Processing and Recoveries.

- Tasmanian Department of Mines, Laboratory and Metallurgist.
- Mr. Keith Parsons, Consulting Metallurgist.
- Amending recommendations by Sala Australia Pty. Limited.

Process Plant Estimates.

- Sala Australia Pty. Limited.
- A. Silver, with supporting cost data from Civil & Civic, Austin Anderson, and others.

Mine Development Timetable.

- Messrs. T.W. Willsteed and A. Silver, using supporting data from various sources.
- Sala Australia Pty. Limited re Mill design and construction.
- Delivery quotations for pumps from Pan Pacific Pumps, Pullen Fluid Dynamics, Kelly & Lewis Limited.
- Drilling progress from Associated Diamond Drillers.

Financial Analyses.

- Messrs. D. Elsworth and A. Silver.

ALLSTATE EXPLORATIONS N.L.

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D.L. Elsworth, Chairman.

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A. Silver, General Manager and Director.

July, 1975.

ALLSTATE EXPLORATIONS N.L.

BEACONSFIELD GOLDMINE PROJECT.

CONTENTS OF FEASIBILITY REPORT - PART "A".

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BEACONSFIELD GOLDMINE PROJECT.

CONTENTS OF REPORT - PART "B".

SECTION 3 - REPORTS ON METALLURGY OF DRILL CORE MATERIAL.

3/1 1974 Metallurgical Report on testing of drill core samples, by H.K. Wellington, Chief Chemist and Metallurgist of Tasmanian Department of Mines.

3/2 Summary Report on metallurgy at Beaconsfield, 1974-75, by Mr. Keith Parsons, Consultant Metallurgist.

3/3 Process flow sheet for ore treatment at Beaconsfield, by Mr. Keith Parsons, December, 1974.

SECTION 4 - MISCELLANEOUS PLANS AND REPORTS.

4/1 Main mine workings and strata by A. Montgomery, 1891.

4/2 Surface geology of Beaconsfield area, by A.J. Noldart, 1963.

4/3 Beaconsfield Deep Lead Gravity Survey, Tasmania Department of Mines, 1964.

4/4 Plan of diamond drill holes, and Longitudinal Projection showing cross-section of drill holes (Middleton, Tricentrol Australia Limited, 1974).

4/5 Development plan of Tasmania Gold Mine, ex old Company records. (Large Scale).

4/6 Longitudinal section of Tasmania gold mine including tabulations of production and data re Level development and grade.

4/7 Longitudinal projection showing stoping extraction, Tasmania Gold Mine, October 1913.

ALLSTATE EXPLORATIONS N.L.

BEACONSFIELD GOLDMINE PROJECT.

PART "A" OF FEASIBILITY REPORT - JULY, 1975.

1. BACKGROUND INFORMATION - OUTLINE.

1.1 Location and Nature of Orebody.

The Beaconsfield goldfield, in northern Tasmania, lies 39 kilometers by road north-west of Launceston and 3 kilometers west of the Tamar River estuary. It is adjacent to the town of Beaconsfield. Refer Figure 1.1.

The main reef (the "Tasmania" reef) was discovered at shallow depth immediately west of the Beaconsfield town site, on the eastern flank of the Cabbage Tree hill. Several smaller deposits were located to the north-west and south-east along a narrow belt centered on the ridge line of the Cabbage Tree hill. Refer Figure 1.2.

Further to the south, the extension of this ridge forms the Blue Tier Ridge with the Salisbury goldfield (now inactive) located at its southern extremity.

The Tasmania reef is a gold bearing quartz reef with the quartz emplaced in a pre-existent fault structure. The reef has an overall length of 1200-1300 feet striking about N 50° E, and the reef has itself been displaced by two major fault zones and by other smaller movements.

Historically the reef has contained small values of copper and silver, as well as gold, but no copper production was reported from the previous processing operations.

The reef has an average width of about seven feet, and dips 50° - 60° from horizontal to the south-east. It plunges to the north-east at 55° from horizontal.

Gold values in the reef were reported by the Department of Mines as consistent laterally, with marked variation with depth. (However, examination of old mine records shows significant variation in values along each Level, as would be expected from a gold deposit). In the upper levels the ore averaged above 20 dwts per ton, but a poorer section was entered at 1100 feet. Ore values diminished, with an average for the 1370 ft. Level of 8.75 dwts; but values were improving on the 1500 ft. Level with an average of 13 dwts. (Refer data on plan 4/6 in Part "B"). Working of the mine ceased in 1914, on the 1500 ft. Level, with the workings still in gold-rich ore.

Five drill intersections of the extension of the orebody, between Mine R.L. 1700 ft. and 1920 ft., indicate that ore grades improve as the reef deepens below 1500 feet.

Later sections of the Report amplify this broad outline of the nature of the orebody.

1.2 Other Mineralisation.

(a) The mineralisation in other parts of the field include the "Moonlight" area which consists of a sub-parallel system of irregular quartz reefs, occupying weak fissures in the Cabbage Tree Conglomerate, with a general strike of about 145^{o} and a steep SW dip. Fair to good grade ore occurred to a depth of about 250 ft., diminishing rapidly with further depth. Occurrences of small saddle reefs were recorded from the richer workings.

(b) A deep lead along the eastern flank of Cabbage Tree Hill had some payable concentrations of gold northerly of the Tasmania reef outcrop at intermediate (100 ft.) Levels, but has not been tested to bottom (400-plus ft.).

Montgomery in 1891 quoted reports from the Ophir Company, of gravel containing gold at between "2 ounces to 4 ounces to the load" at depths of 200 feet and 375 feet. Montgomery mapped the Deep Lead (see Figure 3.1), and a gravimetric survey of the Lead by the Bureau of Mineral Resources in 1964 (attached as 4/3 in Part "B") confirmed the position of the Deep Lead as indicated by Montgomery. The Bureau recommended drilling of the Lead gutter to the north west of the Tasmania reef. No exploration work on the Deep Lead target has been undertaken by Allstate.

(c) Small deposits of the "Moonlight" type occur along the length of Cabbage Tree - Blue Tier ridge.

(d) The 1923 report by J.O. Hudson, Chief Inspector of Mines, (attached as 1/6 in Part "B") also deals with this topic, as does a section of Cundy and Fawcett's report of 1914 (attached as 1/5 in Part "B").

1.3 Previous Goldmining Operations (taken mainly from Report of Chief Inspector of Mines, Hobart, 1923 - see attachment 1/6, Part "B").

Gold was discovered in the Beaconsfield area of Tasmania in 1877, leading to the formation of goldmining companies which worked until 1914. Total production of gold was 854,600 oz. of gold from 1,066,556 tons of ore, giving an average gold recovery over the whole period of 16.01 dwts per ton of ore mined.

From 1877 to 1896 the Tasmania Gold Mining and Quartz Crushing Company treated 299,000 tons of ore for a return of 371,400 ounces of gold by amalgamation. From 1896 to 1903 the Company treated 198,850 tons for a yield of 152,800 ounces of gold by amalgamation, and also obtained 46,600 ounces of gold by chlorination equal to 4.68 dwts per ton, a total of 199,400 ounces of gold equal to 20.05 dwts per ton.

The Tasmania Gold Mining Company Limited acquired the mine in 1903 and worked it until operations ceased in 1914, except for the final few months when the mine was worked on tribute by a co-operative party under arrangement with the Tasmanian Government.

From 1903 to 1913 this Company crushed and treated 524,800 tons for a yield of 251,900 ounces, and no dividends were declared. The residues are stated to have contained from 4 to 5 dwts of gold to the ton.

The following is a summary of ore treated and gold recovered in this period:

Period	Tons Milled	Ounces of Gold Recovered	Mill Recovery Per Ton Milled
			dwts
1903-4	24,238	19,600	16.16
1904-5	43,742	30,648	14.00
1905-6	48,076	32,914	13.66
1906-7	58,339	30,354	10.33
1907-8	70,272	30,302	8.50
1908-9	53,787	21,854	8.08
1909-10	67,113	20,718	6.16
1910-11	53,564	23,143	8.58
1911-12	51,889	21,409	8.25
1912-13	53,812	21,005	7.87
	524,832	251,947	

The original Company obtained its yield above the 815 ft. Level and the latter Company below that Level to the 1500 ft. Level, which was the lowest Level worked. It would appear that a poorer zone was entered at 1100 ft. Level, and that at 1500 ft. Level the values were improving. Sampling from the 1370 ft. and 1500 ft. workings, and from a 20 ft. winze sunk below the 1500 ft. Level, support this view.

The average sampled ore values for a length of 940 ft. at the 1500 ft. Level are given as 13 dwts over an average width of 7 feet. In the 1370 ft. Level the same 940 ft. had an average sampled ore value of 8.73 dwts over a width of 5 feet.

Samples taken from the bottom of the 20 ft. winze gave an assay of 20 dwt per ton for a width of 5 feet.

In June, 1914 the Company decided to close the mine due to a period of poor profitability which had inter alia, depleted financial resources so that it was not possible to undertake the substantial mine development work necessary for continued operations. (See Section 4.6 for more detail). An arrangement was made by the Tasmanian Government to take over the mine on tribute.

Arrangement was made to work the mine co-operatively by the miners, and this method operated until November, 1914. The tributors treated 16,556 tons. The treatment plant assays for the tonnage returned 12.8 dwts per ton of ore. The tributors obtained nearly the whole of the ore from blocks developed by the Company, and which were considered unpayable by the Company.

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There is no record of the value of the residues as no allowance was made to the tributors for this material.

Subsequent operations at Beaconsfield related to retreatment of mine residues (tailings dumps), until the plant finally shut down in 1924.

Over the years the mine workings have flooded, and they are now inaccessible pending dewatering.

1.4 Subsequent Drilling.

Three phases of drilling have been undertaken on the Tasmania reef, to establish continuity of gold mineralisation with depth:

(i) The Tasmanian Mines Department undertook a drilling program (1964-67) and drilled a diamond drill hole B4, which was targetted to a depth of 2000 feet. Due to deviation the actual intersection was obtained at mine R.L. 1700 feet. Two deflection holes were wedged off giving two additional intersections B4A and B4B at R.L. of 1700 ft. and 1750 ft. respectively. Assays of drill core gave 60.2, 42,0 and 26.8 dwts of gold per ton respectively for the three intersections. Copper averaged about 1% in these intersections and the calculated true width of the reef varied from 8.5 ft. to 17 ft.

(ii) In 1969 Allstate Explorations N.L. took an Exploration Licence over the area and initiated a deep drilling program aimed to intersect the lode at 3000 feet.

Drilling was difficult due to limestone caverns, faulted country, and direction control at depth. This phase of drilling lasted from 1969-72.

The original hole A3(i) was drilled to approximately 2760 ft., a second hole A3(ii) was drilled to 3530 ft., and a third hole A3(iii) was wedged off hole A3(ii) at 2156 ft. and drilled to 3790 feet.

No gold values were found in assays of sections of the cores of these holes, but the geological significance of hole A3(iii) is discussed in Section 3.4 below.

(iii) The third phase of drilling was undertaken by Allstate in 1973-74 when the escalation in gold prices indicated that a viable operation could be based upon a relatively small tonnage of ore of the grade indicated in the deposit.

DDH A6 intersected the reef at R.L. 1920 ft.; a core assayed 8.5 dwts per ton of gold and indicated a true reef width of 5.5 feet.

A branch hole DDH A7 was wedged off from hole A6 at a depth of 1534 ft., and this intersected the orebody at R.L. 1870 ft., some 40 ft. to the west of, and 50 ft. above the A6 intersection. The core assayed 23.1 dwts of gold per ton and indicated a true width of 10.5 feet.

1.5 Mineral Titles.

Allstate Explorations N.L. currently holds Exploration Licence 17/73, a copy of which is provided in Section 2/1 of Part "B" of this Report. The area covered by this Licence is also indicated in Figure 1.1 attached.

The Licence has been extended to January, 1976, and no difficulty is anticipated in further extension.

A Mineral Lease will be taken over the vital areas of the Licence before mine development work commences.

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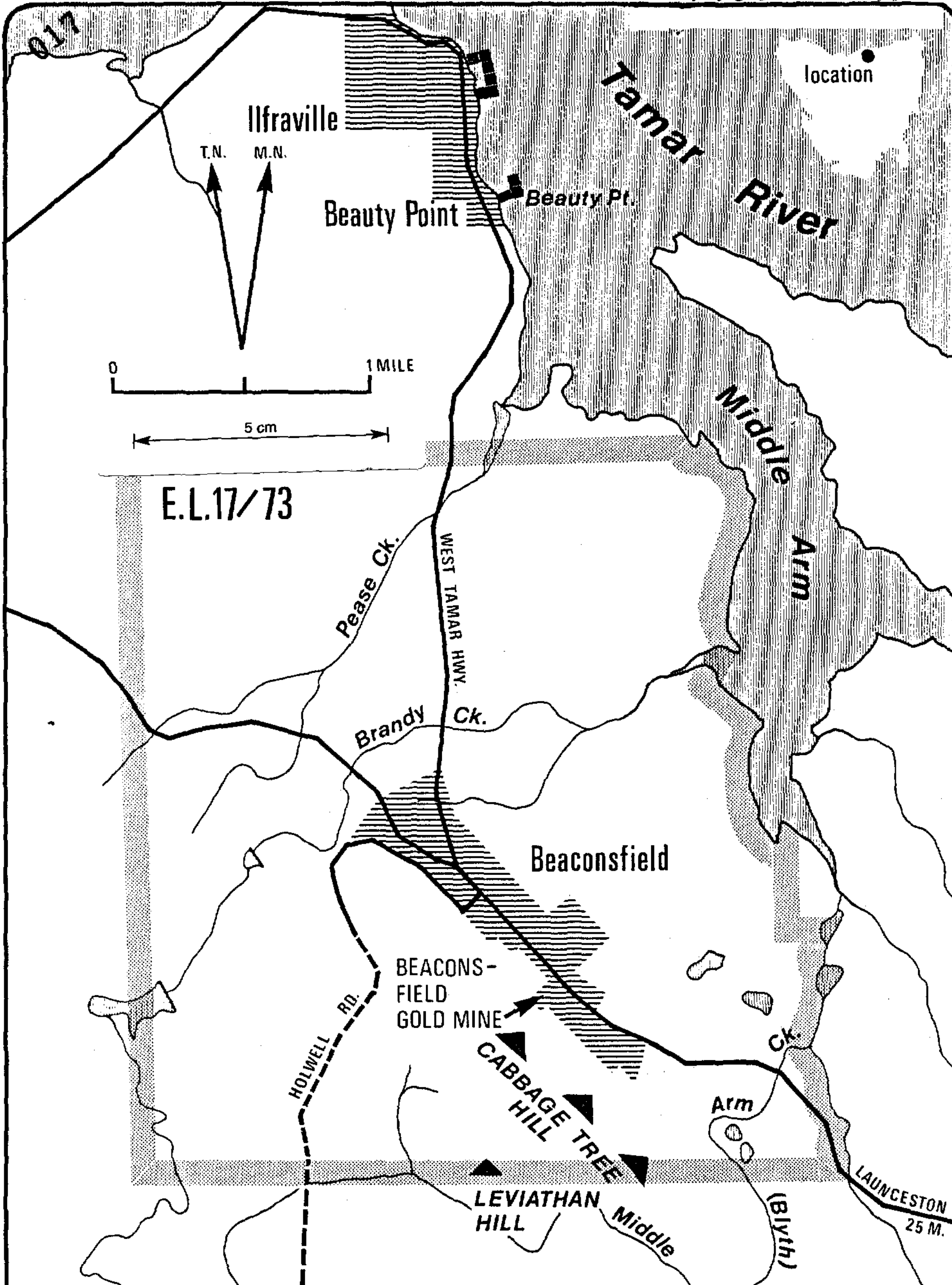
FIGURE 1.1



LOCATION of the Beaconsfield Gold Mine

Prepared by: **Watts, Griffis and McOuat and Associates** for:

ALLSTATE EXPLORATIONS N.L. May, 74

FIG. 1.1

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FIGURE 1.2

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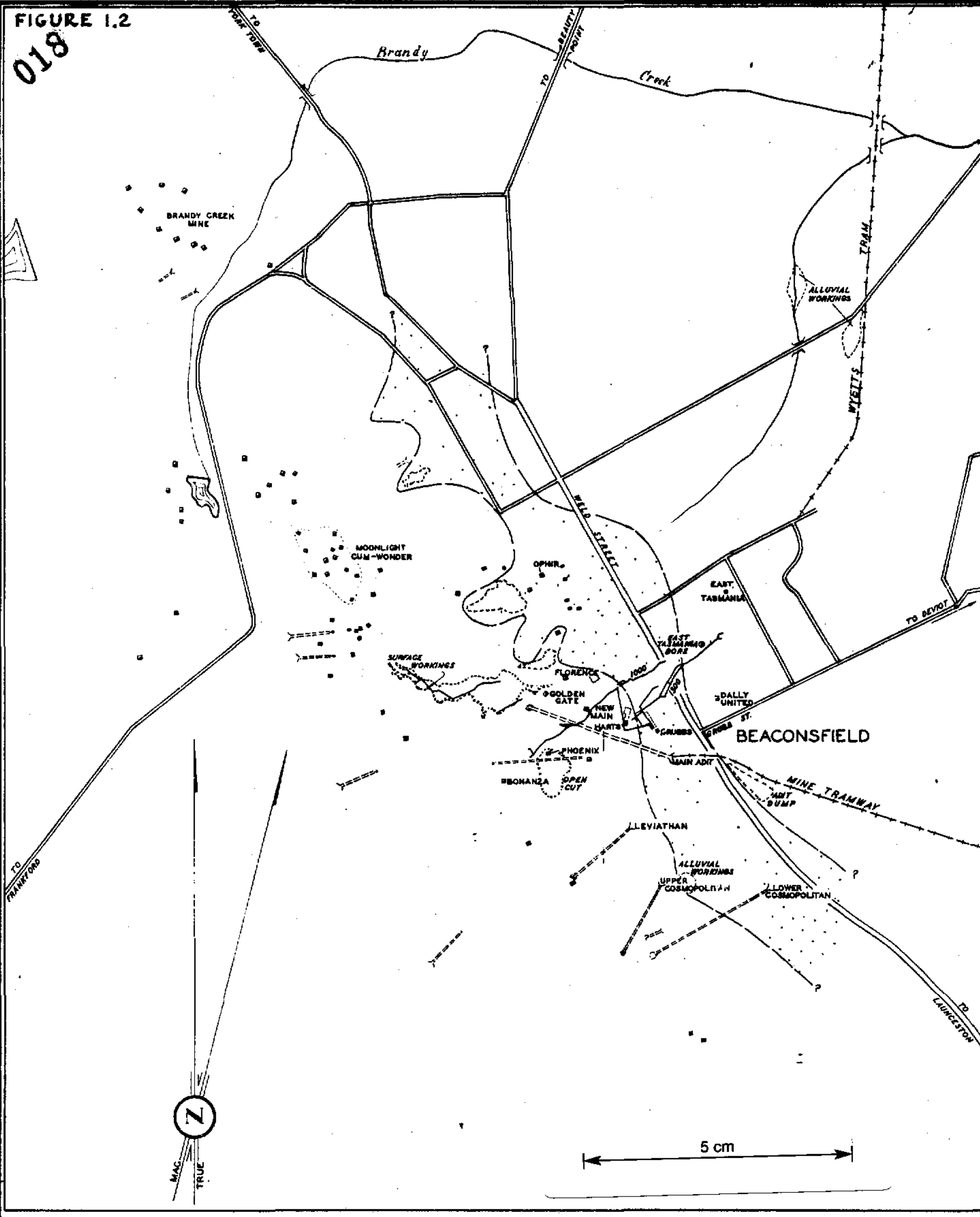

PRINCIPAL MINES

BEACONSFIELD GOLDFIELD

500 0 500 1,000 1,500 2,000 FT.

LEGEND

- SHAFT
- ADIT
- OPEN CUT
- DEEP LEAD AFTER MONTGOMERY 1891

A. J. NOLDART
GEOLOGIST 1963

ALLSTATE EXPLORATIONS N.L.

Fig 1-2

DEPT OF MINES
2206-30

BEACONSFIELD FEASIBILITY REPORT - JULY, 1975.

2. STUDIES AND CONCLUSIONS.

2.1 Objective and Outline of Studies.

The purpose of the Feasibility Study was to investigate the commercial feasibility of dewatering the mine shafts and old workings of the Tasmania reef goldmine, and the subsequent undertaking of mine development and plant construction work, with the aim of bringing the mine into commercial production at the earliest possible time.

In order to best estimate the ore reserve position a full review has been made of the geology of the surrounding country and of extant data from mining of the Tasmania reef up to 1914, to which has been added data gained in recent years from diamond drilling at the site. Certification of ore reserves and mineralisation has been confirmed by Allstate.

As much metallurgical work as is possible has been undertaken on the small amount of material recovered from the orebody via diamon drill cores, in order to assess ability to economically process the ore, and the likely process design. The reports on this work have been used (with some amendment) in assessing gold recovery and determining process design.

As much planning and calculation as can practically and usefully be undertaken regarding the methods, timing and cost of dewatering and repairing the best of the existing mine shafts, has been completed. Positive procedures for handling this phase of activity have been determined and used as a basis for cost estimation; a large contingency allowance has been added to cost and time estimates for this phase.

The methods of developing and operating the mine following dewatering have been planned in detail, including the handling of continuing water inflow. Cost of plant, equipment, labour and other underground requirements has been estimated on the basis of this planning.

Full consideration has been given to the likely trends in future gold prices in arriving at reasonable prices for use in financial analysis.

There are no marketing problems for gold bullion, and none are foreseen for gold-rich copper concentrates.

The current position and likely future position regarding taxation of gold mining profits has been considered. Financial presentations in this Report are shown for both the present position (gold revenue not taxable) and for a situation in which gold revenue is fully taxable.

A timetable for mine development has been prepared. Recognizing the financial advantage of commencing operations as soon as possible it is intended to commence production based on one and one third Levels of mine development, with the second full Level of development coming into production at the start of the second year of production. Achievable production and milling tonnages have been assumed, based on this timetable.

Based on the outcome of these and other complementary studies, financial analyses have been completed to indicate the commercial strength of the project, and its sensitivity to variations in a number of the basic assumptions.

2.2.1 Gold Price

Practically all the revenue for the project is from gold.

The gold price used in financial assessment is expressed in Australian dollars.

Revenue calculations are therefore related to the future U.S. dollar price for gold, and the future exchange relationship between the U.S. dollar and the Australian dollar.

Although moves in the U.S./Australian effective exchange rate are possible in either direction, there does not seem to be any strong reason for assuming other than current exchange rates in financial forecasts for the project.

In the event of a rate of inflation in Australia in excess of that experienced generally in other countries, it is assumed that adjustments in exchange rates will serve to keep a fairly constant relationship between operating costs in Australia and gold revenue in Australian dollars.

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The overwhelming factor in the assumption of revenue in this Report is therefore the world price in U.S. dollars for gold bullion.

Factors influencing the future trend of gold prices are considered in Section 13.1 of this Report.

The financial forecasts in this Report are based on a prime assumption of a gold price 2½ years hence of $A150 per ounce, while sensitivity analyses show the affect of prices of $A130 and $A175 per ounce in two years time. (Equivalent $US prices at current exchange rates, $US200, 175 and 235).

2.2.2 Ore Reserves and Mineralisation.

The calculation of both tonnage and grade of ore reserves and mineralisation is considered soundly based due to the following:

(a) Estimates of tonnage and grade have been made by Mr. T.W. Willsteed, a Consulting Mining Engineer in association with Watts, Griffis & McOuat.

A qualified and experienced Mining Engineer from Allstate has separately checked these calculations and confirms the estimate of tonnages and grades quoted.

These calculations have been made by professional Engineers qualified under the Australasian Institute of Mining and Metallurgy, and meeting the Stock Exchange requirements for the publication of Ore Reserves.

(b) Comprehensive data is available regarding the gold values in the Tasmania reef down to the 1500 ft. Level.

These historical records show Level averages as follows:

	Average
915 ft. Level	27.25 dwt.
1000 ft. Level	19.75 dwt.
1100 ft. Level	15.00 dwt.
1250 ft. Level	11.75 dwt.
1370 ft. Level	8.75 dwt.
1500 ft. Level	13.00 dwt.

The upturn in grade with increasing depth, (comparing the 1500 ft. Level with the 1370 ft. Level), has been strongly supported by recent drill intersections of the orebody extension.

The highest and lowest drill core assays from orebody intersections at depths of 1700 ft. (two), 1750 ft., 1870 ft. and 1920 ft. were 60 dwts and 8.5 dwts respectively. Weighted average of all drill core assays is in excess of 30 dwts.

It is a recognised feature of gold bearing orebodies, evidenced by records of the Tasmania mine, that the grade of gold mineralisation varies considerably throughout such orebodies. The significance of the high (in excess of 20 dwts per ton) gold values from four out of five drill cores which have a good vertical spread over the orebody extension, must be assessed against this background.

These broad considerations support the detailed grade estimates, which are calculated on a "zones of influence" basis. (Refer Figure 5.1).

(c) Historically the mine has made approximately 1000 tons of ore per vertical foot of orebody.

Historical records show Level strike lengths as follows:

915 ft. Level	1421 ft.
1000 ft. Level	1292 ft.
1100 ft. Level	1317 ft.
1250 ft. Level	1231 ft.
1370 ft. Level	1265 ft.
1500 ft. Level	940 ft. *

(* the 1500 ft. Level confirmed the location trend of the Eastern end of the orebody limit, but was not worked to the West of Harts shaft when the mine closed.)

All available evidence of lithological control of the limits of the orebody, including true width calculation from drill intersections of the orebody, and drill hole data at depths of 2400 ft. and 3500 ft., supports the expectation of continuity of the dimension of the orebody extension to at least 2000 ft. (Refer 4/4 in Part "B", and Section 3.4 in Part "A" of this Report.)

Tonnage estimates are based on a calculation which does not take account of tonnage below 2000 ft. depth. However the extension of the orebody is open-ended at depth.

Tonnage estimates are therefore considered to be soundly based.

Taking all of these factors into account it appears that the gold content of the orebody extension down to 2000 ft. has been reasonably estimated in the figures for ore reserves and mineralisation, and that there is a strong possibility of the orebody extending below 2000 ft.

2.2.3 Capital Cost.

Discussion of capital cost in this Report refers only to future expenditures, and ignores expenditure to date as well as capital value of the project as at present.

The Stage I (Dewatering and Confirmatory Drilling) estimate is based on actions and sequences which minimize time to commencement of production (two years from start of work) and total capital cost.

On this basis total capital cost is of the order of $7.0 million, including inflation allowance of $1.57 million and $0.23 million lump sum contingency over and above detailed contingencies. On this basis, cost to completion of confirmatory drilling is $1.16 million.

An alternative basis has been considered based on actions and sequences which minimise cost to completion of confirmatory drilling. This basis necessarily extends the time to commencement of production (from 2 to 2-3/4 years from start of work) and increases total capital cost.

On the alternative basis total capital cost is of the order of $7.85 million (including comparable inflation and contingency allowances), while cost to completion of confirmatory drilling is of the order of $750,000.

2.2.4 Profitability.

(i) The base case for profit calculation assumes the following main factors:

(a) combined tonnage of ore reserve and mineralisation as certified by Willsteed/Watts, Griffis McOuat.

(b) A gold price in 1978 of $A150 per ounce. ($US200 at U.S./Australian exchange rate of 1.33/1).

(c) A milling rate of 74,000 tons in the first year of production, and 100,000 tons per annum from the second year of production onwards.

(d) Gold recovery at 82.5% for the first two years of production, 85% thereafter.

(e) A rate of cost inflation over the next 2½ years of 15% per annum, affecting both capital and operating costs.

The base case is shown on two taxation bases i.e. on the current basis of no tax on gold revenue, and on an alternative basis of full tax.

(ii) In broad outline and on the above bases, one ton of ore will bring revenue of $A87. This compares with an operating cost per ton (after allowing for inflation of costs) of about $A28, giving a cash operating profit per ton before depreciation and tax of the order of $A59.

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At an annual production rate of 100,000 tons, cash operating profit from the second/third years of production would be approximately $5.6/5.9 million per annum before tax; and without extension of the orebody below 2000 ft. the assumed bases will give the mine a seven year life (first and last years not full production).

Extension of the orebody at depth and thus extension of the life of the mine is, however, likely.

(iii) In terms of Discounted Cash Flow on future capital expenditures the only meaningful calculation is after allowing for company tax, as without tax the strength of cash flow is such that a D.C.F. calculation exceeds 50% and therefore becomes of limited significance.

With a 1978 gold price of $A150 per ounce, D.C.F. of the project on future capital expenditure on a maximum tax basis, is 39%.

2.3 Sensitivity of Results to Variations.

Sensitivity is expressed in terms of affect on D.C.F. rate of return on future capital expenditures.

2.3.1 Downside Sensitivity.

Downside risk has been considered relating to the following adjustments to the base case as outlined in 2.2.4(i) above:

(a) An under-estimation of future capital costs.

(b) A reduction of gold price 2½ years hence, from $A150 to $A130. (i.e. to $US170 on July, 1975 exchange rates).

(c) A reduction in grade of mineralisation (pre-dilution) from 15.6 dwt/ton to 13.0 dwt/ton i.e. the sampled average for 940 ft. of the 1500 ft. Level. (Equivalent headgrade reduction is from 13 dwt to 10.83 dwts/ton.)

(d) An increase in annual operating costs of 20%, which could account for one of a number of factors reducing before-tax profit.

Re (a).

An increase in capital cost of 10% (or $A700,000) leaves D.C.F. rate of return in excess of 50% on an untaxed basis; and reduces the D.C.F. rate of return by approximately 3%, to 36% on a fully taxed basis.

Re (b).

A reduction of gold price from $A150 to $A130 per ounce leaves D.C.F. rate of return in excess of 50% on an untaxed basis; or reduces the D.C.F. rate of return by approximately 7.5% to 31.5% on a fully taxed basis.

Re (c).

A reduction in grade of mineralisation (pre-dilution) from 15.6 dwt/ton to 13.0 dwt/ton reduces D.C.F. rate of return to 47.5% on an untaxed basis; or reduces D.C.F. rate of return by approximately 10% to 29% on a fully taxed basis.

Re (d).

An increase in total annual operating costs by 20% (i.e. from $A2.81 million to $A3.37 million per annum in 1978) would leave D.C.F. rate of return in excess of 50% on an untaxed basis; or reduce the D.C.F. rate of return by 4%, to 35% on a fully taxed basis. Such an increase in costs would result from any one of the following:

(i) increase in ore processing costs by 66% to $A14.2 per ton of ore milled in 1978;

(ii) increase in mine operating costs by 37% to $A20.8 per ton of ore produced in 1978;

(iii) increase in overbreak from 20% as allowed, to 48%.

The affect on profits is also equivalent to decreased revenue resulting from a drop in gold recovery in milling from 85% as allowed, to 79.4%.

Allowing for a combination of several of these factors including the lower gold price, and on a taxed basis, a D.C.F. rate of return in excess of 20% would appear to be a reasonable estimate of the downside rate of return on future capital cost.

2.3.2 Upside Sensitivity.

The main factors which would improve the return from the project are:

(a) a higher gold price than $A150 per ounce;

(b) the on-going orebody below 2000 ft. Level;

(c) over-conservative estimation of future capital costs.

Re (a).

If gold price in 1978 and thereafter is at a level of $A175 (i.e. $US235 with July, 1975 exchange rates), operating cash flow increases to approximately $A4.5 million per annum (on a fully taxed basis), or to $A7.26 million per annum (on an untaxed basis). D.C.F. rate of return increases from 39% to 49% on a fully taxed basis. (D.C.F. is above 50% for both prices, on an untaxed basis)

Re (b).

If the orebody continues below the 2000 ft. Level with commercial gold values, the benefit would be felt in a higher rate of production (more levels at one time) and/or in an extension to mine life beyond the seven years on which project calculations are based. Both alternatives would result in a higher level of D.C.F. rate of return.

Re (c).

If capital costs have been over-estimated by 10% (or $A630,000), D.C.F. rate of return remains in excess of 50% on an untaxed basis; and D.C.F. would increase by 3% to 42% on a fully taxed basis.

2.4 Investment Risk.

Confirmatory drilling of the orebody extension is scheduled in the "base case" to follow dewatering of the mine to below the 1370 ft. Level; and will be undertaken from the crosscut between Harts and Grubb shafts at this Level. (Alternatively it could be undertaken from the 1000 ft. Level from the crosscut between shafts at that Level).

The main aim of this drilling is to up-grade ore and mineralisation reserves, to obtain additional samples of drill core material for metallurgical test, and to assist in the planning of mine development.

Positive outcome of this confirmatory drilling will inevitably strengthen the project; and it follows that Stage I of mine development has a higher level of investment risk than subsequent stages.

It is also recognized that the initial dewatering requirement for this mine comprises a risk element.

But compared with bringing into production a virgin orebody, this project carries less risk of the unknown because it has been worked for many years and past operations are well documented.

It has been thought appropriate to consider minimum tonnage or grade of ore reserves and mineralisation which would still give a cash flow sufficient to return invested funds plus 100%. (Tax is ignored in this exercise).

With given tonnage, this level of cash flow would be achieved by a headgrade of 7.9 dwt/ton (compared with 13 dwt as certified).

With given grade, this level of cash flow would be achieved by tonnage of about 254,000 tons (compared with combined tonnage of ore reserve and mineralisation of 493,000 tons, as certified).

The above quoted tonnages and grades are based on a gold price in 1978 of $A150 per ounce. For a gold price in 1978 of $A130 per ounce, the relative figures are 311,000 tons, or 9.1 dwts/ton; and for a gold price in 1978 of $A175 per ounce, the relative figures are 206,000 tons, or 6.7 dwt/ton.

This tonnage and grade minima should be viewed in light of the comments on Ore Reserves and Mineralisation in Section 2.2.2 above.

The investment risk during Stage I of mine development is assessed as being within the bounds of commercial acceptability.

2.5 Current Position on the Project.

Over a period of six years, Allstate Explorations N.L. (with some financial assistance from Tricentrol Australia Limited) has carried this project through a difficult and expensive period of deep drilling as required by artificially restricted gold prices up to early 1973; and since then through a program of diamond drilling targetted between 1500 ft. and 2000 feet. This field work has been followed by detailed examination of old mine records and by the fullest possible analysis of the prospects of the Beaconsfield mine including the aspects of dewatering, metallurgy, mine development, mine operation and ore processing.

The outcome of this work is that the re-opening of the Beaconsfield Goldmine has been shown to involve commercially acceptable initial risks, to be technically practical, and (subject to satisfactory outcome of confirmatory drilling) to be a strongly profitable undertaking.

Tricentrol Australia Limited now has a 7½% interest in a Joint Venture with Allstate on this project, and its technical staff have been kept closely in touch with all aspects of project investigations.

Based on this full understanding of the project, Tricentrol Australia Limited has advised of its preparedness to proceed with the funding of its share of further project expenses as soon as Allstate is able to proceed.

Allstate does not wish to fund further expenditure on the project. Allstate is prepared to transfer part of its equity interest in the project as part of an arrangement with an incoming party which will arrange and/or provide for all further funding requirements relating to the 92½% of the project presently held by Allstate. Terms of participation by the incoming party are subject to negotiation.

Allstate Explorations N.L. was floated in 1969 on the basis of this project, and it remains Allstate's major project. Allstate looks to retain a major interest in the project.

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BEACONSFIELD FEASIBILITY REPORT - JULY, 1975.

3. KNOWLEDGE OF OREBODY.

3.1 Geology.

The Beaconsfield deposits are centred on the Cabbage Tree Hill - Blue Tier ridge, about 39 kilometers north-west of Launceston.

The crest of the ridge is composed of Cabbage Tree Conglomerate (Ordovician) striking about NW-SE, with dips of 45° to 65° NE.

(a) The reef traverses Ordovician sediments. These sediments consist of a basal conglomerate (Cabbage Tree Conglomerate) overlain by sandstones and siltstones (Transition Beds) which are overlain by limestone (Gordon Limestone). The transition beds are approximately 1200 feet thick, and are correlated with the Caroline Creek Sandstone of the west coast.

(b) The reef is an infilled fault, widest in the transition beds but very narrow and low grade in both the limestone and conglomerate, giving lithological ore limits. Where the mineralisation traverses the Conglomerate it consists of narrow branching veins known as the "Moonlight-Cum-Wonder" Reef.

(c) The lode was cut by three faults, all of east-side-down displacement. The two westerly faults outcrop, while the third was first encountered on the 715 ft. Level.

The reef is a fissure reef of about 1300 ft. in overall length, varying from a few feet up to 25 feet in width. It strikes about N 50° E, and dips 50° to 60° SE with the quartz emplaced in a pre-existent fault zone. Movement on the fault is about 100 feet, north side east. The reef in the upper working was displaced some 240 feet, west side north, by a cross fault towards the western end and truncated at the western end by a similar fault.

The reef is occasionally split into two branches to give a footwall and hanging wall lode formation.

The lateral control appears to be lithological, as the reef transgresses most members of the Caroline Creek Sandstone but does not persist at economic grades into either of the adjacent limestone or conglomerate formations. Plunge of the orebody is NE away from the cross faults so that below about 600 feet the whole of the orebody lies to the east of both faults.

The structural plan as produced by Montgomery (Government Geologist, 1891) is attached as Figure 3.1; and Montgomery's report is attached as 1/1 in Part "B".

3.2 Mineralisation of Mined Section of Orebody.

The Tasmania Mine undertook a close, systematic and continuous sampling program on all levels, and from stopes, raises, and winzes connecting the levels.

Sampling intervals were 4 ft. - 5 ft. apart, and it appears from the records that where significant changes in stoping width and/or grade occurred the sampling interval was adjusted accordingly.

The data from this sampling information is available on a projected longitudinal section, prepared by the Tasmania Gold Mining Company, and attached as Figure 3.3. Additional data, including ore values and widths in stoped areas, is available from Figure 3.2 which is believed to have been produced by the old company but is unstamped.

Additional to this, a projected longitudinal section for the whole mine workings is attached (reference 4/6 in Part "B"), from which the following information is provided:

Level (feet)	Strike Length (feet)	Average Width (feet)	Average Grade (dwts./ton)
915	1421	6.25	27.25
1000	1292	5.75	19.75
1100	1317	6	16.00
1250	1231	7	11.75
1370	1265	7	8.75
1500	940 (development incomplete)	7	13.00

This data has been checked against the orebody grade and width data in Figure 3.3, with good correlation between the two.

Gold mineralisation is reported to have changed with depth, the gold obtained from the richer upper levels consisting mainly of free milling auriferous quartz, readily amalgamated. Changes in mineralisation below about the 400 ft. Level showed the presence of pyrite, arsenopyrite, chalcopyrite, sphalerite, galena, etc., in increasing amounts, with a considerable proportion of the gold intimately associated with sulphides, necessitating more specialised and expensive treatment methods.

The high proportion of free milling gold above the 400 ft. Level and possibly extending to a zone below this, can be attributed to secondary enrichment in that area. A degree of deep weathering had taken place, evidenced by the presence of a Tertiary deep lead channel, which is 400 - 500 feet deep. Solution action in the lime-rich beds led to partial oxidation effects, thus creating an apparent deep gold enrichment.

At the 1100 ft. Level the orebody was outside the oxidation enrichment zone, and the average grade for the level was 15 dwts. per ton. Below this level average grade fluctuated in a manner to be expected of an auriferous quartz gold body i.e. both along strike and with depth. Gold values were stronger at the 1500 ft. Level than on the equivalent length of the 1370 ft. Level.

At the date of closure of the mine the strike length of the orebody and its average width were being maintained compared with previous levels, and the lode structure was persisting strongly.

The contemporary (1914) opinion expressed by Cundy and Fawcett on this aspect is as follows: "Boring is not necessary to prove that the reef goes down, that is sure, and it is likely to continue to great depths."

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3.3 Drill Intersections of Orebody Extension.

The broad outline of diamond drilling activity aimed at intersecting the orebody extension below 1500 feet has been given in Section 1.4 above.

A plan of the drill holes, a longitudinal projection showing cross-section of drill holes, and detailed assay results of drill cores where the reef was intersected, are given in Figure 3.5, and in reference 4/4 of Part "B".

3.4 Geological Interpretation of Drill Core Data.

(a) The lithological structure surrounding the orebody, as broadly outlined in Section 3.1 above, has generally been confirmed as continuing beyond the 1500 ft. Level, by analysis of drill core material.

For example DDH B4 and DDH A6 both intersected the complete Ordovician sections, these being black shales, massive limestones, argillites and arenites, black quartzites and upper conglomerate. This same sequence was intersected by DDH A3(iii) at about minus 1850 R.L. Correlation of this intersection with those of the two previous holes indicates an overall dip of 55° of country, which is in line with the data available from old reports.

A major shear breccia zone is evidenced in DDH A3 between 1585 ft. and 1774 feet. Above the shear zone the lithological logging indicates a strongly foliated distorted section of phyllites, calc-silicate rocks and limestones suggesting a sheared upper "Transition Beds" section with overlying limestones. This would indicate an overthrust movement east block up. The longitudinal section (reference 4/4 of Part "B") indicates a possible continuation of Eastern block strata in the region of minus 550 ft. R.L., taking account of this overthrust movement, continuing from in the region of minus 2000 ft. R.L. on the westward side of the shear. ?

Generally the stratigraphy shown in reference 4/4, which is based on detailed examination of diamond drill core, conforms with the overall interpretation for Beaconsfield as described in "The Geological Survey Explanatory Report -

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Department of Mines 1974". A copy of the relevant section of this report is attached as 1/8 in Part "B".

(b) The tendency for the lode to split, which was noted in the lower mine workings, has been emulated in DDH B4, B4B and A6. In the latter, a weak auriferous footwall quartz vein was separated by approximately 3 ft. - 6 ft. of country from the main lode. The footwall section in this case was considered non-payable and hence was not included in the hole assays.

(c) DDH A3 was collared about 1450 ft. each (east?) of the holes which gave intersections of the orebody, and it is now clear that in this location DDH A3 was well toward, if not beyond, the north east limits of the projected reef, based on previous strike length of the orebody (1200 ft. - 1400 ft.).

Based on the best information which is now available, and assuming constant dip and strike, DDH A3 could have been expected to intersect the most easterly extension of the Tasmania Reef about R.L. minus 2700 ft. or should have passed close to this eastern extension. The core of this hole shows carbonate veining, with minor pyrite over the whole increment 2379 ft. - 2393 ft. (about minus 2425 ft. R.L.). Recent examination of this split core has shown pyrite, and assays showed minor copper content but nil values for gold and silver.

The fact that mineralisation did occur, in a limestone environment, is consistent with the experience of the eastern end of the old Tasmania workings. It was reported (Cundy and Fawcett 1914) that as the eastern limits were approached, "the reef becomes erratic in its course and splits up into small stringers with occasional bunches of quartz of no value. Here also the limestone is not far distant. In fact several upper levels have been driven into it."

This data could possibly indicate a manifestation of the eastern extension of the Tasmanian lode structure at a depth of 2400 ft.

(d) Core sections at about minus 3250 ft. and 3400 ft. R.L. from DDH A3(iii) show marked similarity to core sections towards the bottom of holes DDH B4, B4B, A7 and A6. This similarity indicates continuity of the transition beds (hostrock strata), and suggests that the Tasmania lode structure (regardless of metal content) could be continuous to at least 3500 feet.

For detail refer T.W. Middleton Report, December, 1974, attached as 1/9 in Part "B".

3.5 Mineralogical Characteristics.

Reference is made in the following paragraphs to a report by AMDEL, 1974 relating to mineragraphic work on pyrite-rich sections of the core of DDH A7. The sample analysed was 3 - 6 times the average sulphide content of the core. A copy of this report is available for perusal.

The intersections in DDH B4, A6 and A7 indicate a quartz-carbonate lode with variable sulphides and some visible free gold. The AMDEL study indicates the paragenesis as an original quartz-carbonate vein (carbonate probably ankerite), brecciated and invaded by pyrite with further brecciation and invasion by arseno-pyrite, chalcopyrite, galena and sphalerite.

Some coarse specks of free gold were identified in DDH B4 inter-sections (Noldart, 1968) and similarly some free gold particles were visible in the core obtained from A7.

In the sulphide-rich sections examined by AMDEL, they reported fine gold inclusions in pyrite, and some gold locked in pyrite grains.

Chalcopyrite is also present in the concentration of 2 - 3% overall, which is consistent with copper content of the orebody of approximately 1%. Silver occurs in lower concentrations than gold but with apparently no direct relationship. The other significant metal is arsenic which ranges from 1.49% down to .05% and occurs in all the intersections made.

As indicated in later sections of this Report, the presence of arsenic is not of major concern in that the recommended process for gold extraction does not include roasting and calcining. The only problem may arise in relation to the copper concentrate in the event that arsenic cannot be removed by gravitational or flotation methods.

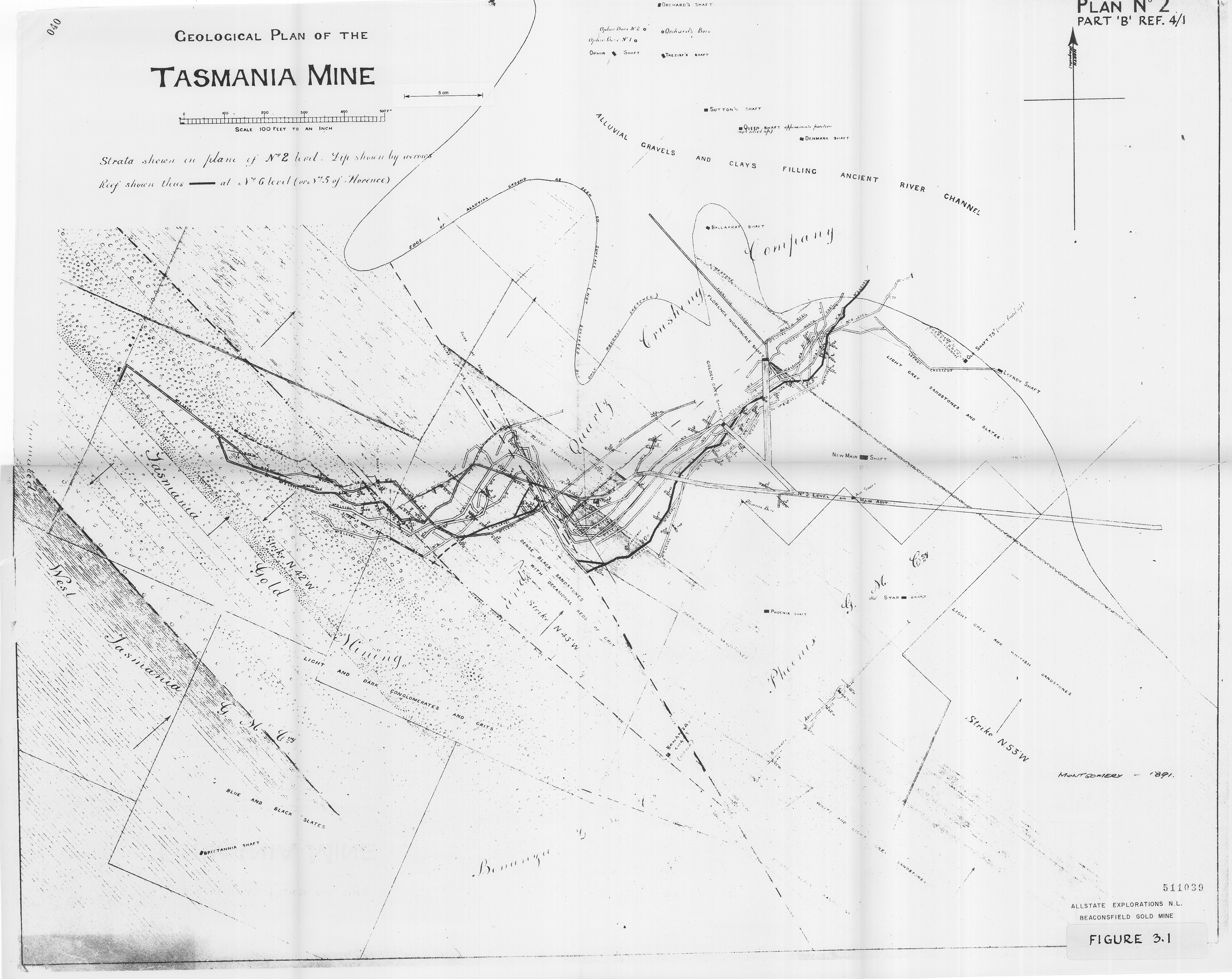
040
GEOLOGICAL PLAN OF THE
TASMANIA MINE
5 cm
SCALE 100 FEET TO AN INCH
Strata shown on plane of No 2 level. Dip shown by arrows
Reef shown thus —— at No 6 level (or No 5 of Florence)
PLAN No 2
PART 'B' REF. 4/1
ORCHARD'S SHAFT
Ophir Bore No 2
Orchard's Bore
Ophir Bore No 1
OPHIR SHAFT
TREZISE'S SHAFT
SUTTON'S SHAFT
QUEEN SHAFT
DENMARK SHAFT
ALLUVIAL GRAVELS AND CLAYS FILLING ANCIENT RIVER CHANNEL
EDGE OF ALLUVIAL GROUND AS SEEN ON SURFACE
BALLAARAT SHAFT
Company
Crushing
Quartz
FLORENCE NIGHTINGALE SHAFT
GOLDEN GATE SHAFT
LEFROY CROSSCUT
LEFROY SHAFT
LIGHT GREY SANDSTONES AND SLATES
NEW MAIN SHAFT
No 2 LEVEL OR MAIN ADIT
DENSE BLACK SANDSTONES WITH OCCASIONAL BEDS OF GRIT
Strike N 45 W
Strike N 42 W
West Tasmania Gold Mining
Tasmania G. M. Coy
LIGHT AND DARK CONGLOMERATES AND GRITS
PHOENIX SHAFT
OLD STAR SHAFT
Phoenix G. M. Coy
LIGHT GREY AND WHITISH SANDSTONES
Strike N 53 W
BLUE AND BLACK SLATES
BRITANNIA SHAFT
Bonanza
WHITE AND LIGHT GREY SANDSTONES
MONTGOMERY - 1891.
511039
ALLSTATE EXPLORATIONS N.L.
BEACONSFIELD GOLD MINE

FIGURE 3.1

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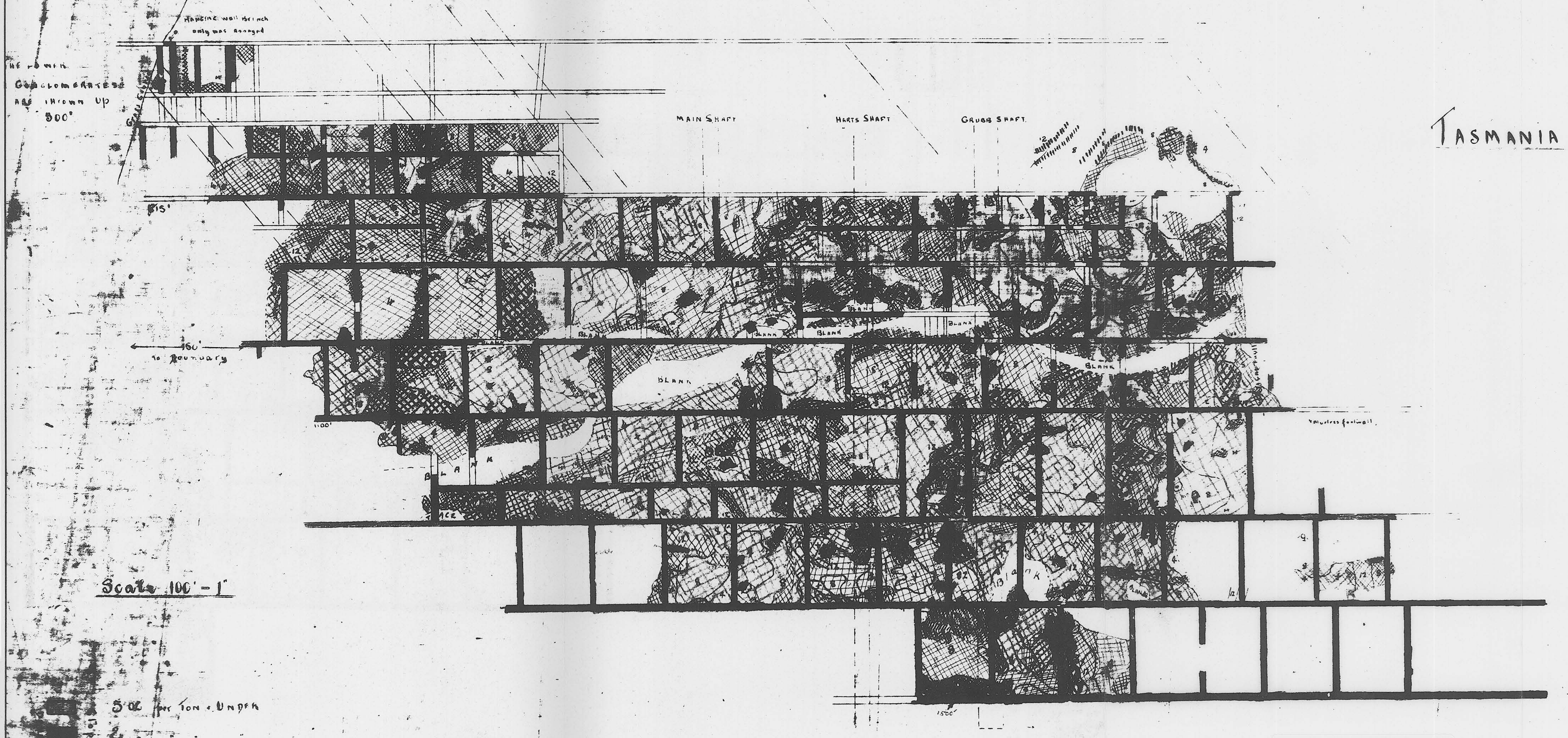
FIG. 3.2

TASMANIA MINE



5 cm

511040

FIG. 3.2

ALLSTATE EXPLORATIONS N.L.
BEACONSFIELD GOLD MINE
FIG 3.2

511041

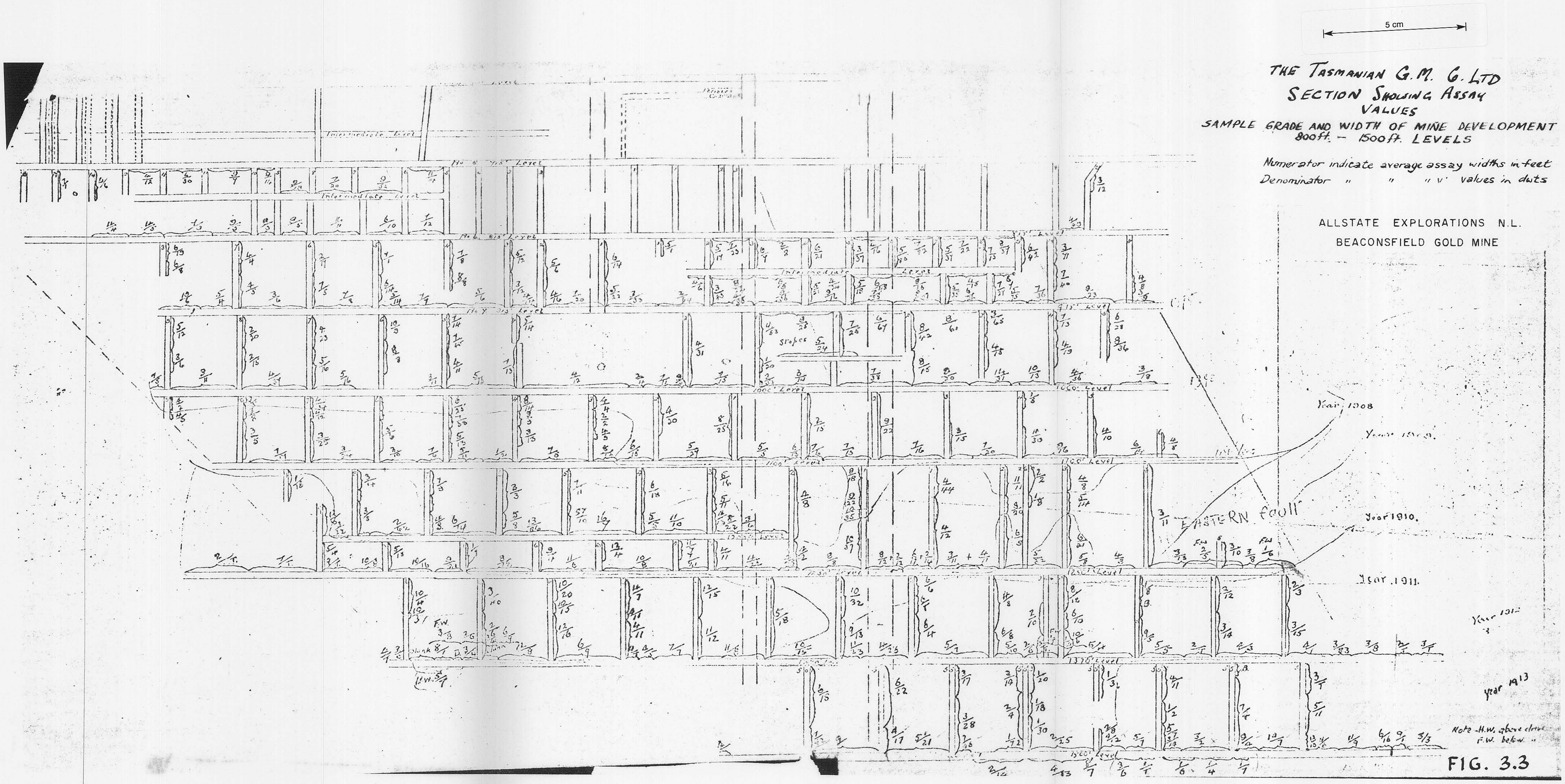
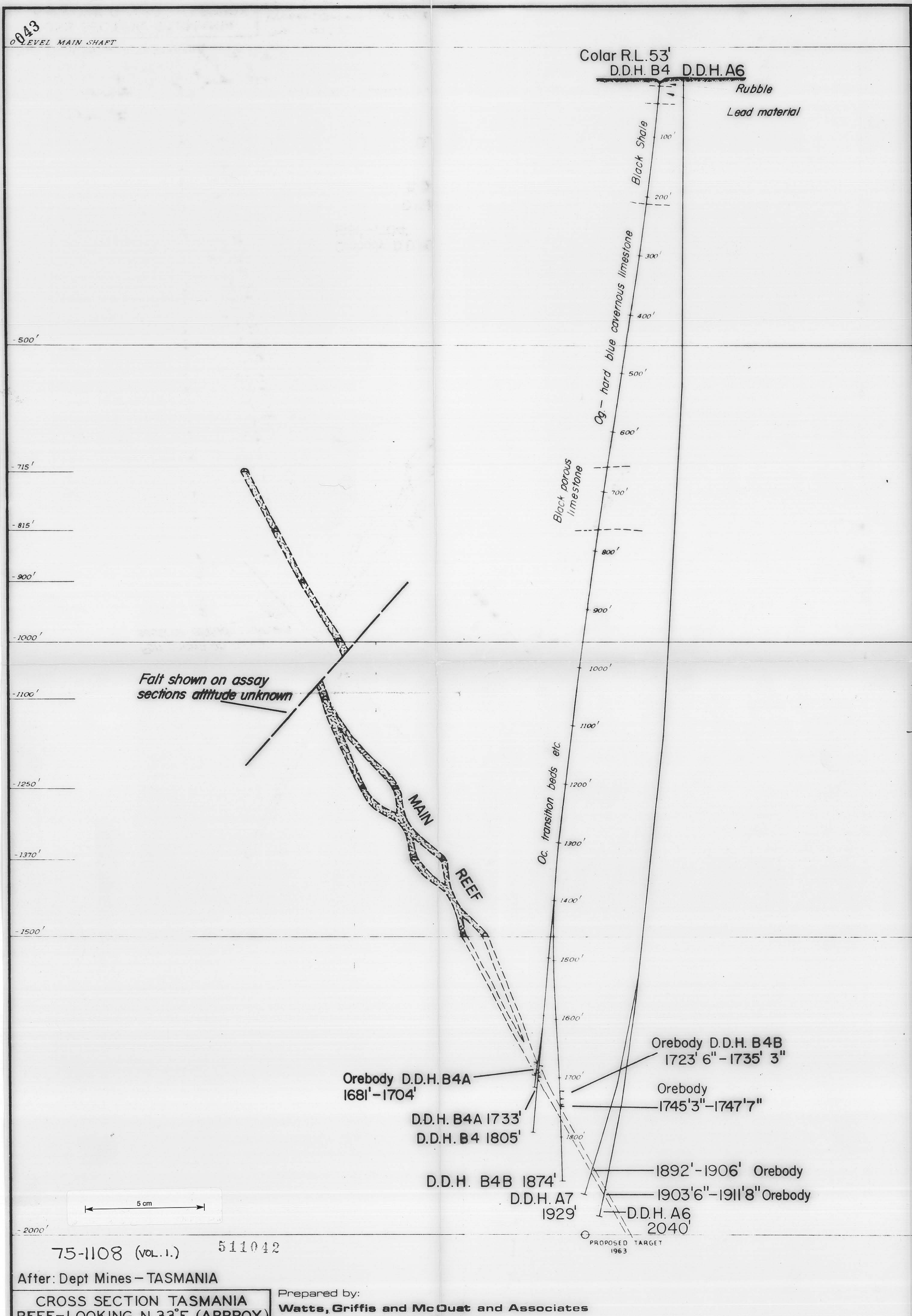


FIG. 3.3



After: Dept Mines – TASMANIA

CROSS SECTION TASMANIA REEF–LOOKING N 33°E (APPROX.)	
DATE *May 1967*	SCALE OF FEET 0 50 100 150
GEOLOGIST *J. Noldart*	
DRAUGHTSMAN *P. Nankivell*	MAP SHEET & Nº

Prepared by:
Watts, Griffis and McOuat and Associates
Consulting Geologists and Engineers
159 Kent St., Sydney
for:
ALLSTATE EXPLORATIONS N.L.

Figure 3.4

ASSAY RESULTS - MINES DEPARTMENT.

D. D. H. B. 4.

Depth				Au	Ag	Cu	As
From feet	in.	To feet	in.	dwts.	dwts.	%	%
1689	0	1691	0	23.0			
1691	0	1693	0	10.7			
1693	0	1695	0	1.7			
1695	0	1697	0	7.5			
1697	0	1699	6	18.3			
1699	6	1702	0	35.3			
1702	0	1704	0	591.0 (76.9 adjusted grade)			
1704	0	1706	0	2.0			
1706	0	1707	8	19.1			
Mines Dept. Composite							
1689	0	1707	8	60.20	4.7	1.06	1.49
Adjusted true width and grade							
	12.5			21.9			

045

ASSAY RESULTS - MINES DEPARTMENT (Continued).

D. D. H. B. 4 A.

Depth				Au dwts.	Ag dwts.	Cu %	As %
From feet	in.	To feet	in.				
1680	10	1683	10	15.4			
1683	10	1686	10	71.0			
1686	10	1689	10	75.0			
1689	10	1692	10	45.0			
1692	10	1695	10	11.0			
1695	10	1698	10	79.2			
1698	10	1701	10	16.0			
1701	10	1704	6	34.2			
Mines Dept. Composite							
1680	10	1704	6	42.1	6.6	0.91	0.43
Adjusted true width							
	17.0			42.1			

ASSAY RESULTS - MINES DEPARTMENT. (Continued).

D.D.H.B.4 B

Depth				Au	Ag	Cu	As
From		To					
feet	in.	feet	in.	dwts.	dwts.	%	%
1723	6	1726	6	21.7			
1726	6	1729	6	17.4			
1729	6	1732	6	41.8			
1732	6	1735	3	39.8			
1735	3	1739	3	0.8			
1739	3	1743	3	1.2			
1743	3	1745	3	3.4			
1745	3	1747	7	16.4			
Mines Dept. Composite							
1723	6	1735	3	26.8	11.0	1.10	0.05
Adjusted true width and grade							
17.0				15.2			

ASSAY RESULTS - ALLSTATE EXPLORATIONS N.L.

D. D. H. A. 6.

Depth				Au dwts.	Ag dwts.	Cu %	As %
From feet	in.	To feet	in.				
1903	6	1904	0	4.2			
1904	0	1905	0	14.2			
1905	0	1906	0	14.4			
1906	0	1907	0	5.0			
1907	0	1908	0	1.8			
1908	0	1909	0	5.4			
1909	0	1909	6	1.0			
1909	6	1910	0	8.4			
1910	0	1911	3	14.4			
1911	3	1911	8	8.8			
Allstate Weighted Mean							
1903	6	1911	8.	8.5	nil	0.76	1.25
Adjusted True Width							
5.5							

ASSAY RESULTS - ALLSTATE EXPLORATIONS N.L.

(Continued)

D. D. H. A. 7.

Depth				Au dwts.	Ag dwts.	Cu 1 %	As %
From feet	in.	To feet	in.				
1891	9	1892	9	2.6			
1892	9	1893	9	6.2			
1893	9	1894	9	10.1			
1894	9	1895	9	2.3			
1895	9	1896	9	10.4			
1896	9	1897	9	19.6			
1897	9	1898	9	11.8			
1898	9	1899	9	6.8			
1899	9	1900	9	32.0			
1900	9	1901	9	37.1			
1901	9	1902	9	8.4			
1902	9	1903	9	17.6			
1903	9	1904	9	71.5			
1904	9	1905	9	81.0			
Allstate Weighted Mean							
1891	9	1905	9	23.1	7.2	0.54	0.62
Adjusted true width 10.5							

4. PREVIOUS GOLDMINING OPERATIONS - DETAIL.

4.1 Tonnage, and Gold Content of Production.

A general summary of the tonnage and gold content of production has been given in Section 1.2 of this Report. However a more detailed look at the figures relating to the last 10 - 11 years of mine operation is given below.

(a) In his special report (reference 1/3 in Part "B") to Messrs. John Taylor & Sons (consultants to the owners), Mr. Arthur Llewellyn in January, 1914 provided the detailed information relevant to the period to September, 1913. (See page 2 of report).

Mr. Llewellyn noted that the occasional inclusion of gold from sources such as plate scrapings and dismantling of the mill tend to vitiate the results for the purpose of yearly comparison, but that the figures served to give a general view of trend.

Mr. Llewellyn also noted however that "the present company's tailings heaps are known to contain 4 - 5 dwts. of gold per ton, a great part of which is profitably recoverable, while a heap of concentrates accumulated during the 10 years working is estimated to contain 10,000 ounces of gold."

(b) In their report of May, 1914 to the Minister for Mines (reference 1/5 in Part "B") Messrs. Cundy and Fawcett give the further information that "for the six months from September 30 1913 to March 31, 1914, there was mined and crushed 20,866 tons, yielding 7,973 ounces equal to 7.64 dwts. per ton. There are also in stock for treatment concentrate and slime with a recoverable content of 24,130 ounces."

Cundy and Fawcett also reported that "a close and systematic method of sampling the reef has been in existence and the results duly recorded on the plant. The samples have been taken at intervals of 4 - 5 feet apart, and the results recorded for all levels, rises, winzes and stopes. Consequently, a complete record of the width and value of the reef can be seen from the 815 ft. to 1500 ft. level, with the exception of the last few months."

"There was a greater width of ground stoped than is sampled and it is found in practice that the tonnage of stone broken and sent to the mill is 50% more than the estimated tonnage worked out on the sampled widths, and the value per ton 33% less."

"The quantity of pyrites in the stone has increased with depth and the proportion of the gold recoverable by amalgamation has gradually fallen with a corresponding increase in treatment costs."

(c) In his report of May, 1923 to the Minister for Mines (reference 1/6 in Part "B"), Mr. J.O. Hudson, Chief Inspector of Mines, comments on the arrangement made via the Tasmanian Government for miners to work the mine co-operatively on a tribute basis, which method operated until November, 1914. "The tributors treated 16,556 tons for a yield of £24,739, being a value of 29s. 10d. per ton treated". (This was equivalent to about seven dwts. of gold per ton). "The Automatic Mine Bin Sampling Assays for 16,556 tons returned an average of 12.789 dwts./ton. The tributors obtained nearly the whole of the ore from blocks developed by the company and which were considered unpayable."

"There is no record of the value of the residues as no allowance was made to the tributors for them, and they were not allowed to take samples. This was very unsatisfactory as the bin samples each week indicated that the mill returns should have been considerably higher. Apparently one sample was taken as an assay return dated 14th December shows 10.9 dwts./ton."

(d) From the above it would appear that the mine records of ore sampling were the result of routine and consistent work over a period of many years. They can therefore be taken as an accurate record of the values in the orebody. Figures 3.2 and 3.3 are reproductions of old mine records on this topic, one of which is stamped by the Company, while the other is unstamped but appears to be authentic.

Another record from the Tasmanian Goldmining Company, being a longitudinal section of the mine but including tabulations of production and data on each level, development and grade, is attached (reference 4/6 in Part "B"). The production and gold recovery statistics included in the various reports which are attached in Part "B", the most detailed of which figures are given on page 2 of Llewellyn's report (reference 1/3), can presumably be taken as extracts from company records no longer available to us.

There is therefore a very strong body of data available from reliable records regarding the worked orebody and gold produced. As noted in Section 3.2 above the orebody data in Figure 3.3 has been checked satisfactorily with the Level data in reference 4/6 in Part "B".

(e) The high gold content of residues and slimes and dumped concentrates, as reported in the above quotations, (from which the company extracted gold from 1914 to 1923), indicates a milling operation up to 1914 which was far from efficient.

There is also ample evidence of excessive overbreak compared with modern practice, and conditions (resulting from the apparent over-capitalisation of the new company in 1903, which would demand maximum mill throughput) which would encourage such overbreak.

These two factors explain the discrepancy between orebody values and gold recovery per ton, in the past.

Strict control of overbreak and an efficient milling operation (aided by factors mentioned in Section 6.1 below) will maximise gold recovery per ton of orebody mined in the future.

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4.2 Water Inflow and Control Measures.

The water inflow into the Tasmania reef persisted throughout the life of the mine, was a significant factor in the cost of production, and was the subject of several reports.

(a) There were many theories as to the origin of the mine water, most asserting that it was accumulated in the surrounding limestone country. In 1903 The Tasmania Gold Mining Company commissioned a very thorough investigation into the matter (by Twelvetrees, the Government Geologist), and reached the conclusion that the large influx was from Blyth's Creek. (Refer 1/2 in Part "B"). This creek was diverted into a sealed deviation channel to reduce the inflow effect.

As evidence, in 1906 the deviation channel at Blyth's Creek broke and overflowed before repairs could be affected. The effect of the water was felt in the mine twenty-three hours after the occurrence, and several days later flooded the mine from the 1000 ft. Level to the 846 ft. Level.

A geological survey of the district disclosed that no places had been located where precautionary work would be any advantage to overcome the water difficulty other than at Blyth's Creek where the water had been deviated by fluming. It was shown that part of the flow from this creek entered the mine by the sandstone beds at the west end of the mine.

(b) The details provided in the 1903 Twelvetrees report are helpful in quantifying the water problem of the mine.

In 1903 the Tasmania mine had a pumping capacity of three million gallons per day. The concern by the directors at that stage was to increase the pumping capacity sufficient to enable the mine to cope with surges of water inflow which occurred when water bearing strata were intersected during mine development work.

However it must be remembered that the major dependence for water pumping was on the old Cornish lift style plunger pumps whereby the pump shaft, comprising timber members, was stroked through a cam system to give a vertical stroke of something of the order of 8 feet. (Supplementary pumps, a pair of Reidler pumps each of 500,000 gallons capacity, were used as auxiliary pumps during stoppages.) Twelvetrees quotes "all these pumps are dependent upon one another in such a way that if any stoppage happens to one of them, then the output of all is immediately affected." In other words, the optimum capacity was something of the order of 3 million gallons per day, and a breakdown in any of the installations substantially reduced this capacity. It was recommended that the pumping capacity should be increased to between 6 - 8 million gallons per day.

Twelvetrees' recommendation was apparently accepted, and it is a reasonable assumption that this was the main reason for Grubb shaft being sunk in such close proximity to Harts shaft i.e. the two shafts were separated by only 150 feet.

(c) Detailed data is available on weekly pumping rates in a report by Hudson dated 1923, over the period from 1911 to the time of the official mine closure in May, 1914. (Refer 1/6 in Part "B"). The average pumping rates are as follows:

1911	3.2 million gallons per day.
1912	2.7 million gallons per day.
1913	2.0 million gallons per day.
1914	1.8 million gallons per day.

At the time of the mine closure water was being pumped from the 1500 ft. Level, and pumping capacity was 6.5 million gallons per day.

Data re pumping rates in the period 1906-1914 are shown in Figures 4.2 and 4.3.

These figures indicate a falling off in pumping requirements with the deepening of the mine, which would be the result mainly of fewer fissures exposed in the Hanging Wall of the lode, as the orebody deepened. However a reduction of development work during 1913 would also have contributed to this result.

4.3 Mining Methods and Shafts.

The reef and the old mine workings are well documented with underground plans, sections, and assay data. Descriptive reports relating to the conditions and the status of the mine at about the time of closure are comprehensive, and are as Figure 3.6 attached and references 4/5 and 4/6 in Part "B" of this Report (plant); also as references 1/3, 1/4, 1/5 and 1/6 in Part "B" of this Report.

The mine was developed through a series of shafts, principally Main shaft (depth 984 feet), Harts shaft 17 ft. x 7 ft. 4 in. (depth 1370 feet plus winze to 1500 feet) and Grubb shaft 32 ft. x 8 ft. (depth 1500 feet). By present day standards, the latter two shafts were large in size, and represented major capital investments. However a significant factor in the shaft sizes was the need to provide compartments for Cornish lift pumps used to handle the significant dewatering requirements.

Pumping techniques today are far in advance of these pumps, and shaft requirements for dewatering are much less.

Stoping was by means of overhand, cut and fill. Level intervals were approximately 130 feet. Development proceeded along the orebody for the full strike length, which was consistently of the order of 1200 ft. to 1400 feet. Stoping floors were then taken on an overhand basis. Quartzite was quarried on the surface, lowered in trucks and tipped down fill passes to provide the back-fill for stabilising the mine, and the working floor for the successive slices.

Timber supports within the stope were also used, and Cundy and Fawcett refer to "stopes being well timbered and in a fairly well advanced state."

There is no precise evidence of the competence of the walls, or of the physical strength of the orebody. However a photograph of a development drive demonstrating a water burst shows a good standing consistent footwall, with no timber for footwall support, whilst the Hanging Wall is supported by round timber. Cundy and Fawcett refer to a Hanging Wall "usually fairly well defined until a hard natured sandstone is met which almost approached a quartzite in character."

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4.4 Overbreak.

The matter of overbreak is worthy of comment in so far as it did affect previous costs, and will have a bearing on future operation

The mine production records in the later years of mine life indicate a consistent overbreak of up to 50%. For example, on the 1250 ft. Level the stoping width was 10½ feet against the true width of 7 feet. On the 1370 ft. Level stoping width was 9.7 feet against a true width of 7 feet, and Cundy and Fawcett state that on the 1500 ft. Level the average width of quartz was 7 feet, and the average stoping width about 10½ feet.

Whether this overbreak was intentional on the part of miners or management attempting to reach output targets, or whether it was uncontrolled is now a matter for conjecture. It certainly had the effect of reducing headgrade by about 33%, and reflected in the recoveries made in the milling process.

On the formation of the new company in 1903 for the purpose of taking over the gold mine, the mine was capitalised to enable treatment of 80,000 - 90,000 tons per year, but the average production over the last 10 years averaged only 52,500 tons per annum. Under these circumstances it is conceivable that the overbreak was a means adopted by mine management of maintaining mill capacity by deliberately breaking non-payable ore, particularly in the latter stages when forward mine development was depleted.

For future operations it would be the intention to control overbreak by strict supervision on grade control (refer 9.1(b)), by the use of timber supports on weak country, by close filling of stoping operations and, possibly, the use of wiremesh with rock bolts for localised areas. By using these techniques it is expected that in future operations overbreak would be confined to 20%, thereby avoiding dilution of headgrade and all the associated mining, haulage and milling costs incurred in processing barren material.

4.5 Ore Processing.

The metallurgical processing as carried out by the old Tasmania gold mine consisted of stamp battery crushing, amalgamation, grinding, pan grinding and further amalgamation.

The grinding pan product was classified and the sands treated on Card tables. Table concentrate was roasted and treated by cyanidation and portion of table tailings were treated by further cyanidation. Flow sheet of the process is appended as Figure 4.4.

Detailed information on processing by previous operators, and particularly the detail of gold recovery, is scanty.

It is known that the company continued in business for about 9 years after closure of the mine, by means of retreating the tailings dumps. The gold content of tailings in previous years must therefore have been substantial. Llewellyn's report noted that the "tailing heaps are known to contain 4 - 5 dwts. of gold per ton."

No detailed descriptions on mineralogy and mineral occurrence are available, but various reports state that with increased depth there was an increase in pyrite content of ore and a diminution of free gold. This would have contributed to a lower recovery in the latter years of mining, considering the complete dependence at the time on the gravity process.

4.6 Closure of Mining Operations.

In June, 1914, the company decided to close the mine due to a period of poor profitability which had depleted financial resources so that it was not possible to undertake the substantial mine development work necessary for continued operations.

This closure of the mine was of major significance to the Beaconsfield area particularly, and the Tasmanian economy in general. Hence the Minister for Mines called for a report by Cundy and Fawcett, and an independent report was prepared by Mr. A. Llewellyn for John Taylor & Sons of London, as consultants to the Tasmania Gold Mining Company. These reports are included under reference 1/5 and 1/3 of Part "B" of this Report.

Both reports concur in their conclusions, and the salient factors relating to the closure of the mine are extracted from these reports:

(a) Labour.

In the final full year of operation the mine had a labour force of 393 men, and this represented a reduction of 54 men since 1911. Llewellyn quotes "labour is dissatisfied and demands for higher pay are made with increasing frequency ... recently a section of this company's employees brought the company before the Arbitration Court on a plaint for raising minimum wages ... this was granted by the Court with the result that wages have now increased 20%. Should this movement extend to other departments of the mine, it will impose an additional heavy burden where the load of costs is already unbearable."

(b) Costs.

Costs in 1913 amounted to £1.15.0 per ton of ore mined which was equivalent to a recovered value per ton of ore mined of 8.2 dwts. per ton.

Historically the mine in 1907 recovered 8.5 dwts. per ton and recoveries had declined to 7.0 dwts. per ton in 1913. The mine therefore was not breaking even and had paid no dividends since 1903.

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The improvement in recovery levels on the 1500 ft. Level in 1914 was apparently not sufficient to provide extra cash flow to catch up on ore development work not undertaken in the previous few years.

(c) Development.

The mine was 18 months behind in its development program and there was not enough ore opened up, regardless of quality, to keep the mill going until a further stage of the shaft was sunk. "Development work has been reduced to a minimum and the idea of shaft sinking has been abandoned, and the best stone has been stoped above the 1250 ft., 1370ft., and 1500 ft. Levels." (Cundy and Fawcett).

(d) Dewatering.

Dewatering costs had been substantially reduced from approximately 20% of the total cost down to 15% of the total cost but still represented a significant cost component in maintaining production.

Accordingly the mine had not proceeded with continued development on the western end of the 1500 ft. Level where it was known that further waterbearing strata would be intersected with a consequent increase in pumping costs. This had a double-barrelled effect. Pumping costs had been reduced by eliminating development which in turn provided insufficient available ore to maintain production.

The consultant's studies indicated that additional development would cost ₤150,000 - 200,000 (in 1914 values), and based on the operating costs, the company would need to develop a further 100,000 tons of ore to cover development and operating costs. At the time of these investigations only about 20,000 tons of ore remained blocked out, and further development was not considered warranted for the conditions pertaining at that time.

There was however an estimated profit of ₤40,000 to be made from retreatment of accumulated concentrate and slimes, and the company undertook this course of action.

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BEACONSFIELD PROCESS UP TO 1914

ROCK BREAKERS
↓
BINS
↓
TRANSPORT TO MILL
↓
STAMP BATTERIES

105 head in 1896

~~40~~ HEAD, 1000 lb STAMPS $4\frac{1}{2}$ TONS/DAY/STAMP
PUNCHED PLATE SCREENS 12 HOLES PER LINEAR IN.
↓
AMALGAMATING TABLES
↓
CLASSIFIERS

- SAND
  ↓
  CARD CONCENTRATING TABLES
  - CONCENTRATE
    ↓ (MERCURY)
    GRINDING PANS
    ↓
    EDWARDS ROASTER.
    ↓ (CYANIDE)
    AGITATORS
    ↓
    FILTER PRESSES
    - SOLUTION
      ↓
      PRECIPITATION
    - TAILING
  - TAILING $1\frac{1}{2}$ dwt.
    ↓
    PART TO PERCOLATION CYANIDATION (UNPROFITABLE AT 33% RECOVERY)
- SLIME +3 dwt.
  ↓
  STORAGE DAM

PREPARED BY K. PARSONS
24-11-74

FIG. 4·1

DRAWN . A.S.

FIGURE 4.2

AVERAGE WEEKLY PUMPING RATE
EX TASMANIA GOLD MINE
MILLIONS OF GALLONS / WEEK
PEAK WEEKLY PUMPING ALSO SHOWN
1911 – 1914

ALLSTATE EXPLORATIONS N.L.
BEACONSFIELD GOLD MINE

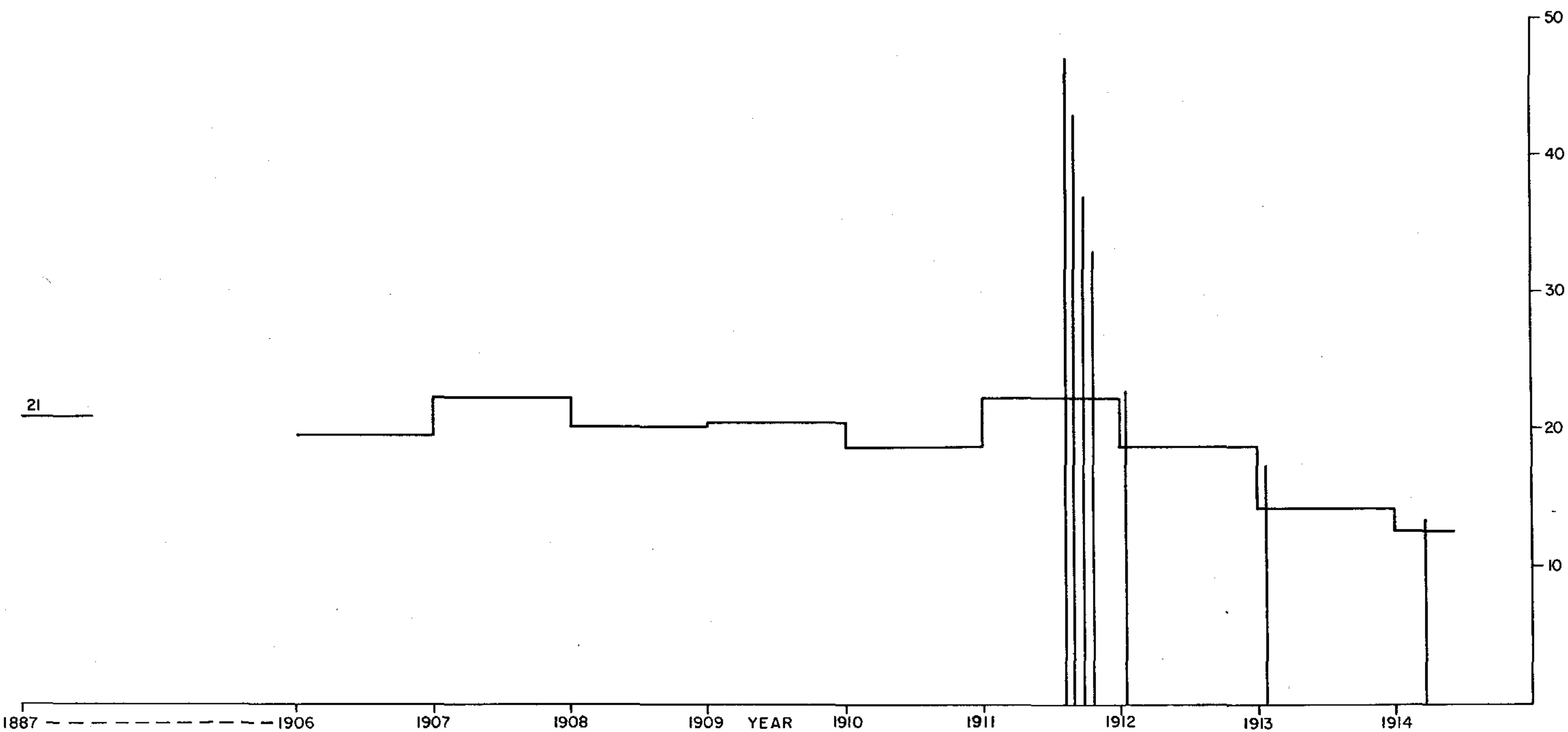

FIG. 4.2

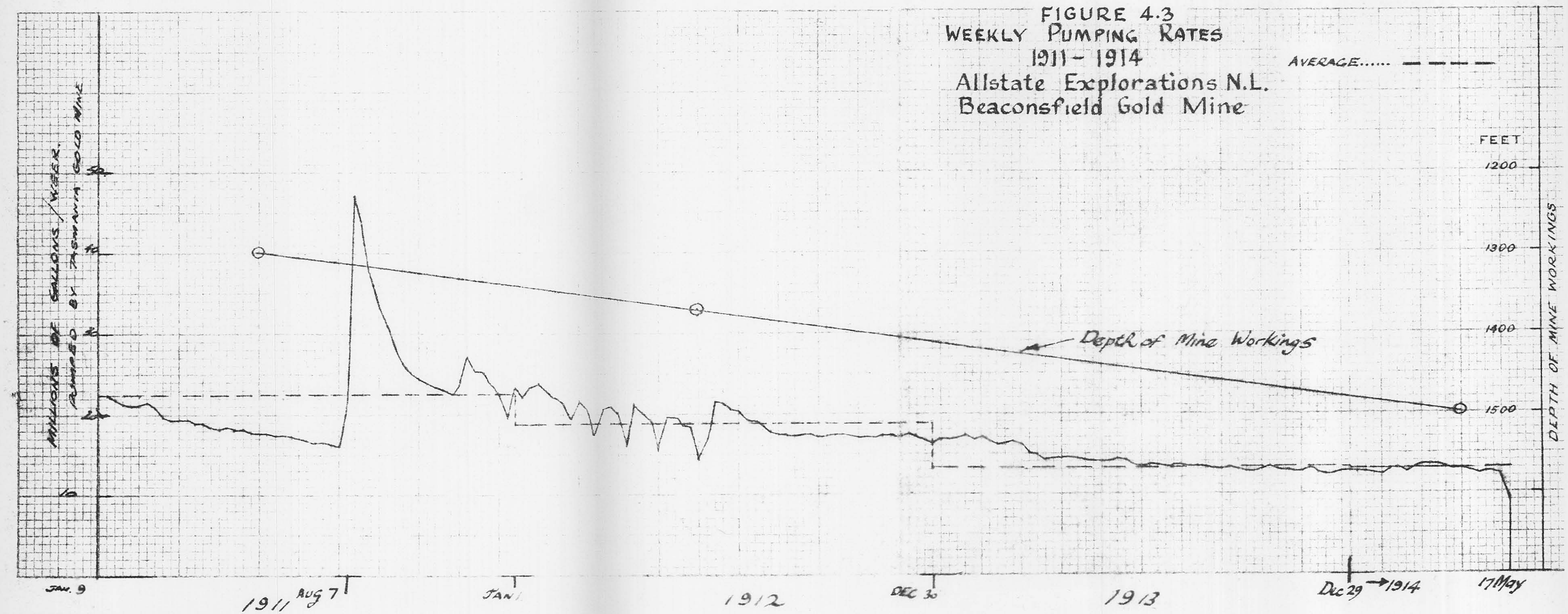

FIGURE 4.3
WEEKLY PUMPING RATES
1911 - 1914
Allstate Explorations N.L.
Beaconsfield Gold Mine

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5. ESTIMATED ORE RESERVES AND MINERALISATION.

5.1 Authority for Reserve Calculations.

Estimates of ore reserves and mineralisation have been made by Mr. T.W. Willsteed, a Consulting Mining Engineer in association with Watts, Griffis & McOuat.

Allstate has separately checked these calculations, and confirms the tonnages and grades quoted.

These calculations have been made by professional Engineers qualified under the Australasian Institute of Mining and Metallurgy, and meeting the Stock Exchange requirements for the publication of Ore Reserves.

5.2 Definitions.

Definitions as recommended by the Australasian Institute of Mining and Metallurgy are accepted.

"Indicated ore reserves" are defined as those for which "tonnage and grade are computed partly from specific measurements, samples or production data and partly from projection for a reasonable distance on geological evidence. The sites available for inspection measurements and sampling are too widely or otherwise inappropriately spaced to permit the mineral bodies to be outlined completely or the grade to be established throughout."

"Possible ore mineralisation" is defined as that "for which the relations of the structure to adjacent orebodies and the geological evidence warrant some presumption that ore will be found but where the lack of exploration and development data precludes it being classed as probable." Such mineralisation is not classified as an ore reserve.

5.3 Calculation of Ore Reserve and Mineralisation.

The following text is extracted from the report of May, 1974, by Mr. T.W. Willsteed in association with Watts, Griffis & McOuat, in which the Ore Reserve and Mineralisation were calculated:

"Ore reserve estimates have been made for the ore extensions below the 1500 ft. Level based on the recorded assay results of diamond drill intersections below that Level.

We believe that the information now available is sufficient to define a probable or drill indicated ore reserve, plus an inferred tonnage of possible mineralisation. We justify this on the following bases:

(i) Assay results on the 1500 ft. Level, and the levels above, appear to have been fully recorded from a close and systematic sampling of the reef prior to extraction. The reported production results appear to confirm the recorded assay information, allowing for the heavy overbreak in stoping which was also recorded.

(ii) The level assay results indicate conformity of mineralisation with levels above, and do not indicate any reason for lack of extensions below the 1500 ft. Level.

(iii) There appears to be no geological evidence to indicate that the reef structure will not extend below the 1500 ft. Level.

(iv) The diamond drill intersections of the lode have been uniformly spaced in section and have given strong, consistent and uniform assay results."

"Ore reserve and mineralisation calculations are based on a lode section as shown in (Figure 5.1).

This is a projection of the reserve parameters on the plane of the vein. The indicated ore is that enclosed by the length of economic mineralisation on the 1500 ft. Level down to the apex of a triangle which carried the influence to the deepest drill intersection (DDH A6) at approximately 1920 ft. R.L. A projection of influence for 50 feet beyond this intersection is allowed for.

The Possible Ore is that additional tonnage enclosed by the projected down strike limits of the original mine workings to a projected level 100 ft. below the drill intersections, and it occurs on either side of the indicated ore section and is classified according to probability.

From an economic assessment which has been completed, a cut off grade of 3 dwts Au per ton is assumed. This is based on breaking and treatment costs of $15 per ton of ore and a gold price of $A100 per ounce. Minimum mining width is taken to be 4 feet. Material of 12 ft. dwt or greater is included as economic mineralisation. A tonnage factor of 12.5 cu.ft/ton is assumed.

The workable lode appears to have had a strike length of about 1500 ft. and an average vein width of 6 to 7 feet. The strike length had been shortened by an eastern fault above the 1370 ft. Level but assays on this end of the 1500 ft. Level indicated lode extension at the eastern end. The western end of this level had not been developed. The 940 ft. driven in lode on the 1500 ft. Level was quoted in the final mine reports (1914) as having an average grade of "just under 13 dwts over an assay width of 7 feet." It was noted at the time that this was an improvement on the average assays for the similar area on the level above where assay has been 8.73 dwts over a width of 5 feet.

The final management report also spoke of a split in the lode developing on the 1500 ft. Level and observed that it was felt that this split would persist for at least 100 ft. below the Level. This, in effect, will increase the mineable tonnage by allowing a split stoping operation in this area.

The average historic head grade in the lower levels of the mine appears to have been 7.9 dwts at probably a 10 ft. width. Thus a 3 ft. overbreak has been taken.

A summary of mine production, which was prepared at the time of closure of the old Tasmania Gold Mine, is as follows:

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Period	Tons Treated	Gold Produced	Average Per Ton
		ozs.	dwts.
1877 to 1896	299,000	371,408	24.84
1896 to 1903	198,850	199,435	20.06
1903 to 1913	524,842	242,143	9.22
Sept. 1913 to Mar. 1914	20,866	7,973	7.64

It is probable that the higher production rates in the later period of the mine's life would have greatly influenced the extent of overbreak. The requirement for tonnage to use the higher capacity facilities would have reduced the desire for tight wall control.

An estimate of approximately 10 dwts. as a reasonable mine average grade at or about the 1500 ft. Level is taken from reports on the earlier mining operation. In this, losses in tailings are variously reported as 4 to 5 dwts. per ton for earlier workings and about 1.5 dwts. per ton at the time of closure. On the reported yield grades at the time of closure this would indicate a head grade of 9 to 9.5 dwts. per ton, from a stoping width of 10 feet, again probably including up to 3 feet of overbreak.

The mineable strike length of ore below the 1500 ft. Level is not clearly defined. Confusing issues include the recorded shortening of the lode above the 1370 ft. Level, and the recorded split in the lode on the 1500 ft. Level. The full length of the lode had not been driven on this level when the mine closed. We recommend that as a conservative measure, a strike length of say 1250 feet be assumed for ore on and below the 1500 ft. Level.

The results of diamond drilling are tabulated in (Figure 3.5). The consistency and strength of the values in all holes indicates the continuity of ore down dip. Because of the uniformity of the high gold assay values, and the frequency distribution of these values, there is little justification to arbitrarily reduce high assays. Core recovery has been 100% through all lode material.

The one exception to this concept is the reduction of the individual very high assay of 591 dwts. in hole DDH B4. This assay has been reduced to the weighted mean of the complete intersection for this hole inclusive of the high assay. This results in an intersection assay of 21.9 dwts. per ton over a true width

5.

not calculated // composite // Lab Assay

of 12.5 feet, as against 60.2 dwts. calculated by the Mines Department.

The full width of intersection of lode in hole DDH B4B is included in the reserve width as it is estimated that the high grade footwall assays would carry the low grade core in the vein. A cut off grade of 3 dwt. per ton is applied.

Each diamond drill hole is assumed to influence its direct zone of influence, which is bounded by the planes bisecting the line of section through the hole intersections. Because of the skew location of the drill hole intersections the zone of influence of the intersections is considered to extend equidistant along strike The eastern limit of the holes consists of a plane which approximately bisects the area of the indicated reserve. The upper limit of influence of the drill holes is assumed to be 50 feet above the upper hole. Because of their proximity DDH B4 and B4A are averaged to represent a single intersection.

To give a more credible projection of reserves the zone of influence of 1500 ft. Level is split according to the weighted average assay values on either side of the bisecting plane. The footwall zone on 1500 ft. Level is assumed to extend half way to the upper drill hole intersection.

The eastern extent of the indicated reserves is determined by the influence of the eastern end of the 1500 ft. Level in the upper zone, and by the average of this zone and the drill hole zone in its lower section.

The values of the possible mineralisation to the east and west of the indicated reserves are assumed to be taken from the adjacent indicated ore zones and are classified according to the distance from available information.

The calculations of ore reserve zones are tabulated in (Figure 5.1). In summary we estimate the reserves and mineralisation as:

Indicated Ore Reserves	=	183,990 tons
Grade/tons - 17.6 dwts. Au.		
Possible Mineralisation	=	309,700 tons
Grade/tons - 14.5 dwts. Au.		

Detailed values for copper and silver content have not been estimated as insufficient assay data is available. We suggest that a figure of 1.0% Cu be used for economic exercises, and that the value of the silver be disregarded at this stage.

For economic analysis and production planning, we recommend that a dilution factor of 20% be applied to these estimates. Such a figure is indicated by the results of earlier workings and also by the occurrence of weak hanging wall material in drill core logging."

Allowing for 20% dilution (refer 4.4):

Indicated Reserves	=	220,800 tons @ 14.7 dwts. of Au.
Possible Mineralisation	=	371,600 tons @ 12.1 dwts. of Au.
Combined ore and mineralisation	=	592,400 @ 13.1 dwts. (weighted average).

BEACONSFIELD GOLD MINE.

ORE RESERVE CALCULATIONS.

Indicated Ore Reserves.

Zone	Tonnage Tons	Width Feet	Grade Au dwts/ton
1500 ft. Level West	30888	5.5	17.5
1500 ft. Level East	42408	9.3	6.7
1500 ft. Level Footwall	10500	4.2	8.5
DDHB4 + 4A	35400	15.0	33.5
DDH4B	25704	17.0	15.2
DDH7	10962	10.5	23.1
DDH6	2640	5.5	8.5
Lower Zone East	25488	11.8	18.3
Total	183990	10.8	17.6

Potential Mineralisation.

East Class I	68000	10.3	11.2
Class II	54600	10.3	11.2
West Upper	39624	7.8	11.2
" Middle	18392	5.5	17.5
" Lower	57628	13.8	24.9
In Depth Class II	71500	10.3	13.2
Total	309744	10.3	14.5

FIGURE 5.1

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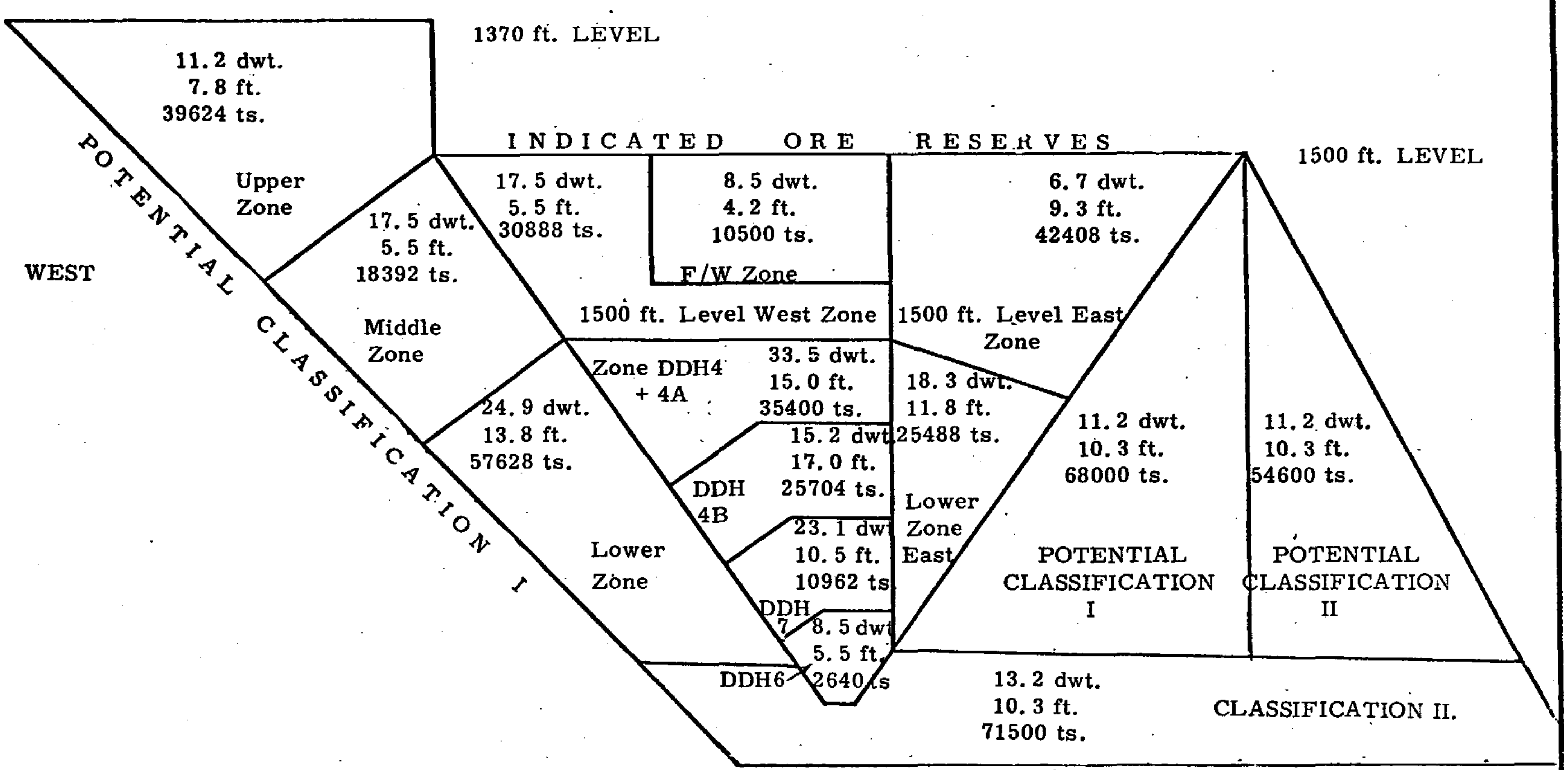

ORE RESERVE CALCULATIONS

by Watts, Griffis & McOuat.

SECTION IN PLANE OF LODE

ORE RESERVE ESTIMATES.

Figure 5/1 Not To Scale.

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ALLSTATE EXPLORATIONS N.L.

BEACONSFIELD GOLD MINE

6. ORE METALLURGY; PROCESS DESIGN.

6.1 Technical Advances in Ore Processing since Closure of Mine.

The major advances in metallurgy since the closure of the mine in 1914, and which are incorporated in the proposed flow sheet, include the following:

(a) Crushing and Grinding Circuit.

The stamp batteries and pan grinding utilised previously for ore reduction have been replaced by Jaw crushers, Gyratory crushers, and Ball Mill installations.

The latter installations require smaller horsepower, and result in improved control and efficiency in the comminution of ore.

Additionally the development of Sieve bends and cyclones, incorporated in the circuit for sizing control, will enable optimum liberation of the free gold, chalcopyrite, and gold-rich pyrite.

(b) Flotation Cells.

The development of Flotation Cells, which were non existent in the previous circuit, will enable selective flotation of deleterious gangue (i.e. talc rich nonmetallic mineral), as well as the selective flotation of a copper and pyrite concentrate.

The preliminary concentration by flotation of the precious metal values into a small bulk for cyanidation, will enable a much smaller plant to be used for cyanidation recovery of gold, than that which was previously required.

(c) Cyanidation Circuit.

The major improvements in this area have been in the development of continuous counter current decantation, the use of zinc dust for precipitation, and improved filtering techniques

(d) General.

The improvements in mechanical and metallurgical installations, will result in a total labour force of 29 men to operate the plant on a continuous basis, compared with an average labour force of 90 men engaged in ore processing in 1913.

6.2 Metallurgical Testing of Drill Core Material.

The metallurgical testing on the drill cores available from drilling of the Beaconsfield orebody (i.e. three intersections made by the Department of Mines and two intersections by Allstate) was carried out at the Launceston laboratories of the Department of Mines.

Allstate engaged Mr. K. Parsons, a metallurgical consultant with some 40 years experience in mill operations and metallurgical plant design, to comment on the results achieved by the Department of Mines.

The comments on metallurgy, process, and recoveries made in this report are based on detailed reports prepared by the Department of Mines and Mr. Parsons, copies of which are attached as 3/1 and 3/2 in Part "B".

All of Mr. Parsons' comments have been accepted as the basis for this report except as regards percentage gold recovery for reasons noted in Section 6.3 below.

The grade assays of the samples tested by the Mines Department, compared with the assay values of the cores from which the samples were taken, are as follow:

DDH No. Sample	B4 Core Assay	B4 Sample Assay	B4A Core Assay	B4A Sample Assay	B4B Core Assay	B4B Sample Assay	A6 Core Assay	A6 Sample Assay	A7 Core Assay	A7 Sample Assay
Au g/t	90	167	60	56.5	40	61	11	14.6	38	35.3
Ag g/t	7	18.1	9	3.8	16	-	-	2.9	11	6.2
As %	1.5	0.95	0.4	0.51	0.05	-	1	0.74	0.7	0.72
Cu %	1.1	1.43	0.0	0.98	1.1	1.3	0.8	0.89	0.5	0.50
S %	7.5	8.4	4.9	4.4	3.6	-	5	5.0	-	13.2

Of the five core samples utilised for test work, the initial two core samples (B4 and B4A) were used to establish the best metallurgical approach for gold recovery and for the production of other mineral concentrates. The final three samples were used for testing related to metallurgical recovery of gold in various products.

After recovery of free gold by gravity means the Mines Department established that it was necessary to produce a pre-flotation concentrate, in order to eliminate a free floating gangue mineral present in the ore. This pre-flotation product carried an unidentifiable gangue mineral, which was suspected to be talc.

Having removed this pre-flotation concentrate, a copper concentrate was produced from the samples assaying 311 grams of gold per tonne of concentrate. A pyrite concentrate was also produced carrying 64 grams per tonne, and a tailing product which assayed 3 grams of gold per tonne.

The pyrite concentrate produced in the test work was then subjected to further tests to recover gold from concentrate.

Initially it was assumed that roasting and calcining of the pyrite concentrate would be necessary, and initial test work proceeded along these lines. The Department demonstrated by roasting and cyaniding the calcine "that about 80% of the gold in pyrite concentrate can be recovered." A roasting process would have involved control and collection of arsenious oxide gases.

However a subsequent test of pyrite concentrate demonstrated that direct cyanidation of pyrite concentrate achieved 86% recovery of gold in solution. This was some 6% superior to that obtained by cyanidation of the calcine product and it eliminated the problem of noxious gas collection. The recommended process design was therefore modified to exclude roasting of pyrite concentrate, and to instal direct cyanidation of this product.

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Both the Department of Mines and Mr. Parsons advise that further test work, preferably on a representative bulk sample of ore, be undertaken before final process design is established. It is for this reason that the obtaining and testing of a bulk sample of ore from the 1370 ft. and 1500 ft. Levels is planned immediately after the dewatering of the mine.

6.3 Projected Gold Recovery.

Based on the test work performed in relation to total gold recovery by all processes from the three core samples tested, it was demonstrated that 86.6% of the gold in ore could be recovered by a combination of gravity, flotation and cyanidation. Where the percentage of free gold is in excess of the 38% (as resulted from this test work) it can be anticipated that higher recoveries than 86.6% could be achieved.

However in predicting gold recovery from ore of a grade certified for Beaconsfield, Mr. Parsons goes on to factor-down this percentage recovery which resulted from sample testing, due to an assumption that tailings assay would remain constant despite a fall in headgrade.

This factoring-down is calculated for the total gold certification for the orebody of 23.9 g/tonne compared with the total gold content of the three cores tested for gold recovery, of 35.6 g/tonne. The following extract is taken from Mr. Parsons' Summary Report:

"Conclusions.

Total Gold Recovery.

a. From a head value of 23.9 grammes Au/tonne the laboratory indications are that the recovery of gold as free gold, gold in cyanide solution, and gold in copper concentrate should total 80 per cent.

b. Past performances indicate that the gold recovered as free gold should be approximately 40 per cent.

c. Improvements on these recovery figures may be possible but have not been indicated in the recent tests.

d. Based on the results of one test, and experience elsewhere, it should not be necessary to use roasting before cyanidation.

e. Improvement in the copper recovery in copper concentrate is necessary to give good cyanidation conditions, this should be possible unless the copper mineral displays unusual characteristics.

f. The mining methods should be such that oxidation of the ore is a minimum.

g. Treatment tests on a sample representative of the ore to be milled to cover the aspects mentioned in the summary will be necessary to make a sound choice of the treatment method to be used."

Allstate has adopted a total gold recovery of 82½% for the first two years of production and 85% thereafter, based on the following considerations:

1. Drill cores B4B and B4 were excluded from the Mines Department's testing for gold recovery. The average percentage of free gold in these two samples was 59.3%, against an average free gold content of 38.2% in the three samples used, and from which a total gold recovery of 86.6% was achieved. Therefore the sample of drill core material on which the Mines Department laboratory test results were based, was not representative of the total drill cores and was biassed in favour of a lower percentage of free gold content.

2. Tests on one of the core samples gave abnormally low recovery of gold from cyaniding of the calcine product - viz. 48.4%, as compared with 82.8% and 79.6% from the other two core samples. This brought the test average for cyaniding of calcines down to 70% compared with Mines Department projections on gold recovery from pyrite concentrate of 80% (see page 6 of Mines Department report).

   It was the actual occurrence of test results on part only of core material recovered, on which Mr. Parsons' figures were based, not on the broader view taken by the Mines Department.

3. One test was performed on direct cyanidation of pyrite concentrate compared with three on cyaniding of calcines. The direct cyanidation test gave 86% recovery of gold from cyanide solution, as against a mean of 70% by roasting and cyaniding the calcine.

   It appears that in adhering strictly to the occurrence of actual test results, Mr. Parsons' forecast has not been based on test results representative of intended future practice.

4. The pyrite concentrate represents 17% of total feed. On a mill throughput of 100,000 tonnes per year this amounts to 17,000 tonnes of product. The cyanide tails as reported in test work represent 12.48% of headfeed i.e. 12,500 tonnes per annum.

   The cyanide tails would be stockpiled separately for subsequent retreatment.

   Assuming that (as per the Mines Department test results and as used by Mr. Parsons in preparing his forecast) the cyanide tails contain of the order of 23.5 grammes/tonne, then this represents approximately 8.2% of the gold in heads.

   Further metallurgical research as advocated by Mr. Parsons would be undertaken on such a product. If only 50% recovery was achieved on the retreatment, then using Mr. Parsons' figure basis this would give an increase of 4% in gold recovery over Mr. Parsons' forecast.

5. On page 25 of his Summary Report Mr. Parsons indicates a 2.7% improvement in overall gold recovery by the use of direct cyanidation on pyrite concentrate from one core sample compared with cyanidation of calcine obtained from the same core sample.

   This is as far as he was able to go, using only Mines Department test figures, in the direction of likely improvement in gold recovery from material subject to direct cyanidation rather than roasting and cyanidation of calcine.

Taking all the above factors into account we consider it reasonable and conservative to adopt in Beaconsfield forecasts, an 82.5% gold recovery in the first two years of production, and an 85% gold recovery thereafter.

6.4 Processing Operations.

Based on the Mines Department test data Mr. Parsons was commissioned to prepare a flow sheet in order that capital costs could be established for the Feasibility Report on this project. This is included as 3/3 in Part "B".

Plant was designed on the basis of 2000 tons of ore being treated per week on a continuous basis seven days per week. Crushing was designed to be carried out on a one-shift basis five days per week and hence the capacity of the crushing section was 50 tons per hour. The rate of throughput planned for the mill proper was 12 tons per hour, continuous operation.

Three principal products were to be extracted i.e. free gold by jigging and amalgamation, a copper concentrate carrying high gold values for shipping direct to a copper smelter, and a pyrite concentrate from which gold was to be recovered by cyanidation.

A general description of the proposed process is as follows:

(i) Crushing and Gravity Section.

Ore will be hauled from the mine by skips and will be discharged into a headframe bin with a capacity of approximately 100 tons. A Diesel truck will be utilised to transfer this ore to the process plant where a limited storage bin will be provided to act as a surge bin. From the surge bin ore will be handled by a feeder to a vibrating grizzly where the plus 2½" product will go directly to a 24 x 36 inch Jaw Crusher. The minus 2½" product will be screened, the undersize going directly to the main mill bins, and the oversize will join Jaw Crusher product to be reduced by two-stage crushing in a Standard and a Shorthead Symons Cone Crusher.

Crushed ore will be stored in 500 ton bins at the head of the mill building which together with the Crude ore bin will provide sufficient ore feed for two days milling.

Primary grinding will be done in a 7' x 8' ball mill and the minus ¼" product treated in a jig for primary concentration of gold.

A tandem sieve bend will be installed in close circuit with a primary ball mill, and the sieve bend undersize will be cycloned.

The spigot discharge will be ground in a secondary ball mill (7' x 8') and the ball mill undersize again treated in a jig similar to the primary circuit.

Jig concentrates from both circuits will be cleaned on a cleaner jig, and the concentrate treated by tabling and amalgamation in a gold room.

(ii) Flotation Copper Concentrates.

Tailings from the jig circuit will be stored in a condition tank and treated by flotation in two stages, in a bank of cells.

The concentrate produced is a low grade copper concentrate carrying deleterious gangue minerals and with minimal gold values in this product.

Tailings from the cleaning stage constitute feed to the copper flotation circuit where rougher and cleaner stages flotation will be utilised to produce a copper concentrate.

This product carries high gold values and the copper concentrate will be thickened, filtered in vacuum pumps, and shipped direct to smelters.

(iii) Pyrite Circuit.

Tailings from the copper flotation circuit will be treated by two-stage flotation to recover pyrite concentrate. This concentrate represents approximately 17% of the head feed tonnage (50 tons per day of product) and contains 60 grammes of gold per ton of concentrate.

The gold in this will be recovered by direct cyanidation along conventional lines. The pregnant solution will be filtered, precipitated with zinc dust, and lead acetate, and precipitate dried, and then roasted and smelted to produce gold bullion.

7. MINE DEWATERING; WATER CONTROL.

7.1 Indications of Previous Rates of Flow.

By reference to Section 4.2 above, it will be seen that for the eight years prior to the closing of the mine, water inflow was at a rate of 3 million gallons per day for the majority of this period, reducing over the last few years to about 2 million gallons per day.

Inflow surges as water bearing strata were cut during mine development were a major problem for mine management, and between 1903 and 1914 the maximum pumping capacity of the mine was increased from 3 million gallons per day to approximately 6½ million gallons per day. This higher capacity was apparently adequate to deal with surges of water inflow at the time.

7.2 Surface Water Control.

The Tasmania Gold Mining Company recognised Blyth Creek as the major source of water flow into the Tasmania goldmine. The creek is thought to be a feeder to the limestone quarries which in turn are connected through waterbearing strata to fissures in the mine walls.

The previous operators diverted and flumed Blyth Creek with round timber poles, a little of which is still evident today. A more sophisticated approach is contemplated along the following lines:

1. A permanent diversion of Blyth Creek by way of an open channel which will be constructed of earthen banks and sealed with an impermeable base, to cut off the wide sweep of Blyth Creek around the limestone quarry area.

2. On the advice of the Water Conservation and Irrigation Commission, the most effective way to seal the diversion of Blyth Creek would be, after the forming up of the new creek course, to lay a bed of clay to a depth of one foot. This clay would be mixed with Bentonite, puddled to an optimum moisture content and compacted and shaped on the two sides, and the base of the creek diversion. This would provide the most effective impermeable seal to obviate water seepage through

limestone beds and is the procedure that would be adopted by the W.C. & I.C. in these circumstances.

It is also possible that, in the event of the ore processing works being located near to Blyth Creek and the limestone quarries (one of two alternative locations being considered), the old quarries could be used for fine tailings dumps and the tailing material could contribute to the sealing of underground watercourses.

The use of dyes is planned as a means of positively confirming the main source of inflow to the mine, and this can be done when dewatering of Harts shaft has progressed some way.

7.3 Estimated Pumping Requirements for Mine Operation.

The historical record of pumping requirements, which diminished with the increasing depth of the mine, would indicate a static requirement at 1500 ft. depth of workings, of approximately 2 million gallons per day to handle the natural inflow into the mine. However provision must be made in the new installations for a capacity sufficient to stand surges of water as water-bearing strata are intersected by developmental headings; also for capacity in the early years to cope with extra flow related to lowering of the watertable in the surrounding countryside.

For these reasons, planned pump capacity is 6 million gallons per day. This will give approximately 4 million gallons per day of surplus capacity over and above the natural inflow into the mine. Such surplus capacity will be fully used initially to dewater the old workings, and will also be used to some extent in the early period after initial dewatering, to handle the extra water inflow related to lowering of watertable. However it is expected that after a period the hydraulic gradient in the surrounding country will be lowered to a point where the rate of inflow into the mine proper will follow the pattern of previous pumping experience.

Following completion of initial mine dewatering, estimates in this Report allow for the removal of 3 million gallons per day for a period of two years, reducing to 2 million gallons per day thereafter

The final pump installation in the dewatered mine will use 3,300 Volt power and will comprise:

(a) three submersible pumps (450 H.P. motors) with a combined 670 ft. lift capacity of six million gallons per day, to lift water from the shaft sump to the 1500 ft. Level and to cope with any water surges which may flood the lower workings. These may be supplemented by centrifugal pumps on the 1650 ft. and 1800 ft. Levels to deal with any continuing flow from these Levels.

(b) Three centrifugal pumps (360 H.P. motors) with a combined 500 ft. lift capacity of six million gallons per day, to lift water from a reservoir at 1500 ft. to a reservoir at the 1000 ft. Level.

(c) Two centrifugal pumps (500 H.P. motors) with a combined 400 ft lift capacity of six million gallons per day, to lift water from the 1000 ft. reservoir to a reservoir at the 635 ft. Level.

(d) Three centrifugal pumps (440 H.P. motors) with a combined 650 ft. lift capacity of six million gallons per day, to lift water from the 635 ft. reservoir to drainage channels at the surface.

This pumping set-up will cope flexibly with initial dewatering, normal inflow, surge inflow, and the unlikely event of flooding of the mine to any depth.

7.4 Initial Dewatering-Procedure and Timing.

(a) General Approach (Base Case).

It is proposed to dewater the mine workings from Harts shaft.

Harts shaft has been chosen as the entry point into the mine because it was the only fully serviceable shaft at the time of mine closure, and the recorded data show the shaft to be in solid ground (exclusive of some detrital material around the collar) right through its full depth. Refer to Figure 7.2 where this feature is shown as part of a geological interpretation of the Deep Lead channel, prepared by Tricentrol in 1974.

The main alternative would have been Grubb shaft which was sunk in the Deep Lead formation. The upper 500 ft. of this shaft had closed in at the date of mine closure to the extent that this part of the shaft was accessible for rope haulage only, not for haulage of men, materials or ore.

Power supply for dewatering will be provided by a diesel generating station of approximately 3000 Kw capacity (refer Section 11.2, items 15 and 16).

A requirement preliminary to the commencement of dewatering is the repair of the collar of Harts shaft, where there has been a collapse of some abutments and foundations of the old shaft collar, and some undermining of the Harts shaft Winder House.

The surface structures remaining at Beaconsfield from previous mining operations include the exterior walls of the Harts shaft winder house, the boiler house, and the Grubb shaft winder house. They have historical significance in the local context. It will therefore be necessary to underpin the walls of Harts shaft winder house at an early stage of repairing the shaft collar.

Repair of the shaft collar, which will include inter alia the removal of the slabs of unreinforced concrete from near the mouth of the shaft and the removal of broken timbers at the mouth of the shaft, will permit the selection of one of the three shaft compartments for the lowering of pumps and effecting of dewatering and lower shaft repairs.

The operation of restoring the shaft collar has been studied by Pearson Bridge Pty. Ltd., which has provided the cost estimate for this item. (Subsequent auger drilling in the shaft area indicates that collar restoration may be simpler and cheaper than forecast by Pearson Bridge).

The attached Figure 7.1 gives a schematic diagram of connections between the three major shafts used for previous mining operations i.e. Main shaft, Harts shaft and Grubb shaft.

The first stage of dewatering would be to the 635 ft. horizon, in Harts shaft. From this horizon there are interconnections to Main shaft, via a Riedler Pump Chamber, and a winze connection between Harts and Main shafts. Therefore at the 635 ft. horizon it should be possible to determine that all upper workings i.e. above 600 ft. R.L. have been drained by the dewatering from Harts shaft.

The 600 ft., 715 ft., 815 ft. and 915 ft. Levels were developed from Main shaft. Thus, once command has been obtained of Main shaft, inspections should be possible from underground of the effective dewatering, by the pumping from Harts shaft, of the 715 ft., 815 ft. and 915 ft. Levels. This is dependent on the extent to which the recorded blockage of Main shaft as an airway at the 1000 ft. Level, caused by a fall of timber from the 200 ft. Level, extends up the shaft.

The old mine plans show dams at each of the main crosscuts from the various shafts to assist in controlling the rate of water inflow to the pumps. The physical construction of these dams is not known, but it is likely that heavy timber boards across the narrowed openings would have been used. If it is found that these dams are holding back water as dewatering proceeds, they can be demolished by an explosive charge as they are progressively exposed by the lowering of the water level in the shaft.

From the 1000 ft. Level Harts shaft was used to develop the mine down to the 1370 ft. Level. Hence on each level there is connection to mine workings and there exist crosscut entries to the orebody up to 300 ft. in length.

These crosscuts are in Hanging Wall (in quartz-sandstone beds), and apart from providing inspection points, the crosscuts can be used for temporary storage dams, and if desirable for positioning extra pumps to supplement the shaft pumps.

From 1370 ft. to 1500 ft. there is a winze connection below Harts Shaft of 8 ft. x 4 ft. dimension. There are also crosscut connections between Harts and Grubb shafts at the 1000 ft. and 1370 ft. Levels.

It is intended that dewatering from the 1000 ft. to the 1370 ft. Level will be undertaken using the submersible pumps in Harts shaft, and that the submersible pumps will probably be transferred to Grubb shaft for the final stage of dewatering below 1370 ft. Level.

The time taken to dewater the old mine workings down to the 1500 ft. Level cannot be calculated with certainty, being a function of rate of water inflow, pump capacities used, and the extent to which temporary repair work is necessary to the timbers of the shaft.

A period of 9 weeks has been allowed for the dewatering operation to the 1370 ft. Level, and the 50% contingency allowed on this item in the cost estimates in effect provides for an additional 4 weeks, as costs are related primarily to timing. A further 6 weeks (including contingency) is allowed for dewatering from 1370 ft. to 1500 ft. Level. These periods exclude intervening holding periods used for stage-pump installations on the exposed Levels.

Given a rate of dewatering of 6 million gallons per day the 13 weeks of dewatering represents about 800 million gallons pumped out. If one assumes that a "normal inflow" of 2 million gallons per day will recommence during draw-down, then the 13 weeks represents at least 500 million gallons of effective dewatering during the 13 week period.

It is also assumed that pumping will be at a rate of 3 million gallons per day for two years after dewatering of the 1500 ft. Level is achieved, representing a further effective dewatering of the countryside of 1 million gallons per day or 720 million gallons over 2 years.

At the end of this two years it is assumed that pumping will diminish to the "normal" level of 2 million gallons per day. The estimated 1320 million gallons of effective dewatering of mine workings and surrounding countryside meets the estimate for this item of several experienced mining men.

Some detail of the planned dewatering operation is given in the following text.

(b) Dewatering Procedure in Detail (Base Case).

Surface installations for the shaft repairs and for mine dewatering are as follows:

a. Surface double drum winch, capable of lowering a sinking stage down to 1500 feet.

b. A temporary headframe over the shaft collar to enable lowering and raising of the stage, and with an independent sheave for a lightframe man cage, to lower men to the sinking stage.

c. A fast winding winch for the man cage installation.

The three submersible pumps will initially be used to progressively lower the water level in Harts shaft. (Later they form part of the permanent pump installation.) The submersibles are suspended on their own columns, and it is intended to use a sinking platform working above the level of the pumps for temporary repair of shaft timbers and installation of services, as described below.

Dewatering of the mine will be done in stages at approximately 500 - 600 feet intervals, and use should be made of the Riedler pump chamber at 635 ft. as the first stage base. There will be four stages of dewatering i.e. to horizons of R.L. minus 635 ft., R.L. minus 1000 ft., R.L. minus 1370 ft., and R.L. minus 1500 feet. These stage intervals are governed partly by practical considerations to maximise control of water inflow surges during mine operation, and partly by the availability of underground sites for locating water dams for temporary storage, and the intention to undertake confirmatory drilling from the 1370 ft. Level.

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Procedures in relation to the dewatering and shaft repairs will be as follow:

a. Clear surface debris from around the mouth of the shaft.

b. Restructure the shaft collar, underpin the winder house, backfill around collar, dewater, test to 150 ft. to determine condition of shaft timbers, inspect at lower depths using T.V. underwater camera.

c. Effect minimal repairs to the shaft timber in the working compartment, using sinking platform.

d. Lower the water level and progressively lower the submersible pumps and the sinking platform, carrying out minimal timber repairs from the platform.

e. Instal the services in the shaft compartment. These will initially comprise the rising mains for the pumps, ventilation duct, electrical cables and perhaps air and water mains. Steel buntons at approximately 50 ft. intervals would also be installed across the short dimension of the shaft to support the rising main.

f. This procedure will be continued until the 635 ft. horizon is reached.

At Harts shaft, R.L. minus 635 ft., the Tasmania Gold Mining Company installed a pump chamber, measuring 12 ft. x 12 ft. in cross-section, and extending 75 ft. toward Main shaft. The pump chamber then extended a further 175 ft. as a crosscut, with a winze connection to the 715 ft. Level from Main shaft.

As this pump chamber is in Hanging Wall, it should be in a condition that requires minimal repairs to re-establish this area as a staging point.

The 635 ft. horizon will serve as a base for locating the first set of centrifugal pumps, and by constructing a concrete wall at each end of the chamber, a storage capacity of 100,000 gallons will be available. The availability of the 635 ft. horizon also provides an underground base for services, and material storage.

Procedure Between R.L. 635 ft. and 1000 ft. Levels.

The dewatering of Harts shaft will be extended from R.L. 635 ft. by lowering the submersible pumps and sinking platform as in the previous stage, and using the submersible pumps to deliver water to the 635 ft. reservoir from which it is lifted to the surface by the centrifugal pumps.

It is planned to advance to the 1000 ft. horizon, where there is a crosscut between Harts shaft and Grubb shaft. This crosscut will be utilised as a water storage dam, and for the installation of a second set of centrifugal pumps. These pumps will raise water (which is delivered to the 1000 ft. Level storage dam by the submersible pumps), to the storage dams at the 635 ft. Level.

A crosscut between Harts shaft and the orebody exists at the 1000 ft. Level, also at the 1100 ft., 1250 ft. and 1370 ft. Levels. These crosscuts can be used for storage purposes, as they are exposed by the lowered water level.

Procedure Between 1000 ft. and 1500 ft. Levels.

From 1000 ft. to 1370 ft., pumping and shaft timber repair will proceed as for previous stages of dewatering.

The dimensions of Harts shaft are 17ft. x 7ft. 4in. to the 1370 ft. Level. Below this there is a winze connection (8ft. x 4ft.) from the 1370 ft. horizon down to the 1500 ft. Level.

To dewater below the 1370 ft. Level some or all of the submersible pumps will be transferred along the crosscut to operate in Grubb shaft, leaving the Harts shaft winze free for inspection and, if necessary, clearing out at lower levels In this event the pump discharge pipes will traverse the crosscut between Grubb and Harts shafts, thence to the 1000 ft Level reservoir off Harts shaft.

The stripping of the winze will take place after installation of the permanent winder and re-timbering of the shaft to 1370 ft. Level.

On completion of winze stripping, a third contrifugal pumping station will be installed off Harts shaft at the 1500 ft. Level, and the submersible pumps will be used to deliver water to the 1500 ft. Level during sinking of Harts shaft to 2000 ft. and development of Levels between 1500 ft. and 2000 ft.

(c) Alternative Procedure and Timing.

In the event that emphasis is put on minimising expenditure and commitment up to completion of confirmatory drilling, (at the cost of delaying commencement of production and increasing capital cost), the following position would apply:

(i) The submersible pumps used would be increased in power (525 H.P.) and number (four) to handle a lift of 1100 ft. at 6 million gallons per day capacity.

(ii) Dewatering would proceed directly to 1100 ft., postponing establishment of the centrifugal pump stations at 635 ft. and 1000 ft. Levels.

(iii) The crosscut between Harts and Grubb shafts at the 1000 ft. Level will be cleaned out and used as a base for confirmatory drilling of the east and west sectors of the orebody extension between 1500 ft. and 2000 ft.

(iv) Diesel generating capacity would be limited to three sets which would be directly connected to three of the submersible pumps. The fourth submersible pump would be powered by H.E.C. supply currently available to the site.

(v) Upon completion of confirmatory drilling orders would be placed for centrifugal pumps, which are 26 - 30 weeks delivery. Delay in placing these orders is the main factor which would cause delay in commencement of production.

(vi) Increase in total capital cost would stem mainly from the effect of inflation on delayed expenditure. Assuming a 9 month delay would result in a 10% inflation increment on capital expenditure of $6 million, the additional cost from this source would be about $600,000.

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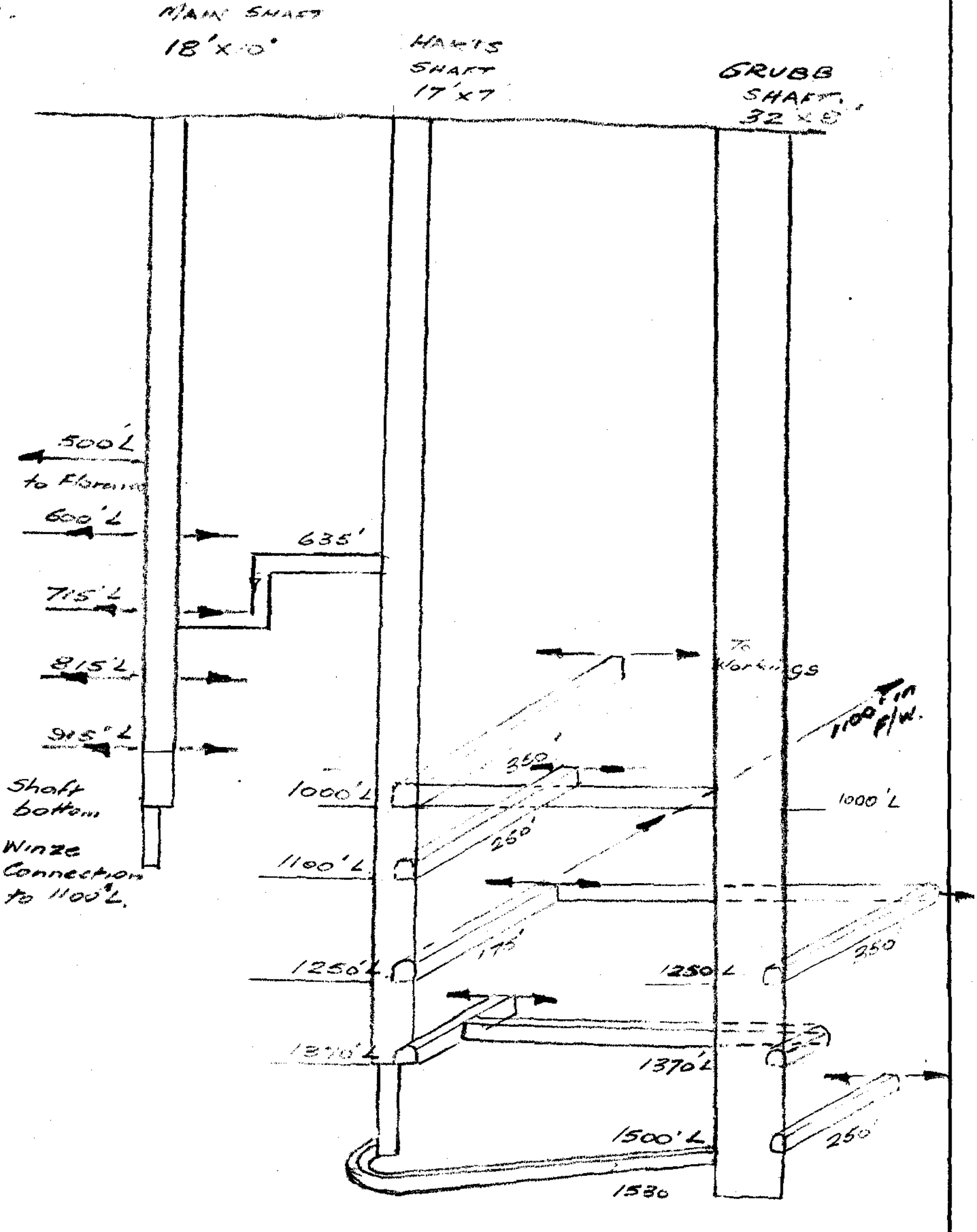
Increased costs would also flow from purchase of higher capacity and greater number of submersible pumps, increased drilling costs due to the 370 ft. of extra drilling per hole drilled, and from purchase of items such as diesel generating sets which suit the initial stage but may not suit longer term requirements.

FIGURE 7.1 511088

093



SCHEMATIC - SHOWING

1. CONNECTIONS FROM OPERATING SHAFTS TASMANIA GOLD MINING CO. TO WORKINGS. (SHOWN THUS ——→)

2. INTERCONNECTIONS BETWEEN SHAFTS IN COUNTRY SHOWN THUS

ALLSTATE EXPLORATIONS N.L.

BEACONSFIELD GOLD MINE

FIG. 7.1

511089

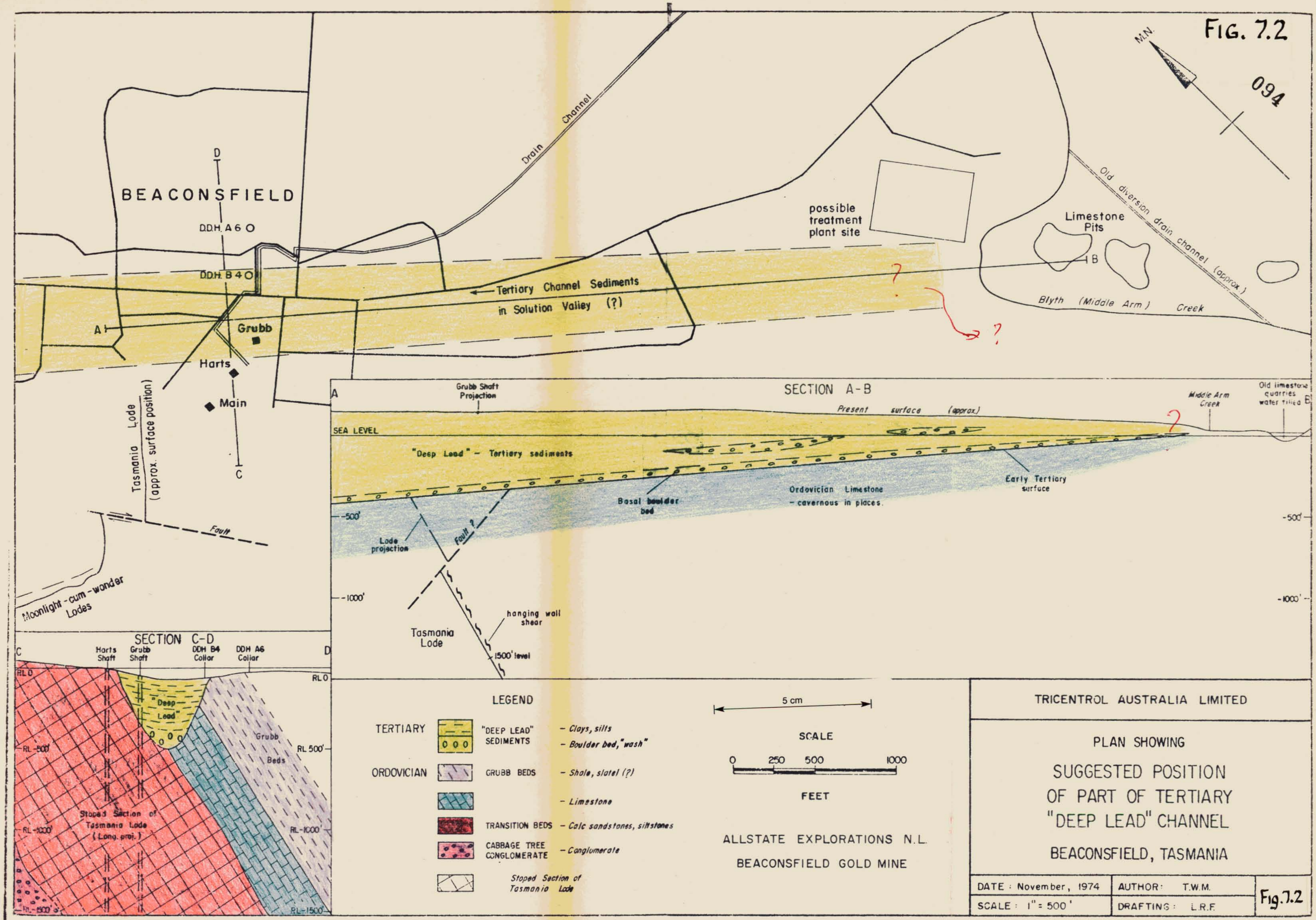


FIG. 7.2
094
M.N.
BEACONSFIELD
D.D.H. A 6
D.D.H. B 4
Drain Channel
Tertiary Channel Sediments
in Solution Valley (?)
Grubb
Harts
Main
A
B
C
D
possible treatment plant site
Limestone Pits
Old diversion drain channel (approx.)
Blyth (Middle Arm) Creek
Tasmania Lode
(approx. surface position)
Fault
Moonlight-cum-wonder Lodes
SECTION A-B
Grubb Shaft Projection
Present surface (approx.)
Middle Arm Creek
Old limestone quarries water filled
SEA LEVEL
"Deep Lead" - Tertiary sediments
Basal boulder bed
Ordovician Limestone - cavernous in places.
Early Tertiary surface
-500'
-1000'
Lode projection
Fault ?
hanging wall shear
Tasmania Lode
1500' level
SECTION C-D
Harts Shaft
Grubb Shaft
DDH B4 Collar
DDH A6 Collar
RL 0
RL 500'
RL-1000'
RL-1500'
"Deep Lead"
Grubb Beds
Stoped Section of Tasmania Lode (Long. proj.)
LEGEND
TERTIARY
"DEEP LEAD" SEDIMENTS - Clays, silts
- Boulder bed, "wash"
ORDOVICIAN
GRUBB BEDS - Shale, slate! (?)
- Limestone
TRANSITION BEDS - Calc sandstones, siltstones
CABBAGE TREE CONGLOMERATE - Conglomerate
Stoped Section of Tasmania Lode
5 cm
SCALE
0 250 500 1000
FEET
ALLSTATE EXPLORATIONS N.L.
BEACONSFIELD GOLD MINE
TRICENTROL AUSTRALIA LIMITED
PLAN SHOWING
SUGGESTED POSITION OF PART OF TERTIARY "DEEP LEAD" CHANNEL
BEACONSFIELD, TASMANIA
DATE : November, 1974
AUTHOR: T.W.M.
SCALE : 1" = 500'
DRAFTING: L.R.F.
Fig. 7.2

8. CONFIRMATORY DRILLING; BULK SAMPLING

8.1 Confirmatory Drilling.

(a) From 1370 ft. Level.

Following mine dewatering to below the 1370 ft. Level, it is intended to carry out a program of confirmatory drilling from the 1370 ft. Level in order to up-grade blocks of ore currently classified as "possible mineralisation", to give data for mine development planning, and to give further information regarding the nature of the orebody from minerological examination of drill core material.

It is not considered economically feasible to carry out this confirmatory drilling from the surface due to limited drilling sites in the Beaconsfield township, difficulty in controlling the direction of holes through the transition beds due to the dip and strike of these beds in relation to the orebody, and the excessive cost of drilling in country which includes cavernous limestone.

Both Harts and Grubb shafts at the 1370 ft. Level are located in the Hanging Wall, and the crosscut between Harts and Grubb shafts gives suitable drilling sites for drilling out both the eastern and western sectors of the orebody between 1500 ft. and 2000 feet.

The technical feasibility of undertaking this drilling has been confirmed by Associated Diamond Drillers, who are familiar with the nature of the country from their previous drilling contract work for Allstate at Beaconsfield, and who have studied mine plans and plans for the intended drilling. Refer their letter, attached as Figure 8.1.

Drilling can commence as soon as the water level is lowered below the 1370 ft. Level and any sediment has been removed from the crosscut between Harts and Grubb shafts at this Level

To expedite this operation it will be possible to employ two drilling teams concurrently, one drilling the western sector and one the eastern sector, both located toward the Grubb shaft end of the crosscut.

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Concurrent with this drilling, dewatering to the 1500 ft. Level will be proceeding, so that there will be ample safety margin on water level, for the drilling operation.

(b) From 1000 ft. Level.

Alternatively, if there is a strong desire to complete confirmatory drilling at minimum cost (at the expense of delay in overall completion and increased total project cost), confirmatory drilling can be undertaken from the crosscut between shafts at 1000 ft. Level, with dewatering to below that Level being achieved as described in Section 7.4(c) of this Report.

Such drilling could also employ two drilling teams concurrently.

The drilling of ten holes (at 100 feet depth intervals near each extremity of the orebody extension between 1500 ft. and 2000 ft.) would involve about 2600 meters of drilling, and some increase in difficulty of direction control due to the run of the country.

The cost per meter would probably be greater than drilling from the 1370 ft. Level, and the drilling period would be longer than a proportional extension of time based solely on increased footage.

8.2 Bulk Sampling.

It is intended to obtain bulk samples (about 50 tons) of ore partly from the western half of the 1370 ft. Level and partly from the eastern (i.e. developed) half of the 1500 ft. Level, for metallurgical testing. This testing is necessary to assist in final determination of the detailed design of the ore processing plant.

Such bulk samples would be taken from the old 1370 ft. and 1500 ft. drives by stripping underfoot. The ore thus obtained should provide a typical sample of gold mineralisation over the full strike length on the upper horizon of the orebody extension.

If the old drives cannot be reconditioned, fresh ore will be broke from the western sector at the 1500 ft. Level beyond where previou mining operations ceased, and a 50 ton sample obtained from this area. The ore thus obtained may not be representative in terms of Au grade, but studies would give valuable mineralogical and mineragraphic data.

Alternatively, the sample could be obtained from the stripping of the Harts shaft winze where it does through the orebody below minus 1370 R.L.

The sample from underfoot stripping on the 1370 ft. Level should be obtained and be tested before the contract is let for the ore process plant, while the 1500 ft. Level sample should be obtained in time for the plant design to be amended (if necessary) based on test outcome. (Refer timetable, Figure 10.1).

Note that metallurgical testing will also be undertaken on drill core material from the confirmatory drilling undertaken from the 1370 ft. Level. This material will be available about the same time as the bulk sample taken from the 1370 ft. Level.

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FIGURE 8.1

Associated Diamond Drillers Pty. Ltd.
DIAMOND DRILLING CONTRACTORS
ADDRESS ALL MAIL TO P.O. BOX 116 PRESTON VICTORIA 3072
Telegraphic Address: "COREDRILL" Melbourne and Sydney
64-74 BELL STREET, PRESTON VICTORIA TELEPHONE 44 0231 6 LINES

MDH/ge

11th June, 1975

Mr. A. Silver,
General Manager,
Allstate Explorations N.L.,
Suite 3910, Tower Building,
AUSTRALIA SQUARE,
SYDNEY, N.S.W. 2000

Dear Albert,

This letter confirms some discussions we had about underground drilling at Beaconsfield.

You advised that you have planned about 1600 m. in ten holes up to 200 m. long to be drilled from a cross-cut near the Grubb shaft at RL1370, possibly in 12-18 months' time. Seeing it is so far off, the following points are meant as guides to your planning, and not as promises.

DIRECTION CONTROL

I believe that we can maintain holes on the planned courses. You have planned them as far as possible to be close to the bearing which ground conditions favoured in previous drilling. They are not long. We can start on the shorter holes and check their directional behaviour and apply it to the longer holes. If necessary, we can drill large diameters and with direction resistant coring assemblies as we used on the surface hole.

MACHINE 1

I suggest that we drill with an electric F30 drill. This has a capacity of -

.... 300m. BQ
.... 200m. NQ

It weighs approx. 1½ tonnes and can be taken apart if necessary. Dimensions are about 3m x 1.3 m x 1.5 m.

TIMING

We would aim to work two drills each on two 10 hour shifts per day if you required. If we count on 8m. per drill per shift, the job would take around 48 days - say nine weeks if we did some Saturday work.

DRILL CHAMBER

The sketch attached gives a rough idea of our needs. Note that the geometry of a diamond drill requires the long axis of the drill to be at right angles to the hole. You will see that the main excavation needs a slot about 2 m. wide to give us head room and space to stack rods.

SERVICES

Electricity We would have drills with 25 h.p. electric motor and pumps with 10 h.p. motors

Drilling Water. Each drill pumps about 2000 litres per hour. We would need drainage from the drill chambers to get rid of this, plus any water inflows.

We usually write the following sentences into agreements for underground drilling. They may not all be applicable here:-

Safety.
The Mine will supply a qualified man to check the safety of drill chambers and access each day.

Ventilation.
The Mine will supply adequate ventilation to the drill chambers.

Lights
The Mine will supply normal underground stores, such as rock bolts, pipe fittings, etc.

Underground Water. Inflows can occur if we strike waterbearing rocks outside the dewatered zone of a Mine. They can affect us in three ways -

... 1. When we first strike a high pressure flow, it can throw the rod string out of the hole and wrap it round the drive with danger to men and equipment.

... 2. A more moderate flow can make drilling difficult as we handle rods and inner tubes against water flows.

... 3. High flows can flood the Mine.

We can take precautions against this including -

... 1. Cement in collarpipe with provision for valve to seal off flow.

... 2. Case through inflow zones.

... 3. Use high pressure pumps.

CHARGES

I would expect charges for drilling to be in the range $30-$40 p/m. at to-day's wage rates depending on hole size, water in-flows, time spent travelling underground, etc. There would be an establishment charge to cover any special conversions we would have to make-but this would not b e a major charge.

I hope that this gives you sufficient basis for planning. Please contact us if you need more information.

Yours sincerely,
ASSOCIATED DIAMOND DRILLERS PTY.LTD.

David

(M.D. HUGHES)
Drilling Manager

Encl.

9. COMPLETION OF MINE DEVELOPMENT; PLANNED MINE OPERATIONS.

9.1 Main Relevant Technical Advances in Mining Since Closure of Mine.

Improvements in mine technology and extraction procedures have occurred in practically every facet of operations as practiced by the Tasmania Gold Mining Company. The principal areas of operations where modern day practice will significantly improve on the results achieved by the previous operators are:

(a) Mine Dewatering.

(i) Electric Pumps.

Mine dewatering relied previously on the use of Cornish lift pumps, which were inflexible by design and location. Accordingly any inflow of water could only be removed when that water gravitated to the mine shafts; this resulted in delays in development, and the lack of flexibility contributed to occasional flooding of the mine shafts with consequent interference with mine production.

In lowering of water levels it is proposed to use electric submersible pumps of high volume capacity. (Pleuger or Ritz pumps, each with a capacity of 2 million gallons per day).

For routine dewatering, permanent electrical centrifugal pump installations will be used (in conjunction with dam storage) on pumping Levels other than the lowest Level where submersible pumps will be standard equipment operating in a sump below the production Levels.

Neither of these types of pumps, nor the portable pumps mentioned below, were available for previous operations.

(ii) Control of Water Surges.

In the development of new Levels, high pressure cement grout will be used to assist in control of the strong surges of water inflow which were the main water problem faced by the previous mining operation.

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Portable electric pumps may also be used to dispose of water surges which do occur, near the point of inflow.

As a safety measure monitoring of water pressure in ground strata will be undertaken by drilling test holes and installing pressure gauges. It is not known whether this technique was used prior to 1914.

(b) Grade Control.

(i) Improved Management.

The limitation of overbreak to the level forecast (i.e. 20%) will be primarily achieved by effective underground supervision, effective use of sampling data, and enforcement of strict directional control on holes drilled in the vicinity of either wall.

Rock broken to provide access only, will be stockpiled and used as fill.

(ii) Use of Rock Bolts.

Use will be made of rock bolts, with wire mesh where necessary, to give support to either wall showing weakness or a tendency to fret. Rock bolts will be used in both stoping and development work. This technique has been developed subsequent to the closing of the mine.

(iii) Use of Hydraulic Fill.

The development of hydraulic fill as a method for stabilising workings represents a major advance on the previous method of quarrying quartzite on surface, lowering broken stone to underground in trucks, and hand filling of extracted areas.

Hydraulic filling will be undertaken promptly after stopes are mined out, thereby giving immediate support to weak sections of walls, and minimising the possibility of wall rock diluting payable ore.

(c) Mechanical Equipment.

The Beaconsfield Gold Mine has been planned as a mechanised mine, geared to the structural limitations of the width of lode, and strike length of the orebody. It is proposed to use mechanical loaders in development headings; Cavo load-haul-dump units and/or mechanical scrapers in stoping; and air legs or drill jumbos in drilling.

None of these units was available in 1914, and the application of mechanical equipment is reflected in a planned mine labour force of 51 (excluding supervision and general engineering services), compared with an underground labour force of 191 at the time of the mine closure.

9.2 Main Steps in Development.

The following are the main steps involved in developing the mine.

(a) Retimber Harts Shaft.

The existing layout for Harts shaft comprises two haulage compartments (i.e. 2 cages and/or skips in balance) with a third compartment for services and as a downcast airway. The overall dimensions of the shaft are 17 ft. x 7 ft. 4 in., and nothing is known about the condition of the shaft timber.

It is understood that Oregon timber was used throughout, and it is probable the shaft timber is in good condition. If so, there is about 60,000 super feet of Oregon timber in the Cornish pump columns in Harts shaft, which could be redressed and made available for shaft repair work.

However a conservative estimate has been taken and new shaft timber has been allowed for full replacement within existing sets. This will provide two hoisting compartments each 5 ft. 6 in. x 3 ft. 6 in., plus two compartments for services and airway each of approximate dimensions 3 ft. x 5 ft. 6 in. It is proposed to use Oregon timber for wall plates, end plates and studdles, and local hardwood timber for liners and dividers between compartments. Shaft runners and guides would be in jarrah or kauri.

106 The estimated time required to retimber Harts shaft is 10 weeks to the 1370 ft. Level.

(b) Stripping Winze in Harts Shaft.

The winze in Harts shaft, from the 1370 ft. Level to the 1500 ft. Level, will need to be stripped. This work will require the prior installation of permanent headframe and winder on the surface. Mullock will be allowed to fall down to the 1500 ft. Level until such time as the winze connection becomes blocked up to stripping level. Broken mullock will then be hauled through to the surface by a Kibble.

(c) Surface Installations for the Deepening of Harts Shaft.

In order to achieve the early opening up of the 1650 ft. Level, sinking operations will initially deepen the shaft (to final shaft dimensions) to a depth of 1700 feet. Permanent headframe and winder will be required for this work.

It is planned to erect a permanent headframe, in steel construction, and to instal the permanent winder immediately following completion of confirmatory drilling. Under normal circumstances a substantial delay may be anticipated in fabrication of a new winder for the Beaconsfield operation. However there are several winders available within Australia capable of meeting the haulage requirements of Beaconsfield. A winder assessed as satisfactory for Beaconsfield is available at Broken Hill, and could be shipped promptly to Beaconsfield. The winder has two 8 ft. diameter drums, capable of single and double drum wind, with the required motor, switch gear, liquid starters, braking system and electrical control gear for winder operation. The winder has been inspected by Mr. A. Goninan who has reported on its condition and cost of restoration to suit Beaconsfield requirements.

It is estimated that the headframe could be erected and winder transferred and installed in a period of 12 weeks.

(d) Sinking of Harts Shaft.

The deepening of Harts shaft would probably be carried out by a shaft sinking contractor. Sinking has been estimated at a progress rate of 20 feet per week, including timbering.

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Dewatering will be maintained during the deepening of the shaft, and if any major inflows are experienced grouting operations will be undertaken.

(e) Development of 1650 ft. Level (and Western Sector of 1500 ft. Level.

When the shaft has been deepened some 50 feet below the 1650 ft. horizon an immediate start will be made on development work, driving east and west of Harts shaft to open up the 1650 ft. Level, and driving west of Harts shaft on the 1500ft. Level.

From historical records 1200 ft. of driving is conservatively expected to be involved in the full development of the 1650 ft. Level, and about 400 feet remains to be driven on the 1500 ft. Level.

This development work (including raises etc.) will provide a broken tonnage of about 9000 tons of ore, which would be stockpiled at surface for subsequent processing. (A further 7000 tons of broken ore will become available in the following year from development work on the 1800 ft. Level).

Valuable information will become available on ore grade on that Level, the mineralogy of the ore, and the percentage of gold occurring with pyrite. The latter data will give confirmation regarding size of the cyanide treatment section of the processing plant.

Grouting will be undertaken to assist control of water inflows during driving operations. Dewatering from the 1650 ft. Level would be by open channel drainage to the shaft sump, with a submersible pump installed in the sump to raise water to the permanent pump installation on the Level above. (As mentioned above, portable pumps may be used on the Level itself to cope with surges).

It is anticipated some 40 weeks would be required to complete the development driving on the 1650 ft. horizon after which the necessary raises will be made for ventilation and stoping requirements.

Completion of this work will enable mine production to commence at a rate of 60,000 tons per annum.

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(f) Development of 1800 ft. Level.

On completion of development of the 1650 ft. Level, Harts shaft will be extended to a depth of 2000 feet, and development of the 1800 ft. Level will proceed promptly.

It is anticipated that both of these operations will be complete at the end of the third year from commencement of work on mine dewatering i.e. commencement of second year of production.

Completion of this step will enable mine production to be increased to a rate of 100,000 tons per annum.

(g) Development Headings.

The size of development headings will be governed by the width of the orebody. However a minimum size for a development drive will be 8 ft. x 6 ft. A 6 ft. width is required to provide adequate gangway clearance for trucks (approximately 2 ft. 6 in. in width), and 8 ft. height to enable sufficient head room for the air, water and electrical services which will be extended along the drives.

(h) Ventilation.

Mention is made in earlier reports of "carbonic" gas generation from the old workings. This is presumed to be CO_2 gas formed by oxidation and acid reaction with carbonate gangues.

This need not be a problem and will be overcome by providing excess volume circulation and by using pressure ventilation to retard emission from old workings, as well as to provide adequate dilution volume to maintain CO_2 content at acceptable safety levels, and to maintain oxygen content. All old workings will be sealed off, at shaft connections.

A pressure ventilation system will be used with a downcast fan located adjacent to the Harts shaft collar. A compartment will be sealed off within the shaft to carry all downcast air, with exhaust through the operating compartments.

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9.3 Outline of Mine Operations.

The planned extraction system for Beaconsfield is the cut and fill operation utilising de-slimed residue for filling purposes. The strike length of 1200 ft. lends itself to three production stopes on each level, each of 400 ft. length. These would be serviced by three raises centrally located in each 400 ft. block, which would serve as a starting point for stoping extraction in both directions from the raises.

It is planned to take horizontal slices of 8 ft. whilst carrying a back height of approximately 13 feet.

Drilling would be undertaken with a self-propelled Upper Waggon, on which are mounted two rock drills equipped with ancillary centralisers and aluminium feed extensions. Holes will be drilled on an incline of 45° - 50° at 3 ft. spacing across the face and 3 ft. burden along the strike.

As approximately 100 tons of ore is required from each firing, a face advance of 13 ft. for each drilling sequence is necessary.

From each side of the central raise it is proposed to have one face being drilled while the alternate face, which has been previously fired, will be mucked out.

On this basis it is planned to achieve 100 tons of ore broken and loaded from each stoping block on a two shift basis.

It is practical to drill out a face sufficient to break 100 tons approximately in three hours with a two man crew. The same crew would then load 100 tons from the alternate face in the remaining five available hours of a shift. Hence the production potential per operating face is 100 tons per shift and therefore 200 tons per day.

For estimating purposes this productivity is factored by 50% so that the production achieved from each stoping area is 100 tons per two shift period.

As the initial milling requirement from mine production is 1200 tons per week (240 tons per day ex mine), two stoping locations on the 1650 ft. Level and one on the 1500 ft. Level should be capable of meeting these production figures on a two shift basis.

Subsequent mine production requirement of 400 tons per day will be met from four active stoping faces on two Levels.

The attached schematic diagram (Figure 9.1) shows the relationship of the three extraction areas along the strike length of the orebody, and the planned program is to have two stoping faces in production per Level, whilst the third is undergoing filling preparatory to the next phase of stoping.

On this basis continuous stoping and filling sequence is undertaken along each operating Level of the orebody with two stopes in production and the third stope either being prepared for filling or ready for the next stage of extraction.

It is proposed to use rubber-tyred shovel loaders of the Cavo 312 design as loadhaul dump units. The output capacity of these on a 200 ft. haul is approximately 50 tons per hour and hence only up to 2 hours of the available shift time will be required to load the planned 100 tons from the stope, even when haul distance is at the maximum.

Although the orebody averages 7 ft. in width from wall to wall, it is known that there have existed wide sections of the orebody (up to 24 ft.) which fully lend themselves to mechanised extraction using Upper Waggons and Cavo type loaders.

However it is also known that some sections of the orebody have narrowed to widths of a few feet.

To avoid excessive dilution of grade in these zones, a degree of flexibility will be necessary in the stoping technique. Conventional hand-held rock drills mounted on air legs may be necessary to drill out narrow sections of the orebody where width restricts the use of the Upper Waggon type assembly.

Similarly, alternate means of mechanical loading will be necessary in these narrow sections. Hence scraper winches will also be carried in the stope as supplementary loading equipment. These scraper winches would be resorted to only where the sections of the orebody are too narrow to use the larger Cavo equipment. Where the narrow zones occur a flexible approach would be adopted in deciding whether to scrape directly to the chute, or merely scrape back from the face to a point where the Davo units pick up the ore and transport back to the ore chute for dumping.

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If, as is likely from the known geological data, the orebody narrows and then widens again, the footwall could be stripped to permit the mechanised rubber tyre equipment to resume operations. Mullock broken in this way would be stockpiled in the stope as back filling, and would not be included as head feed to the mill.

If the payable orebody continued on a narrow width only, then hand operated rock drills, together with scrapers for loading, would be persisted with.

In summary, the overall concept is to use air operated rubber tyred load haul dump units wherever practicable as a means to minimising manpower in stopes, but with a degree of flexibility to have an alternate mechanical means for breaking and loading where stoping and orebody widths are limited. In both situations. mechanised methods of extraction are the key to the situation, together with grade control and minimising dilution.

9.4 Sources and Tonnage of Ore in First and Subsequent Production Years

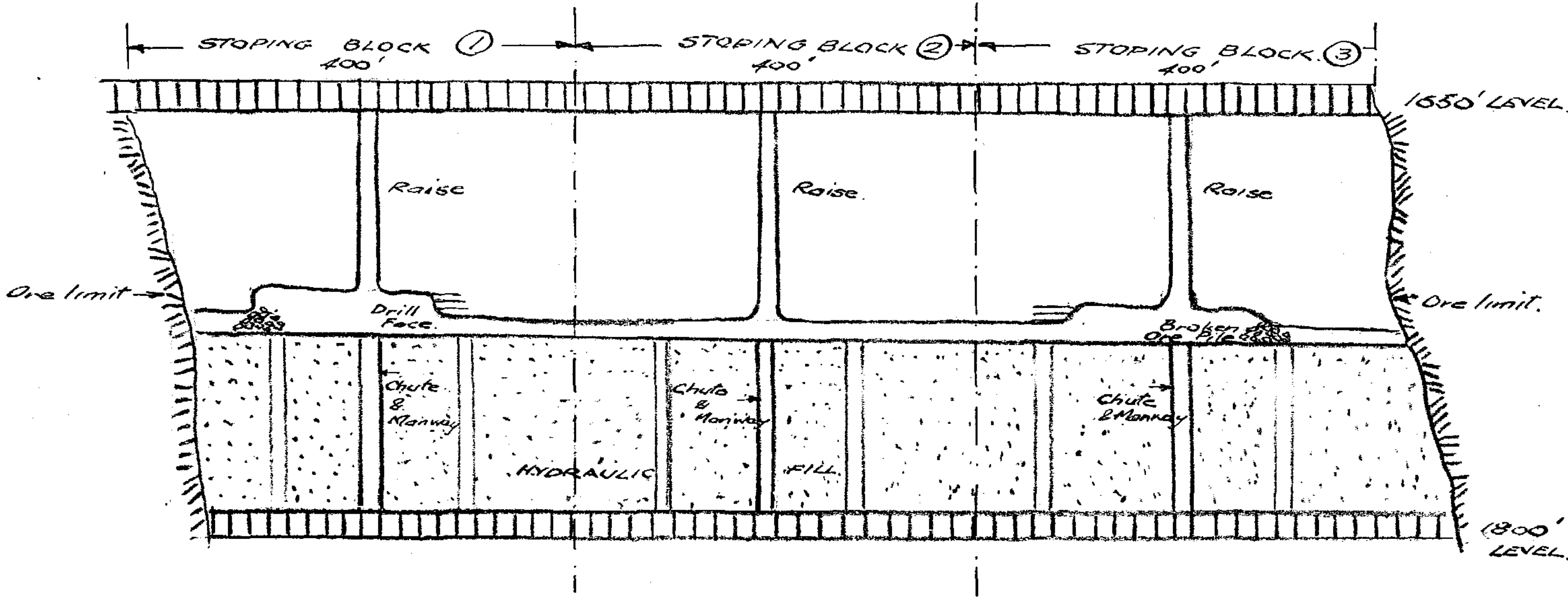
With some factors unpredictable at this stage, the potential output of 200 tons per day per stope (from a two shift operation) has been factored by 50%, to allow a daily production of 100 tons from a 2 man crew, per 2 shifts. This conservative level of productivity would permit 50,000 tons per year to be mined per Level of operation.

In the first year of operations it is planned to process 74,000 tons of ore. Sixty thousand tons will come from operating three stope faces on the 1500 ft. and 1650 ft. Levels, which will have been developed in the previous year. The balance of 14,000 tons will come from stockpiled ore from development work on the 1500 ft. and 1650 ft. Levels in the previous year, and from development of the 1800 ft. Level during the first year of operations.

At the end of the first year of production the 1800 ft. Level will be available for stoping, and the mine will thereafter work from two full productive Levels, each of which should give an output of 50,000 tons per annum.

On an extraction rate of say 1000 tons per week per Level, it would take approximately 8 weeks for a full 8 ft. vertical slice to be extracted along the strike length of the orebody. On this basis a new slice will be commenced approximately every 8 - 10 weeks and approximately 20 slices will be involved in extracting the ore between Levels. The life of a Level would thus be in excess of three years.

Past records of production (and expectations regarding the orebody extension) indicate approximately 1000 tons of ore per vertical foot of orebody. Thus development of the orebody extension down to the 1800 ft. Level will give approximately three years extraction reserves, during which period further Levels will be developed.



SCHEMATIC DIAGRAM SHOWING EXTRACTION AREAS

ALONG STRIKE OF OREBODY.

LEGEND

BLOCK 1.— IN PRODUCTION
BLOCK 2.— FILLED & PREPARED FOR PRODUCTION.
BLOCK 3.— IN PRODUCTION.

ALLSTATE EXPLORATION N.L.
BEACONSFIELD GOLD MINE.
FIGURE 9.1

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10. TIMETABLE.

Figure 10.1 is a bar chart presentation of the timing and sequence for the main steps involved in bringing the Tasmania Goldmine into production.

Its assumed starting point is the date when funds are available to undertake all work involved in re-opening the mine, and this assumes the completion of documentation with an incoming Joint Venture party.

It assumes confirmatory drilling being undertaken from the 1370 ft. Level.

10.1 Dewatering and Confirmatory Drilling.

The first year, approximately, will be taken up in dewatering the mine to the 1500 ft. Level, installing pumping stations at the two intermediate Levels of 635 ft. and 1000 ft., and undertaking confirmatory drilling from the 1370 ft. Level.

The main time controls on this stage of the work are the delivery period for the submersible pumps, and the time taken to draw down the water level, make temporary repairs to shaft timbers, and instal the pumping stations as the sites are exposed by the falling water level.

The program allows for a delay period of approximately 15 weeks due to minimum delivery period for submersible pumps of 26 weeks; if a shorter delivery period can be arranged or if an incoming party can provide submersible pumps from its own stores, part or all of this 15 week delay will be eliminated.

It is also possible (but not assumed in the timetable) that submersible pumps adequate to lower the water level in Harts shaft to below the 635 ft. pump chamber, can be hired on a short term basis in order to expedite the dewatering operation. Suitable pumps are possibly available for hire ex Cobar and perhaps elsewhere.

Pending the delivery of the submersible pumps work can proceed on the reconstruction of the collar of Harts shaft and the diversion and sealing of Blyth Creek.

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Given draw-down of water level in Harts shaft to approximately minus 660 ft. R.L., concrete works (water reservoir and pump beds) can be completed in the 635 ft. pump chamber.

Upon delivery of the first set of centrifugal pumps, installation at the 635 ft. Level will proceed followed by dewatering to below the 1000 ft. Level, installation of the pump station in the crosscut at that Level, and lowering of water level to below the 1370 ft. Level.

The basic approach to sources and timing for electric power is given in Section 11.2 below (items 15 and 16).

Other than the 500 Kva required for preliminary operations, H.E.C. power will not be available until one year after contracting with the H.E.C. We have assumed that such a contract would be entered into on completion of confirmatory drilling (end of first year) and that H.E.C. power in bulk will be available at the end of the second year.

Therefore power for pumping operations in the first two years will be provided by a diesel generating station of about 3000 Kva capacity which will be purchased second-hand. Such a station is currently available.

Upon exposure of the 1370 ft. Level, the crosscut to Grubb shaft will be cleaned out and drilling bases established for confirmatory drilling of both the eastern and western sections of the orebody between 1500 ft. and 2000 ft. R.L. While this drilling is proceeding the mine will be dewatered below the 1370 ft. Level to the 1500 ft. Level.

Six weeks have been allowed for dewatering the final 130 ft., which may include cleaning out of fallen material at the bottom of the Harts shaft winze and/or at the bottom of Grubb shaft. A further three weeks have been allowed for the sinking of a sump below the 1500 ft. Level of Harts shaft, if one does not already exist. These time allowances (and therefore the relevant costs included in capital cost estimates) may be excessive.

The program assumes that sufficient confirmation of ore reserves will be available from drilling carried out in the first six weeks, to enable work to proceed and contracts to be entered into which have been delayed pending this confirmation.

10.2 Mine Development.

Having established dewatering to the full depth of existing workings, the next step is to strip the winze at the bottom of Harts shaft to full shaft dimensions, which will provide passage for the lowering of pump units for installation at the 1500 ft. Level pumping station. Stripping progress is allowed at 30 ft. per week.

Concurrent with this pump installation will be the obtaining from the 1500 ft. Level of the balance of the bulk ore sample required for metallurgical testing, the first part of which will have been obtained earlier from the western end of the 1370 ft. Level.

Retimbering (as necessary) of Harts shaft to 1370 ft. will proceed concurrent with installation of the permanent headframe, winder and compressor, using the temporary winder for this purpose. The period allowed in the program assumes retimbering at a rate of 140 feet per week.

On completion of the installation of the permanent headframe and winder for Harts shaft, stripping of the winze and cutting the sump at 1500 ft. Level will proceed. Timbering of the winze section will be concurrent with stripping.

On completion of these operations work will commence on sinking Harts shaft below the 1500 ft. Level to approximately minus 1700 ft. R.L. (This depth will provide a shaft sump in which the submersible pumps will operate while the 1650 ft. Level is driven). Shaft sinking is allowed at 20 feet per week.

very wet strata will be cut

On completion of this stage of shaft sinking, the 1650 ft. Level will be driven, together with three raises and the ore pass and loading station. Progress is allowed, using two crews, at 80 feet per week for driving, and five weeks each raise. Concurrently the western sector of the 1500 ft. Level, with necessary raises, will be driven.

This will provide 150,000 tons of developed ore from the 1650 ft. Level in addition to ore which remains at the western end of the 1500 ft. Level. The development work will also provide some 9000 tons of production ore, which will be stockpiled ready for completion of the mill.

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expect ver with starts

The sinking of Harts shaft to the 2000 ft. Level will be undertaken concurrent with stoping extraction from the 1650 ft. Level and from the 1500 ft. Level. This shaft sinking will not interfere with production operations, with mullock haulage on night shift and in weekends as necessary.

On completion of shaft sinking, the 1800 ft. Level will be developed similarly to the 1650 ft. Level, enabling production rate to be increased to 100,000 tons per annum at the end of the first year of production.

Delivery quotations for mine plant and equipment are such that there will be no difficulty in acquisition prior to commencement of mine production.

10.3 Mill Construction.

Time estimates provided by Sala Australia Pty. Ltd. indicate that a plant capable of treating 100,000 tons per year (i.e. 300 tons per day throughput) could be constructed and commissioned within 9 months. This period, or a longer period, will fit comfortably into the program which requires commencement of ore processing about 108 weeks after project finance is firmly available.

The program is also such that ample time is available for items such as purchase of mill site.

10.4 Alternative Programs.

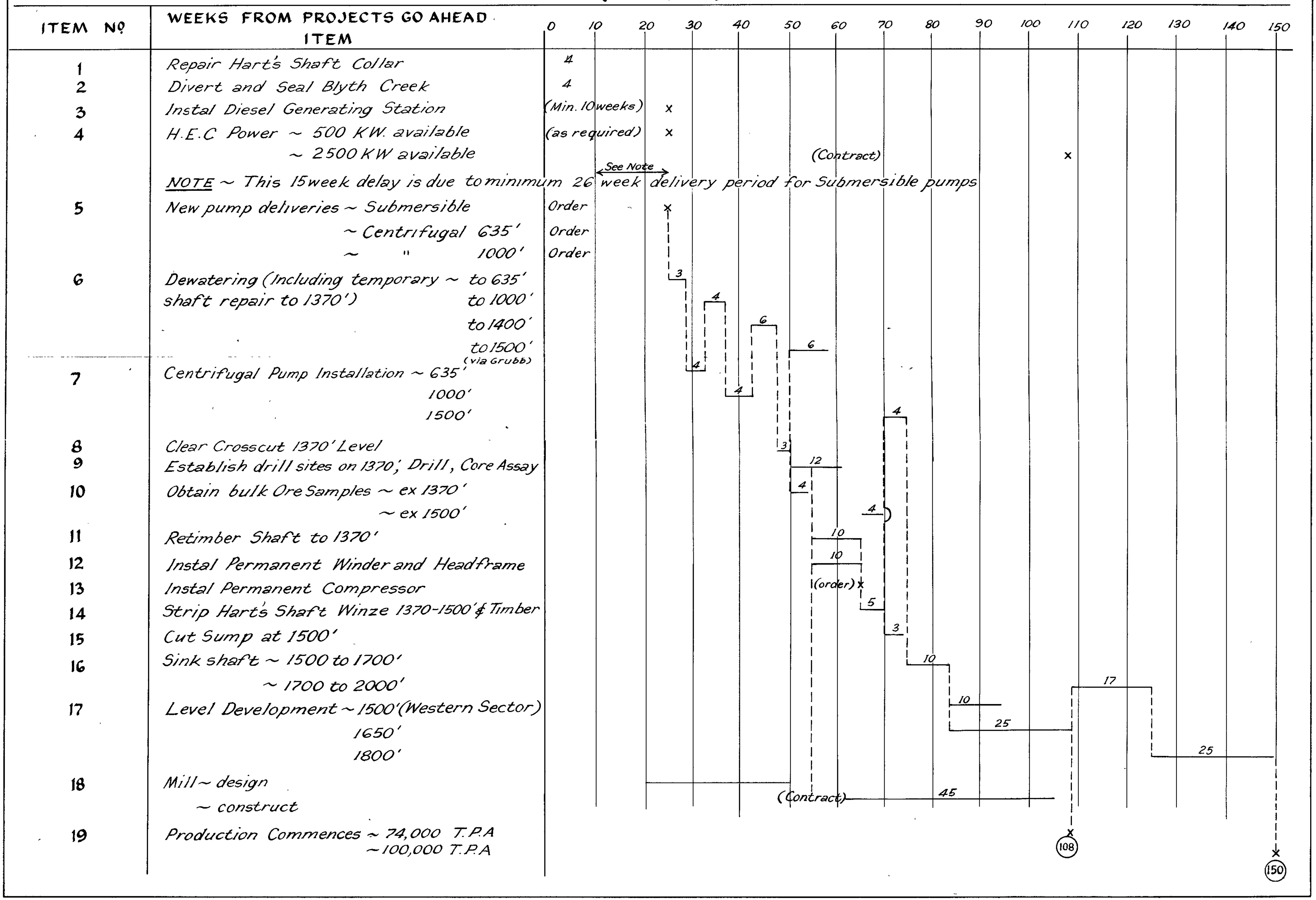
If it were decided to minimise expenditure prior to confirmatory drilling, at the expense of project progress and therefore at some increase in total capital cost, an alternative program is possible based on undertaking confirmatory drilling from the 1000 ft. crosscut between Harts and Grubb shafts.

This alternative would involve a delay of about 37 weeks, due mainly to delay in placing orders for centrifugal pumps and some extension of the period required for undertaking confirmatory drilling from a higher Level.

BEACONSFIELD GOLDMINE – PROJECT PROGRAM – FEASIBILITY STUDY

(BASE CASE) JULY 1975

FIGURE 10.1



Item 2. (Blyth Creek).

The method assumed for sealing of Blyth Creek adopts advice from the W.C. & I.C., and is based on sealing the surface area of the excavated channel with clay mixed with Bentonite to a depth of one foot.

The diversion will follow the land contour, and will bypass the limestone pits.

Item 3. (Diesel Generating Station).

E.A. Marr & Sons (Sales) Ltd. have presented several alternative propositions for the purchase of second-hand diesel generating stations of about 3000 kw capacity, with purchase prices (including any necessary renovation, and delivery to site) ranging from $100,000 to $150,000.

A full power station of seven matched Ruston sets, including switchgear and spares, is most suitable to our requirements and is currently available. The estimated cost includes foundations and building on site.

Items 4, 5, 6 and 7. (Temporary Equipment).

These items cover the hire or purchase of temporary equipment suitable for project requirements up to completion of confirmatory drilling, when permanent winders, headframe, and mine operating equipment etc. will be purchased.

This practice has been assumed in order to minimise capital expenditure prior to completion of confirmatory drilling.

Item 9. (Basic Labour Force).

The basic labour force assumed in this Item covers all significant labour requirements in the first year (other than personnel relating to the diesel generating station, whose costs are included under the heading of "Power Costs"). In the second year allowance under this heading is made for necessary winder drivers and bracemen, also for two miners to cover miscellaneous work in the first 20 weeks of Year Two.

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All other labour requirements have been estimated on a unit-cost basis (e.g. per foot for shaft sinking and level development etc.), for activities which may be undertaken by project personnel or under contract.

Items 11, 12, 13 and 14. (Pumps, Cable, Transformers, Switch Gear).

Costs for inclusion in these Items have been provided by Pacific Pumps in relation to Pleuger submersible pumps, and by Pullen Fluid Dynamics Pty. Ltd., Kelly & Lewis Pumps, and Mr. P. Arnold (Engineering Consultant), in relation to the remaining items.

The final choice of pump capacity and pump supplier could differ from those allowed, as could the power transforming and switch gear arrangements. However the allowed cost would be sufficient to cover any likely alternative to the proposed arrangements.

Items 15 and 16. (Power Cost - Diesel and H.E.C.).

A basic assumption has been made that a contract with the Hydro Electric Commission of Tasmania for the supply of power in excess of 500 kw (which is readily available from the existing supply arrangements at Beaconsfield), will be entered into upon conclusion of confirmatory drilling at the start of the second year, and that this contract will be for a power supply of 2500 kw.

These arrangements with the H.E.C. assume the use of a diesel generating station to provide the bulk of required power in the first two years, and that such a station will be available in subsequent years to provide electric power over and above normal requirements (e.g. to feed the pump installation when it is operating between normal and maximum capacity, in the event of abnormal flows of water into the mine).

This situation is assumed for three basic reasons, as follows:

(i) Execution of a contract with the H.E.C. involves a guarantee to cover capital expenditure by the H.E.C. of an amount which has been indicated in the order of $250,000. It is thought that this amount, which was relevant to an earlier enquiry involving 4500 kw, could be reduced by negotiation with the H.E.C. for a supply of 2500 kw.

123 For prudence, it may be desirable to avoid this commitment until completion of confirmatory drilling.

(ii) The H.E.C. contracts require a minimum demand of about 80% of installed capacity, which makes it uneconomic to use the H.E.C. power supply to cope with occasionally required power surges to deal with occasional abnormal pumping requirements.

(iii) The H.E.C. advise that power installation at Beaconsfield will not be available before one year from signing of contract. To avoid a one year delay in commencement of project work, use of a diesel generating station is required. Enquiries indicate that such stations are not available on a short term lease basis on satisfactory financial terms, therefore purchase of a suitable second-hand station has been assumed.

Item 19. (Confirmatory Drilling).

The letter from Associated Diamond Drillers Pty. Ltd., included as Figure 8.1, reflects the outcome of discussions with Mr. David Hughes, Drilling Manager of this company. It relates to the basic case of drilling from the crosscut at R.L. minus 1370 ft., and is based on extensive experience by A.D.D. in drilling at the Beaconsfield site for Allstate over the past several years.

Mr. Hughes has confirmed that his opinion on matters such as direction control would also relate to drilling from the 1000 ft. crosscut in the event that the alternative program (refer Section 10.4) is adopted.

The capital cost schedule adopts the highest level of the $30-40 per metre range indicated by A.D.D., to which a 10% contingency has been added.

Item 20. (Assay and Metallurgical Test).

To date all processing and testing of drill core material has been undertaken by the laboratory of the Tasmanian Department of Mines at nominal cost to Allstate.

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However an allowance of $5,000 has been included in the capital cost estimate to cover the fees of an outside laboratory, should timing considerations make it desirable to supplement Tasmanian Mines Department facilities with outside facilities.

Item 21. (Retimber Shaft).

It is general experience that timbers exposed by dewatering of flooded mine shafts are in good condition.

The assumption has been made in this instance that the timber wall plates and posts of the shaft will be suitable for retention, but that shaft dividers and the lining of compartments (as required) will be done with new timber to the 1370 ft. Level. This will allow flexibility in design and hoisting arrangements, airways, pump discharge pipes etc. New tallow runners and guides have also been allowed.

Full timbering has been allowed from the 1370 ft. Level downwards.

Item 22. (Winder).

Two winders known to be in storage at Broken Hill could be suitable for use at Beaconsfield. One of these has been inspected by Mr. A. Goninan who has reported fully on the suitability and condition of the winder, together with cost of reconditioning and transport to Beaconsfield.

He has advised a purchase price of $10,000, which has been increased in the cost estimate to $25,000. His cost estimate for the winder building has been replaced by one based on indicative costs provided by Civil & Civic Pty. Ltd. A fifteen percent contingency has been allowed on top of total estimated costs.

A more suitable winder or a more economic purchase could be possible, in view of the 12 months (from commencement of site works) which is assumed prior to commitment to acquisition of a winder.

Items 31 and 36. (Mine Plant).

New plant for use in the mine, both during development and production, has been allowed at a level of mechanisation which is in line with the size of the labour force allowed in operating costs, and which will meet production schedules for development and stoping.

Item 35. (Development of 1500 ft. and 1650 ft. Levels).

Cost estimates per foot included in this Item represent the higher of recent actual cost figures (excluding timbering) for comparable situations, which have been made available to us in relation to West Australian operations at the Scotia Mine and by Central Norseman N.L. It is thought that they will cover any eventuality at Beaconsfield.

Timbering of development headings will proceed after silling-out on the Level has been completed, and this cost is shown as an operating cost.

Item 37. (Mill).

A budget estimate has been provided by Sala Australia Pty. Ltd. for the ore processing mill, with certain exclusions which are allowed for separately in the capital cost summary. Sala's estimate is based on a process flow sheet provided by Mr. Keith Parsons, and both Mr. Parsons and Sala recognise that improvements in the process flow will be made in practice. Such improvements are likely to result in a lower capital cost.

The budget estimate was a firm offer by Sala capable of acceptance by Allstate, and Sala are experienced in this type of work. It should therefore represent an adequate cost estimate for the items covered.

Austin Anderson Pty. Ltd. have had recent experience with another gold mine in Australia, and their estimates on specialist items such as assay office and gold room have been accepted.

Item 41. (Develop 1800 ft. Level).

Ore production between Levels separated by 150 ft., including dilution of 20%, represents approximately 180,000 tons. An allowance of $1.00 per ton represents an expenditure per Level of $180,000, which compares with $175,000 calculated in Item 35(b) for development of the 1650 ft. Level.

This expenditure, of the full $180,000 in the first year of production and subsequently at the rate of $1.00 per ton, is included as an item in operating costs, and is therefore excluded from the capital cost summary.

Item 42. (Working Capital).

Working capital for initial mining operations must be provided from the capital funding of the project. Subsequent increases in working capital (e.g. resulting from the increased level of production in year two) will be available from profit margin and retained cash flow from project operations.

Calculation of working capital takes account of the prompt payment to gold bullion producers of about $A32 per ounce, with an average delay of 2½ months before payment is made in excess of this amount. It also takes account of the direct costs of production in this 2½ month period. The calculation is shown as Item 42 in the capital cost summary (Table 11.1). An amount of $25,000 in excess of this calculation has been allowed as working capital for the project.

Lump Sum Contingency.

An amount of $235,000 has been included in capital costs as a lump sum contingency, additional to contingencies allowed on specific items of capital cost. This has been introduced as an element of overall conservatism in capital cost estimation.

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11.3 Allowance for Inflation.

Cost inflation to the date of this Report is included in basic cost estimates as shown on the first two pages of Table 11.1.

A basic assumption is made that cost inflation in Australia will be of the order of 15% per annum over the coming three years.

A significant part of capital costs is of overseas origin, where inflation is expected to be lower than in Australia.

However inflation has been included in total capital cost estimates at the full rate of 15% per annum, and a six month period has been allowed for negotiation, prior to commencing site work. Timing of expenditure (i.e. average for each quarterly period) has been taken into account in calculating the cost inflation allowance.

Of the total of $7.0 million capital cost estimate, $1.57 million represents allowance for cost inflation, and $0.23 million is a lump sum contingency allowed in addition to contingencies on individual cost items.

11.4 Capital Cost of Alternative Programs.

The alternative program mentioned in Section 10.4 of this Report has not been included as a schedule to this Report but is available for discussion. This comment also applies to the build-up of capital costs for the alternative program.

Broadly however, the alternative program (involving a delay of the order of 37 weeks) involves increased expense mainly relating to the effect of inflation on delayed purchase orders, the extra cost of submersible pumps to handle an 1100 ft. lift, the extra cost of drilling from a higher Level, and the overhead component (including cost of keeping the mine dewatered) of the delay period. The increase in total capital cost would be of the order of $870,000, to achieve a reduction in cost/commitment on completion of confirmatory drilling of about $400,000. i.e. down from $1.16 million to about $750,000.

Table 11.1

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CAPITAL COST SUMMARY (BASE CASE).

($'000)

Stage No.	Stage	Items Included	Estimated Cost	Inflation Allowance (see P.2)	Forecast Total Cost
1	Dewater and Confirmatory Drilling	1 - 7, 8(a), 9(a), 10, 11(a) (b)(c), 12(a)(b) (c), 14(a)(b)(c) (e), 15(a), 16(a), 17(a)(b), 18, 19, 20, 33(a), 34(a), 43(a)	$1,017	145	1,162
2	Complete Mine Development, including Plant for Development	8(b), 9(b), 11(d), 12(d), 13(a)(b), 14(d), 15(b), 16(b), 17(c), 21-32, 33(b), 34(b), 35, 43(b)	$1,509	522	2,031
3	Mine Plant and Equipment (Balance)	36	$ 230	85	315
4	Surface Plant and Buildings	38, 43(d)	$ 163	60	223
5	Mill	37, 43(c)	$1,920	591	2,496
6	Working Capital	42	$ 150	70	220
7	Capital costs after commencement of Production in Year 3, not funded out of Cash Flow	39, 40	$ 206	97	303
8	Lump Sum Contingency (additional to contingency allowances on specific cost items)		$ 235		235
	TOTAL CAPITAL COST		$5,430	1,570	7,000

BEACONSFIELD FEASIBILITY REPORT - JULY, 1975.

INFLATION ALLOWANCE ON CAPITAL COST ESTIMATE.

	6 Months to end 1975	1976				1977				1978	
		1	2	3	4	1	2	3	4	1 & 2	3 & 4
Total Capital Expenditure ($'000)	Nil	545	180	145	135	670	700	1,220	1,245	265	90
Inflate by %	7½	11	15	19	22	26	30	34	37	45	53
Inflation ($'000)	Nil	60	27	27	30	174	210	420	455	119	48
Inflated Capital Expense:											
- by Quarters	-	605	207	172	165	844	910	1,640	1,700	384	138
- by Years	-	1,149				5,094				522	
- Total	-					$6,765,000					
Plus Lump Sum Contingency						235,000					
Revised Total						$7,000,000					

ALLOCATION OF INFLATION ALLOWANCE AND PUMP SUM CONTINGENCY BY EXPENDITURE STAGES

Stage	Description	Allocation ($'000
Stage 1	Dewater and Confirmatory Drill	145
2	Complete Mine Development	522
3	Balance of Mine Plant	85
4	Surface Plant	60
5	Mill	591
6	Working Capital	70
7	Capital costs after commencement of Production	97
		$1,570

BEACONSFIELD GOLDMINE - CAPITAL COST - FEASIBILITY REPORT - JULY, 1975.

Item No.	Item	Units	Unit Cost	Cost $	Contingency %	Contingency $	Total Cost $	Timing Notes	Data Source
1	Repair Shaft Collar				Included in Estimate		50,000	4 weeks	Estimate by Pearson Bridge
2	Divert and seal Blyth Creek			12,500	50	6,500	19,000	4-5 weeks	- Method as per W.C. & I.C. - A.S. estimate
3	Purchase Diesel Generating Station	Second hand station 3000 KW capacity Includes Switch-board			Included in Estimate		168,000	10 weeks from order	E.A. Marr - highest of several alternatives.
4	Purchase Two Temporary Winders incl. ropes		$14,000 Plus Transport etc.	16,000	Included in Estimate		16,000	Pending installation permanent Winder	E.A. Marr
5	Hire 1 x 3000 C.F.M. Compressor	40 weeks	$175 per week	7,000	15	1,000	8,000	25-65 Weeks	E.A. Marr
6	Purchase 1 Fan and Air ducting	5,000 C.F.M., 24 Axial Fan		2,000	15	300	2,300		Aerex Fans
7	Purchase 2 Kibbles and 1 Sinking Stage			2,000	15	300	2,300		E.A. Marr
8	<u>Air & Water Pipes down Shaft:</u>								
8(a)	To 1370' Level	1370 ft.	Material Cost $22 per foot (Labour in item 9.)	30,000	10	3,000	33,000		A.S. using quotes (for materials) from Tubemakers Aust.
8(b)	1370' to 1650' Level	280 ft.		7,000	10	700	7,700		

BEACONSFIELD GOLDMINE - CAPITAL COST - FEASIBILITY REPORT - JULY, 1975.

Item No.	Item	Units	Unit Cost	Cost $	Contingency %	Contingency $	Total Cost $	Timing Notes	Data Source
9	Basic Labour Force:								
9(a)	Year 1 - 2 Winder) 4 Miner) 2 General)	30 weeks	$200/wk/man incl. O/H for Winder & Miner $150 for General & Brace men	45,000	-	-	45,000	25-55 weeks	A. Silver
9(b)	Year 2 - 3 Winder) 2 Brace-) men)	50 weeks	As above	45,000	-	-	45,000	2nd Year	
	Year 2 - 2 Miners	20 weeks	As above	8,000	-	-	8,000	50-70 weeks	
10	Small tools - initial supply						5,000		A. Silver
11	Pump Purchase (all 3,300 K.V. motors):								
11(a)	- Submersibles	3x2 m.g.p.d. 450 H.P. Pleuger	$19,400	58,200	5	3,000	61,200	26 weeks delivery	Pan Pacific Pumps
11(b)	- Centrifugal 635'	3x440 H.P.	$39,000	117,000	-	-	117,000	do.)	Pullen Fluid Dyamics Pty. Ltd.
11(c)	- Centrifugal 1000'	2x500 H.P.	$38,500	77,000	-	-	77,000	do.)	
11(d)	- Centrifugal 1500'	3x360 H.P.	$31,000	93,000	-	-	93,000	do.)	

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Item No.	Item	Units	Unit Cost	Cost $	Contingency %	Contingency $	Total Cost $	Timing Notes	Data Source
12	Cable Purchase and Instal:								
12(a)	- Submersibles	Includes $20,000 for instal-lation		17,000	10	1,700	18,700	As for Pumps	P. Arnold - Consultant (data from Laurenson & Hansen)
12(b)	- Centrifugal 635'			25,000	10	2,500	27,500		
12(c)	- Centrifugal 1000'			22,000	10	2,200	24,200		
12(d)	- Centrifugal 1500'			34,000	10	3,400	37,400		
13	Tranformers, and Sub Station Civil Works:								
13(a)	- 22,000 to 3,300 at Mine	Transformers 38, Instal 12		50,000	Included		50,000		P. Arnold, Consultant
13(b)	- 22,000 to 415 at Mill	Transformers 19, Instal 6		25,000	in		25,000		do.
13(c)	- 3,300 to 415 at Mine	(Use initial trans-formers for diesel station as item 14)		-	Estimates		-		do.
14	Pump Switch Gear:								
14(a)	- Submersibles	(Includes initial transformers for Diesel Installation)		24,000	Included		24,000	As for Pumps	P. Arnold, Consultant
14(b)	- Centrifugal 635'			24,000	in		24,000		do.
14(c)	- Centrifugal 1000'			15,000	Estimates		15,000		do.
14(d)	- Centrifugal 1500'			24,000			24,000		do.
14(e)	- 4x1000 KVA Trans-formers (415 - 3,300)			36,000			36,000		do.

BEACONSFIELD GOLDMINE - CAPITAL COST - FEASIBILITY REPORT - JULY, 1975.

Item No.	Item	Units	Unit Cost	Cost $	Contingency %	Contingency $	Total Cost $	Timing Notes	Data Source
15	Power Cost - Diesel:								
15(a)	- Year 1		$44,000 p.a.				80,000		Fuel - B.P. Tasmania
15(b)	- Year 2		+ 1.6 cents/ KWH				219,000		A.S. and P. Arnold for unit cost and usage
16	Power Cost - H.E.C.:								
16(a)	500 Kw - Year 1		$24,000 p.a.				34,000		H.E.C. Tasmania for power rates
16(b)	- Year 2		+ 1.01 cents/ KWH				68,000		A.S. for usage.
17	Pump Installation:							Weeks	
17(a)	- On 635' Level)	Water Sump and Foundations		6,000	20	1,200	6,200	29-32	A. Silver
17(b)	- On 1000' Level)			6,000	20	1,200	6,200	37-40	
17(c)	- On 1500' Level	Foundations (sump, see item 29)		4,000	20	800	4,800	70-73	
	(Work done by Basic Labour Force)								
18	Clear Crosscut 1370' (Harts - Grubb)	300 feet	100 ft/week	(Work done by Basic Labour Force)			Nil	Weeks 48-50	
19	Drill from 1370' Crosscut (Cut drill chambers with basic labour force)	1600 meters	$40/meter	64,000	10	6,400	70,000	Weeks 51-62	Associated Diamond Drillers
20	Process and Test drill core and bulk samples (Extract with basic labour force)						5,000		A. Silver

BEACONSFIELD GOLDMINE - CAPITAL COST - FEASIBILITY REPORT - JULY, 1975.

Item No.	Item	Units	Unit Cost	Cost $	Contingency %	Contingency $	Total Cost $	Timing Notes	Data Source
21	Retimber Shaft:								
21(a)	To 1370' Level - new dividers and Liners	Materials labour	$24/foot $10/foot	47,000	50	23,500	70,500	Weeks 55-64	A. Silver (Timber cost from Timber Merchant)
21(b)	1370-1500' Level-full timber	Materials Labour	$110/foot $20/foot	17,000	10	1,700	18,700	65-70	
22	Purchase, renovate and Instal Winder	Purchase Renovate and Instal (incl. design)	-$25,000 102,000	127,000	10	12,700	139,700	Weeks 55-64	A. Goninan
23	Fabricate and erect Headframe	Foundations Fab. & erect	3,000 31,000	34,000	15	5,000	39,000	Weeks 55-64	Cost per ton of structural steel ex Civil and Civic
24	Cages	2	$5,000	10,000	10	1,000	11,000		Watts Griffis McOuat
25	Winder Building	50'x40' = 20 Squares	$1000/ Square	20,000	10	2,000	22,000	Weeks 55-64	Cost per Square ex Civil and Civic
26	Surface Ore Bin	1		10,000	10	1,000	11,000		A. Silver
27	Compressor 1200 C.F.M.	1	Comp. $50,000 Building $5,000	55,000	10 on Bldg.	500	55,500		Atlas Copco
28	Strip Winze, 1370' to 1500'	130 feet	$100/foot	13,000	Included in Estimate		13,000	Weeks 65-70	A. Silver

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Table 11.

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BEACONSFIELD GOLDMINE - CAPITAL COST - FEASIBILITY REPORT - JULY, 1975.

Item No.	Item	Units	Unit Cost	Cost $	Contingency %	Contingency $	Total Cost $	Timing Notes	Data Source
29	Cut Sump on 1500'	20,000 C.Ft.	$1/C.Ft	20,000	Included in Estimate		20,000	Weeks 71-83	A. Silver
30	Sink Shaft 1500' - 1700'	200 ft.	$3.50/ft. incl. timber	70,000	10	7,000	77,000	Weeks 74-83	Allied Constructions
31	Plant for Mine Development:								
	- Rocker Shovels	3	$14,000	42,000	-	-	42,000		Simco Machinery
	- Drill Units	4	$ 2,000	8,000	-	-	8,000		Atlas Copco
	- Level Loco & Trucks	2 sets	$16,000	32,000	10	3,200	35,200		Watts Griffis/Gemco
	- Rails	3000 ft.	$2.33	7,000	10	700	7,700		A. Silver
	- Air & Water Lines on 2 Levels	6000 ft.	$3.00/ft.	18,000	-	-	18,000		A. Silver
32	Main Fan Installation 30,000 C.F.M.		$3,200 + installation	5,000	10	500	5,500		Fox Manufacturing
33	Site Supervision:								
33(a)	- Year 1	Mgr. + Foreman	20+12	32,000	-	-	32,000		A. Silver
33(b)	- Year 2	Mgr. + 3 Foremen	20+36	56,000	-	-	56,000		
34	Project Management:								
34(a)	- Year 1			NIL			NIL		As per current agreement with Tricentrol
34(b)	- Year 2			50,000			50,000		

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Item No.	Item	Units	Unit Cost	Cost $	Contingency %	Contingency $	Total Cost $	Timing Notes	Data Source
35	Level Development:								
35(a)	- 1500' Level (West)-		$/ft.						
	- Drive	400 ft.	90	36,000	Included			Weeks	The higher of
	- Raise	150 ft.	50	7,500					
				43,500	in		43,500	84-93	Central Norseman
35(b)	- 1650' Level -								and
	- Crosscut	150 ft.	90	13,500	Price				
	- Drive	1200 ft.	90	108,000					
	- 3 Raises	450 ft.	50	22,500	per				Scotia cost data
	- Vent Raise	150 ft.	50	7,500					
	- Ore Pass	50 ft.	70	3,500				Weeks	
	- Sump	As item 29		20,000	Foot			84-98	
				175,000			175,000		
36	Mine Production Plant: (for 1500' and 1650' Levels)								
	Cavo Stope Loader	3	22,000	66,000	-	-	66,000		Atlas Copco
	Scraper Winches	2	8,000	16,000	-	-	16,000		Joy Manufacturing
	Upper Wagon Drill Rigs	2	13,000	26,000	-	-	26,000		Atlas Copco
	Stope Drill Rigs	4	2,000	8,000	-	-	8,000		Atlas Copco
	Level Loco. and Trucks	1 Set	16,000	16,000	10	1,600	17,600		Watts Griffis/Gemco
	Skips & Tipplers	2	10,000	20,000	10	2,000	22,000		Watts Griffis
	1650' Loading Station			50,000	10	5,000	55,000		A.Silver/C.G.F.A.
	Miscellaneous:								
	- Ducting		5,000						
	- Aux. pumps		3,000						
	- Fill System		5,000						
	- Balance Small Tools		5,000	18,000	10	1,800	19,800		A. Silver

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Item No.	Item	Units	Unit Cost	Cost $	Contingency %	Contingency $	Total Cost $	Timing Notes	Data Source
37	Mill:								
37(a)	Ore processing plant of 100,000 T.P.A. capacity including crushing, grinding, flotation, cyanidation and gold recovery processes						1,715,000		Budget price, Sala Australia, 2/6/75
37(b)	Mill Switchboard				Included in estimate		35,000		P. Arnold
37(c)	Land Acquisition	Say 5 acres adj. limestone quarries	$200/acre	1,000	-	-	1,000		A. Silver
37(d)	Site clearing and foundation excavation	11,000 C.Yd.	$3/C.Yd.	35,000	10	3,500	38,500		A.S./Civil & Civic
37(e)	Concrete works			30,000	10	3,000	33,000		A.S./Civil & Civic
37(f)	Assay Office & Equpt.	600sq.ft.		30,000	10	3,000	33,000		A.S./Austin Anderson
37(g)	Gold room security	Fence & Vault		10,000	10	1,000	11,000		Austin Anderson
37(h)	Security fencing, paving, internal roads			8,000	10	800	8,800		Cyclone/Paving contractors
37(j)	Tailings Dam and Environmental aspects			25,000	20	5,000	30,000		A. Silver

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Item No.	Item	Units	Unit Cost	Cost $	Contingency %	Contingency $	Total Cost $	Timing Notes	Data Source
38	Surface Plant etc.:								
38(a)	Admin. Building	Existing building ex Beaconsfield Council		15,000	20	3,000	18,000		Discussion - Beaconsfield Council
38(b)	Change Room	For approx. 85 people		30,000	10	3,000	33,000		A.Silver/Renison
38(c)	Workshop, misc. tools			20,000	10	2,000	22,000		A. Silver
38(d)	Explosives magazine			5,000	10	500	5,500		A.Silver/Renison
38(e)	Batteries & Charging station			2,000	10	200	2,200		A. Silver
38(f)	Ambulance Facilities	Existing, adj. mine		Nil	-	-	Nil		
38(g)	Vehicles	3	$5,000	15,000	-	-	15,000		A. Silver
38(h)	Staff Houses	3		60,000	-	-	60,000		A. Silver
38(j)	Mine Fencing			3,000	10	300	3,300		Cyclone
39	Extra Mine Production Equipment: (for 2nd full operating Level)								
	Cavo Stope Loaders	2	22,000	44,000	-	-	44,000	Start of Year 4	Atlas Copco
	Scraper Winches	2	8,000	16,000	-	-	16,000		Atlas Copco
	Upper Wagon Drill Rigs	2	13,000	26,000	-	-	26,000		Atlas Copco
	Stope Drill Rigs	2	2,000	4,000	-	-	4,000		Atlas Copco
40	Sink Harts Shaft 1700' to 2000'	300 ft.	$350/ft.	105,000	10	10,500	115,500	Weeks 108-125	Allied Constructions

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Item No.	Item	Units	Unit Cost	Cost $	Contingency %	Contingency $	Total Cost $	Timing Notes	Data Source
41	Develop 1800' and subsequent Levels	Covered by allowance in Operating Costs		-	-	-	-	During Mine Operation	As item 35(b) spread over 18 months output.
42	Working Capital for initial Mine operations	- Based on 2½ month delay in payment for bullion beyond $32 per ounce - Monthly output in Year 1 = 5,000 Tons .˙. 2½ months output = 12,500 tons - Direct cost of production = $26/ton in Year 3 less initial bullion payment 16/ton = Un-recouped cost $10/ton x 12,500 Tons = $125,000					Say 150,000	Required initially pending establishment of Cash Flow	
43	Design Work (where not included in Cost:								
43(a)	Electrics & Pumps						10,000		A. Silver
43(b)	Headframe, Cages, Winder building, Ore bin, Compressor building	$86,000	10%				8,600		Assume 10% fee.
43(c)	Mill Construction works excluding Sala quoted items	$154,000	10%				15,400		Assume 10% fee.
43(d)	Surface Plant - items 38(b) and (c)	$35,000	10%				3,500		Assume 10% fee

12. OPERATING COSTS.

12.1 Labour Costs.

Labour costs for an output of 100,000 tons per annum are based upon the number of men for each operation as shown in a separate column in Table 12.3. The number of men allowed in the first year of operations, when mine output and mill throughput are less than 100,000 tons, is generally shown in the relevant column of Table 12.3.

The cost per annum for all employees (as indicated in Table 12.3) includes a loading for overheads of 30% for underground personnel 25% for surface wages personnel, and 20% for salaried staff. This loading includes holiday pay, sick pay, payroll tax and workers compensation insurance premium. Examples of the build-up of annual cost for several classes of labour are given in Table 12.1.

12.2 Power Costs.

The basic approach to sources of electric power and timing of an H.E.C. contract is fully covered in Section 11.2 above - see comment on items 15 and 16.

The calculation of power costs for the first and second years of operation is shown in Table 12.2. Power costs for subsequent years of operation are as for the second year of operation, by which time the assumed pumping rate is at 2 million gallons per day i.e. the "normal" inflow to the mine from the surrounding countryside, as illustrated by the last few years of operation prior to closing of the mine in 1914.

It is assumed that 2500 Kw of power will be available from the H.E.C. from the commencement of the first year of production (i.e. the third year of the project). This allows for the execution of a contract with the H.E.C. upon completion of confirmatory drilling, and a one year period between executing the contract and the power supply being provided by the H.E.C.

Power usage from commencement of operations is within the capacity to be provided by the H.E.C. except for a small usage in the first year of operations. Calculated usage from the H.E.C. has been costed at the rates indicated by the H.E.C. were a contract to be signed with the H.E.C. in the near future ($50 per Kw per annum plus .5 cents per KwH); while usage of diesel generated power has been costed at 1.6 cents/KwH. It is assumed that from start of operations, maintenance of diesel generating sets will be handled by the electrical and fitting personnel employed for general mine and mill maintenance. (Separate personnel were allowed for this job prior to commencement of operations.)

Inflation of the power expense item has been covered in the overall inflation allowance for operating costs at a rate of 15% per annum (see first page of Table 12.3).

An average five percent usage of pump capacity throughout the year, over and above that required to handle the three and two million gallons per day assumed as normal inflow, has been allowed for the handling of surges in water inflow as a result of cutting of new water bearing strata during development. This represents 2½ weeks per year pumping at full capacity of six million gallons per day, which more than covers the period of such surges shown by old pumping records in the latter years of mine operation.

12.3 Other Items.

Generally the basis for calculation of operating cost items is shown in Table 12.3.

12.4 <u>Overall Checks on Order of Accuracy</u>.

A check on the overall order of accuracy has been possible through the availability of cost data from the Scotia Mine in Western Australia, the Carr Boyd Mine in Western Australia, and from Central Norseman N.L. A check on a number of detailed items has also been possible through the cost breakdown provided by Central Norseman N.L.

The overall checks, after allowing for the main identifiable differences between the operations being compared, indicates that the operating costs for Beaconsfield are of the right order.

The available data also indicates that the sum allowed for project management fee is also of the right order.

Table 12.1

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BEACONSFIELD FEASIBILITY REPORT - JULY, 1975.

INDICATIVE CALCULATIONS OF LABOUR COST PER ANNUM.

	WEEKLY COST					ANNUAL	
Classification	Award *	Average over - award	Sub Total	Overhead Allowance	TOTAL	Total x 52	Including Allowance for Overtime
Top Miner (on Winzes/ Raises)	112	15	127	(30%) 40	167	8700	9500
Contract Miner	As Top Miner (127), plus 1/3rd =		170	(30%) 50	220	11500	11500
Fitter/Electrician	113	15	128	(30%) 39	167	8700	9500
Average Millhand	92	15	107	(25%) 27	134	7000	7000
Winder Drivers	112	15	127	(25%) 32	159	8300	9500
Bracemen	95	15	110	(30%) 110	143	7400	8000

* Gold and Metalliferous Award.

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Table 12.2

BEACONSFIELD FEASIBILITY REPORT - JULY, 1975.

CALCULATION OF POWER COSTS

Year 3 (First Year of Operation)	Total Absorbed H.P.	Average Usage %	Absorbed H.P. Usage on Average
Pumps (3m.g.p.d.)	4000	55%	2200
Winder	350	75%	262
Surface	115	80%	98
Mining	370	74%	275
		Total Mine	2835
Milling (5 day week)	710	70%	500
Crusher	250	30%	75
		Total Mill	575
			3410
		Plus 10% Losses	340
		TOTAL	3750 Abs. H.P.

K.W. Used:

Mine (2835 + 280) x .75	=	2340
Mill (575 + 60) x .75	=	475
	Total	2815 KW

H.E.C. Supply: (2500 KW)

	Mine	Mill
Demand Charge @ $50/KW		
- 475 KW Mill	-	23,750
- 2025 KW Mine	101,250	-
Energy Charge @ .5 cents/KWH		
- 475 x 8740 x $\frac{5}{1000}$	-	20,750
- 2025 x 8740 x $\frac{5}{1000}$	88,500	-
Diesel Supply: (Balance 315 KW to Mine)		
Operating Cost @ 1.6 cents/KWH		
- 315 x 8740 x $\frac{16}{1000}$	44,050	-
	$233,800	$44,500

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Year 4. (Second Year of Operation)	Total Absorbed H.P.	Average Usage %	Absorbed H.P. Usage on Average
Pumps (2 m.g.p.d.)	4000	38%	1520
Winder	350	75%	262
Surface	115	86%	98
Mining	370	74%	275
		Total Mine	2155
Milling (7 day week)	710	100%	710
Crusher	250	30%	75
		Total Mill	785
			2940
		Plus 10% Losses	294
		TOTAL	3234 Abs. H.P.

KW Used:

Mine (2155 + 215) x .75	=	1780
Mill (785 + 78) x .75	=	650
	Total	2430 KW

H.E.C. Supply:

	Mine	Mill
Demand Charge @ $50		
- 650 KW	-	32,500
- 1780 KW	89,000	-
Energy Charge @ .5 cents		
- 650 x 8740 x $\frac{5}{1000}$	-	28,400
- 1780 x 8740 x $\frac{5}{1000}$	77,600	-
	$166,600	$60,900

BEACONSFIELD FEASIBILITY REPORT - JULY, 1975.

OPERATING COST AND STAFFING SUMMARY

	Cost p.a. for 100,000	Cost/ton (100,000 T.P.A.)	Cost Year 3 (Mine 60,000 Mill 74,000)	Number Men (100,000 T.P.A.)
	$'000		$'000	
Mining	1,069	$10.69	1,054	61
Milling	604	6.04	464	29
Admin. & Management	303	3.03	284	12
Totals (1975 dollars)	1,976	$19.76	1,802	102

ALLOWANCE FOR INFLATION DURING NEGOTIATION (6 MONTHS) AND MINE DEVELOPMENT (2 YEARS). ($'000).

(a) First Year of Operation

(Mine 60,000 tons, Mill 74,000 tons.)	Mine	Mill	Admin.	Total
July 1975 Costs	1,054	464	284	1,802
+ 7½% to end 1975	79	35	21	135
	1,133	499	305	1,937
+ 15% 1976	170	75	46	291
	1,303	574	351	2,228
+ 15% 1977	195	86	53	334
	$ 1,498	660	404	2,562

(b) Second Year of Operations and thereafter.

(Mine and Mill 100,000 Tons)

	Mine	Mill	Admin.	Total
July 1975 Costs	1,069	604	303	1,976
+ 7½% to end 1975	81	45	23	149
	1,150	649	326	2,125
+ 15% 1976	173	97	49	319
	1,323	746	375	2,444
+ 15% 1977	197	112	56	365
	$ 1,520	858	431	2,809

Note: Inflation of operating expenses has been taken into account up to start of commercial operations, in line with forecast of gold price 2½ years ahead.

Table 12.3 148

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BEACONSFIELD GOLDMINE - OPERATING COST - FEASIBILITY REPORT - JULY, 1975.

Group	Item No.	Item	Cost Data (100,000 T.P.A.)	Annual Cost (1975 dollars) 100,000 T.P.A.	Annual Cost (1975 dollars) Mine 60,000 tons Mill 74,000 tons Year 3.	Cost/ton (100,000 T.P.A.)	No. Men (100,000 T.P.A.)	References
Mining	1	Stoping labour	33 tons/man/shift 4 Faces x 4 men = 16 @ $11,500 (see Note)	184,000	(3 Faces 12 men) 138,000	1.84	16	
	2	Stope Maintenance	3 men @ $9,500 Supplies $22,500 p.a.	51,000	(2 men) 35,000	0.51	3	
	3	Stope Filling	6 men @ $9,500 Supplies $10,000 p.a.	67,000	(4 men) 45,000	0.67	6	
	4	Explosives	Cost/ton as Central Norseman	26,000	(60%) 16,000	0.26	-	
	5	General Stores	(Say)	10,000	(60%) 6,000	0.10	-	
	6	Timber for Levels	2 men @ $9,500 Timber $14,000 p.a.	35,000	(2 men) 35,000	0.35	2	
	7	Compressed Air (excl. Power)	Supplies $18,000 p.a.	18,000	12,000	0.18	-	
	8	Maintenance, Air and Water Lines	2 men @ $9,500 Materials $6,000 p.a.	25,000	(2 men) 23,500	0.25	2	
	9	Rock drills and Steels	15c./ton Steels) 20c./ton Drills) plus 15% contingency	40,000	(60%) 24,000	0.40	-	

Table 12.3

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BEACONSFIELD GOLDMINE - OPERATING COST - FEASIBILITY REPORT - JULY, 1975.

Group	Item No.	Item	Cost Data (100,000 T.P.A.)	Annual Cost (1975 dollars) 100,000 T.P.A.	Annual Cost (1975 dollars) Mine 60,000 tons Mill 74,000 tons Year 3.	Cost/ton (100,000 T.P.A.)	No. Men (100,000 T.P.A.)	References
Mining	10	Hoisting Labour	4 Winder Drives @ $9,500 2 Bracemen) 4 @ $8,000 2 Platmen)	70,000	(8 men) 70,000	0.70	8	
	11	Development	$180,000 per level of 180,000 tons (including dilution) = $1.00 per Ton	100,000	(1800 ft. Level full development) 175,000	1.00	5	Refer Table 11.1 Item 35(b)
	12	Pump and Diesel Generator Maintenance - Labour	3 men @ $9,500	28,500	(3 men) 28,500	0.28	3	
	13	Haulage	2 Loco. Drivers @ $10,500 2 Truckers @ $9,500 Supplies $22,000 p.a.	62,000	(3 men) 46,500	0.62	4	
	14	Diamond Drill (excl. assay)	1000 meters p.a. @ $30/ meter	30,000	(full rate) 30,000	0.30	2	
	15	Supervision	3 Shift Foremen @ $11,000 2 Day Foremen @ $13,000	59,000	59,000	0.59	5	
	16	Power to Mine	Includes power for pumps	166,600	233,800	1.67	-	Refer Table 12.2

Table 12.3

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BEACONSFIELD GOLDMINE - OPERATING COST - FEASIBILITY REPORT - JULY, 1975.

Group	Item No.	Item	Cost Data (100,000 T.P.A.)	Annual Cost (1975 dollars) 100,000 T.P.A.	Annual Cost (1975 dollars) Mine 60,000 tons Mill 74,000 tons Year 3.	Cost/ton (100,000 T.P.A.)	No. Men (100,000 T.P.A.)	References
Mining	17	Engineering Services for Mine Plant	1 Foreman @ $11,000 2 Fitters @ $9,500 2 Offsiders @ $8,500 Materials $50,000 p.a.	97,000	(5 men 60% Material) 77,000	0.97	5	
			TOTAL "MINING"	$1,069,000	1,054,300	1.07	61	

Note: Labour and Salary costs per annum include:

30% for overheads on underground personnel
25% for overheads on surface wages personnel
20% for overheads on salaried staff.

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BEACONSFIELD GOLDMINE - OPERATING COST - FEASIBILITY REPORT - JULY, 1975.

Group	Item No.	Item	Cost Data (100,000 T.P.A.)	Annual Cost (1975 dollars) 100,000 T.P.A.	Annual Cost (1975 dollars) Mine 60,000 tons Mill 74,000 tons Year 3.	Cost/ton (100,000 T.P.A.)	No. Men (100,000 T.P.A.)	References
Milling	1	Supervision	4 Shift Supervisors @ $11,000	44,000	44,000	0.44	4	
	2	Assay	1 Chief Chemist @ $15,000 2 Assayers @ $7,000 Supplies $12,000 p.a.	41,000	(75% Supplies) 38,000	0.41	3	
	3	Mill Labour	18 men @ $7,500 (7 day week	135,000	(5 day week 14 men) 105,000	1.35	18	
	4	Gold Room	2 men @ $9,500	19,000	19,000	0.19	2	
	5	Maintenance - general	2 men @ $9,500 Materials $80,000 p.a.	99,000	(2 men 75% Mat/ls) 59,000	0.99	2	
	6	Re-agents	Usage ex Lab. tests priced ex Cyanamide	140,000	(75) 105,000	1.40	-	
	7	Ball-mill lining and replacements	Balls 51c/ton/p.a. Liners 50c/ton/p.a.	65,000	(75¢) 49,000	0.65	-	
	8	Power to Mill		60,900	44,500	0.61	-	Refer Table 12.2
			TOTAL "MILLING"	603,900	463,500	6.04	29	

BEACONSFIELD GOLDMINE - OPERATING COST - FEASIBILITY REPORT - JULY, 1975.

Group	Item	Cost Data (100,000 T.P.A.)	Annual Cost (1975 dollars) 100,000 T.P.A.	Annual Cost (1975 dollars) Mine 60,000 tons Mill 74,000 tons Year 3.	Cost/ton (100,000 T.P.A.)	No. Men (100,000 T.P.A.)	Comment
Administration and Management	Resident Manager	1 man @ $23,000	23,000	23,000		1	House provided
	Department Heads and Geologist	1 Mill Manager) 1 Mine Manager) 4 @ 1 Chief Engineer) $18,000 1 Accountant)	72,000	72,000			Two of these have houses provided
		1 Geologist	11,000	11,000	0.83	5	
	Clerks	3 @ $8,000 average	24,000	24,000	0.24	3	
	Typists	3 @ $7,500 average	22,500	22,500	0.23	3	
	Office Supplies, telephones, rates, etc.		50,000	50,000	0.50	-	
	Consultants		15,000	15,000	0.15	-	
	Management Fee	Payable to company responsible for Joint Venture Management	70,000	(75%) 52,000	0.70	-	Approximately as per current draft agreement with Tricentrol

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BEACONSFIELD GOLDMINE - OPERATING COST - FEASIBILITY REPORT - JULY, 1975.

Group	Item	Cost Data (100,000 T.P.A.)	Annual Cost (1975 dollars) 100,000 T.P.A.	Mine 60,000 tons Mill 74,000 tons Year 3.	Cost/ton (100,000 T.P.A.)	No. Men (100,000 T.P.A.)	Comment
Administration and Management	Management Travel		15,000	15,000	0.15	-	
		TOTAL ADMINISTRATION AND MANAGEMENT	$302,500	$284,500	3.03	12	

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BEACONSFIELD FEASIBILITY REPORT - JULY, 1975.

13. METAL PRICES AND REVENUE.

13.1 Gold Price.

13.1.1 Assumptions.

The financial forecasts in this Report are based on a prime assumption of a gold price 2½ years hence of $A150 per ounce, while sensitivity analyses show the affect of prices of $A130 and $A175 per ounce in two years time.

13.1.2 Background.

(a) Figure 13.1 shows gold price movements over the past seven years.

The rise in gold price since early 1972 is generally accepted as an inevitable adjustment to take account of currency devaluation since the fixing of gold price at Bretton Woods in 1942, especially since restrictions were removed on the free market price of gold.

The price of gold appears recently to have reached some sort of stability in the region of $A130, which is the approximate Australian currency price at which France revalued its gold reserves. It is also approximately the price at which U.S. Treasury gold sales have been made at auction. This price is taken as the low point in a range of gold prices two years hence, used in this study.

(b) Future movements in gold price will depend upon several main factors including its final position in the international banking system, the extent to which inflation is controlled in the Western economies, and the strength of recessionary forces in Western economies.

Despite the pure logic in the United States' approach to the place of gold in the international monetary system and gold's recent demonetisation, it is likely that the general human regard for gold as a store of value (based on thousands of years of acceptance), and the difficulty of establishing a world-wide trust in paper currencies

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the backing for which does not include gold, will ensure that gold retains a meaningful position (via Central Bank reserves) in the international banking system.

The factors of inflation and recession could well be linked in the foreseeable future, in that control of inflation could require moves which promote recession, or the avoidance of recession could mean that inflation is inevitable. Balance in these factors has yet to be seen in most Western economies.

Examination of gold futures markets, in which gold is bought and sold for delivery up to sixteen months ahead, shows a consistent upwards movement in price with increasing length of delivery. Extrapolated to 24 months ahead this movement indicates a price of about $US200. (Refer Figure 13.2).

To be conservative in forecasting, a price of $US200, or $A150, has been taken as the primary price projection for gold 30 months hence.

(c) We believe that one of the leading gold mining companies in the world adopts gold prices up to $US250 for projections of sales in about 3 years time.

This reflects the view that only temporary restraints have limited the rise in gold price during 1975, including such factors as U.S. Treasury gold auctions.

To take account of such views, sensitivity studies include figures resulting from a gold price of $US235 or $A175.

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13.1.3 Mechanics for Sale of Gold Bullion.

All gold bullion produced in Australia is sold in the following manner:

(a) It is delivered to one of four accredited buyers:

(i) Matthey Garret
(ii) Engelhard
(iii) Perth Mint
(iv) Electrolytic Refining and Smelting Co.

The buyer pays the seller a price which is fixed by the Reserve Bank (currently $32.25 per ounce).

(b) The buyer passes the gold to the Reserve Bank and declares quantities, grade and company to which gold is credited.

(c) Every two weeks the Reserve Bank advises the Gold Producers Association of quantities of gold lodged with the Bank.

The Gold Producers Association then sells this quantity of gold on the open market.

(d) Approximately 6 - 8 weeks after lodgement with the initial buyer, the Gold Producers Association credits the initial seller with the difference between the sales price achieved on the open market and the price paid by the initial buyer.

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13.2 Copper Price.

The price range for copper in the last 12 - 15 months has varied from £stg.1400 to near £stg.500 per tonne, and is currently on an upward trend at approximately £stg.565.

International copper producers are determined to achieve some degree of price stability for the metal, and a conservative figure for 2½ years hence, of £stg.700 per tonne, has been used in the Feasibility Study.

13.3 Revenue Calculation.

Revenue, as summarised in Sect.13.3.3, is calculated on the following bases.

13.3.1 Gold.

Tonnage and grade as per Watts, Griffis, McOuat/T.W. Willsteed Reserve Calculations for "drill indicated ore" plus "possible mineralisation":

494,000 Tons at 15.6 dwt/ton (Refer 5.4).

Dilution at 20%. (Refer 4.4 and 5.4).

Tonnage and grade after dilution:

593,000 Tons at 13 dwt/ton.

Gold recovery, first two years of operation, 82½%, thereafter 85%. (Refer 6.3).

Mill output, first year of operation, 74,000 tons, thereafter 100,000 tons per annum. (Refer 9.4).

Gold price, $A150 ($US200) per ounce in 2½ years time. (Refer 13.1).

Gold revenue per ton of headfeed, first two years of operation = 13 dwt x 82½% x $A7.50 per dwt = $80.44.

Gold revenue per ton of headfeed thereafter = 13 dwt x 85% x $A7.50 per dwt = $82.87

Total Gold Revenue:

Year 1 of Operation	74,000 x $A80.44	=	$5,953,000
Year 2 of Operation	100,000 x $A80.44	=	$8,044,000
Thereafter	100,000 x $A82.87	=	$8,287,000

13.3.2 Other Metals.

Analysis of assays on core recovery indicates a head grade of silver of 5.8 dwts per ton of headfeed, and a .75% of copper in headfeed.

Watts Griffis McOuat/Willsteed ore reserve and mineralisation estimates, on which this Report is based, "suggest that a figure of 1% Cu be used for economic exercises, and that the value of the silver be disregarded at this stage."

Revenue calculations in this Report are therefore based on 1% copper in ore, and ignore silver.

The W.G.M./Willsteed report also assumes an 85% recovery of copper, whereas subsequent metallurgical test work by the Mines Department on core material indicated 53% recovery. The Mines Department reported that recovery could be improved in practice, their limitation on further work being availability of test material.

A copper recovery of 60% has been used in revenue calculations in this Report.

The W.G.M./Willsteed report assumes a concentrate grade of 30% compared with Mines Department test results on core material of 15.5%. The Mines Department reported that concentrate grade could be improved in practice (as with recovery).

A concentrate grade of 25% has been used in revenue calculations in this Report.

The price of copper 2½ years hence is taken as £stg.700/tonne, equivalent to $A1,190 per ton at current exchange rates.

Value of copper concentrate at 25% grade is approximately $A300 per ton.

Cost of smelting, freight and transit losses is approximately $A100 per ton, leaving a net smelter return per ton of concentrate of $A200/ton, or 36 cents per pound of copper.

Contained copper in one ton of headfeed (i.e. after 20% dilution) .83% = 18.6 lbs.

With recovery at 60%, recovered copper per ton of headfeed is 18.6 x 60% = 11.2 lbs.

Net value of recovered copper per ton of headfeed = 11.2 lbs. x 36 cents = $4.03.

∴ Total revenue from copper:

Year 1 of Operation:

74,000 x $4.03 = $298,000

Year 2 of Operation and thereafter:

100,000 x $4.03 = $403,000.

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13.3.3 All Metals.

Total revenue for all metals, per ton of headfeed, is as follows:

	Year of Operation		
	1.	2.	3.
Gold	80.44	80.44	82.87
Other Metals	4.03	4.03	4.03
All Metals	84.47	84.47	86.90

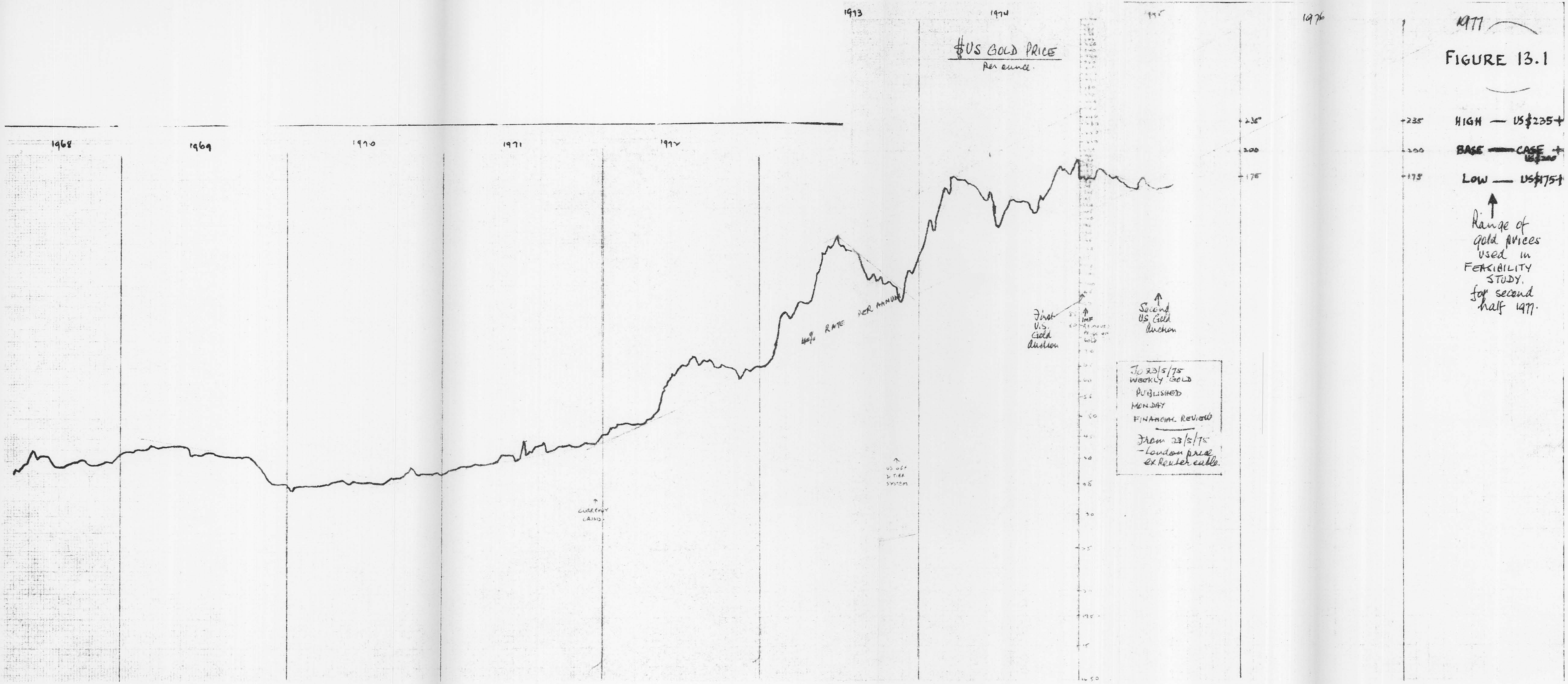
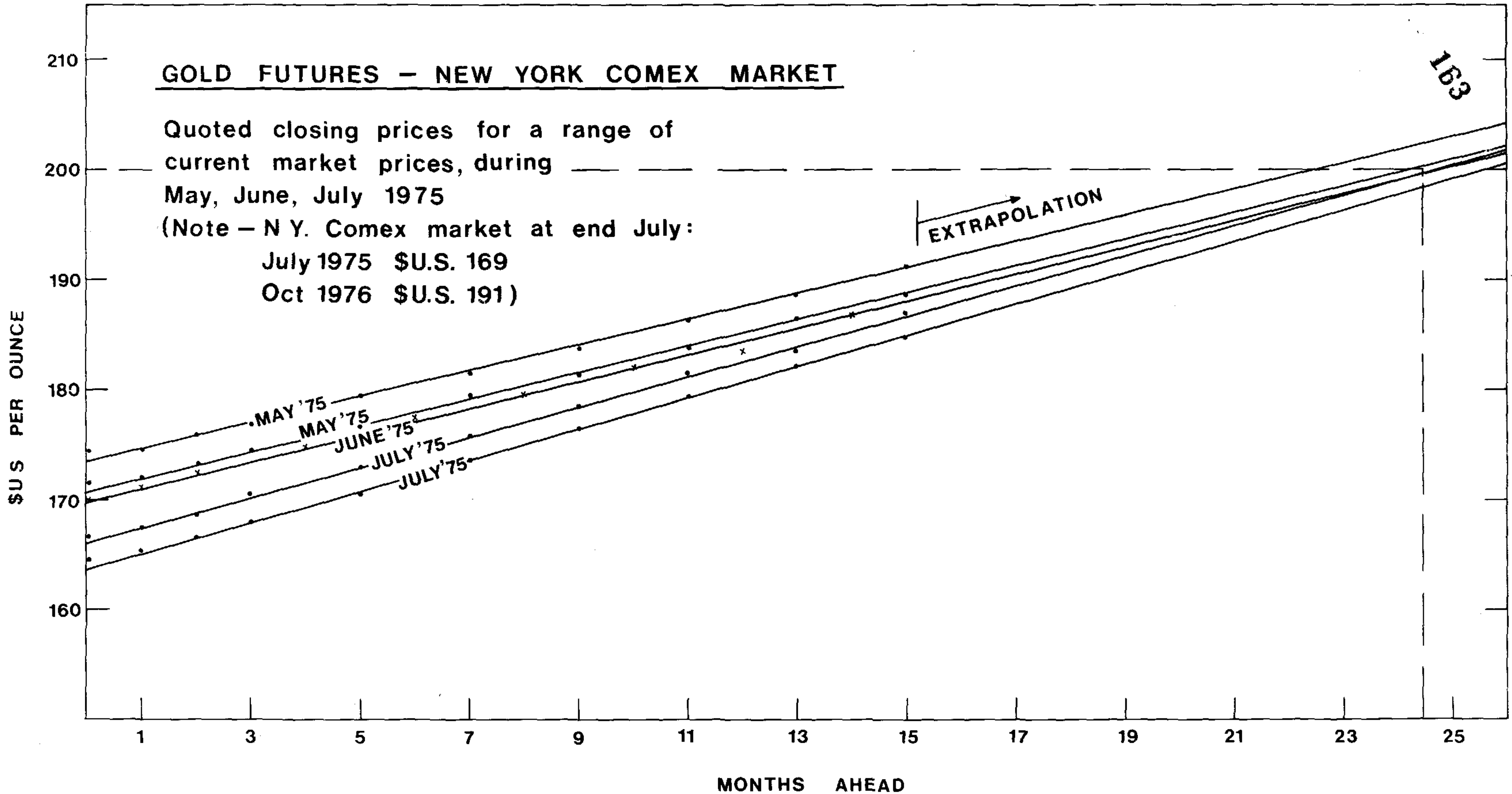

$US GOLD PRICE
Per ounce.
1968
1969
1970
1971
1972
1973
1974
1975
1976
1977
CURRENCY CRISIS
US GOT 2 TIER SYSTEM
4% RATE PER ANNUM
First U.S. Gold Auction
IMF Removes Price on Gold
Second US Gold Auction
To 23/5/75 WEEKLY GOLD PUBLISHED MONDAY FINANCIAL REVIEW
From 23/5/75 - London price ex Reuter cable.
235
200
175
HIGH — US$235+
BASE — CASE + US$200
LOW — US$175+
Range of gold prices used in FEASIBILITY STUDY. for second half 1977.

FIGURE 13.1

FIG. 13.2


GOLD FUTURES – NEW YORK COMEX MARKET
Quoted closing prices for a range of current market prices, during May, June, July 1975
(Note – N Y. Comex market at end July:
July 1975 $U.S. 169
Oct 1976 $U.S. 191)
163
EXTRAPOLATION
MAY '75
MAY '75
JUNE '75
JULY '75
JULY '75
210
200
190
180
170
160
$US PER OUNCE
1
3
5
7
9
11
13
15
17
19
21
23
25
MONTHS AHEAD

14 TAXATION OF GOLD REVENUE.

14.1 Present Position.

Revenue from goldmining is currently not taxable in Australia, but the position regarding future taxation of such revenue is presently under review.

In December/January 1974/75 the Industries Assistance Commission conducted an enquiry as to "whether the Australian Government should accord assistance including assistance by way of taxation treatment for the production in Australia of gold".

The Report from the I.A.C. went to the Government towards the end of May and this report will be published at the end of July. In the following month (i.e. August) public comment will be received, and the report will be considered by a Standing Interdepartmental Committee on Assistance to Industry.

This Committee reviews the I.A.C. recommendations in the light of Government policy, and makes its recommendations to Cabinet. The decision by Cabinet on this matter is expected at the earliest in late August, but probably in September. It is understood that Cabinet decisions are usually in line with the Committee's recommendation.

The Government decision could well be influenced by political considerations i.e. the sensitivity to this decision of the Kalgoorlie electorate which is presently represented by a Labour Member of Parliament.

It is our experience on another matter, where employment was at stake, (regarding tariff on calcium carbide production at Electrona) that the Australian Government decision was contrary to the recommendation of the I.A.C. and in favour of continued protection.

Therefore we consider that there are reasonable grounds for expecting a Government decision to continue the present non-taxable status of revenue from goldmining; or in the event that such revenue is to be taxed, that there would be some substantial benefit to gold mining compared with the taxation levied on mining generally.

14.2 Assumptions.

The taxation treatment of gold revenue is one of the most vital factors influencing the investment outcome of the Beaconsfield Goldmine project.

We have therefore shown the results of financial analyses on two bases i.e. taxed on maximum basis, and non-taxed.

Fortunately the project remains an attractive investment proposition even with the application of full income tax to its reveneu.

Note: Since the above text was prepared the Press has published the recommendations of the I.A.C. report, as shown in the following cuttings.

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Aust. Fin. Review 29/7/75

IAC urges Govt to stop aid to gold-mining

By ROBERT HAUPT

GOVERNMENT assistance to the gold-mining industry, which has been a burning issue in the Labor Party Caucus at the instigation of the member for Kalgorlie, Mr Fred Collard, should be discontinued, the Industries Assistance Commission has recommended.

The commission has recommended the ending of the Government assistance both by way of subsidy and through tax concessions.

Yesterday the Government also announced a decision on a textile case before the IAC which may set a trend back towards a more stringent attitude towards assistance.

On the residual items of the last major textiles report, the Government has decided to ignore the IAC's recommendation that it seek voluntary restraint under the GATT agreement.

The Government decided, on the basis of a decline in imports of the relevant goods since the textiles authority report, that it would defer action "for the time being."

On gold, the IAC said: "Australian Government provisions for assistance to the gold-mining industry have comprised subsidy assistance under the Gold-Mining Industry Assistance Act 1954-1972 and concessions under Sections 23(0), 23c(1) and 23c(2) of the Income Tax Assessment Act which exempt the income received by persons and companies engaged in gold mining.

"In the commission's opinion, these measures are inappropriate for the current situation.

"With respect to the Gold-Mining Industry Assistance Act, no significant payments have been made since 1972-73. The concessions provided under this Act were designed for the situation where the price of gold was maintained at an artificially low level.

"Unless there is a major reversal of world monetary policy, it is unlikely that the price of gold will ever again be subject to the constraints which were applied in the past.

"Consequently, the need for direct financial assistance in the form of a subsidy to compensate producers for artificially depressed prices appears to have been permanently removed.

"With respect to the taxation concessions, there are two principal reasons why assistance in the form currently provided is inappropriate.

"First, with the large increases in the price of gold which have occurred in recent years, the cost of these tax concessions in terms of the taxation revenue forgone has greatly increased.

"In 1973-74 the cost to the general community of these tax concessions totalled over $12-million and, by 1980, they could cost in excess of $40 million annually, depending on movements in production costs and in the price of gold.

"Second, and more importantly, the taxation provisions provide assistance to the gold-mining industry which is not available to other forms of mining.

"Under the current situation where gold prices are not depressed by Government action, the taxation concessions mean that gold-mining receives more assistance than other forms of mining activity.

"It would be undesirable if such a situation were to be continued on a permanent basis since it must eventually result in a misallocation of resources between gold mining and other forms of mining.

"The commission considers that these concessions can no longer be justified and recommends their removal.

"In order to avoid undue disruption in the industry, the commission has recommended the phasing-out of the existing taxation measures over a period."

The textile decision covered the following items:

- Woollen and worsted fabrics and mixtures (including interl.n-ings) other than fabrics containing a mixture of discontinuous man-made fibres and wool.
- Blankets and rugs.
- Cotton yarns, discontinuous man-made fibre yarns and gimped yarns;
- Cotton fabrics, other than *unraised sheeting for* use as or in the making-up of bed linen: including denims and drill.
- Elastic and elastomeric fabrics. fabrics.
- Woven labels, badges and the like; and
- Embroidery.

The Government also announced yesterday a temporary assistance authority inquiring on plywood imports

Australian 29/7/75.

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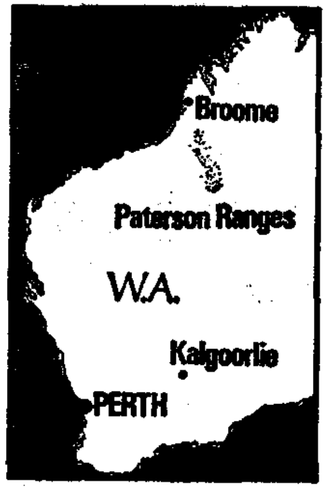
IAC urges end of tax aid for gold miners

Industry warns of collapse



AUSTRALIA'S gold producers were told virtually to support themselves without Government assistance yesterday in a special report by the Industries Assistance Commission, which recommends the ending of special taxation concessions.

At the same time Broken Hill Proprietary announced it would take part in a new gold mine in the Paterson Ranges in north-west Western Australia, with an initial investment of $27 million.

It is the first major gold mine development in 40 years (Story below).

Industry leaders said last night the IAC recommendations would kill the industry and particularly the city of Kalgoorlie, whose 23,000 people rely almost entirely on gold mining.

The IAC said the assistance was not necessary because the artificial restraints imposed on the industry no longer applied. It recommended phasing out the concessions, which exempt gold mining income from tax.

Concessions could be phased out over a five-year period, which would mean that by July 1980 gold miners would be taxed at the same rate as other producers of minerals.

The IAC also rejected a claim that gold miners should be given special treatment, such as long-term low interest loans to finance new housing. The commission saw no reason why prospecting for gold should be distinguished from prospecting for minerals in general.

The special Minister of State, Senator D. McClelland, said the Government would consider the IAC report, which pointed out that the tax concessions provided to the gold mining industry were unique to that industry.

"In the commission's opinion, these concessions are inappropriate and their continuation cannot be justified," the report said. "As these concessions are not available to other forms of mining, the additional assistance available to gold mining must eventually result in a misallocation of resources within the mining sector."

The IAC recommended the phasing out of the concessions to allow gold miners to adjust to the proposed new taxation plan. It estimated that phasing out the concessions would cost the industry between $60 million and $130 million.

LABOR COSTS

But the commission also suggested that restrictions on gold be lifted. The Banking Act puts firm restrictions on the private sale and ownership of gold in Australia.

"The commission notes that in recent years various countries have repealed such legislation," the report said.

Industry spokesmen in Kalgoorlie said last night most mines would collapse if the tax concessions were dropped. One spokesman said the town was fuming.

"How the hell can they call it an assistance commission?" he asked.

The West Australian gold mining industry is worth about $24 million to Australia, but few companies are making much profit and the industry is reeling under inflation and labor costs and the petrol equalisation tax, the spokesman said.

Since the abolition of tax concessions on shareholders' dividends, much of the incentive had been lost.

The mining companies were begging for assistance, he said. In the past six weeks, more than 100 mine employees have been retrenched.

About 2000 Kalgoorlie people work in the mines. Most of the town's residents rely on the industry.

The town clerk of Kalgoorlie, Mr D. Morrison, said last night: "The industry has been struggling for some time. The last boom, in 1967-69, lifted shareholders a little but things have been very quiet. The town depends on the gold mining industry. We are aware of the downtrend at present and view it with concern."

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15. FINANCIAL ANALYSES.

All D.C.F. rates of return given below are based on future capital expenditures only. i.e. there is no allowance for the present value of the project as it presently exists.

15.1 Discounted Cash Flow Method, on Total Ore and Mineralisation.

Tables 15.1 and 15.2 show the cash flow from the project given the chosen level for the various bases for calculation, as discussed and supported in the text of this Report. (Referred to as the "Base Case").

To summarise, these items are as follow:

Combined tonnage of ore reserves and mineralisation	493,000 tonnes
Grade in situ	15.6 dwt of gold per tonne, and 1% Copper.
Dilution by barren rock in headfeed to mill	20%
Mill throughput	
- first year of operation:	74,000 tonnes
- second year, and subsequent years of operation:	100,000 tonnes
Gold price in 1978	$A150 per ounce
Copper price in 1978	£stg.700 per tonne
Mill recovery of gold	
- first two years of operation	82½%
- thereafter	85%
Timetable for mine development	as per 10.1
Capital costs	as summarised in Table 11.1
Operating costs	as summarised in Table 12.3

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Calculation of income tax (in the taxed alternative)	assumes worst possible condition of full (45%) company rate of tax on gold revenue, with deduction of capital cost over full life of mine.

On the above bases, the project cash flow before tax is as follows:

First year of operation	$3.17 million
Second year of operation	$5.64 million
Subsequent years of operation	$5.88 million.

D.C.F. calculation on cash flow before tax is in excess of 50%. Present value, discounted at 15% = $9.9 million; discounted at 20% = $7.5 million.

Project cash flow after tax is as follows:

First year of operation	$2.00 million
Second year of operation	$3.54 million
Subsequent years of operation	$3.73 million.

D.C.F. on a maximum-tax-basis is 39%. Present value, discounted at 15% = $4.6 million; discounted at 20% = $3.1 million.

15.2 Sensitivity Analysis.

15.2.1 Changes in Future Capital Cost.

The basic future capital cost of $7 million includes 2½ years of cost escalation at 15% per annum, and a lump sum contingency of $235,000 in excess of contingencies for specific items.

Nevertheless it is interesting to consider the effect on D.C.F. percentage of a further 10% increase in capital costs. The outcome is a reduction in after-tax D.C.F. of 3%, to 36%. (D.C.F. level on a before-tax basis continues to exceed 50%).

It is also of interest to consider the possibility that capital cost has been overestimated by say 10%. This is a possibility in view of the amounts included for inflation and lump sum contingency, also because although capital cost estimates include several major plant items on a second-hand basis (where such plant is known specifically to be available), many other items will probably be acquired second-hand in practice. The increase in after-tax D.C.F. level would be 3%, to 42%. (D.C.F. level on a before-tax basis continues to exceed 50%).

15.2.2 Changes in Operating Cost due to one of a variety of Causes.

Total annual operating costs include allowance for cost inflation over 2½ years to commencement of production, at the rate of 15% per annum.

An increase in total annual operating costs by 20% (i.e. by $560,000 per annum) would leave D.C.F. rate of return on an untaxed basis in excess of 50%; or reduce the D.C.F. rate of return on a fully taxed basis by 4%, to 35%.

Such an increase in costs would result from any one of the following:

(i) increase in ore processing costs by 66% to $A14.2 per tonne of ore milled in 1979;

(ii) increase in mine operating costs by 37% to $A20.8 per tonne of ore produced in 1979;

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(iii) increase in mine overbreak from 20% as allowed, to 48%.

The effect on profits is also equivalent to decreased revenue resulting from a drop in gold recovery in milling from 85% as allowed, to 79.4%.

15.2.3 Changes in Grade.

Records from the old Tasmania Gold Mine Company include the result of detailed sampling of ore grade on each operating level of the mine. The sampled average for the 940 ft. of driving on the 1500 ft. Level was 13 dwt per ton, which was a significant increase on the sampled average for the same 940 ft. of the 1370 ft. Level.

Due to the consistently high gold grades of drill intersections of the orebody extension below 1500 ft., the Willsteed/W.G.M. certification of ore and mineralisation grade for the orebody extension averages 15.6 dwt per ton.

In the event that the orebody extension was of a gold grade equivalent only to the 1500 ft. Level, the reduction in orebody average gold value would be from 15.6 dwt/ton to 13 dwt/ton i.e. a 17% reduction. (The equivalent reduction in gold value of headfeed would be from 13 dwt to 10.83 dwt/ton.)

Such a reduction would reduce D.C.F. rate of return to 47.5% on an untaxed basis, or by 10%, to 29%, on a fully taxed basis.

15.2.4 Changes in Price of Gold.

The most objective available indicators of the future price of gold are the Gold Futures Markets which arrange forward sales for gold bullion up to about 16 months ahead. Figure 13.2 shows price quotes for future gold deliveries, for a range of current gold prices expressed in U.S. dollars, over the past 3 months. They indicate, by extrapolation, a price of $US200 per ounce of gold in 24 months time.

A price of $US200, converted at July, 1975 exchange rate (1.33 : 1), has been conservatively taken as the base price for gold 30 months hence, for use in the Feasibility Study for Beaconsfield.

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It is however, for reasons given in Section 13.1 of this Report, of interest to look at a future gold price both above and below this assumed level.

A reduction of gold price from $A150 to $A130 per ounce leaves D.C.F. rate of return in excess of 50% on an untaxed basis; and reduces the D.C.F. rate of return by approximately 7.5% to 31.5% on a fully taxed basis.

An increase of gold price from $A150 to $A175 per ounce leaves D.C.F. rate of return in excess of 50% on an untaxed basis; and increases the D.C.F. rate of return by approximately 10% to 49% on a fully taxed basis.

15.2.5 Change in Life of Mine.

The Willsteed/W.G.M. certification of ore reserves and mineralisation at Beaconsfield is for 493,000 tons in situ.

The orebody extension is however open-ended at depth, and as indicated in Section 3.4 of this Report, there is geological interpretation of drill core data which suggests the extension of the strata in which the orebody is contained at higher levels, to at least a depth of 2400 feet.

In the event that the orebody extension continues below the 2000 ft. Level, the life of the mine could be significantly increased, the profitability of mining could be continued for many years, and the financial attraction of the initial investment could be significantly increased.

However in view of the already high level of D.C.F. percentage on the ore and mineralisation estimate limited by a depth of 2000 ft., no attempt is made to estimate the effect on D.C.F. of an extension of the orebody below 2000 feet.

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15.3 Minimum Grade or Tonnage to give Return of Capital plus 100%.

Confirmatory drilling of the orebody extension is scheduled to follow dewatering of the mine to below the 1370 ft. Level; and will be undertaken from the crosscut between Harts and Grubb shafts at this Level. (Alternatively it could be undertaken from the 1000 ft. Level from the crosscut at that Level).

The main aim of this drilling is to up-grade ore and mineralisation reserves, to obtain additional samples of drill core materials for metallurgical test, and to assist in the planning of mine development.

Positive outcome of this confirmatory drilling will inevitably strengthen the project; and it follows that Stage I of mine development has a higher level of investment risk than subsequent stages.

It is also recognized that the initial dewatering requirement for this mine comprises a risk element. But compared with bringing into production a virgin orebody, this project carries less risk of the unknown because it has been worked for many years and past operations are well documented.

It has been thought appropriate to consider minimum tonnage or grade of ore reserves and mineralisation which would still give a cash flow sufficient to return invested funds plus 100%. (Tax is ignored in this exercise). Table 15.3 gives calculations relevant to this consideration.

With given tonnage, this level of cash flow would be achieved by a head grade of 7.9 dwt/ton (compared with 13 dwt as certified).

With given grade, this level of cash flow would be achieved by reserves of about 254,000 tons (compared with 493,000 tons as certified).

In this exercise, full capital costs have been taken into account, and gold grade has been taken at the average for the 593,000 tons of combined ore and mineralisation (after dilution) as certified.

In practice, if it was found that the orebody was limited in tonnage to, for example, the 254,000 tons necessary to meet the 100% surplus parameter of this exercise, capital cost would be reduced to cope with a smaller annual mine output and mill throughput; and the gold grade of recovered ore would more likely approximate the higher grade attributed by Willsteed/W.G.M. to the "drill indicated" ore.

The above quoted tonnages and grades are based on a gold price in 1978 of $A150 per ounce. For a gold price in 1978 of $A130 per ounce, the relative figures are 311,000 tons, or 9.1 dwts/ton; and for a gold price in 1978 of $A175 per ounce, the relative figures are 206,000 tons, or 6.7 dwt/ton. This tonnage and grade minima should be viewed in light of the comments on Ore Reserves and Mineralisation in Section 2.2.2 above.

TABLE 15.1

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BEACONSFIELD FEASIBILITY REPORT - JULY, 1975.

PROJECT CASH FLOW - BEFORE TAX.

Years from Start of Work on Site	1	2	3	4	5	6	7	8	9
Mill Throughput - (tons	-	-	74,000	100,000	100,000	100,000	100,000	100,000	19,000
Au Grade of Headfeed (20% dilution, dwt/ton)	-	-	13.0	13.0	13.0	13.0	13.0	13.0	13.0
Au Mill Recovery (%)	-	-	82.5	82.5	85.0	85.0	85.0	85.0	85.0
Au Recovery in bullion and Cu Concentrate (ozs.)	-	-	39,680	53,620	55,250	55,250	55,250	55,250	10,500
Revenue from all metals - (per ton headfeed)	-	-	$84.47	$84.47	86.9	$86.9	$86.9	$86.9	$86.9
Revenue from all metals - ($/000)									
Total	-	-	6,251	8,447	8,690	8,690	8,690	8,690	1,650
less delay in receipts	-	-	(220)	-	-	-	-	-	220
= Revenue receipts in the year	-	-	6,031	8,447	8,690	8,690	8,690	8,690	1,870
less Operating Expenses	-	-	2,562	2,809	2,809	2,809	2,809	2,809	600
= Operating Surplus (before depreciation and Tax)	-	-	3,469	5,638	5,881	5,881	5,881	5,881	1,270
Capital Expenditure									
- Dewater and Confirmatory Drill	(1,162)								
- Complete Mine Development		(2,031)							
- Balance Mine Plant		(315)							
- Surface Plant & Buildings		(223)							
- Mill		(2,511)							
- Capital Costs in Year 3			(303)						
- Lump Sum Contingency		(235)							
Total Capital	(1,162)	(5,315)	(303)						
Cash Flow before Tax	(1,162)	(5,315)	3,166	5,638	5,881	5,881	5,881	5,881	1,270

D.C.F. in excess 50%. Present Value discounted at 15% = $9.9 million; discounted at 20% = $7.5 million.

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Table 15.2

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BEACONSFIELD FEASIBILITY REPORT - JULY, 1975.

PROJECT CASH FLOW - AFTER MAXIMUM TAX.

Years after Start of Work on Site	1	2	3	4	5	6	7	8	9	
Operating Surplus before working capital and depreciation and Tax (ex Table 15.1)	-	-	3,689	5,638	5,881	5,881	5,881	5,881	1,050	
less minimum rate of Depreciation	-	-	1,100	1,100	1,100	1,100	1,100	1,100	180	See Note 1.
= Maximum Taxable Project Income	-	-	2,589	4,538	4,781	4,781	4,781	4,781	870	
Tax at 45%	-	-	1,165	2,100	2,150	2,150	2,150	2,150	400	See Note 2.
Project Cash Flow after maximum Tax	(1,162)	(5,315)	2,001	3,538	3,731	3,731	3,731	3,731	870	

D.C.F. = 39%

Present Value discounted at 15% = $4.6 million; discounted at 20% = $3.1 million.

Note 1. If gold revenue becomes taxable, tax may be less than shown due to:

a. phasing-in period for incidence of tax on gold revenue;

b. some other benefit to gold miners compared with general mining;

c. earlier incidence of depreciation due to "life of mine" calculation.

Note 2. Gold revenue is not presently taxable.

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Table 15.3

BEACONSFIELD FEASIBILITY REPORT - JULY, 1975.

CALCULATIONS RELATING TO CAPITAL RETURN PLUS 100%.

1. Tonnage necessary, with gold grade as given:

To recover Capital Cost of $7 million
plus 100% 7
$14 million

Gold Price of:	$A130	$A150	$A175
Operating Cash Surplus -			
Year 1. ($'000)	2,895	3,689	4,681
Year 2. ($'000)	4,566	5,638	6,979
	7,461	9,327	11,660
Balance of $14 million	6,539	4,673	2,340
Operating surplus per ton in Year 3 onwards	$47.76	$58.81	$72.62
∴ Tons to earn balance	137,000	80,000	32,000
Add tons in Years 1 and 2			
= Total Tonnage required	311,000	254,000	206,000

2. Gold Grade necessary, with tonnage as given:

To recover: $14 million

Required Operating cash surplus per ton = $14,000,000 / 593,000 tons

= $24/ton

Add operating cost per ton (say) $30

$54

less Copper contribution $ 4

= Required Gold Contribution $50 per ton

	Dwt/ton Required	
	Recovered Gold	x 100/85 = Headfeed Grade
At Gold price $130/oz. = $6.50/dwt	7.7	9.1
At gold price $150/oz. = $7.50/dwt	6.7	7.8
At Gold price $175/oz. = $8.75/dwt	5.7	6.7

16. ENVIRONMENTAL CONSIDERATIONS.

Submission of an Environmental Impact Study to the Department of the Environment in Tasmania is a necessary prerequisite for the Beaconsfield project to proceed.

Some relevant points of interest taken from a draft Study are as follows.

1. Loading of Copper/Gold Concentrate.

   Existing road and shipping facilities will be used. Concentrate of approximately 2000 tonnes per annum will be shipped either from Beauty Point or from Launceston.

2. Power Generation and Transmission.

   The H.E.C. will be requested to provide 2500 KVA which will be utilised partly at the mine site and partly at the processing plant. One substation, and several transformer stations will be required in the vicinity of the mine and mill sites. It is understood that the H.E.C. plans to supply this power from a source near Bell Bay and to bring it by underground cable across the Tamar Estuary, thence by overhead line to the site.

   A diesel generating station with 3000 KVA capacity will be located near the mine shaft, and will be in continuous operation for two years (assuming commitment to the H.E.C. contract after completion of confirmatory drilling), thereafter on an occasional basis as required to provide power to pumping capacity over and above that required for normal mine dewatering, in periods of major water inflow into the mine.

3. Water Supplies.

   Water supply for the underground operations and for hygiene purposes at the mill site will be drawn from a fresh water supply installed by the local Authority (i.e. Waters & Rivers Commission).

   Water for the processing plant (estimated at between 50,000 and 100,000 gallons per day) would be drawn from a surface dam

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fed by the mine dewatering pumps. Mine water pumping is estimated in the range of 2 - 3 million gallons per day.

Controls will be maintained on the pH level of mine water, also on minerals in solution and other biochemical constituents

4. Waste Products.

For a mining operation of 100,000 tonnes per annum, approximately 50,000 tonnes of deslimed residues will be disposed of underground. This material would be the coarser fraction of the mill tailings and would comprise mainly quartz dolomite.

Approximately 35,500 tonnes of waste produced per annum (representing the slime fraction of tailings) would be stock-piled in an appropriate tailing dam. The possibility exists of utilising existing limestone pits for mill tailings disposal in which case no interference with surface topography will be made. If it becomes necessary to store mill tailings above ground, regeneration and tree planting would be an integral part of the tailing dam design.

Approximately 12,500 tonnes of cyanide tailings which would carry significant gold values, would be stockpiled for further processing, subject to metallurgical research.

An evaporation pond would be provided to store the cyanide tailings from mill operations. The quantity of this product has been calculated at approximately 3000 gallons per day at a concentration of .25% NaCN. An adequate area would be provided to ensure that this quantity of water would evaporate or seep into the soil. The location of this evaporation pond would also be such that if any break occurred it would still be empounded by an earthen bank to obviate any entry into the estuarian waters.

The other waste products would be minor dust from the coarse crushing section of the mill plant. To obviate any dust emission to the atmosphere and because of the potential value of this dust, extractors would be incorporated in plant design.

5. Affect on Wildlife and Local Amenities.

There is no wildlife in the immediate vicinity of Beaconsfield and nothing in the proposed project would interfere with any wildlife in the bushland surrounding Beaconsfield or on the Tamar estuary.

Remains of two old winder houses and of the old boiler house building exist near the top of Harts shaft. Other than the historical value of these surface structures, which it is proposed to preserve, there is no significant scenic or recreational value in the site.

Beauty Point, which is approximately five miles from Beaconsfield, is a recreational/sporting area but nothing in relation to the development would detract from the facilities at Beauty Point.

6. Traffic Generation.

Currently, up to 200 trucks a day carrying payloads in excess of 30 tons traverse the main road through Beaconsfield.

It is anticipated that approximately 14 truckloads or ore will be transferred per day from the mine to the processing plant. This will where possible be confined to daylight hours, with occasional minor transport between 4 p.m. and midnight in case of emergency.

Of the 90 - 100 employees on the project, approximately half will be engaged on day shift. This will involve 30 - 40 cars, used mainly between 7 - 8 a.m. and 4 - 5 p.m. The balance of employees would be spread between the mine location and process plant location, and therefore the number of cars operating outside of the above hours will be relatively minor at each location.

Overall, the project will not generate any significant increase in road traffic usage.

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7. Noise Level.

At the mine site, the main sources of noise will be the compressor station, the tipping of ore from skips into a surface ore bin, discharge of ore from the bin into trucks, and the diesel power generating station.

The compressor station will operate 24 hours a day, involving intermittent cutting in of compressors. Noise insulation of the compressor building will be undertaken if necessary.

Ore haulage to the surface will involve approximately 200 dumps per day, occurring mainly on the day shift. There will be approximately 14 discharges of ore from the surface bin into trucks, mainly during the day shift with occasional operation between 4 p.m. and midnight.

If necessary, the siting and housing of the diesel generating station will be especially arranged to keep noise to a minimum level.

Minor disturbances will occur when explosive charges are detonated from underground operations. Firing in relation to development headings would take place at either mid-shift or end of shift (i.e. at four-hourly intervals between 8 a.m. and 12 midnight).

Underground explosions for extractive operations would be confined to the end of shifts.

Approximately 8 firings per day for development headings and four firings per day for extraction operations would be involved. With mining operations and development at a depth exceeding 1500 ft., no significant vibration disturbance of surface structures is expected.

Noise from the process mill site would be occasional major noise from the crushing plant mainly in daylight hours, with noise from electric motors and grinding plant on a continuous basis.

The location of the process plant is however unlikely to be near to existing residential areas.

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8. General.

Generally, environmental factors are not expected to create any major problems, and solution of any problem areas is expected to be facilitated by the local desire to see the Beaconsfield goldmine back in production.

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17 INFRASTRUCTURE.

17.1 Industrial Development of the Region.

The geographic location of Beaconsfield is a major favourable factor which minimises the infrastructure requirements for the project.

The Beaconsfield township itself is a mainly agricultural community with 1,000 people, and supports no secondary industry.

However the city of Launceston, located 26 miles to the south-east, has a population of 62,000 and has a balance of agricultural and industrial undertakings. Within the major manufacturing industries located in Launceston are Repco, which manufactures automotive parts for car companies in Australia and employs 600 people; and Phoenix Foundry, a subsidiary of Johns-Waygood, which is a long-established foundry with a steel processing works. This company manufactures castings as well as fabricating mine processing plant and undertakes general steel fabrication.

Launceston also supports a number of textile plants such as Coats Patons Aust. Ltd., James Nelson Aust. Pty. Ltd. and Kelsall & Kemp Tas. Ltd. The implications of these textile plants is that these companies, apart from the process workers, employ a fair range of artisans such as welders, fitters, electricians, etc. who could constitute a source of skilled labour requirements for the Beaconsfield undertaking.

Launceston is the centre for the largest aluminium plant in Australia, located at Bell Bay on the Tamar estuary, which is directly opposite the Beaconsfield location. Comalco Aluminium employs approximately 1,200 people, and the plant produces 94,000 tons of primary aluminium per year.

B.H.P.'s subsidiary, Tasmanian Electrical Metallurgical Co. Pty. Ltd. (Temco), is also located at Bell Bay and produces 75,000 tons of ferro manganese per year. Temco is currently planning a major expansion into ferro silicon. The current labour force at Temco is approximately 600.

17.2 Availability of non-mining Labour and of supporting Services.

The importance of the Aluminium and Ferromanganese industries is that they provide a source of process operators with similar skills to that required for the metallurgical processing required for Beaconsfield.

Approximately 30% of Tasmania's manufacturing industry is concentrated in the Launceston area and, as a result of this, the area has developed the service industries such as electrical workshops, small machining and engineering shops, and the building construction services, required to support major industrial undertakings.

The whole area is well serviced by sealed roads, and the labour force is reasonably mobile within the area. This is exemplified by people living in the Beaconsfield region and working either at Bell Bay or Launceston. For the above reasons it would not be necessary to offer accommodation at Beaconsfield itself in order to attract labour. One of the fastest growing suburbs of Launceston is within the boundaries of the Beaconsfield Shire Council, and is only 20 miles by road from Beaconsfield township.

It is considered that approximately 40% of the labour requirements at Beaconsfield will be for process workers and qualified tradesmen.

Labour requirements apart, the concept in the development of the Beaconsfield operation is to provide for minimum project installations, sufficient for routine maintenance, and to utilise the already established service industries in the Launceston area to undertake major maintenance requirements.

17.3 Availability of Miners.

For both mine development and mine production it will be necessary to have skilled miners with some experience in similar type operations.

Tasmania has a strong mining industry and the latest statistics provided by the Bureau of Census and Statistics indicate that people employed in mining number 4,200. The range of mining covers both opencut and underground operations and the centre of mining activity is on the west coast of Tasmania.

The underground operations cover both open stope and cut-and-fill mining, and of recent years the west coast of Tasmania has been a leader in the development of mechanised techniques. The consequence of this is that the miners in the west coast are skilled and adaptable in a range of mining techniques and could be drawn-on to fill mining vacancies at Beaconsfield.

Although labour rates are attractive on the west coast, climatic conditions are arduous and it is considered that the moderate rainfall of Beaconsfield and the relative proximity to a capital city should be attractive compared with the west coast locations.

Within 60 miles of Launceston are located the mining developments of Rossarden and Storeys Creek which employed 200 miners and mill workers. These mines have in recent years been for a time on a care and maintenance basis due to depressed metal prices, and there has been a pool of labour available from this location. It is believed that these mines have a limited life.

The nature of the deposit at Rossarden and Storeys Creek (in that they are narrow vein operations of a similar type to the Beaconsfield orebody) would provide labour readily adaptable for Beaconsfield.

Rossarden/Storeys Creek mines could also provide the specialised labour (such as timbermen, platmen, bracemen, loco drivers, and winch operators) which is required for the Beaconsfield mine.

In this context the geographic location of Beaconsfield would be considered an attractive alternative to both the west coast and the Rossarden area.

18 GOVERNMENTAL CONSIDERATIONS.

18.1 Local and State Governments.

The re-opening of the old Tasmania Gold Mine at Beaconsfield is of significant interest and importance to the Local Government Authority and the State Government of Tasmania, both of which have been and will continue to be most co-operative in matters concerning the project.

Until 1968 the mineral leases for the project area were held by the Department of Mines, which expended considerable funds in exploring and drilling the area and which has a strong belief in the future of the project. The Department will regard a commercial undertaking as the culmination of its previous efforts and expenditure.

The project will provide direct employment for approximately 100 people and indirect employment for many more. It is located near Launceston, where unemployment is significantly higher than the State average.

Although on the opposite side of Launceston, it is sufficiently close to Storeys Creek/Rossarden to provide a practical source of alternate employment for miners and mill hands who have (or will in the future) become unemployed due to the run down of operations at those locations. The mines and mills at Storeys Creek/Rossarden are owned by Aberfoyle Ltd. They have in the recent past been on a care and maintenance basis because of depressed metal prices, and although both are currently in operation the ore reserve position of both mines will limit the life of these operations to only a few more years.

The Beaconsfield mine will have an output valued in excess of $8 million per annum, and will infuse into the Beaconsfield and Launceston Municipalities in excess of $1 million per annum in salaries, wages and local purchases.

The project can therefore reasonably expect co-operation and good will from the relevant Departments of the Tasmanian State Government and from the Beaconsfield Shire Council.

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18.2 Australian Government.

The sale of gold bullion, which will comprise in excess of 90% of the value of output, is arranged through the Gold Producers Association and is not complicated by any export or other controls administered by the Australian Government. Nor are gold and copper considered to be strategic minerals and therefore subject to policy actions by the Australian Government. Copper concentrates containing some gold may be sold to an overseas smelter, but no problem is envisaged in obtaining any necessary export permits for this product.

The Australian Government is however in a position to affect the project via any transfer of project ownership and control as part of financing arrangements, should such transfer be to a "foreign" company as defined by Australian Government legislation.

This is a matter of which notice must be taken in the event of discussions between Allstate and a "foreign" company.

19 MANAGEMENT OF PROJECT.

19.1 Organisation Structure.

The Beaconsfield project will be owned and operated through a Joint Venture to which Allstate Explorations N.L., Tricentrol Australia Limited and the incoming company will be parties.

The project assets will be owned jointly by the Joint Venture Parties under conditions specified in the Joint Venture Agreement. The Exploration Licence is presently held in the name of Allstate Tasmania Pty. Limited, and appropriate arrangements will be made for the transfer of this title to the Joint Venture interests, or for the holding of the title in trust for those interests. Mining leases will be taken out in the names of the Joint Venture Parties.

If it is desirable to have the project assets owned by a Joint Venture Company, in order to facilitate borrowing for the project, this will be possible.

19.2 Management Responsibility.

The project will be managed by one of the Joint Venture Parties in terms of an agreement satisfactory to all parties.

Allstate Explorations N.L. is presently manager of the project, and is capable of expanding its nucleus of executive personnel to handle all aspects of the project.

Future management of the project may however be subject to discussions with an incoming financing party. In the event of such party being a "foreign" company in terms of the Australian Government definition, management arrangements will have to be acceptable to the Committee on Foreign Takeovers, as one part of the overall proposition to be presented to that Committee.

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ALLSTATE EXPLORATIONS N.L.

E.L. 17/73

BEACONSFIELD GOLD MINE PROJECT

Material Additional to the Feasibility Report of August, 1975,

Comprising:

1. Revised Cash Flow Schedules with covering notes - dated February 1976.

2. Supporting Data re Estimated Grade of Ore Reserves and Mineralisation - February 1976.
   - to which is attached a calculation relating to stoping width and overbreak percentage for past workings.

3. Statistical Analysis on Gold Assays on which Reserve grade estimates are based - October 1975.

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A[illegible] EXPLORATIONS N.L.

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RE BEACONSFIELD PROJECT

Some notes and cash flow figures additional to the Feasibility Study of August 1975 have been prepared, and a copy is attached.

The figure revisions, compared with the Feasibility Study cash flows, include the following:

1. Gold price at US$150 per ounce which, converted at 1.26 = A$119. This appears to be the generally accepted price currently used for feasibility projections commencing 2 - 3 years hence.

2. Inclusion of Tasmanian Government Royalty on the basis specially negotiated for this project.

3. In the "first revision" we maintain the previous bases of approximately 600,000 tons mill feed with headgrade 13 dwt. (These bases are further supported by the notes attached).

4. In the "second revision" we take account of the general comment made to us that the orebody can be expected to continue to at least 2500 feet depth, or say one million tons.

   Our second revision therefore allows for 1,200,000 tons mill feed, the main result of which is to allow for a production rate of 150,000 tons per annum.

   We are satisfied that this higher production rate is achievable, although we have amended a number of annual production costs upward in arriving at the revised production cost per ton. We have for example allowed for a third working level in the mine.

   We have also, in this revision, added a broad-brush estimate of $900,000 to capital costs (including $270,000 for inflation on the extra items included). Details of these figures can be advised as required.

5. Also in this second revision we have allowed for a reduced average grade in the orebody on the arbitrary assumption that the second 500,000 tons of ore may show a grade of only 10 dwt per ton. This, combined with the previous grade for the previous ore, would give an average ore grade of 12.8 dwt. and an average head feed grade of 10.7 dwt.

   We have not taken advantage of the cash flow benefit of attributing the higher grade to the earlier production, which could reasonable have been done.

6. Although we expect gold revenue to remain free of tax, we have shown an after-tax position for each revision.

   In calculating tax we have assumed that any introduction of tax on gold revenue would be (at worst) on the phase-in schedule recommended by the I.A.C. in its report of 1975. We have also used the current company tax rate.

   (We note that use of the phase-in schedule would make it desirable to spread write-off of capital expense evenly over the period of mine life as shown by drill indicated ore ahead (say 7 years); whereas otherwise it would probably be acceptable and adviseable to charge all expense up to completion of confirmatory drilling against first revenue, and spread the remaining cost over life of mine. Our figures are based on the latter approach).

A summary of the main figures resulting from the revisions is as follows:

		Annual Cash Flow	DCF %	Present Value @ 15%	Present Value @ 20%
1st Revision	- without tax	$4m	42%	$5.4m	$3.6m
	- after tax	$2.6m	32%	$4.0m	$1.7m
2nd Revision	- without tax	$5.1m	43%	$9.0m	$5.9m
	- after tax	$3.2m	32%	$4.2m	$2.5m

ALLSTATE EXPLORATIONS N.L.

FEBRUARY 1976.

Sheet 1.

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Gold at $150 US/ton
Copper at 700 Stg/ton

FIRST REVISION TO

BEACONSFIELD FEASIBILITY REPORT - FEBRUARY 1976

PROJECT CASH FLOW - BEFORE TAX

Years from Start of Work on Site	1	2	3	4	5	6	7	8	9
Mill Throughput - (tons	-	-	74,000	100,000	100,000	100,000	100,000	100,000	19,000
Au Grade of Headfeed (20% solution, dwt/ton)	-	-	13.0	13.0	13.0	13.00	13.0	13.0	13.0
Au Mill Recovery (%)	-	-	82.5	82.5	85.0	85.0	85.0	85.0	85.0
Au Recovery in Bullion and Cu Concentrate (ozs)	-	-	39,680	53,620	55,250	55,250	55,250	55,250	10,500
Revenue from all metals - ($) (Per ton headfeed)	-	-	67.8	67.8	69.75	69.75	69.75	69.75	69.75
Revenue from all metal - ($/000)									
Total	-		5,017	6,780	6,975	6,975	6,975	6,975	1,325
Less delay in receipts			(220)						220
= Revenue receipts in the year	-		4,797	6,780	6,975	6,975	6,975	6,975	1,545
Less Operating Expenses	-		2,562	2,809	2,809	2,809	2,809	2,809	600
= Operating Surplus (before depreciation & Tax & Royalty)	-		2,235	3,971	4,166	4,166	4,166	4,166	945
Royalty to Tasmanian Dept. of Mines - 2% Revenue			-	-	-	(138)	(138)	(138)	(26)
Capital Expenditure:									
- Dewater and Confirmatory Drill	(1,162)								
- Complete Mine Development		(2,031)							
- Balance Mine Plant		(315)							
- Surface Plant & Buildings		(223)							
- Mill		(2,511)							
- Capital Costs in Year 3			(303)						
- Lump Sum Contingency		(235)							
Total Capital	(1,162)	(5,315)	(303)						
Cash Flow before Tax	(1,162)	(5,315)	1,932	3,971	4,166	4,028	4,028	4,028	919

D.C.F. 42% Present Value discounted @ 15% = $5.4 million. Present Value @ 20% = $3.63 million

511801

FIRST REVISION TO
BEACONSFIELD FEASIBILITY REPORT - FEBRUARY 1976

PROJECT CASH FLOW AFTER TAX AS PER I.A.C. RECOMMENDATION.

Years from Start of Work on Site	1	2	3	4	5	6	7	8	9
Operating Surplus after Royalty & before Tax & Depreciation ($000) (Ex Sheet 1)	-	-	1,932	3,971	4,166	4,028	4,028	4,028	919
Less Depreciation			1,964	802	802	802	802	802	802
Project Income before Tax			(32)	3,169	3,364	3,226	3,226	3,226	117
Tax payable as per I.A.C. Schedule @ 42½% Company Tax			-	673	1,072	1,371	1,371	1,371	49
Project Profit	-	-	(32)	2,496	2,292	1,855	1,855	1,855	68
PROJECT CASH FLOW	(1,162)	(5,315)	1,932	3,298	3,094	2,657	2,657	2,657	870

D.C.F. 32% Present Value @ 15% Discount = $4.02 million. Present Value @ 20% = $1.71 million

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Gold at $150 US/oz
Copper at 700 Stg/ton.

SECOND REVISION TO
BEACONSFIELD FEASIBILITY REPORT - FEBRUARY 1976
PROJECT CASH FLOW - BEFORE TAX

Years from Start of Work on Site	1	2	3	4	5	6	7	8	9	10	11
Mill Through-put -(tons)	-	-	74,000	120,000	150,000	150,000	150,000	150,000	150,000	150,000	106,000
(Mine Insitu Grade 12.8 dwts/ton)											
Au Grade of Headfeed (20% dilution, dwt/ton)	-	-	10.7	10.7	10.7	10.7	10.7	10.7	10.7	10.7	10.7
Au Mill Recovery (%)	-	-	82.5	82.5	85	85	85	85	85	85	85
Au Recovery in bullion and Cu Concentrate (ozs)	-	-	32,560	52,800	68,250	68,250	68,250	68,250	68,250	68,250	48,230
Revenue from all metals - ($) (per ton headfeed)	-	-	56.4	56.4	58	58	58	58	58	58	58
Revenue from all metals -($/000) Total	-	-	4,173	6,768	8,700	8,700	8,700	8,700	8,700	8,700	6,148
Less delay in receipts			(200)								200
= Revenue receipts in the year	-	-	3,973	6,768	8,700	8,700	8,700	8,700	8,700	8,700	6,348
Less Operating Expenses	-	-	2,562	3,120	3,450	3,450	3,450	3,450	3,450	3,450	2,438
= Operating Surplus (before depreciation, Tax & Royalty)	-	-	1,411	3,648	5,250	5,250	5,250	5,250	5,250	5,250	3,910
Royalty to Tasmanian Dept.of Mines	-	-	-	-	-	(174)	(174)	(174)	(174)	(174)	(122)
Capital Expenditure:											
- Dewater & Confirmatory Drill	(1,162)										
- Complete Mine Development		(2,031)									
- Balance Mine Plant		(315)									
- Surface Plant & Buildings		(223)									
- Mill		(2,911)									
- Capital Costs in Year 3			(803)								
- Lump Sum Contingency		(235)									
Total Capital	(1,162)	(5,715)	(803)								
Cash Flow before Tax	(1,162)	(5,715)	608	3,648	5,250	5,076	5,076	5,076	5,076	5,076	3,788

D.C.F. 43% Present Value discounted @ 15% - $9.02 million; Present Value @ 20% = $5.88 million.

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SECOND REVISION TO

BEACONSFIELD FEASIBILITY REPORT - FEBRUARY 1976

PROJECT CASH FLOW - AFTER TAX

AS PER I.A.C. RECOMMENDATION

Years from Start of Work on Site	1	2	3	4	5	6	7	8	9	10	11
Operating Surplus after Royalty & before Tax & Depreciation ($000) (Ex Sheet 1)	-	-	608	3,648	5,250	5,076	5,076	5,076	5,076	5,076	3,788
Less Depreciation			1,886	748	748	748	748	748	748	748	578
Project Income before Tax			(1,278)	2,900	4,502	4,328	4,328	4,328	4,328	4,328	3,210
Tax payable as per I.A.C. Schedule @ 42% Company Tax			-	548	1,913	1,839	1,839	1,839	1,839	1,839	1,364
Project Profit	-	-	(1,278)	2,352	2,589	2,489	2,489	2,489	2,489	2,489	1,846
Project Cash Flow	(1,162)	(5,715)	608	3,100	3,337	3,237	3,237	3,237	3,237	3,237	2,499

D.C.F. = 32.5%

Present Value @ 15% = $4.2 million

Present Value @ 20% = $2.47 million.

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BEACONSFIELD GOLD MINE PROJECT

Notes on Supporting Data re Estimated Grade of Ore Reserves and Mineralisation - February 1976

The tonnage and grade calculations of T.V. Willsteed/Watts Griffis and McOuat dated May 1974, have formed the basis of Allstate's Feasibility Report of August 1975.

The following notes refer to a point not made specifically in that Report i.e. the relevance in reserve estimating of the ore grade of the previous 600 ft. depth of workings, which was outside the influence of any secondary enrichment zone and which could well be comparable to the grade of the orebody extension.

The Secondary Enrichment Zone

From a combination of the grade and the recorded nature of ore removed from the old Tasmania Mine, it appears that the zone of secondary enrichment did not extend much beyond a depth of 400 - 500 ft. (Refer Noldart, Ref. 1/6A.)

From the upper levels (i.e. 815 ft. level and above), recovered values were of the order of 25 dwt. per ton (1877 - 1896) and 20 dwt. per ton (1896 - 1903).

An increasing presence of primary sulphides below the 400 ft. level is recorded with a "considerable proportion of the gold intimately associated with the sulphides".

Comparability of Previous 600 ft. of Workings.

Approximately 550,000 tons was produced from the mine in the last eleven years of its operation (1903-14) which represents the last six levels of the previous workings. (Ref. 1/3 and 1/5 of Part B).

These levels were below the 815'level and this production is unlikely to have been affected by secondary enrichment.

Both Llewellyn and Cundy & Fawcett quote average gold recovery from this production as being in the order of 10 dwt. per ton.

In general terms the upper part of this mine segment yielded higher values per ton than the lower part, but the following factors must be taken into account:

a) Gold orebodies arevariable in grade concentration.

b) It is known that there was a major discrepency between yield and assay for the 1914 tributors' production; and that sampling of the 1500 ft.level over 940 ft.of strike length showed 13 dwt/ton, a significant increase over sampling from the same 940 ft.of the previous level.

c) Drill intersections of the orebody extension indicate that this increasing grade continues into the extension.

It therefore does not seem unreasonable to take the previous 685 feet of orebody as having some relationship to the next 500 feet.

The insitu ore grade of this previous segment can be calculated by reference to several well established data given in past reports, as follows:

1) Sampled Grade

The sampled grade on levels 915' to 1500' (i.e. insitu grade without dilution and recovery factors) is calculated from the past company's detailed records at 0.81 ozs/ton.

2) Work-back from Recovered Grade

Recorded gold recovery over the years 1903 - 1914 was approximately 10 dwt/ton.

Factored for mill recovery and dilution, insitu ore grade can be calculated.

(a) Mill Recovery:

Heathcote's report of 1913 (Ref. 1/4, Part B) refers to the grade improvement to 13 dwt/ton on the 1500' level, and adds:

"This improvement is encouraging but as the average stoping width in this mine is 50% greater than the assay width, and allowance has to be made for loss in tailings of 1.5 dwts per ton crushed, these values are not payable."

On this basis, Heathcote was anticipating or achieving a head grade from the 1500' level, after dilution of $13.0 \times \frac{1}{1.5} = 8.67$ dwts/ton.

The loss anticipated in recovery was 1.5 dwts/ton, which gives recovery at $\frac{(8.67 - 1.5)}{8.67} \times 100\%$

$$\therefore \text{Recovery} = \frac{7.17}{8.67} \times 100\%$$

$$= \underline{82\tfrac{1}{2}\%}$$

An 82½% mill recovery seems high in view of the limited milling techniques then available, but is used in this exercise because it gives a conservative result.

(b) Dilution:

The extent of dilution indicated by Heathcote is 50%.

However records available to us relating to the last ten years of the company workings, provide data which indicates that 30% dilution was more likely.

(From tables on Ref. 4/6 in Part B. of Feasibility Study, mean stoping width of 8.3 feet compares with mean assayed width of 6.45 feet, giving overbreak of 29%; while overbreak calculated on a volume basis as attached gives a figure of 28%).

(c) Calculated Grade:

For a range of dilution from 30% to 50%, the calculated grade is as follows:

(i) 30% Dilution:

$$\text{Headgrade} = 10 \text{ dwt} \times \frac{100}{82.5} = 12.1 \text{ dwt}$$

$$\text{Oregrade} = 12.1 \text{ dwt} \times \frac{1.3}{1.0} = 15.7 \text{ dwt}$$

(ii) 50% Dilution:

$$\text{Oregrade} = 12.1 \text{ dwt} \times \frac{1.5}{1.0} = 18.2 \text{ dwt.}$$

3. Summary re Grade

Ore grade and mineralisation grade for the orebody extension from 1500' to 2000' depth at Beaconsfield is thus the subject of three assessed or calculated figures.

(a) Detailed back sampling by the old operating company, from all levels from 915' to 1500' inclusive, averages 16.2 dwts per ton.

(b) Calculated ore grade as above, relating to the previous eleven years of production, gives 18.2 dwts per ton, or 15.7 dwts per ton. (depending on assumed dilution factor).

(c) Ore grade as calculated by Willsteed/Watts Griffis & McOuat in 1974, was 15.6 dwts. per ton.

In calculating ore reserves, Willsteed/W.G.M. took cognisance of the assay data on the 1500' level, plus the assay data on 1370' level for the undeveloped length of the 1500' level.

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The zone of influence attached to each drill hole is given schematically in Figure 5/1 Part A of the Feasibility Report, and the influence of the 1500' level, where the grades in both F/W and H/W drive sampling has been given full recognition.

On this basis the Mine grade before dilution, for Indicated Ore and Possible Mineraliation, has been calculated at 15.6 dwts/ton.

BEACONSFIELD GOLD MINE PROJECT - ATTACHMENT

CALCULATION OF STOPING WIDTH 815 to 1250 FT. OF OLD WORKINGS AND OF OVERBREAK PERCENTAGE

(a) Summary.

This calculation is based on a comparison of volumes, with stoping width as the unknown factor.

The known factors are as given by past company records, with an assumed 60° dip to the orebody and a specific gravity of 12.5 cubic feet to the ton.

It has also been assumed that the recorded intrusion of barren rock was approximately 20 - 30 feet in vertical depth and was not included in production figures.

In summary:

Increment of Workings	Volume (c.ft.) calculated from Production Tonnage	Strike Length x Down Dip Depth x W. (Width)
815 - 915	1,358,000	164,830 W.
915 - 1000	1,057,000	115,040 W.
1000 - 1100	928,000	136,600 W.
1100 - 1250	1,687,000	210,870 W.
	5,030,000	627,340 W.

$$W. = \frac{5,030,000}{627,340} = 8.02 \text{ feet}$$

Assayed orebody width - mean over this increment = 6.25 feet

Overbreak = 1.77 feet

$$\text{Overbreak Percentage} = \frac{1.77}{6.25} \times 100 = 28.3\%$$

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(b) Calculations

	815 - 915	915 - 1000	1000 - 1100	1100 - 1250
Data * Tonnage extracted	108,671	84,550	74,222	134,885
* Strike Length (mean)	1,421	1,331	1,280	1,274
* Assay Width (mean)	6.25'	6.00'	5.87'	6.50'
* Vertical depth	100'	85'	100'	150'
Assumed Barren Rock				
* Length	nil	700'	200/250	300'
* Vertical depth	nil	19'	24/17	31
* Down-dip depth	nil	22'	28/20	36
* Length & down-dip depth	nil	15,400	10,600	10,800
Calculations				
* Volume broken (Tons x 12.5) ('000 tons)	1.358	1.057	928.	1.687
* Down-dip depth $\frac{\text{(Vertical)}}{(\text{Sin } 60^o)}$ (Feet - whole increment)	116'	98'	115'	174'
* Strike length x down-dip (Sq.Ft.- whole increment)	164,830	130,440	147,200	221,670
Less Barren Rock (Sq.Ft.)	nil	15,400	10,600	10,800
= Production Rock	164,830	115,040	136,600	210,870

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BEACONSFIELD GOLD MINE PROJECT

NOTES ON STATISTICAL ANALYSIS ON GOLD ASSAYS - OCTOBER 1975

Two separate analyses have been prepared on assay data from the old Tasmania Gold Mine.

Case 1. Assay data from the five drill hole intersections is available in 2 feet increments, over the whole length of the intersections.

In total there are 37 separate sample assays, from which is calculated a Mean Assay of 23.8 dwts/ton.

Applying a statistical analysis to these 37 readings, the probable error in these assays = $\pm$ 6.4 dwts/ton with 90% confidence limits.

Expressed another way there is a 90% probability that the gold content of the orebody, based on 5 drill holes, will fall in the range of 30.2 dwts and 17.4 dwts.

Accepting the lower limit there is a 90% probability that gold content of the orebody will not be less than 17.4 dwts/ton. This compares with W.G.M. calculations of 17.6 dwts/ton.

Case 2. A separate analyis has been made incorporating all the assay data on the 1500' level (both H/W and F/W), the winze data below the 1500' level, and the 5 drill hole intersections.

To give equal weight to the drill hole data (in 2 feet assays) and the 1500' level, the 1500' level assay have been weighted according to the strike length recorded, divided by 2 feet.

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For example, the first recorded assay in H/W lode on the 1500' level, shows 21 dwts over 100 feet. This has been accepted as 50 readings of 21 dwts. Where Nil assays or traces have been recorded, these have been take into the calculations at "no value", for the appropriate number of readings.

There are, on this basis, 591 readings of assay on the 1500' level and the winze below.

Combining these assays with the drill hole intersections, (the 591 assya on the 1500' level have a' high influence on the drill hole assays, which number 37), the following results are calculated.

The Mean Assay on 628 readings = 14.3 dwts/ton.

The probable error, with 90% confidence limits = $\pm$ 1.1 dwts/ton.

Again accepting the lower limits, there is a 90% probability that the assay value should not be below 13.2 dwts/ton.

The statistical method applied in both cases has been verified with a Statistician who confirms that the method has "statistical integrity". The only comment he makes is that it would be incorrect to compare the proabable error in Case 1 (i.e. $\pm$ 6.4) with the probable error in Case 2 ($\pm$ 1.1), because in Case 2 there has already taken place a normalising, or averaging out, in obtaining the assyas relating to strike length.

ALLSTATE EXPLORATIONS N.L.
OCTOBER, 1975.

ALLSTATE EXPLORATIONS N.L.

BEACONSFIELD GOLDMINE PROJECT.

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SECTION 1 - REPORTS RE PREVIOUS MINING AND DRILLING ACTIVITIES.

SECTION 2 - MINING TITLE.

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SECTION 3 - REPORTS ON METALLURGY OF DRILL CORE MATERIAL.

3/1 1974 Metallurgical Report on testing of drill core samples, by H.K. Wellington, Chief Chemist and Metallurgist of Tasmanian Department of Mines.

3/2 Summary Report on metallurgy at Beaconsfield, 1974-75, by Mr. Keith Parsons, Consultant Metallurgist.

3/3 Process flow sheet for ore treatment at Beaconsfield, by Mr. Keith Parsons, December, 1974.

SECTION 4 - MISCELLANEOUS PLANS AND REPORTS.

4/1 Main mine workings and strata by A. Montgomery, 1891.

4/2 Surface geology of Beaconsfield area, by A.J. Noldart, 1963

4/3 Beaconsfield Deep Lead Gravity Survey, Tasmania Department of Mines, 1964.

4/4 Plan of diamond drill holes, and Longitudinal Projection showing cross-section of drill holes (Middleton, Tricentrol Australia Limited, 1974).

4/5 Development plan of Tasmania Gold Mine, ex old Company records. (Large Scale).

4/6 Longitudinal section of Tasmania gold mine including tabulations of production and data re Level development and grade.

4/7 Longitudinal projection showing stoping extraction, Tasmania Gold Mine, October 1913.

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Part B. Reference 1/1.

REPORT ON THE GEOLOGICAL STRUCTURE OF THE BEACONSFIELD GOLDFIELD.

Geological Surveyor's Office,
Launceston, 10th July, 1891.

Sir,

I have the honor to report to you the results of my geological examination of the Beaconsfield District.

Maps.—In order to illustrate and render the report more intelligible, there are sent herewith the following maps:—Plan No. 1, a plan of the Beaconsfield Goldfield, showing the position of the principal mine workings and the strata in which they occur; Plan No. 2, a plan of the Tasmania mine and enclosing strata, the latter being shown on the horizontal plane of the main adit; Plan No. 3, a similar plan of the underground workings of the Little Wonder, Moonlight, and Amalgamated West Tasmania mines, and a section across the Cabbage-tree Hill, showing the order of superposition of the strata. These maps are compiled from the official maps of the sections held under lease and otherwise in the district, the plans of the underground workings of the mines furnished by the owners annually to the Inspector of Mines, and my own surveys. Mr. G. T. Eddie's valuable maps of the Tasmania mine were also used at times, and I have to acknowledge Mr. Eddie's great courtesy in allowing me the use of his original plan.

My first and principal examination of the district was in January and February, 1890, though several visits were made to it subsequently, hence the state of the mine workings shown on the maps is generally as seen at that time, later workings not having been in all cases plotted. The extensions of workings made since the beginning of 1890 have not, however, to my knowledge resulted in giving any further information as to the geological structure of the ground.

Previous Reports.—Two reports on this district have been previously made to the Government—the first by the late Mr. Charles Gould in 1866, entitled "Geological Surveyor's Report of the country near Ilfracombe, in the West Tamar District"; and the second in 1883 by Mr. G. Thureau, F.G.S., on the "Beaconsfield and Salisbury Mining District." Mr. Gould's excellent report deals with the general geology of the country, and more particularly with the large deposits of iron ore near Ilfracombe and at Anderson's Creek. A full narrative of the attempt to work these ores for iron is given by Mr. T. C. Just in the "Tasmanian Official Record, 1891." Analyses of the ore, the iron manufactured from it, and the slags from the smelting are given by Mr. R. M. Johnston in his "Geology of Tasmania," on page 28; and another analysis made for Mr. Gould by Mr. G. Foord, of Melbourne, is given in the "Monthly Notices of Papers and Proceedings of the Royal Society of Tasmania, 1866," page 84. To the full information as to these interesting ironstone deposits given in these papers, I have nothing to add further than the remark that the increasing use of chromium steel bids fair to render what was formerly the defect in the iron made from them, namely, its percentage of chromium, its principal merit at some future, but perhaps not distant date.

As regards the present mining district of Beaconsfield, gold does not appear to have been found in it till long after Mr. Gould's survey, and consequently his report mentions only the general features of the country, and does not go into further detail than giving the succession of the strata seen in the Middle Arm Creek (now called Blyth's Creek). As a general report it is most excellent, and should be consulted before the later and more detailed and restricted examinations by Mr. Thureau and myself are taken up by anyone desirous of studying the district.

Mr. Thureau's report deals more particularly with the portion of the country forming the Beaconsfield and Salisbury goldfields, and gives valuable information about the structure of the district as revealed at the time, and more especially as to the alluvial workings on the surface, and in the "deep lead" which runs along the eastern base of the Cabbage-tree Hill. As giving a description of the mines in their earlier stages of the principal features of the reefs, and of the Salisbury portion of the field which the present report does not deal with, it also should be read before this one is taken up.

The report which I now have the honour to submit to you is the result of a still more detailed examination of the Beaconsfield Goldfield, with the following objects in view:—(1.) The obtaining of more definite knowledge as to the relations of the various beds of country rock to each other and to the auriferous reefs, and their influence on the gold value of the latter; (2) The determination as far as possible of the position and effect on the reefs and the country of the various crosscourses or faults that disturb them; (3) The collection of further information as to the "deep-lead" or buried river channel running along the eastern base of the Cabbage-tree Hill; and (4) The noting in general of all facts connected with the geological structure of the district likely to be of interest and value either practically or scientifically.

General Topography and Geology.—Without going over the ground already traversed by the reports of Messrs. Gould and Thureau unnecessarily, it seems advisable to begin by recapitulating the principal facts as to the general topography and geology of the goldfield. The main feature in both respects is the low range known as the Cabbage-tree Hill, running N.W. and S.E. about two miles inland from the Middle Arm of the River Tamar, and approximately parallel to it. The hill is a little over two miles in length, and averages from 350 to 420 feet in height above sea level. The same range continues on to the south-east under the name of the Blue Tier, this and the Cabbage-tree Hill having once formed a continuous range, which has been cut into two parts by erosion of the deep gorge of Blyth's Creek, which now separates them. At its north-western end the Cabbage-tree Hill is separated by the Brandy Creek from rolling country, which forms the watershed between the latter and Anderson's Creek. From the Middle Arm the ground rises pretty evenly with a gentle slope to the base of the hill where the Town of Beaconsfield is situated. As the main street of the town is only about 100 feet above sea level, the slope from it seaward is so slight that the country may be called a plain. South of the Cabbage-tree Hill about two miles lies another hill known as the Blue Peaked Hill, and between them the ground is flat and somewhat marshy in parts. This flat extends up the Flowery Gully, an eastern tributary of Blyth's Creek, to a point about due east from the Blue Peaked Hill.

Several geological formations are represented in the district. The hills mentioned are all composed of hard metamorphic sandstones, slates, grits, and conglomerates, of probably Lower Silurian age. About a mile west of the northern end of the Cabbage-tree Hill old volcanic rocks are found, forming a large patch of serpentine country. The stratified Lower Silurian rocks are generally inclined at rather high angles of dip, and form some synclinal and anticlinal folds. Their general strike is about N.W. and S.E., corresponding with the long axes of the hills, the existence of the latter as hills being clearly due to the greater resistance offered to erosion by the hard sandstones and grits of which they are composed as compared with the softer slates skirting them. The distance to which the Silurian formation extends towards the Tamar cannot be accurately estimated as yet, as the surface is much covered with later deposits of gravels and clays. From the West Arm to the Middle Arm the shore is found to consist of sandstones, mudstones, and limestones of Carboniferous age, except at Beauty Point, where Tertiary basalts are found. The Carboniferous rocks cannot extend more than a short distance inland before the Silurian formation crops up from under them, but the junction of the two is obscured by more recent superficial deposits. These are of various ages, ranging from the early Tertiary to the Recent period, the gravels of the "deep lead" being probably of the former age, while the shallower surface gravels are more recent. The deep alluvial ground of Flowery Gully and the flat between the Cabbage-tree and Blue Peaked Hills may contain deposits belonging to the older Tertiaries as well as the recent ones visible at surface, and may perhaps in places also cover the Carboniferous formation.

The general history of the locality may be sketched thus:—Sediments of gravel, sand, mud, and calcareous matter laid down on the floor of a sea in the Lower Silurian period were hardened into grits, sandstones, slates, and limestones, crumpled into highly inclined folds, elevated into mountain ranges, and greatly worn away by sub-aerial and marine erosion in the immense interval of time intervening between their deposition and that of the later Carboniferous beds. The latter were deposited as shell-banks and beds of sand and mud at a much later date on the upturned edges of the older strata at a time when the surface of the land was relatively lower than at present, and the sea came well up to the flanks of the Cabbage-tree Hill, which at that time must have been an island or peninsula, as the Carboniferous beds are found now nearly surrounding it. No great contortion of the strata has taken place since these beds were laid down, as they still lie almost horizontal, and show no signs of metamorphism or strain due to pressure. However, between the date of their deposition and that of the early Tertiary deposits, elevation of the land must have taken place to a height probably quite 300 feet above the present level, as the next evidence of geological work which is met with in the district is the formation of the channel of the "deep lead." This is an old river channel, and the water that scooped it out must have run downhill to the sea; hence, as the bottom of the lead is proved by the Ophir Company's borings to be now 270 feet below sea level, it must at one time have been more than that distance vertically higher than its present position. This elevation of the land subsequently to the laying down of the Carboniferous beds very probably took place during the Mesozoic period, when the immense outflows of diabase greenstone, which are so prominent a geological feature, throughout the whole colony, were being emitted. This greenstone occurs very abundantly on both sides of the River Tamar from Middle Island up to Launceston. The channel of the lead was doubtless eroded deeper and deeper as long as the movement of elevation proceeded, but after a time, the ground remaining stationary or beginning to subside, it began to become filled up with deposits of gravel. A movement of subsidence now appears to have set in, for the old channel became more and more filled up. At one stage it appears to have been a swampy estuary or valley, as there is in it a deposit of black mud mixed with fragments of timber, leaves, and other vegetable remains. It is from this portion of the alluvial beds that specimens were obtained of fossil fruits (see Johnston's Geology of Tasmania, page 278), which enable the age of the lead to be certainly referred to the Older Tertiary (Palæogene) epoch. This lead is therefore contemporaneous with some of the oldest deep leads of Victoria and New South Wales. As subsidence went on the old channel at last became entirely filled up. There is reason to believe that the subsidence did not cease when the ground had reached its present level, but continued until the sea reached a point on the flank of the Cabbage-tree Hill at least 250 feet above the present tide-mark. The evidence for this is that we find at 250 feet above sea-level on the Moonlight Company's section (No. 349) heavy rounded water-worn stones and coarse gravel, and at the same height on the eastern slope of the range there are heavy deposits of similar gravel in Eastman and King's and Bruen's old workings. The deposits in these consist of boulders, heavy gravel, and clay, all more or less horizontally bedded or dipping slightly seaward. Their being found on both sides of the hill up to a fairly constant level creates a great likelihood that they are remnants of a large body of gravel which once surrounded the hill up to that level, but has since been almost entirely removed by erosion. The level of the highest occurrences of heavy gravel, therefore, probably represents the sea level at the end of the last period of subsidence. From the evidence obtainable in other parts of the colony it is known that the extensive outflows of basalt which cover a large area in the northern districts, forming the best farming land, took place towards the end of the Palæogene or Older Tertiary period, and it is possible that the disturbance caused by these resulted in elevating the land again to about its present level. During the progress of this elevation (a slow movement in all probability) the deposits of gravel which had accumulated round the Cabbage-tree Hill have been swept away nearly down to the bed-rock, leaving only the remnants above mentioned and occasional gravel mounds on the plain to attest their former existence. There is some of the basalt just mentioned at Beauty Point and at Point Ellingham, in the immediate vicinity of the Beaconsfield District, and a little further away, at Lefroy and Back Creek, it is again found, and this time covering auriferous deep leads. There is a possibility that the Beaconsfield deep lead may also run under the basalt towards Ilfracombe.

From a consideration of this history the obscure and patchy character of the alluvial deposits will be understood, and it will be seen that the configuration of the present surface can afford little, if any, indication of where the deep ground lies.

Deep Lead.—While this history is fresh in mind, it is well to finish our consideration of the "Deep Lead." It has long been known that along the eastern base of the Cabbage-tree Hill there exists deep alluvial ground, this being proved by numerous shafts and prospect holes. The main street of Beaconsfield (Weld-street) is almost fairly upon the centre of this ground. The principal workings have long since been abandoned, and it is hard now to fix the sites of even some of the old shafts with any certainty. Such as I could determine are shown on Plans Nos. 1 and 2. All these old workings were either on the "high reef" or sloping edge of the lead, or on false bottoms. The lowest workings were 112 feet from the surface on a false bottom of black ligneous clay. None of the workings have yet reached the "gutter" or bottom of the old river channel. I have not been able to obtain much really reliable information about the old mine workings, the accounts given by various presumably well-informed persons being very conflicting. It would seem, however, that the workings on the "high reef," that is, on the Silurian bed-rock forming the sloping sides of the channel, were fairly payable, and that there was also a good deal of gold on the black false bottom. No good section of the lead has yet been obtained. The diamond drill bores recently executed by the Ophir Company do not give a satisfactory section, as the greater part of the boring was done without bringing up any solid core, and consequently the exact nature of the strata passed through is somewhat doubtful. Two bores were put down, marked D and E on Plan No. 1, and Ophir bores No. 1 and 2 on Plan No. 2, to a depth of 375 feet and 286 feet respectively. The following section of the deeper bore was given by Mr. Bowen, the Director of the Company, who superintended the boring:—

"First bore, 375 feet, passed through from surface sandy clay with gravel to 40 feet, then pug 200 feet, then gravel containing gold at two ounces to load, then boulders to 300 feet intermixed with clay, then black clay, 50 feet, then decomposed timber, &c., then wash to bottom." Elsewhere in his report Mr. Bowen says that at the bottom of this bore there were 9 feet of wash with gold at the rate of 4 ounces to the load. The second bore bottomed on limestone at 286 feet, and had "about 12 feet wash, giving returns at 2 ounces to load." If these results are reliable the richness of the lead would be phenomenal.

The Ophir shaft, which was sunk to a depth of 300 feet in the alluvial and bottomed on sandstone, ought to have given an excellent section of the lead; but I have not been able to obtain any more definite account of it than that it passed through a succession of beds of gravel, sand, and clay, occasionally containing a little gold. The surface of the sandstone bottom sloped to the north-east, showing the shaft to be on the south-western side of the gutter. Orchard's shaft (see Plans 1 and 2) struck limestone bottom at … feet, dipping south-westerly, and between it and the Ophir shaft the bores E and D strike bottom at … feet and 375 feet; hence the "gutter" is evidently close to bore D. Owing to the swelling nature of the ground the Ophir shaft has become twisted and more or less useless, and no work has been done on the … from it. The policy of sinking a shaft in the solid rock to a depth well below the lead, and then … ing out under it being adopted now by the Ballarat Company, is a much safer one than that of … mpting to sink in the drift itself, and much more conducive to the economical working of the gutter.

To the east of the Tasmania mine workings the lead evidently passes not very far from the Lefroy … The No. 4 and No. 5 levels of the Florence Nightingale mine were driven out into it, encountering … and gravel in the face, at depths of 270 and 330 feet. The Lefroy shaft itself seems to have been in … al material for about 70 feet, and then to have been sunk in soft clayey slate. The East Tasmania … and the workings of the Daliy's United mine prove that the alluvial channel keeps close in to the … of the hill going south-east from the Lefroy shaft, and it most probably hugs the foot of the hill right … Blyth's Creek in this direction. Alluvial material was passed through in the first 300 feet of the

Tasmania main adit, and for 412 feet in the mouth of the lower Cosmopolitan adit. Past this point I have not been able to trace it as yet to the south-east; but from the outcrops of solid bed-rock it almost certainly must pass under Blyth's Creek near the bridge on the road from Beaconsfield to Launceston. The northerly extension of the lead from the Ophir bores probably runs N.W. and then N.N.W. out under Brandy Creek. Mr. Thureau in his report gives a sketch map showing two leads coming together from the N.W. and S.E. at the Ophir ground, and then breaking out north-easterly towards the sea. There is deep ground in this latter direction, for J. T. Allen bored some 170 feet at a spot approximately marked on Plan No. 1, and was forced to stop without reaching the bed-rock by coming upon a bed of hard conglomerate boulders. The drives from the East Tasmania and Dally's United shafts, the shaft of the old Duke of Edinburgh mine, and the occurrence of solid bed-rock near the intersection of Weld and Crowther streets, together with several wells and shallow borings put down from time to time by various persons, have, however, pretty conclusively proved that there is no outlet for the lead in this direction, and Allen's bore must be in a different lead. It is probably connected with a run of deep ground found to the eastward of the Beaconsfield Public School, and perhaps this second lead may prove to be a branch of the Ophir one.

The further tracing of the lead will probably be left until such time as the works now in progress will have demonstrated the payableness or otherwise of the part about to be worked. Should the result be favourable the extensions of the lead will be of very great importance. The tracing will be rather a slow and expensive matter, as it will require series of lines of borings across the most probable courses of the old channel.

As to the prospects of the gutter proving payable there is much difference of opinion. There is no doubt that the upper portions of the gravels have been auriferous, but a very general opinion appears to prevail locally that under the false bottom on which the deepest workings are situated there will be no payable gold. The grounds for this notion are difficult to imagine. Rightly or wrongly a belief is very prevalent in the district that the good results got from the recent Ophir bores, and from a previous bore called Orchard's bore on Plan No. 2, were not genuine,—in fact, that the bores were "salted." No good reason is ever given for this belief, which is a most unfortunate one for the district, as a mere suspicion of such a thing acts as an absolute bar to investment of capital, and I quite fail to see any reason why gold should not have been genuinely got from the bores. On the contrary, there are good *a priori* reasons for thinking that the lead ought to be richly auriferous.

The channel has been eroded through the auriferous Silurian rocks. We know that the reefs existed before this erosion, for the Tasmania reef, as above stated, has been found to be cut through by the lead in the most easterly workings. It is clear, then, that a large piece of this reef has been cut out and sluiced in the old channel. But all the time that the latter was being eroded the surface of the Cabbage-tree Hill and its numerous contained auriferous veins was being also worn down, and in natural course the contained gold would find its way into the channel. The fact that the gravels filling the lead are all derived from the Silurian grits and sandstones disposes of any objection that perhaps at the time of formation of the channel the adjacent hill slopes were covered with coatings of more recent rocks, of the Carboniferous period for example. If these had existed they would have contributed their share of the gravels. Now, all the time that the river channel was being cut down, which must have been considerable, it acted as a sluice to concentrate the gold brought down from the adjacent hillsides. The later gravels derived from these same hillsides have been proved to be auriferous, then why not also the earlier ones? Further, it may be observed that while the later gravels were deposited during a period of subsidence when the channel was slowly filling up, and therefore the gravel once deposited was but little disturbed afterwards, in the earlier stage the gravel never could accumulate, but was swept out by the force of the stream which was cutting its way down through the bed-rock. While this action must doubtless result in carrying a great deal of gold down the river, at the same time it affords much greater opportunity of concentrating the gold into the bottom of the gutter than in the subsequent stages when the latter was filling up. In my opinion, therefore, there is every reason to expect that the bottom of the lead will be rich. Like all such leads it is to be expected to vary very much in quality, and no surprise and alarm should be felt if the first truck-load of wash extracted from it does not realise expectations. Patient work may be required before rich deposits are found. This is no news to anyone knowing anything of deep leads, but may be useful to other investors.

Cabbage-tree Hill.—Turning our attention now to the rocks of the Silurian system, we find that it is in those of the Cabbage-tree Hill that the most important gold discoveries have been made. Auriferous reefs have also been found at the Blue Tier, but the mines here have practically been at a standstill for a long time past. Not having examined this portion of the field, this Report will not refer to it further. The ridge of the Cabbage-tree Hill is composed of coarse-grained grits or fine quartz conglomerates, alternating with beds of hard metamorphic sandstone. On either side of the hill softer slates and sandstones are found. The general strike of the formation may be taken as that of the ridge, N.W. and S.E., though, as shown by the Plan No. 1, this is not uniformly preserved. Though a certain amount of folding of the strata can be seen in the crown of the ridge (see section) the general dip is to the north-east, the average angle of dip lying between 45° and 75°. The lowest beds of the series are therefore those seen on the western side of the Cabbage-tree Hill. The plans and section show the succession of the strata. The bluish slate, which is the lowest rock seen, is found in the Britannia shaft and in some old shafts to the north-west of the Little Wonder workings. In the extreme end of the deep crosscut at 422 feet level from the Moonlight shaft a band of fine-grained black sandstone is encountered, which must lie between the slate and the next band seen on the section, namely, a very peculiar jointed, shining, graphite-like slate. This is made up of small fragments polished brightly and striated on every face, showing the results of extreme pressure. The black colour of this rock is due to carbonaceous matter. It is encountered again in the drive south-west from the Moonlight No. 3 or most westerly shaft, and in the mouth of the Little Wonder adit, though here somewhat altered by loss of its black colouring, also in the old Garfield shaft. Upon it lies a bed of soft clayey tenacious slate, locally known as "pug." This is found with the graphite-like slate in the mines just mentioned. The next beds in the ascending series are shown by the workings of the Little Wonder, Moonlight, Amalgamated West Tasmania, and Tasmania mines to be a succession of layers of metamorphic

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sandstones, grits, and conglomerates. There appears to be, as shown on Plan No. 3, a layer or series of layers of black grits overlaid by another series of similar beds of much lighter colour. The lower black grits are highly charged with carbonaceous matter. In a new crosscut now being driven east from the Moonlight shaft, and in the No. 5 level of the Tasmania Mine, west of the second crosscourse, there is a great abundance of a substance, to which I can give no better name than a soft coal, mixed up with the gravel and sand forming the grits and sandstones, and often thus preventing the material from cementing together into the usual hard stone, and lying between the layers and in the joints of the rock. Though all apparently containing a large percentage of earthy matter, pieces can be readily got which will take fire in the flame of a candle, and burn like charcoal, without flame, for some time after being withdrawn from it. Heated in a closed tube a little tarry matter is given off, but so little as to show that the substance must be almost all carbon. The purest pieces are bright and shining, very soft and friable, and burn to a white ash. From the way in which the carbonaceous substance is not only interspersed through the substance of the rock, but also through joints and fissures in it, I am in some doubt as to its origin, but think that most probably there was a great deal of organic matter among the sands and gravels when originally deposited as sediments, and that, during the processes of metamorphism to which the rock has been subjected, part of this has been volatilised as oil or tar through the joints in the rock, and afterwards been completely carbonised, while part remained as carbon where originally deposited in the sediments.

Overlying the beds of grit and conglomerate comes a large band of very hard black crystalline sandstone, seen very well between the two main cross-courses in the Tasmania mine. In parts this contains layers of coarser grit, and sometimes the grain is so coarse that the rock would be rather called a grit than a sandstone; still, on the whole, sandstone predominates in this portion of the strata. The differences in lithological character exhibited by different parts of the same bed of sediment often render it difficult to be sure of their identity. On the plan I have shown by similar marking the beds which appear to belong to the same horizon in the series, even though they do not appear to be altogether the same in character at different points. This black sandstone, for example, is more a grit towards Blyth's Creek, and also towards Brandy Creek. On the eastern side of the main cross-course in the Tasmania mine it is met with close to the fault in No. 6 level, and the diamond drill bore of the Phœnix Company went into it after passing through the reef. The lower position of this black sandstone with regard to the strata met with in the main body of the Tasmania mine workings is therefore well assured. The Phœnix diamond drill bore, the workings of the Tasmania mine, and the long adits of the Olive Branch, Bonanza, Leviathan, and Cosmopolitan Companies, give numerous excellent sections of the strata lying above the black sandstone. Three bands may be distinguished—the lowest, a white sandstone often containing numerous but very imperfect fossil casts; next, a dark bluish sandstone which forms a well marked band in the Phœnix bore, but owing to variation in colour is not always easily recognised in the other sections; and above this, a long series of small beds of light bluish, light grey, and yellowish sandstones, with thin partings of mudstone or slate. The higher (more easterly) beds become more and more slaty in character, and thin beds of impure limestone make their appearance. These limestone beds, however, do not as yet appear to be continuous over long distances, the limestones lately struck in the Ballarat shaft, at and below 250 feet, not apparently having extended into the Tasmania mine, but been changed to mudstone and sandstone. The strata lying between the eastern workings of the Tasmania mine and the next section, that seen in the East Tasmania diamond drill bore, may be seen in places on surface in the alluvial workings west of the Ophir shaft, and are still sandstones and slates. The workings of the East Tasmania bore, the Dally's United mine, and the East Tasmania mine nearly complete the section visible. The most notable feature in this portion is the thick band of limestone met with for over 500 feet in the bottom of the bore. This is again struck in the southern drive from Dally's United shaft, and probably is identical with the large band of limestone worked in Dally's quarry, on Blyth's Creek. Beds of slate, impure limestone, slate, and sandstone succeed the main limestone mass. The next known rock in the series is a bed of hard blue limestone met with in the bottom of the East Tasmania shaft. On this again lies an arenaceous, often calcareous slate, with bands of soft schist. From this point eastward the exact succession of the strata has not been revealed. A shaft to the north-west of the Police Station shows blue slate, and the Middle Arm Creek shows a few exposures of schist, sandstone, and limestone, from which we may conclude that the higher beds are a succession of these rocks. The whole formation is evidently of immense thickness, the section now given showing over 3000 feet of rock, all on one side of an anticlinal axis running somewhere to the west of the Cabbage-tree Hill. The beds seen in the Middle Arm Creek render it probable that the thickness is very much greater still.

A very hurried visit to the Blue Peaked Hill showed it to be composed of sandstones similar to those of the Cabbage-tree Hill, and it seems possible that the strata there may be those of the latter repeating themselves on the other side of an anticlinal axis. Against this supposition, however, is the occurrence of a large quantity of solid blue crystalline limestone at the head of the Flowery Gully, which has the same strike and the same north-easterly dip as the strata of the Cabbage-tree Hill. The exact similarity of the stone to that in Dally's quarry and the East Tasmania bore leads one to suspect some connection between these beds, but as yet none has been demonstrated. I hope to have an opportunity of running a section over the Blue Peaked Hill from Beaconsfield to the Flowery Gully caves in order to elucidate this matter. It is of practical importance, as a recurrence of the auriferous strata of the Cabbage-tree Hill would probably be accompanied with similar quartz veins.

Flexures of the Strata.—Though, on the whole, the strata seen in the section across the Beaconsfield field are dipping to the north east, there are several flexures in them which in places reverse the usual dip. As seen in the section these flexures lie under the crown of the ridge of the Cabbage-tree Hill. From Plan No. 2 it will be seen that the strata in the extreme western workings of the Tasmania mine are dipping south-westerly, and from Plan No. 3 it is seen that the south-westerly dip continues to be found throughout the Amalgamated West Tasmania Mine and in the workings of the Moonlight Mine east of the shaft. Throughout the Little Wonder workings, however, the strata dip to the north-east, as they do also in the Moonlight deep south-westerly crosscut. That there is more than one fold in the beds between the synclinal axis shown on plan No. 3 and the Tasmania Mine is seen from the section exposed in the Garfield Company's old adit. In this the strata are seen to dip towards the north-east for about 130 feet from the mouth; at 145 feet their dip is to the south-west, and continues so to about 320 feet, when they become much broken, and the dip is not clear. At the end of the adit, 447 feet from the mouth, the strata are again dipping to the north-east. Near the south boundary of Section 112, two shafts sunk by the Garfield and Little Wonder Companies show proof of further flexures, the dip of the beds in the former being south-westerly, and in the latter north-easterly. The eastern crosscut from the Moonlight shaft now being made will, if carried far enough, throw a great deal of light upon the folding of the strata under the ridge.

The only other deviation from the general north-easterly dip of any importance seen during my examination of the field was in the East Tasmania Mine, in the northern drive at the 100-feet level. One of the small limestone beds appears to be folded back upon itself where cut in a small crosscut, but the section is not long enough to show if this is more than a small local fold. It is mentioned, however, to show the possibility of further flexures being encountered in the strata in this direction.

Fossils.—I was exceedingly unfortunate in obtaining fossil remains from the auriferous formation, only getting a few broken and imperfect specimens of species already catalogued in Johnston's "Geology of Tasmania." Very good specimens of a species of *Orthis* were obtained by Mr. Davies from 242 feet in the new Tasmania main shaft, but the fossiliferous bed proved to be a very small one. Imperfect and much broken casts and impressions are not uncommon in the bed of white sandstone lying just east of the main cross-course in the Tasmania mine, and also found in the Cosmopolitan shaft. The most interesting organic remains found were the carbonaceous deposits above described as occurring in the Tasmania No. 5 level, west of the second large cross-course, and in the new eastern crosscut from the Moonlight shaft. It is noteworthy that the only specimen of a vegetable fossil yet got in this country, or, to the best of my knowledge, in Australia, in rocks of Lower Silurian age, was found at Beaconsfield, in the Cabbage-tree Hill grits, viz., *Licrophycus Tasmanicus* (see Johnston's Geology of Tasmania). Very few undoubted plant remains have anywhere in the world been got in rocks older than the higher members of the Upper Silurian system, though the occurrence of graphite in the Laurentian rocks of Canada is generally believed to be due to vegetable matter; hence the carbonaceous beds of Beaconsfield are of very great interest scientifically. I think I am not wrong in saying that these are probably the most ancient beds containing anything of a nature approaching to coal that have yet been discovered.

The fossil evidence as to the age of the auriferous rocks of Beaconsfield is scanty, and therefore somewhat unsatisfactory, but, such as it is, it points to their being of Lower Silurian age, or even older. The crystalline limestones among them have not yielded the fossil remains that might have been expected, though Mr. Gould noted the occurrence in these of imperfect remains of what were once probably corals. Mr. R. M. Johnston, in his "Geology of Tasmania," refers the limestones found at the head of the Flowery Gully to a "Primordial Calciferous Group" of probably Cambrian age. The lithological resemblance of these limestones to the comparatively adjacent ones of Dally's quarry and Dally's United mine workings, which have now been proved to overlie and be conformably bedded with the sandstones and grits of the Cabbage-tree Hill, is so strong as to lead one to feel nearly sure of their being of the same age—a conviction strengthened by their having the same strike and dip as the Beaconsfield beds. It seems to me not unlikely that the limestones of Railton, Tarleton, and Chudleigh are also contemporaneous. As, according to Mr. Johnston, the limestones of the Primordial Calciferous Group appear to immediately overlie the trilobite beds of Caroline Creek, it would therefore seem likely that the Beaconsfield auriferous rocks are contemporaneous with the latter, and should be referred to the Cambrian Period, which has yet to be done to establish clearly the relations of their various formations to each other.

Faults.—The rocks of the Cabbage-tree Hill are traversed by several faults which disturb the country very considerably, and, as they are of the greatest importance to the miner, I have devoted much attention to them and to the effects they have upon the lodes. The most important is that generally known as the "main crosscourse," running about N. 30° W., and heaving the Tasmania Reef a distance of about 240 feet. Where exposed in the workings of the mine this fault is found to be a well-defined fissure, with polished and striated walls. It is often as much as six feet wide, but varies a good deal, and is filled with crushed and slickensided masses of rock. There are often several parallel polished surfaces between the walls of the fault, forming false walls Everything gives the impression of repeated motion having taken place along this fissure at intervals of time. The striations on the slickensided surfaces are not always vertical, and are sometimes inclined at considerable angles, showing horizontal as well as vertical movement, but I was not able to detect anything certain as to the direction of the motion from these. Proof of repeated motion is seen in the occurrence of fractured quartz along portions of the fault, which must have been formed in it, and subsequently crushed and broken by pressure during a later displacement. Some of this quartz contains gold, and in places there is enough of it to have led to the quartz being stoped out and sent to the battery. It has been considered that this quartz has been mechanically broken off from the reef and carried down into the fault fissure; but after seeing the continuous sheets and strings in which it generally occurs, I cannot accept this explanation, and must conclude that the quartz has been formed by deposition in the fault fissure from solutions. Somewhat similar quartz is found in parts of the other fault fissures in the district, and occasionally carries gold. The fault is met with again in the Olive Branch adit and in the Bonanza adit, though in these it is not so clear as where opened up by the Tasmania Mine workings, and probably it extends for a long distance. Though the dip is to the westward, it is clear that the eastern is the downthrow side of the fault, which is therefore a "reverse fault." On driving westward through it, with the exception of a small patch of white sandstone passed through in the No. 2 level of the Tasmania Mine, all the country met with in the workings is dense hard black crystalline sandstone. On the east side of the fault this is found in the lowest workings of the mine, dipping under the white sandstone, and in the Phœnix diamond drill bore after passing through the reef. The downthrow of the eastern side is therefore clear, or would appear so at first sight. The same appearance would, however, be presented if the western side of the fault had been bodily heaved northward, and this I have come to believe is the true state of the case. As seen from the plans, there is a second crosscourse to the west of the one just described. The black sandstone is found in the Tasmania mine all between these two faults, but on driving westward through the second one grits and conglomerates are encountered, showing that though this fault also dips westerly the eastern is the downthrow side. Now, as the effect of a downthrow of the eastern side of the main fault would be to heave the reef to the north of the line of the portion on the western side, it would have been expected, in accordance with the law of faults, that the drives from the Golden Gate shaft ought to have turned to the left or south-east in order to recover the reef after passing through the crosscourse. But, as a matter of fact, it was necessary to go some 240 feet to the right or north-west. This heave of the reef to the right is incompatible with a downthrow of the eastern side of the main fault, and as the evidence of the strata is conclusive as to there having either been this downthrow or else a bodily heave of the wedge of country lying between the crosscourses to the northward, we must conclude that the latter action has taken place. This lateral displacement is probably the result of several more or less vertical movements, accompanied in every case with a considerable amount of lateral thrust to the north-west. The direction of the striæ on the slickensided surfaces shows that the general direction of movement was more or less up and down, and not horizontal. In the Bonanza adit, however, a small branch drive along a slide, which is probably connected with the main fault, reveals striated surfaces with the striæ inclined towards the south-east at angles of only 12° from the horizontal, and, as above remarked, inclined striæ (in one case making angles of 30° with the horizontal) are found on some surfaces of the walls of the main crosscourse. These show that the lateral movement was at times considerable. The resultant effect of several up and down movements, accompanied with strong lateral thrust, might very well be such a sidethrow or heave as is found in the mine.

Were it not for the position in which the reef is found, there would be no reason to search for a further explanation of the position with regard to each other of the strata on each side of the main fault than the apparent one of a downthrow of its eastern wall, but, as above remarked, this is incompatible with a heave of the reef to the northward. The unusual nature of the case, therefore, leads us to inquire if there is no other possible explanation than that just given. The occurrence of quartz, occasionally gold-bearing, along the slide in considerable quantities suggests that the break found in the lode is really a "deviation," not a true "heave," and this suggestion gains further probability when the behaviour of the reef at the second crosscourse is examined into, for we find that immediately after passing this the reef appears to run off in quite a new direction, changing its course from S. 48° W. to about N. 56° W., thus turning through an angle of 76°. The current local belief that the Little Wonder, Moonlight, and West Tasmania lodes are part of the main Tasmania reef is an expression of this theory. If it be true the reef fissure must have been formed subsequently to the faulting of the country by the crosscourses, and the deviation would be due to the fissuring force partly rending open the old fractures, and being altered in direction thereby. Such deviations of lodes are not uncommon.

The question as to whether these breaks in the Tasmania reef are due to true faults or to deviations is not one of merely scientific importance, and to be regarded as of no practical moment by the commercial mining world,—on the contrary, a very practical issue is involved in it. It is this: if the breaks are only deviations it is most probable that the Tasmania and Moonlight lines of reef are one and the same; but if, on the contrary, the reef has been faulted, the western extension of the Tasmania reef has never been seen on the west side of the second crosscourse, and an important part of it has yet to be discovered.

As far as the main crosscourse is concerned, the following considerations seem to me very conclusive as to the reef having been faulted and not deviated:—(1.) The heave, or lateral displacement, of the reef remains as nearly as possible constant at the various levels from the surface down to the deepest or No. 6 level. This is characteristic of true faults, but almost, if not quite, unknown in deviations. (2.) It happens that the main crosscourse has cut through the Tasmania reef at a place where it has, in mining parlance, "taken horse," that is, has divided into two branches which have united again further on, enclosing a mass of country rock. The fault goes fairly through the middle of this horse. At every level the distance between the two branches, where they abut against the fault on its eastern side, agrees almost exactly with the distance between them, where they are found again abutting against its western side. (This is also an argument in favour of the belief that the resultant effect of the faulting motions has been simple horizontal displacement of the country northwards, as it is not likely that in any other case the widths of the horse at different levels would correspond when brought opposite to each other by vertical displacement.) It is quite incredible that a fissuring force should split the country on each side of the cross-course deviating it with such accuracy. But it is easily understood that the ends of the branches must correspond if the horse has been cut across by a fault. The horse is shown on Plan No. 2, at No. 6 level, but not at the other levels, except partially. (3.) The reef is cut cleanly through by the fault, and does not turn partly into it or drag along it as is usual in deviations. The quartz above mentioned as being found in the fault does not appear to me to be similar to the quartz of the main reef, and I believe it to be a quite separate growth.

The position of the beds of country rock on either side of the crosscourse is compatible with either theory, as they could be brought into their present relations either by simple downthrow of the eastern side or by sidethrow of the western one. If the faults existed before the reef the former supposition is most likely to be true; if not, the latter must be true. As we have seen, the evidence at the main crosscourse is all in favour of its being a true fault, formed subsequently to the reef and cutting through it.

Taking now the second cross-course, we must consider how it bears upon the question of deviation or faulting of the reef. Like the main fault it is clearly a fault as far as the country is concerned, its eastern side being black sandstone and its western one grit and conglomerate, where cut through by the mine workings. The downthrow or northerly sidethrow of the eastern side is here as plain as at the main crosscourse, and again either motion would explain the relative positions of the strata on either side of the fault. The latter has beautifully defined well-polished walls, runs about N. 46° W., and dips S.W., at an angle of about 80°, being thus apparently a "reverse fault," like the main cross-course. If the reef has been faulted by this slide the continuation of it on the western side would have to be looked for to the left on going through the latter, that is the levels should turn off to the southward. They have, however, gone to the northward, and a reef has been found which has been assumed to be the continuation of the Tasmania reef. The evidence in favour of the deviation theory seems stronger here than at the main fault, for the heave is different at different levels, an unusual thing in case of faults but normal for deviations. The reef appears to be heaved 40 feet at No. 1 level, 61 feet at No. 2, and 100 feet at No. 5. This increasing heave at different levels may, however, to my mind, be accounted for without accepting the theory of the reef having deviated, by the explanation that the stone met with on the western side of this cross-course belongs to a different reef from that left on the eastern one,—in fact, that the stone on the western side is the Moonlight reef, and on the eastern one the Tasmania reef. It will be seen from Plan No. 1 that, taking the general line of the series of viens forming the Little Wonder, Moonlight, and West Tasmania reefs, it is due to strike the cross-course very near to where the Tasmania workings have struck quartz as above described. The increasing heave of the fault would then be simply due to the differences of dip of the intersections of the two lodes with the plane of the fault. On this supposition it would only be a coincidence that the two bodies of quartz were found so near to one another at the fault. The explanation gains in plausibility when the evidence at the main cross-course is taken into account, for this, as has been shown, almost certainly proves that the reef with its enclosing country has been heaved to the northward. This implies the existence of two faults, one on each side of the heaved country. The pre-existence of the two faults is equally implied by the deviation theory, and their general or approximate parallelism would render their being of contemporaneous origin probable in any case. If we assume this to be true, and believe that the wedge of ground between the two faults has been heaved nearly horizontally northward, it will be seen that the Tasmania reef would thus be brought 240 feet nearer to the Moonlight one. If the wedge is supposed to be forced back southward till the Tasmania reef is again continuous at the main cross-course, the ends of the two reefs at the second cross-course would be 280 feet apart at No. 1 level instead of 40. There are certain considerations with regard to the nature of the reefs themselves that make it more likely that the reef west of the second cross-course is the Moonlight one and not the Tasmania. East of the main cross-course the latter, though subject to minor bendings and sinuosities, prestrves a fairly straight line of strike (N. 48° E.) throughout its length. Between the two cross-courses this same strike is fairly well preserved, though several small heaves break the continuity of the line. We may therefore say that all the reef east of the second cross-course preserves a straight line of strike. In the same way the reef, or run of reefs—for there appear to be a number of more or less parallel veins—in the Little Wonder, Moonlight, and West Tasmania mines, which for convenience I have been calling the Moonlight reef, preserves a fairly straight course (N. 56° W.) up to the cross-course. It seems almost impossible that this fault should cause the reef to veer through such a large angle as 76°. The theory that the reefs are different ones seems far more probable. This probability is increased when the difference in the character of the reefs is taken into account. The Moonlight reef is notoriously bunchy, consisting of blocks of quartz which rapidly thin out to mere strings both in strike and dip, while the Tasmania reef, though sometimes pinched, preserves with great uniformity a continuous body of stone. It would be a curious and unlikely thing that the mere deviation of a lode from its former course should effect such a change in its character. The only feasible explanation of such a fact would lie in the statement that while the Tasmania reef runs across the strata of the country, the Moonlight one runs nearly with them (though crossing them on the underlay); and while the former traverses sandstones, the latter lies in grits and conglomerates, which might be expected to break more irregularly. The workings of the Tasmania mine on each side of the second cross-course show the difference in the nature of the two reefs; on the eastern side the stone is fairly continuous, but on the western one it had the same character as the Moonlight line—blocky and irregular. Immediately west of the cross-course there was good auriferous quartz from surface down to No. 2 level, but below that point the stone gave out, and at No. 3 and No. 5 levels only a "track" of the reef was visible. On driving westward also along the reef at No. 1 level the quartz soon dwindled to strings, and though these were found to lead on to several bunches or blocks of quartz, no regular body of stone could be got; in fact, the behaviour of the lode in this part of the Tasmania mine has been exactly the same as in the mines on the Moonlight line of reef. From all these considerations taken together, I feel nearly certain that the workings of the Tasmania mine, by going northward at the second cross-course, have left the Tasmania reef and struck a portion of the Moonlight line.

If we suppose, then, that these two lines of reef are separate, and existed before the faults broke them, they must have either joined together or crossed one another, the latter being the more probable on account of their directions being so nearly at right angles. Should they have done so, traces of their extensions past the intersection should be met with. No sign has yet been seen of the Tasmania reef west of the second cross-course, but no great amount of prospecting for it has been done. There appear to me, however, to be reasons to believe that extensions southward of the Moonlight line of reef have been found. Ever since the opening of the field a good deal of gold in quartz has been got along the eastern slopes of the Cabbage-tree Hill from end to end. The long drives into the ridge of the Beaconsfield, Bonanza, Leviathan, and Cosmopolitan companies testify to the general local belief in the existence of lodes running with the ridge, that is, across the line of the Tasmania reef. Auriferous quartz veins have been cut in various places, the most important workings on them being those of the Cosmopolitan Company. Bearing in mind that the Moonlight reef appears to consist of a number of veins running with the ridge, and that these are very irregular and bunchy, it seems very probable that the leaders found to the south of the line of the Tasmania reef are an extension of the run of veins found north of it.

If, then, the Tasmania reef has been lost west of the cross-course, where would be the most likely place to find it again? Owing to the sidethrow above mentioned, and to a twisting round of the strata east of the main fault so as to make their strike more westerly than that of those on its west side, it is not possible to accurately estimate the amount of downthrow of it, and hence the heave cannot be predicted. Any heave, however, should be to the southward, hence the most probable position of the faulted portion is to the south of a line connecting the Golden Gate and Britannia shafts. It is most likely to pass through either the most southerly section held by the Amalgamated West Tasmania Company or that of the Bonanza Company, but it may even be heaved to the south of the Bonanza section altogether, though this is not likely. It may seem incredible that the reef should have remained so long undiscovered if it passes out through any of these sections, but an examination of the ground shows that very little systematic prospecting has been done, and on the western slope of the hill there is often a good deal of surface *débris*. Should the reef happen to be thin at its outcrop it might easily escape notice. The prevalent notion, too, that the Tasmania and Moonlight reefs are one and the same might have a good deal to do with diverting attention from this part of the ground. The first discovery of gold, if I have been correctly informed, was made west of the second cross-course, and on what I consider to be the Moonlight lode; the fault was then encountered, and soon afterwards the Tasmania reef was picked up on the other side of it. The belief once held (if my information is correct) that the Phœnix shaft was on about the line of the reef shows that it was not till a good while after the lode had been first found that its true direction was ascertained. The discovery of the identity of the reef near the Golden Gate shaft with the one in the workings near the top shaft, as it is called (see Plan No. 2), was required to disprove the belief that it had not continued on or about the line of Dally's first discovery.

Before leaving the subject of these two cross-courses, a rather noticeable feature in them should be mentioned. The hade of both gets flatter towards the north-west end of the workings on them, consequently the drives at the different levels diverge fan-like, when seen in plan. This divergence is plainly seen on Plan No. 2 at the second cross-course, but that at the main cross-course is not shown. Here, indeed, it would appear that the dip of the fault changes from easterly to westerly, the Nos. 3, 4, and 5 levels lying to the east of the line of fault at No. 2, while No. 6 is to the west of it. The plan of this part of the mine is taken from the surveys sent annually to the office of the Inspector of Mines, and ought to be correct, but I have grave doubts of its being so. In all the levels the hade of the fault is to the westward, and the underground captain of the mine, Mr. Swanston, told me he had never known it to be otherwise in the stopes on the quartz found there. The large plan at the mine made by Mr. Davies, the mining manager, shows all the drives on the cross-course as lying successively further and further to the westward of its outcrop on the surface, and also exhibits the fan-like divergence of the levels at the north-west end, just as in the second cross-course. I cannot but think there is some mistake here in the official surveys from which my plan is taken.

Besides these two principal faults there appear to be a great number of smaller ones running more or less parallel to the ridge of the Cabbage-tree Hill. A somewhat larger one, but apparently of no great importance, seen in the Cosmopolitan mine, is shown on the plan, and traces of numerous others are met with in all the workings along the hill from the Garfield adit to the Cosmopolitan mine. Quartz is often found on these slides, and occasionally carries payable gold. As the axis of a syncline running N. 37° W., which as seen on Plan No. 3 passes just west of the Moonlight main shaft, has much the course of the generality of these slides, there is some foundation for suspecting that these fractures were caused by the force which caused the flexures seen in the section. There is a certain amount of probability that the veins of the Moonlight line of lodes themselves are connected with this series of fractures. I have not, however, been able to detect any faulting of the beds of country rock by these lodes, yet lying as they do almost in the same line of strike as the beds, which in this part of the field are very similar to one another, a considerable amount of faulting might easily escape observation.

Going now to the eastern end of the Tasmania mine, we shall inquire if there is any evidence of faulting here which would give a clue to where to look for the continuation of the reef in this direction, for hitherto it has not been discovered. By actual working the lode has been followed to a short distance east of the boundary between the old Lefroy and Florence Nightingale sections. The deep lead previously described cuts it off in all but the deepest level. As the lead is manifestly of much later formation than the reef, and is simply a river channel cut through it, it is absurd to ascribe to it, as is often done, any inimical influence upon the lode. The simple erosion of a river channel cannot fault or disturb in any way the continuity of the solid country from which it is carved out. Something, however, has happened to this reef, for it is not cut either in the long cross-cuts of the East Tasmania Mine or in those of the Dally's United. If the reef had continued on its course it would pass a little to the north of the East Tasmania shaft, which would strike it at about 350 feet. But no sign of it has been seen in the cross-cuts from the shaft. It is evident that one of three things has taken place, (1) either the reef has been heaved a very long way to the north or to the south, or (2) it has dipped deep in the strike with what is known as an "endlong dip," that is, instead of the outcrop coming to surface it has only come partly up through the ground, and the drives have therefore passed over it, or (3) the reef has died out altogether. The first of these is the most likely supposition. The second was to have been tested by the East Tasmania Diamond Drill bore, but unfortunately this had to be abandoned without proving anything on account of breakage of the rods, and loss of a portion of them in the bore. The third supposition is possible enough, but not to be contemplated until the other two have been proved to be impossible.

My belief that a large fault exists between the Lefroy shaft and the East Tasmania bore is founded on the relative positions of the limestones met with in the bottom of the Ophir Company's bores, and in the Dally's United Mine and East Tasmania bore, also on the marked difference in the strike of the country in the East Tasmania Mine from that in the Tasmania Mine. A glance at the section given herewith shows that in this part of the field there are two principal limestone bands, one met with in the bottom of the East Tasmania shaft, the other, and much larger one, struck at 458 feet in the East Tasmania bore, and also cut in the south drive from Dally's United shaft. From the plan it may be seen that there is no room for another large limestone band to exist between this one and the eastern workings of the Tasmania Mine. Now, what appears to be a very large body of limestone exists under the deep lead in the Ophir ground, for the Ophir diamond drill bore bottomed on limestone, as did Orchard's shaft also. The assumption that this large limestone body is the same met with in Dally's United Mine is therefore a very reasonable one. To confirm this belief, another band of limestone is found at the crossing of Weld and Crowther-streets, at about the same distance from the Ophir limestone as that in the East Tasmania shaft is from that in the Dally's Mine. While this gives great probability to the theory I am about to bring forward, it is only fair to point out that the whole argument depends on the identity of these limestone bands. When we come to try to connect the Ophir limestone with Dally's it is at once apparent that something is out of joint. An extensive series of observations of the strike of the country rock in the Olive Branch adits, Tasmania Mine and other long adits running into the Cabbage-tree Hill, of which more hereafter, has proved that the average strike of the strata in these is N. 53° W., while a careful examination of the north drive from the East Tasmania shaft shows that there the strata strikes N. 42° W. This difference of itself would suggest the existence of a fault. But neither of these lines of strike will connect the two bodies of limestone, thus again rendering the presence of a fault between them probable. I have accordingly on the plan drawn each limestone as having the strike of the nearest strata to it which have been measured. The exact position of the fault is of course doubtful, and its direction can only roughly be ascertained by noticing that it has not been cut in the East Tasmania, Dally's United, Tasmania, or Cosmopolitan workings. As it must, therefore, have much the same course as the main Tasmania fault, it has been drawn parallel to the latter on the plan. The commonness of the occurrence of parallel faults in disturbed districts makes it likely to begin with that these would be roughly parallel. Looking at the positions of the faulted portions of the main limestone on the plan, and remembering that the workings of the East Tasmania and Dally's United mines have shown that the strata here still have a north-easterly dip, it will be seen that the western must be the downthrow side of the fault to bring the strata into their present position. A downthrow of the western side of the fault would have the effect of heaving the Tasmania reef so that its continuation on the east side would be to the south of its line on the west. The amount of heave of the strata not being extremely great, the heave of the reef could not be very extreme either, but could easily be sufficient to allow of its passing between the north end of Dally's United and the south end of the East Tasmania drives. A somewhat unexpected confirmation of this theory was furnished on plotting the position of the limestone struck in the East Tasmania bore and that got in the Dally's United mine. The dip of the strata (63°) was easily ascertained by measuring the cores brought up by the drill, and consequently the position of the top of the limestone at the level of Dally's United workings was easily found. But on joining the horizontal traces of the bed in the two places they are found not to correspond with the average strike of the country, and lines drawn through them parallel to this are found to be 104 feet apart. It would seem, therefore, that there is a fault of some sort between these two points. Now, in examining the Tasmania reef I have frequently remarked that the beds of country on the hanging wall do not correspond with those on the footwall; that the reef itself, in fact, is formed in a line of fault. This is a common enough occurrence in lodes, movement of the walls having taken place after the opening of the fissures. At No. 6 level of the Golden Gate section of the Tasmania mine a good opportunity is afforded of measuring the amount of heave of the beds of country. For some little distance westward from the shaft dark bluish sandstone is found on both sides of the reef; on the footwall side this is underlaid by white sandstones, but the dark stone continues on the hanging wall for 104 feet further, when the white sandstone comes in under it also; thus the country rock beds are heaved 104 feet westerly by the reef. As above seen, this is just the amount of heave of the hypothetical fault lying between the East Tasmania bore and Dally's workings, which is strong evidence that the reef lies somewhere there. If, now, we further remark the difference of strike of the portions of country on each side of the large north-westerly fault, which for convenience we may call the Lefroy fault, it is seen that the amount is 11°. Let us also notice that the strike of the country west of the Tasmania second cross-course is as nearly as possible identical with that of the strata east of the Lefroy fault, and that the country between these two large faults has fallen downwards—the faults being thus "trough faults." It has been seen that there is evidence of strong thrust from the southward having accompanied the downthrow of this piece of ground, and it is likely that the difference of strike of the portion in the trough between the faults from that east and west of them is due to an oblique thrust having screwed it round more to the north-west. If, now, the country containing the Tasmania reef be screwed back again so as to have its strata with their original strike of N. 42° W., the course of the reef would be N. 59° E. instead of N. 48° E., as at present, and this is the course we should expect it to have east of the Lefroy fault, and in the extension west of the Cabbage-tree Hill as well. This line is laid down on the plan as the position of the hypothetical fault occupying the probable position of the reef above referred to, and it will be seen that it goes easily between the ends of the East Tasmania and *Dally's United drives. The ground lying between these ought most certainly to be tested by extending* them, best towards one another. The reasoning as to the reef lying between the ends of the two drives may appear to be only ingenious theorising, and I am quite well aware that many objections could be raised to the argument: still the belief that such is the case has been forced upon me in the endeavour to reconcile and explain existing facts. The strongest point in favour of it is that it is difficult to imagine where else the reef can go. A very great heave would be required to take it either south of the Dally's or north of the East Tasmania cross-cuts. The deductions from the position of the country rocks thus agreeing with *primâ facie* probability, I feel as certain as the nature of the case permits that the eastern extension of the Tasmania reef lies where indicated on the plan.

There appear to be at least two other large faults in the Beaconsfield district—one crossing the Cabbage-tree Hill to the south-east of the Cosmopolitan and Peru mines, and another running along the Brandy Creek between the Little Wonder and Brandy Creek mines. The existence of these is indicated by the difference in strike of the strata on each side of them and by the want of agreement in position of various recognizable beds. It is not claimed that either the position or direction of these faults is correctly represented on the plan, the lines being drawn to indicate only approximately where they may be met with. To locate them with accuracy would require a very close survey of the ground in their neighbourhood, and as sections, and even exposures of the solid rock are rather rare there, it might even then prove impossible to do so. The whole district seems to be much faulted, and there are probably many faults yet to be discovered.

Strike and Dip.—In the course of my survey of this field it was found necessary to ascertain as exactly as possible the strike and dip of the strata, and some hundreds of observations of these were taken. Though, as might be expected, there were considerable local variations due to petty disturbances, it was found that on the whole the average strike of the various blocks of country separated by the main slides was fairly constant for each, though that of each block was generally different from its neighbours. The average direction of the strata as thus found is shown on the general plan, and actual observations in the mines are plotted on Plans Nos. 2 and 3, and some taken in the gorge of Blyth's Creek on Plan No. 1. Owing to the small scale of the latter, the actual observations in the Garfield, Olive Branch, Leviathan, and Cosmopolitan adits are not figured on it. The mean of the measurements in the Olive Branch gives a strike of N. 58° W., in the Leviathan N. 51° W., in the Upper Cosmopolitan adit N. 54° W., and in the Lower Cosmopolitan adit N. 47° W. These altogether give the same average strike as is found in the eastern part of the Tasmania mine, N. 53° W., but they also give reason to believe that the strike of this block of country becomes more westerly towards its northward end. This would accord with a flattening of the dips of the faults towards their northward ends, as observed above in the case of the two Tasmania cross-courses, and with the somewhat rotatory motion previously mentioned as being the course of the difference of strike of the strata east and west of the Lefroy fault. In the Garfield adit the rock is much disturbed and the strike very variable in consequence, but the measurements give an average strike of N. 40° W., which agrees very well with that calculated from those in the West Tasmania, Moonlight, and Little Wonder mines, viz., N. 42° W.

Influence of Country Rock on the value of the Reefs.—One of the objects of this survey was to ascertain as far as possible if there was any difference discernible in the gold value of the reefs as they passed through the various strata. It has often been noticed in the history of mining that certain bands of country appear to enrich the reefs passing through them, while others have the contrary effect. In the present instance the evidence on this head shows no marked influence of the different beds on the value of the reefs, except in one case mentioned below. The Tasmania reef has been auriferous throughout all the strata traversed by it. The richest stone is found in a number of distinct "shoots" or "chutes," which, according to Mr. Davies, the mining manager, are often distinctly separated from the rest of the quartz by small sandy and clayey partings or "selvages". Outside of the "shoots," however, the quartz has been generally payable. The shoots dip easterly, conforming pretty closely with the dip of the country rock. Some remain narrow and constitute roughly parallel bands in the quartz, but two or more widen out very much in the lowest levels of the mine. The strata that have proved "favourable country" for gold in this mine may be said to be all those between the lower beds of grits and conglomerates and the main limestone bed. Owing to the loss of the quartz west of the second cross-course below No. 2 level, it is not known if the black carbonaceous grits there carry any gold. In the Moonlight and Little Wonder mines these black strata have been almost altogether barren of gold, though recent discoveries of the metal in quartz leaders cut in the new eastern Moonlight cross-cut at 422 feet give hope that they are not always so. In the mines on the Moonlight line of reef rich stone has been got in the upper levels of all, and as long as the quartz was found in the light coloured grits and sandstones, but on getting down into the black country the stone has become unpayable in every case, and, with the exception just spoken of, very rarely contains any gold at all. According to experience up to the present, therefore, the lower beds of the grits and conglomerates have proved to be "unfavourable country."

The gold-producing capabilities of the strata lying east of the present Tasmania workings and of the slate underlying the grits, have yet to be ascertained. With the exception of the limestone bands, which are not generally considered to be favourable for gold, the bands of slate, schists, and sandstone seen to the east of the Cabbage-tree Hill are similar to those proved to be auriferous a little lower in the series, and will probably therefore also be favourable. The value of the reef in the limestone remains to be proved by actual workings. Gold has sometimes been found in limestone, and it is quite possible that the reef will not suffer in value in passing through it in the present instance; still, the difference between slate or sandstone and limestone country is so great, both chemically and physically, that some change in the contained reefs may be expected, and no great hopes should be entertained until the metal is actually proved by working to be present in them.

As regards the blue slate found west of the Cabbage-tree Hill there is no proof yet as to whether it is favourable or unfavourable country. No known auriferous veins have yet been found in it, but none of the proved auriferous reefs have yet been traced into it, and on almost every goldfield there are numbers of barren veins even in the most favourable country rock. Slate of similar character is generally regarded as good country for gold, and there is therefore reason to be hopeful as to the future of the reefs when traced into this. Should a western continuation of the Tasmania reef be discovered it will soon pass into this slate, and the auriferous nature or otherwise of this will then be soon proved. The same country will be found at a depth of from 800 to 1000 feet in the Little Wonder, Moonlight, and West Tasmania mines. It seems to me that the most useful work these companies could now do would be to combine to sink the Moonlight shaft, which is the deepest and best constructed of the three, to below the "black country" in the hope of the reefs improving when they pass through it into the slate. It is probable also that by sinking thus the reefs would be got further away from the contorted strata, and there would then be more hope of having them solid and continuous, and of the numerous veins combining into one lode. The upper levels are practically exhausted, and the present lower ones are barren, hence all that remains for these mines is to find new veins or to sink for better country. The latter course commends itself as a genuine mining enterprise of the sort that has saved many a mine from being given up, and led to great success.

Remarks on the Prospects of the Mines.—Passing on to the consideration of the light thrown by this survey on the future prospects of the mines as to permanency and value, it is seen that the Tasmania mine stands in a most favourable position. There is an undisturbed body of stone from the main cross-course to the Lefroy fault, going down in favourable country to at least 1000 feet, and probably much further than it can ever be followed by mining operations. The favourable country is dipping to the eastward, as also are the shoots of gold, and it is to the eastward that the ground held by the owners of the mine stretches furthest in the direction of the dip of the reef, and consequently contains it to the greatest depth. The Phœnix diamond drill bore proved that the reef was as strong as ever at over 700 feet and rich in gold. The lode is evidently a true fissure vein, or it could not have faulted the enclosing strata. There seems no reason to doubt that it will be as large a gold producer as in the past for many years to come. The powerful new pumping appliances now in course of erection will easily cope with the water which has hitherto retarded progress so much, and to be able to overcome a still greater inflow should such occur.

The Phœnix Company should have a valuable property, but have difficulties to overcome in working it, as it lies beneath the Tasmania workings, and hence the mine will be liable to get the water from these. Owing to the shape of the boundary between the two holdings the workings of the Tasmania mine will not be of much benefit to the Phœnix until they are down to about 800 feet, as there is only a small triangular portion of the reef belonging to the latter company above that level. There are thus years of work before the Tasmania Company, if they choose, before their pumping will help the Phœnix mine. The owners of the latter, if they wish to work it, will have to provide drainage plant of their own. This would require to be equally powerful with the large new plant of the Tasmania Company, for there can be little doubt that the lower workings would soon drain the upper ones dry. The strata are very loose and open, and water gets through them freely. This was shown by the diamond drill bores both of the Phœnix and East Tasmania Companies; in neither case could the bore be kept full of water, even when tubed to the bottom with iron tubes: this, too, when the bottom of the bore was 200 and even 400 feet below the level of the water in the Tasmania mine. Another proof of the porous nature of the country is seen in the fact that there is rarely any water in the bottoms of the Moonlight, West Tasmania, and Little Wonder shafts, though these are all a long way below sea level. During the recent flooding of the Tasmania mine, however, water rose in the Moonlight shaft and remained until the former was unwatered, when it drained out of the latter also. If water can communicate so easily through the rocks with points so distant from the Tasmania mine as the East Tasmania bore and the Moonlight shaft, it seems incredible that it would not get from the Tasmania into the Phœnix mine, on the same reef and closely contiguous. It would probably be possible enough to sink the Phœnix shaft to 900 or 1000 feet without a very large pump, but when the mine was opened out, and the "bleeding" or "weeping" surface much increased thereby, the open nature of the rock is such that it would be impossible to prevent the water from the Tasmania mine finding its way down. Some water might be cut off, and led to the Tasmania's shafts to be raised, but only a small proportion, and one getting less and less as the Phœnix workings are extended. The question of drainage would be a serious one between the two companies, even if both had powerful pumping appliances, as it would be perfectly impossible to fairly allocate the burden of pumping expense between them. Only by a mutual friendly arrangement between the two as to sharing the expense could endless disputes and litigation be avoided, as the extensions of the workings of both mines would cause the inflow of water into each to be continually varying, so that no hard-and-fast rule could with justice be laid down as to how much was the proper share of each party. The best way out of the difficulty lies in an amalgamation of the two claims. All the pumping could then be done from the New Tasmania main shaft. The Phœnix ground, lying further west than the Tasmania's future deep workings, is nearer than the latter to the lower deep strata of the country, hence in the western workings the Phœnix will encounter the grits and conglomerates long before the Tasmania will. The ground is so pegged out, indeed, that the latter mine will carry the bulk of the most favourable country with it eastward. While, therefore, the former mine will doubtless have a good deal of gold in it, it must be remembered that as it goes down it gets nearer and nearer to the unfavourable black grits, and to the yet unproved slates, and its future has nothing of the same certainty about it that its neighbour's has, in consequence.

The Amalgamated West Tasmania, Moonlight, and Little Wonder mines, being all on the same line of reef, may be dealt with together. Their prospects and future policy have already been referred to, but a few more remarks may be made upon them. The Little Wonder and Moonlight have been the best producers of gold, the upper levels having been very rich. There are several veins more or less parallel to each other in these mines, and as these often pinch to mere strings, and the ground is greatly broken and disturbed in addition, the following of them has been a matter of great difficulty, and they have been often lost and not always found again. The exact number and relationship of them is therefore doubtful. In the main workings of the Little Wonder two larger branches diverge northward from near the shaft, and in the opposite direction come together and run into the Moonlight ground. The old Olive Branch Company had a small portion of this reef in the south-west corner of their western section, and worked it with considerable success till it dipped away into the Moonlight ground. The general dip of this line of reef is to the south west. The Moonlight Company have been very persevering in their attempts to find gold at a depth, and have done a great deal of prospecting at their 422 feet level, assisted in this by a subsidy from the Government. Their long south-western cross-cut, though unsuccessful in cutting any reefs of value, has proved a considerable stretch of ground, and has afforded an excellent section of the strata that has been of the greatest service to me in preparing this Report. The old Olive Branch sections having fallen into their hands, they are now driving eastward from the main shaft, and have cut some leaders and bodies of quartz that give promise of greater success in the future. The development of these discoveries is suspended until the completion of a contract for driving this eastern cross-cut. One leader contains a little gold. In the workings near the surface the Moonlight reef appeared to dip north east—that is, away from the shaft,—then it became vertical, and finally turned towards the shaft and passed through it, dipping south-westerly. This has often led to a suspicion that there were two "legs" to this lode, one dipping north-east and another south-west. Owing to the proximity of their boundary line, the Moonlight Company were not able to test this supposition until lately, when the Olive Branch ground was acquired, and these discoveries in the eastern crosscut give some support to it. The Olive Branch Company also drove easterly from their shaft, but did not get any lodes of importance. The belief in an easterly leg of the Moonlight lode seems borne out by last year's (1890) workings of the West Tasmania Company, who, after picking up the reef in the old surface workings of the Moonlight, followed it to the south-east, and extracted 507 tons of quartz, which yielded 549 ounces of gold. This stone was found to dip to the north-east, and at 200 feet passed through the West Tasmania shaft. Workings on it at the 150 feet level were carried on up to the Olive Branch boundary. This is the only gold-bearing stone of any consequence yet got in the West Tasmania Mine, with the exception of a little that was obtained in the 315 feet level where it joined the Moonlight 250 feet. All the other workings have been on thin veins and tracks not worth stoping out. This lode was of a very peculiar character, and extremely difficult to follow. The quartz was found in the most irregular bunches, connected by nothing but clayey "tracks" and occasionally thin veins of quartz, and was enclosed in a jumbled mass of sandstone and grit fragments, mixed with clay and sand. On examination I came to the conclusion that there was here a rather wide lode fissure filled with broken fragments of the wall-rock, amongst which the quartz had been deposited, thus accounting for the very irregular distribution of the latter. The filling of the lode is mainly composed of loose angular fragments of sandstone, grit, and conglomerate, sometimes crushed or disintegrated into loose sand and gravel. In places open spaces were found into which an arm could be thrust up to the shoulder. Throughout the broken formation no trace of regularity of stratification could be observed, but in one or two crosscuts through it the solid regularly stratified rock was to be seen, proving the nature of the occurrence to be of the "mullock lode" type. Where the main shaft had been sunk through it there was no definite body of quartz, and consequently the fact of there being a lode was not noticed. From the size of this formation I should expect that the fracture of the rocks shown by it is an important one, and, as lodes filled with fragments of wall-rock frequently are found on being followed to narrow and become filled with quartz or other proper lode material, I think that it would be well worth while to trace this one further, in the hope that it would so change. All the quartz so far found in it has been highly payable, and, should the filling change to a defined quartz reef, there would be every hope of it proving payable also.

There is a somewhat similar lode to that just described, and also dipping to the north-east, found in some workings from what is known as the Moonlight No. 3 shaft. A drive on this lode at the 60 feet level showed it to be filled with a broken mass of country rock and occasional pieces of quartz. No gold was got in these, however. This mullock lode also should be traced further, in the hope of its changing to quartz.

As above said, the future of these three mines depends on finding new lodes or branches of those already known, or on sinking deeper. A good prospecting work would be a drive from the Little Wonder shaft south-west into the Moonlight ground far enough to make sure that none of the veins worked upon in the No. 5 level of the latter mine have turned off along the boundary of the two sections. A crosscut north-east from the same shaft would also prove some very likely ground. Sinking, however, seems to me the best policy of all.

Only two other mines now working have found reefs—the Cosmopolitan and the Brandy Creek. The latter was shut down on the only occasion when I had an opportunity of visiting it, and I was only able to go hurriedly through the former once, consequently I have little to say about them. The Cosmopolitan workings are unfortunate in being situated in what appears to be a regular network of small slides, which have cut off the quartz repeatedly. A great deal of driving has been done to get to the reefs, but very little real work on them. Good gold-bearing stone has been at times obtained, and, if the mine were opened up better, it might produce a good deal, but the numerous slides have hampered work very sadly. It will be necessary to get the reef in less broken country before work can be profitable.

Marble.—The blue-black crystalline limestone found in the East Tasmania bore takes a very good polish, and is really a very handsome marble. When polished it is nearly black, the bluish shade in the colour being only perceptible on close inspection, and numerous veins of pure white calcite give variety and beauty to it. The stone works well, being close-grained and hard. There should be no difficulty in getting blocks of uniform texture and any required size from Dally's quarry on Blyth's Creek (where this marble is being burned for lime), or more easily still from the large masses cropping out at the head of the Flowery Gully. If worked by skilled marble-workers I have no doubt that this stone could be sold profitably in considerable quantities for ornamental and monumental purposes. The quantity easily got at is very large, and the facilities for quarrying are good; while the proximity to a shipping place, and ease with which tramways could be constructed from it to the quarries, are very favourable for cheap transport to either the local or the Australian markets.

The pale bluish white marble found at the old limekiln at the second bridge on the road from Beaconsfield to Launceston would also be of commercial value if easily obtained. As the workings of the old quarry on this bed are now full of water I cannot speak as to the size of it, or as to whether large blocks could be got of uniform texture and free from flaws. If, however, the loose stones lying about fairly represent the general quality of the marble it would be well worth quarrying.

Besides the marbles there is another stone in the Beaconsfield district that would be very useful for ornamental work, namely, the Serpentine that occurs abundantly in Anderson's Creek. This is found of great variety and beauty of colouring, and could be made into a great number of highly ornamental articles. Various shades of green and very pretty mottled serpentine are quite common.

Appendices.—As no topographical features are shown on the plans accompanying this Report, I append a number of heights above sea level of various points throughout the district. These were taken by means of an aneroid barometer, and consequently are only rough approximations to the true levels, but will be useful for purposes of comparison.

I also append sections of the Phœnix and East Tasmania diamond drill bores.

Thanks.—During the course of my survey of the district I received much useful information and many courtesies from the mining managers and other gentlemen throughout the district, whose help and kindness is now gratefully acknowledged.

I have, &c

A. MONTGOMERY, *M.A., Geological Surveyor.*

The Secretary of Mines, Hobart.

Appendix No. 1.

HEIGHTS above Sea Level of Points in the Beaconsfield Goldfield, approximately determined by Aneroid Barometer.

	Above H.W.M. Feet.
Weld-street, opposite Club Hotel	105
Top of Lefroy shaft	125
Florence Nightingale shaft	153
Dally's United shaft	117
Golden Gate shaft	186
Ballarat shaft	168
Ophir shaft	127
New main shaft, Tasmania	159
East Tasmania shaft	86
West Tasmania main shaft	352
Moonlight main shaft	334
Moonlight No. 3 shaft	262
Little Wonder shaft	321
Little Wonder shaft on Garfield Section	321
Mouth of Little Wonder main adit	204
Garfield adit	213
Bonanza adit	172
Leviathan adit	195
Cosmopolitan upper adit	204
Cosmopolitan lower adit	96
Top of Old Britannia shaft	276
Phœnix bore	213
East Tasmania bore	86
Ophir bores	120
Denmark shaft	110
Bonanza air shaft	231
Cabbage-tree Hill on line of Bonanza tunnel	411
Old shaft near Phœnix and Bonanza boundary	330
Leviathan air shaft	321
Kohinoor shaft	375
Cosmopolitan shaft	258
Hematite shaft	195
Air shaft on lower Cosmopolitan adit	163
Old New Providence prospecting shaft	348
Tasmania open cast workings	348
King and Eastman's alluvial workings	249
Old Stanley shaft	101
J. T. Allen's bore	65
New Brandy Creek shaft	155
Pease's shaft	164
Old Brandy Creek shaft	177
Excelsior shaft	190
Dundee shaft	191
Mouth of London adit	155
Brandy Creek old surface drive	159

Appendix No. 2.

SECTION of Strata afforded by the Phœnix Company's Diamond Drill Bore.

Strata.*	Thickness. ft.	in.	Total Depth. ft.	in.
Alluvial surface matter	4	6	4	6
Hard, brittle, whitish, and yellowish fine-grained sandstone, breaking into small rhombohedral fragments; very much jointed; would not form core; contained a few very thin quartz veins; flinty and somewhat crystalline in texture	368	4	372	10
Fine-grained, dark bluish, somewhat crystalline, sandstone, much jointed, and yielding little solid core; contained occasional specks of pyrites; a band of grey micaceous sandstone at 472 feet, but only very thin	119	8	492	6
Light grey and whitish sandstone, with occasional bands of grey slate; impressions of fossils rather numerous, but imperfect; sandstone more granular and porous than the preceding bands; gave a few inches of solid core at times	213	0	705	6
Tasmania reef—Quartz containing iron and copper pyrites; gold freely visible	24	6	730	0
Dense hard crystalline dark blue or black sandstone, with a good deal of pyrites in it	50	7	780	7
Total	780	7	780	7

* Dip of strata 58° (average of 12 measurements of angle of dip visible on cores).

SECTION of Strata afforded by the East Tasmania Company's Diamond Drill Bore.

Strata.*	Thickness.		Total Depth.	
	ft.	in.	ft.	in.
Surface soil and clay	12	0	12	0
Brown, grey, yellow, and bluish sandstone, with small quartz veins; some grit as well as sandstone, also thin bands of slate	68	10	80	10
Slate similar to that found in the 100 feet level of the East Tasmania mine, rather soft, greenish, arenaceous, and calcareous; vein of calcite and pyrites 2 inches wide at 116 feet	35	2	116	0
Impure silicious limestone with calcite veins; vein of dense whitish grey hard quartz at 120 feet	64	0	180	0
Dark black, bluish grey, light grey, and dense blue grey fine grained slates, containing a little pyrites and a few small quartz and calcite veins	278	0	458	0
Dense dark blue fine grained crystalline limestone or marble, very solid as a rule, but occasionally fractured. The drill ceased working while still in this rock	520	4	978	4
TOTAL	978	4	978	4

* Average dip of strata 63° (mean of 19 measurements of angle of dip shown by cores).

PART 'B'
Reference 1/2

REPORT UPON THE PRESENT POSITION OF THE TASMANIA MINE, BEACONSFIELD.

—:o:—

Government Geologist's Office, Launceston,
8th May, 1903.

SIR,

Agreeably *to your instructions, I proceeded to* Beaconsfield on the 30th April, in order to carry out the wish of the directors of the *Tasmania Gold Mining and Quartz Crushing* Company Registered that I should examine the mine, and give an opinion upon its present state and requirements so far as these relate to the special reports recently furnished by the general manager. I understand that the Board does not require me to supply an exhaustive and full report, but only one which shall cover the questions involved in the special reports mentioned. Consequently, I need not describe the company's property, its history, its geology, or the characteristics of the reef, or other well-known features of this important mine. It is sufficient for the present report to recall that, from the inception of the work to date, £772,071 15*s.* dividends have been paid on the extraction of 563,307 ozs. of gold.

The handling of the water in the reef has always formed a serious task. For the last nine years, since the cutting of the reef at the 600 and 700 feet levels, three million gallons per 24 hours have had to be raised, the water-level being lowered about 25 feet per annum. In driving on the reef at the 1000-feet level, a burst of water took place in August last, which stopped work there, and increased the water raised in that half-year by 240,000 gallons a day. The sinking plant in the new main shaft was unable to cope with the increase of water, which rose in three days to the 818-feet level, and which at the present time appears to be standing at about 64 feet below the 818-feet level.

Under these circumstances the company has to decide whether to attempt to overcome the water for a time by readjusting the pumping plant of the different shafts, or to provide for a fuller and more economical development of the mine by the installation of plant adequate to both present and future requirements.

ORE RESERVES.

The necessity for prompt decision is apparent from the present state of the ore-reserves. The estimate of ore existing above the 818-feet level, made 30th June 1901, by Mr. Joseph Davies, the late general manager, and those made by Mr. C. F. Heathcote, the present general manager, in his half-yearly reports, are nearly co-incident. The ground still unworked above this level may be relied upon as containing

about 15,000 tons of stone, which, opened up and extracted, will keep the battery going at the present rate for seven months. There has been rarely sufficient stone available to keep the 105 heads of stamps going; last half-year the number of stamps averaged 55, and there are now only 30 head at work. The mine, therefore, must be considered as nearly exhausted above the 818-feet level.

The future of the mine as an ore-repository, consequently, depends upon the quantity and quality of stone existing below the 818-feet level.

WORK ON THE REEF BELOW THE 818-FEET LEVEL.

This consists of—

1. Five winzes in course of sinking from the 818-feet level, the deepest of which is now down 64 feet. Careful assay sections have been kept of the reef in these winzes, and show the average gold contents of the stone to be as follows:—

Winze. No.	Depth assayed. ft.	Average width of stone assayed. ft. in.	Average assay yield. ozs. gold per ton.
2	18	4 6	0·8109
3	10½	5 0	1·152
4	27½	6 0	1·075
5	43	3 8	1·826
6	4½	1 0	6·477

The latest assays show that No. 6 winze, as it is descending, has entered a part of the reef, averaging 2½ ozs. gold per ton. They are as follow:—

Winze. No.	Depth of assay. ft.	Width of stone assayed. ft.	ozs. gold per ton.
5	61	5	1·404
6	6	2½	2·766
6	8½	4	2·425
6	11	4½	2·318
2	26	5	0·823
4	44½	7	0·359

Having examined these winzes, I find the reef going down strong with stone, ranging from 5 to 8 feet in width, corresponding in all its physical features with the reef which has been worked above the 818-feet level, and showing no signs whatever of deterioration at the deepest point attained. The winzes are well-designed, as they not only prove absolutely the value of the stone at that depth, but will also provide the means of putting men immediately on ore-winning when the further development of the mine is decided upon.

2. *Drive on the reef for 50 feet at the 1000 feet level.*— I examined this as well as was possible with the heavy flow of water. The east drive, which is 22 feet in length, shows a lode-channel of 7 feet in width, with 3 to 4 feet of quartz,

of a kindly, laminated appearance, closely resembling the stone in the upper parts of the mine. The west drive, 19 feet, shows in its end the apparent beginning of a horse in the reef. The large quantity of water prevents thorough examination of these drives, but I saw enough to satisfy myself that the reef is living and strong at this depth. This reef was cut 29th January, 1902, and samples taken while driving, and assayed at the mine, yielded the following results:—

East Drive.

2 ft. east of cr[illegible]		hanging wall	2 ft.	0·489	ozs. per ton
12 ft.	„	„	1 ft. 6 in.	1·161	„
23 ft.	„	„	2 ft. 6 in.	2·193	„

West Drive.

4 ft west of crossc[illegible]		face	2 ft. 6 in.	0·371	ozs. per ton
7 ft.	„	„	2 ft.	0·244	„
12 ft.	„	hanging wall	3 ft.	0·032	„
	„	footwall	1 ft. 6 in.	0·299	„
19 ft.	„	hanging wall	2 ft.	0·146	„
	„	footwall	2 ft	0·489	„

Boreholes 3 to 5 feet were put into the lode when it was first struck, and the stone assayed, with the following results:—

305 ft. 4 in. to 309 ft. 8 in. from shaft.
1 oz. 7 dwts. 10·56 grains.
1 oz. 0 dwts. 21 grains.

309 ft. 8 in. to 310 ft. 8 in. from shaft.
9 dwts. 3·52 grains.
2 dwts. 14 grains.

Estimates of the average width of the reef in the Tasmania Mine have been made from time to time by competent persons. The reef in its widest part is 26 feet across, but the calculations of average width are from 5 to 8 feet. It would appear therefore that its size at the 1000-feet level is equal to the average of the higher levels.

Quality of Stone.

The published returns of the crushings are, of course, tabulated in order of date only, and do not afford the means of ascertaining the returns from individual levels. Consequently, the yield from the upper parts of the mine have to be stated with reference to the time when they were worked. A further difficulty arises in connection with the corrections for tonnage, but, beginning with a rate of 7 per cent., and increasing to 19 per cent., the yield of the reef in ounces per ton of retorted and smelted gold has been as follow:—

From 1877 to April, 1894, 1 oz. 3 dwts. 19 grs.
To September, 1900, 18 dwts. 21 grs.
To January, 1902, 1 oz. 2 dwts. 14 grs.
To December, 1902, 14 dwts.

The recent falling off in the quality of stone is due to the poorer nature of the reef in the west end of the mine above the 818-feet, where it was also poor above the 700-feet, though said to have been good at the 600-feet. There are only 270 feet to drive west at the 818-feet level, when the main crosscourse will be reached, and the present *run of stone* terminated. But from September, 1900, to date, the average yield, according to the best means at our disposal for ascertaining it, has been below 18 and 19 dwts.; and using the same means of calculation for the whole period of six years' work before that, from May, 1894, to August, 1900, the yield was the same (within a few grains).

The records which have been kept of the output from the 818-feet level, subjected to such tonnage corrections as it is possible to apply, show that the reef between the 718 and 818 feet levels has yielded an average of between 17 and 18 dwts. retorted and smelted gold, of a value of £3 15*s*. 6*d*. per ounce.

While, therefore, the general manager is strictly correct, having regard to the uncertainty of *the corrections for tonnage*, in refraining from an opinion as to decrease or increase in the average value of the reef from the surface downwards, I believe the nearest approximation to the facts would be that for the last nine years the mean yield has remained about the same. The fall to 14 dwts. for the last year must not be taken by itself, but included in the average of *recent years*. In *the present* year the average yield from quartz and concentrates has decreased to 11 dwts., and that from the mine output exclusively to between 10 and 11 dwts. This is perhaps what might be expected for a time, while at the poor end of the reef, and when the quality cannot be reinforced by stone from ground opened up elsewhere (except a trifle from the current breakings in *the lower winzes*).

Production.

The 818-feet level will have returned over 94,000 tons of quartz from the reef between it and the 718 feet, and, as the reef-chute in the distance between the two levels has increased in length by 75 feet, viz., from 1475 feet to 1550 feet, it is reasonable to suppose that the increase will continue as the reef descends. The General Manager expects the reef between the 818 and 1000 feet to yield 180,000 tons quartz, irrespective of any increase in the length of the reef. This is a conservative estimate, and in basing his calculations on 90,000 tons as the probable output of each 100-feet level, I am of opinion that he has taken a just and moderate view.

It is true that the quartz has been *exposed in the* 1000-feet level only for a length of 50 feet, so far, but the regularity in its occurrence at so many levels, in its contents and general features, its size and value in the winzes below the 818-feet, and its aspect and value in the 1000-feet where driven upon, render its full development in unimpaired quality at the 1000-feet almost certain.

Water.

The water now apparently stands about 64 feet below the 818-feet level, having been lowered by that vertical distance during eight months' pumping. The history of the mine is that in the course of driving on the reef intermittent bursts of water occur, followed by casing off after draining away the influx. A heavy burst took place about six years ago in the 600-feet level, when it took nine or ten months' pumping to drain the water, which rose above the 500-feet level. At the 718-feet level west a burst stopped developmental work in that end for nearly three years. In the 1000-feet level, last August, the water broke out of the face, filling the drive, and rising in the reef to the 818-feet within the space of three days.

Much of the country driven through is open and fissured, affording numerous channels for the passage of water, which flows into the levels by tricklings, or even strong gushes, as driving on the reef proceeds. As the reef traverses the strata, it naturally collects the water along its walls, and often receives it into friable or fissured portions of its own substance. The tighter parts of the reef and strata hold back the water till it is suddenly released by driving, and sometimes with inconvenient results. A good deal of the water may have found its way thither from the main cross-course, which in its turn received it from the limestone-beds at each end of the mine, though I am inclined to believe that the Blyths Creek or eastern belt of limestone is responsible for most of it. Limestone is the most soluble rock in the district—the subsidence of limestone country in the neighbourhood of the mine, presumably—though the Tasmania drainage is well known. The burst in Dally's United proved the limestone line of country to be heavily charged with accumulated water, and the east end of the Tasmania Mine used to be wet, though most of the water has been in the western part. Although the mine is so near the Tamar, the water is not an infiltration from the river, as the strata are dipping towards the latter, and not away from it. The drainage of the Little Wonder Mine strata by the Tasmania shows some of the flow to have taken place from that direction. *The Tasmania water-logged strata may be* regarded as a channel of rather open country, running north-west and south-east, flanked by a highly-permeable broken sandstone and limestone belt on the east, and less pervious

slates on the west. Unfortunately, the store of underground water in the reef-channel shows at present no indication of approaching exhaustion. The signs of drainage of the surrounding country encourage the hope that the continued *pumping is having some effect on the great subterranean reservoir*. The subsidence of some of the limestone country appears to be due to the work at the Tasmania, but some of the superficial drying-up of swamps, &c., may be due to the succession of unusually dry seasons during the better part of the past decade. The constant pumping certainly is lowering the level of the ground-water in the mine. The *permanent ground-water* in any district is the residue of the rainfall not discharged at lower levels, or by streams or by evaporation, but percolating downwards through the strata until it is arrested by some impermeable rock. It then becomes stationary, or moves gently in the direction of easy flow. In the Tasmania there must come a time when, at an increased depth, the country will become tighter, the fissures which now act as water-conduits will close up, and the rock will be found comparatively dry.

How soon this will happen cannot be predicted; it may be at no great depth below the deepest level, or a good deal lower. But, obviously, it would be unsafe to lay out the work of the mine on the assumption that the water will henceforth begin to be a negligable factor.

There is therefore nothing for it but to proceed steadily with the attempt to obtain command of the water, under conditions that will admit of profitable working. It is possible that, by readjusting the present plant, the normal water-store may be handled, and work, *after a certain delay*, and at an undesirable cost, be carried on for a time as now; but this expedient would be temporary, and subject to the following drawbacks:—

1. Although the normal water-supply could be controlled, the same liability to heavy increases would continue as at present.

2. The average crushings for the last five years have *not* exceeded 30,000 tons per annum (the highest, 34,389 tons, last year). The standing charges can only be reduced by *increasing* the output, and opening the mine by two levels at a time.

3. In a few years the present state of things would recur, and the question of providing permanent plant would have to be faced, with this serious disadvantage, that there would be no proved reef underfoot, as now, to rely upon as the warranty for future expenditure.

The *present* pumping machinery is distributed between Hart's shaft and the main shaft. At the main shaft the Cornish plant (two 24-inch columns with three sets of 24-

inch plungers), of 3,500,000 gallons capacity, [illegible] from the 718-feet level the water delivered to it by t[illegible] inch hydraulic pumps at the 818-feet. At Hart's sha[illegible] [illegible] water is raised to the 600-feet level by a Cornish plant ([illegible] 1,000,000 gallons capacity), and then transferred to the main shaft, 718-feet level, whence it is pumped as mentioned above. The pair of Riedler pumps (each 500,000 gallons capacity) are used as auxiliaries during stoppages. All these plants, as the general manager points out in his last half-yearly report, are dependent on one another in such a way that if any stoppage happens to one of them the output of ore is immediately affected.

The water at present being raised each 24 hours amounts to something less than 3,000,000 gallons. Any new scheme for equipping the mine with a permanent pumping plant for continuous work, regardless of water, and on a scale commensurate with the requirements of the mine as a dividend-paying concern, must provide for a pumping capacity of from 6,000,000 to 8,000,000 gallons per day. Discussion of the particular description of plant, and of its motive-power, is not within the scope of this report.

Costs.

The yearly crushings of quartz for the last few years have been—

In 1898	26,370 tons
1899	23,350 „
1900	22,340 „
1901	24,930 „
1902	36,609 „

and the last half-year's mining, battery, and general charges have been £2 8*s*. 1*d*. per ton of quartz, equal to £3 10*s*. 5*d*. per ounce of gold.

Mr. Heathcote's estimate, that the gross cost will be reduced to £1 17*s*. 10*d*. per ton of quartz when the mine is properly equipped, appears to me to be based upon reliable factors. The last half-year's costs, as ascertained from the books at the mine, appear in comparison with it, as follows:—

	Half-year ending 31 Dec., 1902, on 1[illegible],893 tons quartz. £ *s*. *d*.		Mr. Heathcote's estimate of cost on 62,400 tons per annum. £ *s*. *d*.	
Mining..................	1 17 6	per ton	1 8 3	per ton
Reduction Works ...	0 8 3½	„	0 7 4	„
Head Office and Management	0 2 3½	„	0 1 7	„
Total..............	£2 8 1	„	£1 17 10	„

As the [illegible] of the quartz taken out from between the [illegible] and [illegible] feet levels has been at 13·6[illegible] dwts. gold per

ton) £3 6*s*. 3*d*. per ton, the profit per ton of quartz and ounce of gold is readily seen. A further reduction of 3*s*. 5*d*. per ton is estimated as possible when the output reaches 90,000 tons per annum, which, as said above, is expected to be the product of each 100-feet level.

I have gone through the description of the mine, and all the estimates of production and cost given in the General Manager's special report, and have carefully compared them in detail at the mine with the data upon which they are founded. I can endorse them as being substantially and reasonably accurate.

Summed up, the position is this, that between the 818-feet and 1000-feet levels there is every reason to believe that 180,000 tons of quartz exist, which, provided the mine is properly equipped with pumps, can be profitably extracted, and will give three years' work for the battery (of 105 heads).

The crosscut at the 1100-feet has not reached the reef yet, so that the only data for anticipations of the yield from greater depths are the past returns from the mine, and the appearance of the reef at the 1000-feet. If the reef descends with the regularity which it has always observed so far, many years' working may be predicted as the outlook for the future.

From my examination of the mine, I arrive at the conclusion that it can only be worked to advantage when it is equipped with appliances for raising the quantity of water indicated in this report. It has been shown to me that for this purpose the directors have invited competitive designs from large manufacturers of pumps and machinery in Europe, America, and Australia; and I understand that tenders for plant, based upon the full requirements of the mine, are now due at the office of the company's London agents. In order that the best engineering advice may be secured, the designs submitted will, I am told, be referred to the most eminent professional authority available in London, and that the final order is only to be given after the guidance of such opinion has been obtained. I am of opinion that this is a highly judicious and effective safeguard.

I have the honour to be,

Sir,

Your obedient Servant,

W. H. TWELVETREES,

Government Geologist.

W. H. Wallace, *Esq.*,

Secretary for Mines, Hobart.

JOHN VAIL,
GOVERNMENT PRINTER, TASMANIA.

Part B.
Reference 1/3.

511215

Special Report of Mr. Arthur Llewellyn, M. Inst. M. M.

TASMANIA GOLD MINE,
BEACONSFIELD, TASMANIA,
17th January, 1914.

MESSRS. JOHN TAYLOR AND SONS,
Managers,
6 Queen Street Place,
London, E.C.

GENTLEMEN,

In pursuance of your instructions, I have made a careful examination of this Mine—more especially with a view to future operations—and I have the honour to submit the following report thereon:—

In order to better appreciate the present position of the Mine, it may be well to take a cursory view of its past history and achievement.

Formed in 1877, the original Company, The Tasmania Gold Mining and Quartz Crushing Co., had, in the 20 years to August, 1896, crushed 299,000 tons of ore for a yield of 371,408 ounces of gold, equal to 24.84 dwts. per ton, by amalgamation only. In the next seven years, to June, 1903, the same Company crushed 198,850 tons for a yield by amalgamation of 152,813 ounces (equal to 15.37 dwts. per ton), and by chlorination of 46,622 ounces (equal to 4.68 dwts. per ton), or, together, 199,435 ounces, equal to 20.05 dwts. per ton.

The total value of this gold was £2,094,833, and the amount paid in dividends £772,072.

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Taken over by The Tasmania Gold Mining Company Limited, in 1903, the Mine during the past ten years has produced as follows:

Period.		By Amalgamation. Tons.	Ounces.	Per Ton dwts.	By Other Processes. Ounces.	Per Ton dwts.	Total Yield. Ounces.	Per Ton dwts.
1903-4	..	24,238	16,040	13.6gr.	3,560	2.22gr.	19,600*	16.4grains
1904-5	..	43,742	25,904	11.20gr.	4,744	2.4gr.	30,648	14.0gr.
1905-6	..	48,076	24,353	10.3gr.	8,561	3.13gr.	32,914	13.16gr.
1906-7	..	58,339	23,434	8.0gr.	6,920	2.8gr.	30,354	10.8gr.
1907-8	..	70,272	25,037	7.3gr.	5,265	1.9gr.	30,302	8.12gr.
1908-9	..	53,787	17,943†	6.16gr.	3,907	1.10gr.	21,850	8.2gr.
1909-10	..	67,113	15,334	4.13gr.	5,384	1.15gr.	20,718	6.4gr.
1910-11	..	53,564	14,741	5.11gr.	8,402‡	3.3gr.	23,143	8.14gr.
1911-12	..	51,899	11,569§	4.11gr.	9,840	3.19gr.	21,409	8.6gr.
1912-13	..	53,812	11,218	4.4gr.	9,987	3.17gr.	21,205	7.21gr.
		524,842	185,573	7.13gr.	66,570	2.14gr.	242,143	10.3gr.

* Fifteen months, July, 1903, to September, 1904.
† Including 2,906oz. from plate scrapings.
‡ Includes 797oz. from dismantling mill.
§ Includes 453oz. from dismantling mill.

The total value of this gold was £1,071,414, and no dividends were paid during the period.

Although the irregular plate scrapings and occasional inclusion of gold from extraneous sources, as noted, somewhat vitiate the results for the purpose of yearly comparison, it will be seen that the general tendency has been towards a lower grade, especially in the amalgamation yield, the latter being due to increasing sulphides. This fall in grade was, during the earlier years of this Company's history, more than redeemed by increased tonnage, and for a time good profits were made, amounting in one instance to £46,700 a year. But in 1906 a serious inundation was experienced, which, besides calling for excessive outlay, interfered greatly with the Mine developments, and in 1906-7 the profits had fallen to £17,600. This set-back the Mine has never recovered, and in 1908-9 there was a loss on the year's working of £2,300. The annual loss has continued, and in 1912-13 amounted to £3,030.

The Mine has in the meantime reached a depth of 1,500 feet, and the amount of ore in reserve at the end of 1913 was about 40,000 tons, or less than one year's crushing. This ore is considered to be not of higher average grade than that crushed during the past year, and further loss on the current year's operations is anticipated.

It is apparent that, apart from further accidents, the question of future success or continued failure must depend on one or more of the following considerations: Is the grade of the ore likely to improve in greater depth? Can a larger tonnage be secured to compensate the fall in grade? Can working costs be decreased to the extent necessary to convert the present loss into a future profit?

Taking first the question of improved grade: It will be seen from the figures given that whereas the old Company's average yield over the first 20 years of its existence works out at 1 oz. 4.84 dwts. per ton, the present Company's average during the past 10 years is slightly over ½ oz. per ton, falling from 16 dwts. 4 grs. in 1903-4 to 7 dwts. 21 grs. in 1913. The old Company's operations were practically all above the 815 feet level, while the present Company's have been practically all below.

The figures referred to, although they serve sufficiently for purposes of comparison, are not correctly representative of the actual gold contents of the ore, as the old Company's mill residues must at times have carried abnormally high values, whilst their unit ton has not been free from suspicion. On the other hand, the present Company's tailings heaps are known to contain 4 dwts. to 5 dwts. of gold per ton, a great part of which is profitably recoverable, while a heap of concentrates accumulated during the 10 years' working is estimated to contain 10,000 ounces of gold.

The amount of the old Company's gold losses is unknown, and it is therefore impossible to compare same with the present Company's unrecovered gold. I think it probable that they would be quite as great, so that the comparison may be allowed to stand. In any case, we know that above the 815 feet level the Mine produced £2,095,000, and that in the 685 feet below the production has been £1,071,000.

The recoverable values in shillings and pence per ton, as given by development samplings at successive levels below the 815 feet level, have been as follows:

Position Above.	Length Sampled.	Average Assay. s.	d.
915 feet Level	1,200 feet	76	9½
1,000 feet Level	1.200 feet	54	6½
1,100 feet Level	1,200 feet	33	2
1,250 feet Level	1,200 feet	22	8½
1,370 feet Level	1,200 feet	23	2½
1,500 feet Level	900 feet	28	7

From this it will be seen that there was a slight improvement in value from the 1,250 to the 1,370 feet level, and a marked improvement in the 1,500 feet. The latter, however, has been driven along only three-fourths of the available length, the remaining one-fourth at the west end being normally the poorest part of the shoot. If we assume for this one-fourth a value of 17s. 3d. per ton as contained in the corresponding section at the 1,370 feet level, the average for the whole 1,200 feet is reduced to 25s. 9¾d. per ton.

This still constitutes an all-round improvement, and as such may be regarded as a favourable point. There is, however, an unsatisfactory feature in connection therewith, for whereas in the levels above the values have been fairly evenly distributed along the entire length, with a gradual diminution towards the western end, in the 1,500 feet level the high values are included in a length of 300 feet, while outside this the values average but 12s. 4d. per ton. In view of this fact, and of the fact also that even in the richer section described, the value is still very much below the rich sections in the upper levels, I do not think the outlook is favourable for an improvement of the lode to anything approaching its former productiveness, and an examination of the reef in the lower levels of the Mine has confirmed me in that view. I do not, however, see any reason why a partial recovery should not obtain in depth, and in this respect should have no hesitation in recommending deeper sinking.

Can a larger tonnage be obtained? The factors which affect this question are (a) the possibility of increase in length or breadth of the lode, and (b) the rate of development thereon. As regards the former, the position of the Tasmania Mine is unusual. The geological conditions have been described by other observers, notably in Professor Frecheville's Report of October, 1904, and it is unnecessary to deal with them at length in this Report. The outstanding fact is that the lode, although beyond all doubt a fissure vein, and as such evidencing a considerable amount of movement along its walls, is singularly short for a vein of this description, at least in its ore-bearing portions, and is only productive where it traverses a comparatively narrow series of sandstones and conglomerates.

These it traverses almost at right angles to their bedding, and at the 715 feet level has a productive length therein of 1,450 feet, which at the 1,370 feet has decreased to 1,300 feet. At the 1,500 feet level it is probably less, but the drift has not yet been advanced to the western limit. Beyond the central sandstone strata, and where the conglomerate beds predominate, the vein splits up, and the branches become less and less productive. Beyond this again the conglomerates are succeeded by limestone beds, and into these the vein has not been traced. Some of the shallower levels of the Mine were carried forward into the limestones, but as they liberated carbonic acid gas in dangerous quantities, which interfered with the working of the Mine, no further attempts in this direction have been made in the lower levels.

It is considered that the lode, even if distinguishable as such in the limestone, will continue to be unproductive therein; but as other sandstone beds are known to exist beyond these limestones, it seems a reasonable assumption that if the fissure survives it may again become productive on entering these congenial strata. This view has encouraged a good deal of shallow and deep prospecting east and west of the Tasmania Mine in past years, not only by the parent Company, but by independent workers. Such work has been invariably disappointing, and one is reluctantly forced to the conclusion that the Tasmania reef does not continue into these beds, or, if existing, ceases to be productive. Apart, therefore, from the danger and inconvenience of exploratory work into and beyond the limestone beds in the deeper levels of the Mine, there seems every reason to believe that such work would fail in the desired discovery.

In regard to (b), quicker development in depth, the outlook is equally unfavourable. The average output of the Mine during the past 10 years has been 52,500 tons per annum. Taking the overall length of the ore-shoot at the 1,500 feet level as 1,200 feet, and assuming an average width of 7 feet, each 100 feet of depth may be expected to disclose 70,000 tons of ore, or, say, 16 months' supply for the mill. Ordinarily there should be no difficulty in deepening a well-equipped mine like the Tasmania at the rate of 100 feet per annum, or 200 feet if necessary; but here the conditions are exceptional, through the enormous burden of water.

Judging from past experience, it may be expected that each foot in depth the Mine is sunk will entail the pumping of 21 million gallons of water. As the united capacity of the three pumps installed is 6½ million gallons per day at normal speed, it follows that for every foot sunk over three days must be occupied in pumping alone, so that in order to sink 100 feet a year the work must be practically uninterrupted, and pumping proceed at

full capacity without intermission. The task is, perhaps, not an impossible one; but the figures given leave little margin for accidents, and under such conditions it will be conceded that sinking even 100 feet a year would be a creditable performance. As I shall presently show, it is not under existing conditions practicable; but I thought it well to present the case in its most favourable aspect, in order to make it clear that even under the most favourable circumstances there would be little prospect of increasing the speed of development in depth.

Finally, as to (c) Working Costs: These are admittedly high, and have varied during he past three years as follows:

Heads of Expenditure.	Cost per ton, 1911.			Cost per ton, 1912.			Cost per ton, 1913.		
	£	s.	d.	£	s.	d.	£	s.	d.
Administration and Survey	0	1	1.60	0	1	2.74	0	1	2.13
Development Work	0	4	4.74	0	3	10.09	0	2	10.07
Mining and Stoping	0	10	5.64	0	10	5.28	0	10	11.88
Pumping	0	7	4.98	0	6	11.14	0	5	6.13
Ventilation	0	0	5.50	0	0	3.12	0	0	1.86
Crushing and Tramming	0	0	9.26	0	0	9.58	0	0	9.72
Milling and Cyaniding	0	9	6.89	0	9	0.25	0	8	8.75
Surface Costs	0	1	0.88	0	0	11.70	0	1	1.67
Repairs	0	0	9.90	0	1	2.50	0	1	2.35
Railway	0	0	10.66	0	0	9.00	0	0	8.53
General Expenses*	0	1	3.26	0	1	2.49	0	1	9.39
	1	18	3.31	1	16	7.89	1	13	0.48
Credit, Freights and Sundry	0	1	2.25	0	0	11.87	0	1	4.62
	£1	17	1.06	£1	15	8.02	£1	13	7.86

* Rents, Insurance, Workmen's Compensation, etc., etc.

These figures seem to indicate a steady reduction, but it will be seen that the decrease is chiefly on Development work, which is an unfavourable feature, and on Pumping, which is largely adventitious, the flow of water having decreased since the draining of the 1,500 feet level. On further drivage of the 1,500 feet west, which has not yet been completed, or resumption of sinking, the flow would immediately increase. Nevertheless, it reflects great credit on your Engineer that, in spite of the reduced flow and increased depth, the efficiency of the pumping plant has continued to improve, being 76.525 in 1913, with an average flow of 12,929,748 gallons per week, as against 75.775 in 1912, with an average flow of 17,277,140 gallons. These pumps are steamed from a nest of boilers, which also supply steam to the compressor and feed pump, the coal consumption being 2.845 lbs. of Australian coal* per indicated h.p. hour, equivalent to 2.498 lbs. of Welsh coal. It is, I think, hardly probable that these excellent figures will be much improved upon, and consequently but little reduction of cost can be looked for under this head.

The other item which shows a steady though small decrease through the three years recorded is Milling and Cyaniding. The present cost, however, is still high, and the chief reason for this is the intractable character of the ore, which increases with the growing depth of the Mine. Only about 40 per cent. of the gold content is now recoverable by amalgamation, the remainder being associated with sulphides, which require grinding and roasting before cyanidation. Experiments have been made with sliming of the ore followed by direct cyaniding and vacuum filtration, but they have so far been unsuccessful. The treatment of this ore has for several years been the careful study of your Reduction Officer, who is a close observer and skilful chemist, and I do not anticipate that any great improvement on present methods is possible in the near future. At the same time, continued progress along present lines may result in further economies, and I think that with the Mill running at its full capacity treatment costs might probably be reduced to 8s. per ton.

Mining and Stoping charges, here as in most countries the chief item of expenditure, are also high, the contributing causes being the insecure character of the ground, high wages, and, apparently, inefficient labour. The only remedy which suggests itself is the employment of light stoping drills, and experiments are now being made in that direction, which seem to point to success. I do not think much benefit is likely to accrue, working on the small scale permitted by your present ore-reserve; but it is probable that with extended operations the general use of stoping drills of modern and efficient type would materially reduce the present high cost per ton appearing under this head. Development costs, on the other hand, are unusually low, due to the relatively small amount of development work carried on during the past two years. Under normal conditions the development charge could not be expected to be less than the average of the past 10 years, viz., 4s. per ton, and with deeper work might exceed that figure.

The following table of monthly costs for the first three months of the present financial year, and subsequent to the period of the previous table, will show that a gradual reduction in costs per ton still continues. It is probable that working under existing con-

* Australian coal = 12.733 B.T.U. Welsh coal = 14.500 B.T.U.

ditions here, with a regular output of 4,250—4,500 tons per month, and continuance of the present excellent local management, the net revenue expenditure might be maintained at the minimum figure shown, viz., 30s. per ton; but I do not think it would be safe to estimate on a lower figure for future work than the average of the past three years, which is 35s. 6d. per ton.

Costs per ton Milled.

	October. 4,190 tons.			November. 4,180 tons.			December. 3 947 tons.		
	£	s.	d.	£	s.	d.	£	s.	d.
Administration and Surveying	0	1	2.80	0	1	2.95	0	1	3.18
Development Work	0	1	3.81	0	2	7.74	0	1	3.65
Stoping and Mining	0	11	2.06	0	9	7.85	0	10	6.51
Pumping	0	4	10.38	0	4	8.10	0	4	7.43
Ventilation	0	0	0.70	0	0	2.55	0	0	6.52
Crushing and Tramming	0	0	9.10	0	0	9.15	0	0	7.73
Milling and Cyaniding	0	8	3.10	0	8	5.86	0	9	4.41
Surface Costs	0	0	11.82	0	0	11.07	0	1	2.85
Repairs	0	1	2.43	0	1	0.63	0	0	8.60
Railway	0	0	6.59	0	0	6.47	0	0	5.28
General Expenses	0	0	8.86	0	0	7.86	0	1	0.55
	£1	11	1.65	£1	10	10.23	£1	11	9.12
Deduct Freights and Credits	0	0	10.56	0	0	9.28	0	0	10.18
	£1	10	3.09	£1	10	0.95	£1	10	10.94

As previously noted, the chief reason for the present low figures is the abnormally low charges for pumping and development, due almost entirely to the cessation of sinking. The former is and must always remain an enormous burden on the working of the Mine, varying from £15,000 to £20,000 per annum according to the flow. As far as I am able to form an opinion, there is little hope of a permanent diminution in the amount of water to be pumped, and as the Mine becomes deeper the cost of pumping must proportionately increase.

There remains to consider a factor of paramount importance to all mining enterprise in this country, and to yours in an unusual degree—that of labour. Not only is the cost of labour excessive as compared with most mining fields, but from the conditions maintaining in this Mine the proportion of unproductive labour is very large. There is at the same time usually a shortage of good miners, and your Manager informs me that much of the highly-paid labour is inefficient. The following table, constructed from Mr. Heathcote's carefully prepared statistics, will show the average number of men employed daily in the various Departments during the past three years, and the average earnings:

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Particulars.	1911.	1912.	1913.
Underground—			
Number of Men Employed..	226.53	205.10	191.11
Wages Paid	£30.302.9	£27.732.5	£26.717.2
Average Per Shift..	9s. 2.8d.	9s. 4.2d.	9s. 8.6d.
Surface—			
Number of Men Employed..	117.07	113.26	111.79
Wages Paid	£13.244.8	£12.753.0	£13.306.0
Average Per Shift..	7s. 0.5d.	7s. 3.8d.	7s. 5.4d.
Reduction Works—			
Number of Men Employed..	104.10	97.3	89.8
Wages Paid	£12,316.8	£11.341.3	£11.066.8
Average Per Shift..	7s. 1.0d.	6s. 11.6d.	7s. 6.9d.
Totals *			
Number of Men Employed..	447.75	415.70	392.68
Wages Paid	£55.864.5	£51,826.8	£51,090

* Table does not include *Railway Department, which is run separately.*

This is a formidable wages bill for such a moderate output, and unfortunately the conditions do not point to any amelioration. As in other parts of the Commonwealth, labour is dissatisfied, and demands for higher pay are made with increasing frequency. Quite recently a section of this Company's employees, through their Association, brought the Company before the Arbitration Court on a plaint for raising the minimum wage in their Department. This was granted by the Court, with the result that wages in that Department have now increased 20 per cent. Should this movement extend to the other Departments of the Mine, as appears likely, it will impose an additional heavyburden where the load of costs is already unbearable.

The output of ore per man per shift worked is as follows:

Particulars.	1911.	1912.	1913.
Tons Crushed	53.564	51.899	53,812
Tons Per Shift, Underground	0.813	0.875	0.979
Tons Per Shift Underground, Surface, and Reduction Works	0.388	0.405	0.453

The production figure is small for white labour, due chiefly to the large proportion of unproductive labour already referred to. The apparent improvement is not real, but due to diminished development, as previously noted.

To sum up the general position as set forth: The Mine has not shown a profit since 1908. The grade of the ore mined has fallen during the present Company's working from (approximately) one ounce per ton in 1903 to (approximately) half-an-ounce in 1913. The length of ore ground has decreased from

1,450 feet at the 715 feet level to 1,300 feet at the 1,370 feet level, without any compensating increase in width. To compensate for decreased value by increased tonnage is considered impracticable. Working costs are high, and will probably remain so, while the growing demands of labour will in this respect constitute a continual menace.

The one redeeming feature in recent years has been the slight improvement in the general value of the lode between the 1,250 feet and 1,370 feet levels, and continuance of that improvement over a partial length in the 1,500 feet. It is not much to build on, but in view of the interests at stake and the magnitude of the establishment which has grown up here, I should under ordinary circumstances consider it sufficient ground on which to recommend you to sink at least another 100 feet, on the chance that this recovery might continue and lead to better fortune.

The position, however, in regard to further sinking is extraordinary; and although Grubb's shaft is an exceedingly good one and magnificently equipped, it appears practically impossible to sink it to another level. As previously explained, each foot of depth in sinking the Mine entails the pumping of 21 million gallons of water, the main channel of which is the lode. The lode traverses the enclosing rock at right angles to the stratification, and the harder strata serve to hold back the water, while the softer, porous bands afford ready channels for its flow. These latter when approached by drivage on the lode cause sudden bursts, and it is therefore necessary to construct flood gates between the shaft and the lode, to regulate the flow of water to the pumps. Until the ground between a new level and the level above is drained, drivage at the new level is marked by periods of moderate flow (two to four million gallons per day), leading up to an overwhelming burst, when the flood gate has to be closed and drivage suspended.

Grubb's Shaft is in the hanging wall of the lode, and has hitherto been in hard and only moderately pervious strata. It is now approaching the lode, and at the same time one of the loose water-bearing strata, which has unexpectedly taken a flatter dip It is therefore almost certain that before the shaft reached the depth of the next level a heavy burst would be encountered Apart from the danger attaching to this inundation, the water could not be dealt with by the two pumping units now in Grubb's Shaft, and Hart's Shaft would require to be brought into operation. To keep all three units supplied would require six 22-inch drawlifts in Grubb's Shaft, and further sinking under these conditions would be almost impossible.

The only alternative seems to be another temporary shaft below the 1,500 feet level, in impervious strata. To remain in such strata the shaft would require to be sunk on the dip of same (about 45 per cent.), and here Cornish pitwork would be unsuitable. This shaft would be in the footwall of the lode, to which crosscuts would be driven at successive levels, flood gates constructed, and the water drained by electrically operated centrifugal pumps. Grubb's Shaft would then be sunk as each successive level became dry, for the installation of the permanent pitwork.

It is estimated that the cost of the necessary plant in position, including the erection of electric generating station, would be £25,000; while the cost of sinking and pumping each "lift" of 100 feet, drivage of crosscuts, and construction of floodgates, would probably amount to an additional £25,000. There would then remain the cost of sinking Grubb's Shaft from level to level, extension of the permanent Cornish pumps, and the drivage of successive levels along the lode.

I find that in the past each level has cost about £65,000 for development, pump extension, and pumping. Under the proposed new scheme the cost would exceed this figure by at least £5,000, so that apart from the additional capital expenditure of £25,000 on plant, etc., you would be faced with a development charge of £70,000 on each ore block between successive levels.

The present financial position of the Company will not permit of your undertaking an expenditure amounting in the aggregate, probably, to £100,000; and, as shown in the preceding pages, the Mine is unfortunately not in a position to render you any assistance. The ore reserve is less than sufficient for one year's work, and if extracted in bulk without selection it is probable that the average value will scarcely pay to keep the Mine open under present conditions during the interval which must elapse before you could get to work on the proposed scheme.

To attempt the deeper exploration of the Mine on the lines proposed with the resources at the Company's disposal, including even the slowly realisable value of the tailings heaps, would, I think, be too risky an undertaking, and likely to end in failure before the next level has been attained. You would then be so far committed to the enterprise that the raising of fresh capital would be a necessity, while the future prospects would still remain as indefinite as they are to-day.

The question before me is, therefore—*not* whether the recent improvement in the lode sufficiently justifies the sinking of a winze, or the deepening of a shaft, or the opening up at ordinary and moderate cost of one more level; but whether, *in* face of the past history of the Mine, I am prepared to recommend the expenditure of £100,000 further capital and the possible reconstruction of the Company on the prospects now before me.

To this there can be but one answer, and I regretfully find myself in complete accord with the views recently expressed by your Manager. I therefore support his recommendation that after extraction of the richest and payable portion of the ore now developed, the Mine should be closed down.

I am, Gentlemen,

Yours faithfully,

ARTHUR LLEWELLYN.

5112221

Part 'B'
Reference 1/4

The Tasmania Gold Mine Ltd.

25 Patterson Street, Launceston,
March 3, 1914.

TO THE SHAREHOLDERS ON THE TASMANIAN REGISTER.

DEAR SIR (OR MADAM),

The Local Advisory Board have recently notified the Shareholders through the courtesy of the Press, as to the instructions issued to the Superintendent "to discontinue all development work and take out the payable ore reserves." The time that will be occupied depends upon the output that can be profitably maintained.

In order that Shareholders on the Tasmanian Register may be put upon the same footing as those on the London Register, the following cabled information is now given:

The Annual General Meeting will be held in London on the 19th March, when the Directors' Report and Annual Accounts will be presented. The latter show that the loss on working for the year ending 30th September, 1913, was £3,030, whilst the Cash Assets on that date amounted to £15,100.

Incidentally we may remark that this balance has now been reduced through loss on working during the past five months to about £11,000. It is estimated that the treatment of accumulated products at the Company's Reduction Works will occupy a period of three or four years, and will yield a total profit to the Company of from £35,000 to £40,000.

The Directors will, at the Annual General Meeting, propose that the final decision as to procedure should be deferred for a few months, in order to see if any alternative proposal to that of liquidation can be recommended. They will suggest as a possibility that they might be able to find a new property in Tasmania.

Herewith will be found the final part of the Superintendent's Annual Report—the earlier portion of which deals with the details of routine work—and a copy in full of the Special Report of Mr. Arthur Llewellyn, M.Inst. M.M., who was deputed by the London Board to visit the Mine and report as to its future prospects, which Report has been received this day.

Yours faithfully,

AUG. SIMSON,

Chairman of the Local Advisory Board.

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Final Portion of Superintendent's Report for Year ending 30th September, 1913.

ORE RESERVES.

Ore Reserves remaining on the 30th September, 1913, above the 1,500 feet level are now estimated as follows:—

	Tons.
Ore developed ready for stoping	36,061
Partly developed but yet to be drained	5,328
To be expected, when western part of lode is drained, a further	21,000

GENERAL.

During the past year Development Work at the 1,500 feet level has not been continued west, as it would entail cutting heavy water, but the Eastern Drive or Drives have been kept going continuously. At times values appeared to be improving, but these improvements did not continue, and, as may be seen from the values given in the body of the Report, the Eastern part of the Mine has been very disappointing.

The "horse" or mass of waste rock which split the lode into two parts at the 1,250 feet level dies out below that level, but another just below and beyond it was found on the 1,370 feet level. During the past year this has proved to extend about 50 feet below that level, and where the branches joined into one lode a zone of better values was met with. Unfortunately the lode was found to be again split at the 1,500 feet level, and over a greater length than in any of the levels above, excepting only the 500 feet level. From its appearance this "new horse" is not likely to cut out in less than 100 feet of depth. The remarks made in last year's Report hold good, except that this split is a new one, and not as was then supposed, the bottom of the one at the 1,370 feet level. The average value of the 1,500 feet level over the 940 feet driven to date has been just under 13 dwts. over an assay width of 7 feet; the same 940 feet in the level above had an average of 8.73 dwts. over a width of 5 feet of quartz. This improvement is encouraging, but as the average stoping width in this Mine is 50 per cent. greater than the assay width, and allowance has to be made for loss in tailings of 1½ dwts. per ton crushed, these values are not payable.

Each level costs about £20,000 for development and about £45,000 for pump extension and pumping. In the past the output from each 100 feet of depth has averaged about 100,000 tons. Reduction of output by selection for mining of only parts of the lode must increase the cost for pumping and development in inverse proportion. Such selection has been necessary in the last two levels. The continued improvement in the two lowest levels in the value of that part of lode near the main crosscuts shows in my opinion that increase in depth is not affecting values. The increase in depth is, however, affecting the mode of occurrence of those values: the percentage of contents that can be obtained by amalgamation is gradually decreasing, and the percentage contained in the concentrates and slimes is gradually increasing. The best extraction that could in the past be obtained from slimes tailings was 40 per cent. During the past few months Mr. Arthur F. Hosking, Assoc. M.Inst. M.M., the Company's Chief Reduction Officer, has succeeded after numerous experiments in working out a method of treatment which, on experimental lots of 1½ cwts., promises a further extraction of 85 per cent. from these slimes at a cost of about 8s. per ton. Extensions of Plant would be required to treat the whole of the current slimes. It will be seen from the above statements that it is difficult to come to a decision as to whether funds should be provided for the Plant necessary for deeper sinking, as proposed in the last Annual Report. The programme laid down three years ago has been carried out, and has not proved remunerative, but having in view the past expenditure on equipment, the Mine would appear to be worth further trial.

At the beginning of the financial year an increase in wages of 6d. per day was given to all adult employees, and it was not anticipated that the Company would be brought under the provisions of the Commonwealth Conciliation and Arbitration Act, an Act brought into being during a time of exceptional prosperity that has since continued, founded on popular misconceptions, and controlled as to its administration upon the anomaly, or rather mathematical absurdity, of fixing from time to time a "minimum" upon the basis of an average in which the controlling function is a prior lower "minimum."

The Company has been brought before the Court as regards certain of its employees, and steps have been taken that will in the near future bring all employees under the Act. As the relative rates of pay of the various classes have become adjusted during a period of 35 years' work, it is certain that any increase will be at the same average rate. The proposed award has now been published, and as far as can be determined beforehand by applying it to recent time-books, the increase is equal to over 30 per cent. on the Company's recent rates of pay, and to over 20 per cent. on present rates of pay.

Owing to the large amount of machinery required for pumping and treatment, the proportion of non-productive labour is excessive, and as regards that labour as a whole, no further efficiency can be obtained, for the mining conditions do not allow of any greater output.

I now estimate that the cost of obtaining and installing the necessary machinery for deeper sinking would, on account of the hampering conditions imposed, exceed £25,000; that the extra expenditure on wages during the minimum time of two years that would be required to instal and prove would amount to a further £20,000. In addition to this sum of £45,000, a loss on mining operations in the immediate future will also have to be faced.

These facts in my opinion render but one decision possible. I therefore recommend that the Mine be closed down, that such parts of the pitwork as will pay for removal be withdrawn as soon as the best of the ore now in sight is worked out, and that while the accumulated concentrates and slimes are being treated in the Grinding Plant, the Company's assets be realised as opportunity permits.

The necessity of awaiting the outcome of the Arbitration Court proceedings has been a source of worry to the whole staff, and my thanks are therefore especially due to them for their assistance throughout the past year.

I remain, Gentlemen,

Yours obediently,

C. F. HEATHCOTE, M.Inst. M.M.,

Assoc. M. Inst. C.E.,

Superintendent.

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511223

The Hon. J. E. Ogden, M.H.A.,
Minister for Mines,
HOBART.

Part B.
Reference 1/5

Dear Sir,

Following instructions contained in letters dated April 3rd and 7th, 1914, we have completed a detailed inspection of the Tasmania Company's Mine at Beaconsfield and beg to submit the following report:

Before entering into the details of the report itself, it is necessary to state the conditions laid down, which may be summarised as follows :-

1. That there was no desire to question the conduct of the Mine either under former or present management.
2. That the stability of the shafts and underground workings be reported upon with estimates to be furnished of probable cost of further sinking.
3. "That the report should not only deal with the future prospects as to the occurrence of gold at a depth, but should also consider the question of the financial resources required to fully carry on the working of the Mine at Lower levels."
4. The report to be of a general character throwing as much light as possible on the commercial prospects of the Mine in view of its further development at greater depth.
5. That the fullest information should be given to the general public in as clear a manner as possible.
6. That special attention should be given to the advisability or otherwise of extending the upper workings in either direction, and the probability or otherwise of opening up shoots of stone beyond the limits of the present drives.

7. And further, that consideration be given to the question of securing additional ventilation by connection with adjoining Mines.

These instructions are clear and definite and in the following report we deal with the position as we find it, without reference to, or criticism of, the past policy of management.

FINANCIAL POSITION.

A summary of the financial results since the Mine was taken over by the English Company may be given by way of prelude.

In 1903 the new Company started with an amount to credit of working capital of	£180,000
Since reconstruction in 1910 there has been called up	25,000
	£205,000.

The position on March 31st, 1914, was that this working capital (less cash in hand) had been expended. 545,708 tons of stone had been mined and treated for a return of 250,116 ozs. of gold - 9 dwts. per ton. No dividends had been paid and the Company had at that date a credit balance of £10,500.

In addition, there is an estimated profit on treatment of accumulated concentrate and slime, &c., of £40,000.

It is under these conditions that we are called upon to report as to the commercial prospects of the Mine involving chiefly consideration of the probable value of the reef below the present workings and cost of further development. (Schedule A. & B.)

GEOLOGICAL CONDITIONS.

For the purposes of this report it is not necessary to enter into any lengthy detailed statements regarding the geological conditions, seeing that numerous reports have been written from time to time in

connection with the Mine itself and the surrounding
Some reference is, however, necessary in order to expla
deductions arrived at in other portions of this report.

The country in which the reef has been worked consists of sandstones and conglomerates striking N.W. and S.E. and dipping about 45° N.E. The strike of the reef is N.E. cutting the country practically at right angles and dipping at varying angles but approximately 50° to the S.E.

The reef varies in width, and, in places, is split in two, and occassionally three branches, by "horses" of country rock. It is in varying widths from 1 ft. to 20 ft. - this is given in a general way - the actual stoping widths being dealt with later on. The average length of the reef worked is about 1400 ft. and may be regarded as one shoot, for, although values vary, there have been few blanks and practically the whole length has been stoped down to the 1370 ft. level.

Regarded as one shoot of stone, it shows a decided pitch to the N.E. and a gradual shortening for the last few levels from 1420 ft. at the 900 ft. to 1200 ft. at the 1370 ft. Level. The middle portion on the length of the reef contains the best values which lessen towards each end. The upper portions of the reef were undoubtedly richer than those at a depth and it is evident that the rich shoots were more regular in their downward course than they have been in the lower levels - say below 900 ft. Rich portions have frequently been met with in the lower levels, but erratic in their deposition, and no definite line of pitch could be laid down.

In the upper portions of the workings a strong fault cuts obliquely across the country rocks in a north westerly direction displacing the lode channels some 240 ft. northward. This fault has been followed along its course and subsequently the displaced portion found and followed for some hundreds of feet in a west and north-west direction. This extension of the reef channel is well defined, but the quartz is very small as a rule, though fairly large masses have been found but of a very low grade.

It is also to be noted that with increasing depth the character of the reef has altered in the respect that it contains a larger percentage of sulphide minerals, chiefly iron pyrites, than in the shallower levels. The immediate effect of this is that a lesser quantity of gold is recovered by amalgamation, and a larger percentage is accounted for in the slime and concentrate.

The western portion of the reef channel ends in black gritty sandstone and conglomerate country with the hanging wall fairly defined but, for a considerable distance after leaving the payable shoot, the ore channel becomes wider, consisting of country rock, small spurs, and threads of quartz. There is abundant evidence that the lower levels in the western portion of the Mine are in close proximity to the limestone as deposits of iron and copper sulphates and carbonate of lime are forming on the walls of the drives. It may be noted that the western portion of the Mine is generally in softer country, and more cheaply worked than the eastern.

The eastern ends of the lower levels are of a somewhat different character in that they show the hanging wall usually fairly well defined until a particularly hard natured sandstone is met with which almost approaches a quartzite in character. In this the reef invariably becomes erratic in its course and splits up into small stringers with occasional bunches of quartz of no value. Here also the limestone is not far distant, in fact, several of the upper levels have been driven into it.

The occurrence of such large bodies of water has frequently been commented upon and numerous theories advanced as to its source and extent. It seems tolerably certain, however, that the water being drained is contained within the limits of two parallel limestone deposits, some 1700 ft. apart, and running in a N.W. and S.E. direction These limestones probably have bands of impervious rocks or pug seams bounding them, sufficient to hold the water back from other parallel water channels that exist on either side. There is justification for this conclusion when it is remembered that with all the water raised by the Tasmania Company the mines of the Tasmania West Extd. on one side, and the East Tasmania, on the other, are unaffected by it.

PRODUCTION AND YIELDS.

For the 10 years, 1903 to 1913, the ore raised and crushed amounted to 524,842 tons which yielded 242,143 ozs., equal to 9.22 dwts. per ton.

For the 6 months from September 30th, 1913, to March 31st, 1914, there was mined and crushed 20,866 tons, yielding 7973 ozs., equal to 7.64 dwts. per ton.

Total tons crushed 545,708, yield ozs.250,116, equal to 9.16 dwts. per ton.

There are also in stock for treatment Concentrate and Slime with a recoverable content of 24,130 ozs. which added to the above gold yield gives a total of 274,246 ozs. equal to 10 dwts. per ton.

This Concentrate and Slime will yield on treatment an estimated profit of £40,000 as shown on Schedule annexed and the treatment with the present roasting, grinding and cyanide plant will extend over four to five years. (Schedule C).

WATER AND PUMPING.

The large quantity of water which has to be dealt with in the Mine has been the cause of the greatest outlay, and is the most serious item in working costs.

During the last 8 years it has varied from £21,000 to £15,000 per annum - the latter during the year 1913, when development had practically ceased.

With a fair proportion of development the cost of pumping would be at about £20,000 per annum - a serious charge on a yearly output of only 50,000 odd tons.

The pumping plant on the Mine is in 3 units of a combined capacity of 6½ million gallons per 24 hours running at normal speed.

Two units are in Grubb Shaft with the permanent plungers at 1500 ft. and one unit in Hart Shaft down to 1370 ft.

The beds of sandstones and grits cut through by the reef vary in composition in that some are fairly hard and impervious while others are comparatively soft and porous, forming channels for the underground waters.

These latter are the so-called "wet strata" through which most of the water comes into the mine workings.

The reef itself in this Mine is not the water channel (a fact which we desire to emphasise) though where the reef is a bit more open and fissured than usual there would be some flow of water into the advancing level, before the wet stratum was cut.

As these strata are cut when driving along the reef, heavy bursts of water occur, to control which it has been found necessary to fix concrete dams in the main crosscuts near the shafts at each level, so that the flow of water to the sumps can be regulated to the capacity of the pumps until that particular wet stratum has been drained and driving can be proceeded with. Apart from the coming water or daily flow, there is a reservoir of stored water in the country which has to be pumped. The west end, i.e., west of the main crosscuts from Grubb and Hart Shafts, is the wet part of the Mine in which most of the wet strata are met. The quantity of water to be dealt with may, and probably will, gradually lessen with depth, but it will still be heavy, and have to be pumped from increasing depths.

Since the management has contemplated ceasing operations and decided until closing down only to mine the stone already developed which was likely to be payable, no driving has been done West at the 1500 ft. level, the reasons being that the values underfoot at the 1370 ft. level West were low, and that there was the certainty of cutting heavy water and so adding to current expenditure on Pumping. (Schedule 1).

Under these conditions, with no development going on except the blocking out of ore at the two levels for stoping, only one unit of pumping plant at Grubb Shaft has been at work, raising 1,600,000 gallons per day, and pumping costs have been low, amounting to £6175 for the half-year ending 31st March, 1914. (Schedule E).

VENTILATION.

If subject only to ordinary mining conditions, the Tasmania Mine would to-day, having three big shafts only open, be a well ventilated Mine. The occurrence, however, of heavy Carbonic Acid Gas, probably from the limestones known to exist at both ends of the Mine just beyond the workings, complicates the matter. The difficulty of keeping the old workings open is elsewhere referred to and, during the past year or so when all expenditure has been reduced the old Main Shaft which is down to the 1000 ft. level and is connected by direct winze with the 1100 ft. level, has been blocked up. Several sets of timber have fallen away just below 200 ft. and this with general debris has fallen to the bottom at 1000 ft. and blocked the air way that was formerly conducted from the lower levels by a connecting rise from 1100 ft. It may be stated that all the necessary gear is in position to admit of this shaft being put into thorough repair, and the cost would not be a matter of much consequence.

This matter of Gas in the Mine workings has always been a serious one and naturally increases in depth. It has caused loss of time and has, therefore, increased the costs, as at certain changes in atmospheric pressure when the gas increases in the Mine, men and contractors have to be withdrawn from their working faces and moved to emergency places which are kept open for the purpose.

This trouble has never been made much of in the Mine Reports for obvious reasons connected with the supply of labor.

It is doubtful if any system of ventilation by currents of air between shafts, however well controlled and directed, will effectively clear the Mine of such a heavy gas and it is probable that if the Mine is worked to greater depths actual pumping of the gas will have to be resorted to.

The suggested scheme for improving the ventilation of the Mine by effecting a connection with the Bonanza Shaft is a feasible one but not considered advisable. The latter shaft is 1030 ft. from Hart Shaft in a south westerly direction, and according to reliable information, it is 1180 ft. deep. The shaft is 205 ft. from the west end of 1000 level in the Tasmania Mine, and 250 ft. from the end of the 1100 ft. level. The Collar of the Bonanza Shaft is 190 ft. above that of Hart Shaft, consequently the bottom of the shaft is approximately on the same level as the 1000 ft. drive, but as the latter level is now inaccessible, it would be very costly to connect these two points, besides which the pitch of the ore shoot takes it further from the Bonanza each lift. Another disadvantage is that the bottom of the shaft is in limestone.

For the satisfactory working of the Mine in the future, it is essential that airways must be established and maintained.

FURTHER SINKING.

Under favourable conditions, and with a big output to keep up, the Mine would have another lift sunk by this time, levels out draining the country, winzes sunk blocking out the roof ready for stoping, and all preparations made for sinking the next lift.

The Mine is now quite eighteen months behind in its development, and there is not stone enough opened up, regardless of quality, to keep the mill going until another lift is sunk.

During the last six to twelve months, since it became evident that the developments at the 1370 and 1500 ft. levels were not furnishing profitable stone and that the Mine was still working at a loss, development work has been reduced to a minimum, all idea of sinking has been abandoned, and the best stone has been stoped from above the 1250, 1370 and 1500 ft. levels.

In 1912, when further sinking was being considered, as a necessary work and imperative even at that time if the Mine was to be kept going, the whole question was thoroughly gone into by the superintendent and his staff, several alternative methods considered, much correspondence passed between the local office and the London Board, advice and tenders were sought from eminent engineering firms who specialise in pumping, and a mass of information obtained which resulted in a definite scheme being evolved and its cost ascertained.

Owing to the inclination of the sandstone beds cut by the reef any vertical shaft in a suitable position to work the known length of reef, whether sunk in the footwall or hanging wall must at some depth cut wet strata.

Grubb Shaft at 1500 ft. is close to one of them and early in the next sink must enter it.

The idea of sinking Grubb Shaft another lift under these conditions was abandoned by the Management as impracticable and an alternative scheme of sinking an underlie shaft on the footwall side of the reef in a comparitively dry stratum, pitched so as to keep in the stratum, and within a short distance of the reef at each successive level, was finally adopted although not carried into effect. A drive has been made at the 1500 ft. level to the proposed site for this shaft.

The Underlie shaft equipped with electrically driven Rees Roturbo Pump would be sunk direct to a vertical depth of 250 ft. before opening out. At that depth a crosscut would be put out towards the reef and a floodgate put in, after which the crosscut would be continued to the reef and a level driven to cut the water strata.

The water would then be pumped to a cistern at 1500 ft. where the plungers would take it to the surface and the country be drained until Grubb Shaft could be sunk comparitively dry to 1625 ft., it being proposed to make each lift 125 ft. vertical.

In further development the underlie shaft would always be sunk one lift ahead of the vertical pumping shafts. This would give a dry lift which could be blocked out and mined rapidly if labour were available, further sinking being pushed on at the same time.

Details of this scheme and estimated costs are given in Schedules F & G.

Under it, with machinery to order and install it would not be safe to reckon on developing the 1625 ft. level sufficiently to keep the mill going, in less than two years.

During and after that time all ore above the 1500 ft. level which would pay costs of mining and treatment, would be broken out, l after that the treatment of concentrate and slime in stock would go on.

It seems to us probable that the amount of water to be dealt with ll gradually lessen with depth but in providing for further sinking would not be safe to reckon on this.

As each lift of 125 ft. vertical will develop approximately 0,000 tons of stone on the anticipated length of reef, the cost of 72,000 plus further development on the level, blocking out, proportion f current pumping charges while mining the ore and contingencies ill amount to about £1 per ton on 100,000 tons.

The total charges against ore developed by next lift, assuming it to be 100,000 tons would then be

Development per ton	£1: 0: 0
Mining & Treatment and all charges except development.	1: 5: 0
	£2: 5: 0

Allowing for a loss of 1½ dwts. in tailings, the whole of the block for a length of 1200 ft. by a width of 10 ft. and the depth of lift 125 ft. vertical would have to contain 12½ dwts. per ton with a recoverable value of 11 dwts.

Any quantity less than 100,000 tons of stone of quality good enough to mine and treat would increase the cost per ton- for instance, if only 50,000 tons were available the cost per ton would be £2 and pro rata for developing and pumping.

The total recovery per ton on the 53,812 tons mined and treated in 1913 was 7 dwts. 21 grs. per ton. The total recovery from 350,447 tons mined and treated in six years 1908 - 1913 was 7.91 dwts. During the last six months 20,866 tons yielded 7973 ozs. equals 7164 dwts. per ton and the Mine has made a loss.

This last return is subject to qualification. The output being small the costs per ton for General Expenses and administration have been heavy, against which, little development work having been done it has not borne its fair share of costs for development and pumping

It will be seen that considerable improvement on these results ld have to take place at the next level to make it profitable.

In estimating the costs as above no account has been taken of the capital outlay on power and pumping plant etc., on which ry/interest and redemption would amount to an appreciable charge annum and per ton of ore, as it is assumed that the Mine would start ew campaign with a clean sheet.

(Schedule F & G).

SINKING GRUBB AND HART SHAFTS.

If the Company were in a good financial position and paying dividends that an extra capital cost of £20,000 to £25,000 would not matter much, would join in approving of the underlie shaft proposal as being safe d thorough - though costly.

If further sinking is undertaken, as matter stand, it has to be ne as economically as possible.

We have given the subject very serious consideration and have rived at the conclusion that it is quite possible to sink another ift of 125 ft. vertical with the present shafts, by doing so utilizing e present splendid pumping plant and incurring no additional capital itlay for plant.

The plan we suggest would be to complete Hart Shaft at once to a amp below the 1500 ft. level and install the permanent plunger workings here, so rendering available a unit of pumping plant capable of dealing ith 2,500,000 gallons per day, which, at present, is not in use.

The Western Level at the 1500 ft. to be driven to cut the water ede so draining the Mine to that level.

The sinking of Grubb and Hart Shafts to be then proceeded with, ith draw lift workings to handle the water as has been done in previous inks.

All three units of pumping plant would then be in operation when a urst of water took place and after it was mastered the shafts could be unk concurrently or one sent ahead of the other as conditions suggested. e position is assisted by the fact that two floodgates are already ilt, one in the main 1500 ft. Crosscut, the other in the drive, leading the rise which connects with the bottom of Hart Shaft.

THE RECONSTRUCTION IN 1910.

When the Company was reconstructed in 1910 it was after

7 years of work during which the whole of the working capital of £180,000 has been expended and no return whatever had been made to the Shareholders.

Grubb Shaft was then down to 1500 ft., Hart Shaft to 1370 ft. and the reef had been cut by the Main Crosscut from Grubb Shaft at the 1370 ft. level.

The circular issued by the Board at the time gives the reasons for reconstruction, shewn by the following excerpts:-

> "Necessity to provide further working capital if the deepen-
> "ing of the Mine and the exploration of the reef at the
> "1370 ft. ~~and~~ and 1500 ft. levels are to be regarded as essential
> "for the proving of the mine below the poorer zone in which
> "during the last three or four years the operations have
> "been carried out".
>
> "The Mine has a splendid equipment".
>
> "It is not an uncommon experience for Mines to pass through
> "an impoverished zone and to eventually recover their
> "former value".
>
> "Our financial position is that we have about £5000 in hand
> "out of which we shall have to pay for Filter Press, &c.
> "£1100.

Under these conditions the Company was reconstructed so as to provide a further working capital of £62,500 of which £25,000 has been called up.

The definite purpose for which the additional capital was provided, viz., the exploration of the 1370 and 1500 ft. levels, has been achieved, and in the course of that work the Superintendent has come to the conclusion that further exploration in depth is not warrented; his opinion has been confirmed by Mr. Llewellyn after investigation, and has been adopted by the Board.

REDUCTION WORKS.

The stone as it comes from the Mine to the elevated brace at Hart Shaft passes through rock breakers to bins, whence it is conveyed by electric tram to the mill a mile away.

At the mill 40 heads of 1000 lbs. stamps crushing 4½ tons a day each through punched screens having 12 holes to the linear inch, deal with the present output.

The old 65 heads have been dismantled but if the output were increased additional 1000 lbs. stamps would easily be added- shed, foundations, &c., being there.

The pulp, after passing over the amalgamating tables, is classified and concentrated on Card Tables.

The Concentrate is ground and amalgamated in Forwood- Down Pans, roasted in Edwards Furnaces, cyanided by agitation and filter pressed the extraction being 90% of the assay value.

The tailings from the concentrating tables is classified into sand and slime. The latter going to the storage dam and average about 3 dwts. per ton, though latterly it assays higher than this.

The plant under the new process of treating this cannot deal with the current output from the Mill, but on what is treated makes an extraction of 80% of the content at a cost of 8/- per ton.

Portion of the classified sand is cyanided by percolation but no profit is made. This sand averages a content of 1½ dwts and ½dwt is extracted at a cost of 2/- per ton.

It is stated that better extraction cannot be made without fine grinding, agitation and filter pressing, a process which would cost more than the value of the recovered gold.

(Schedule H).

EXPLORATORY WORK.

Regarding the question of parallel reefd existing or likely to exist to the North and South of the main Tasmania lode it may be pointed out in this connection that extensive prospecting work has been done by the Tasmania and adjoining companies without meeting with any tangible success. The chances appear remote for finding any parallel reefs, seeing that the country is practically driven across for many hundreds of feet. Some 1600 ft. to the south of Hart Shaft the Tasmania Company had been prospecting from a shaft 100 ft. deep and at this depth crosscuts have been driven covering over 700 ft. of country. At varying depths from near the surface down to 1250 ft. the footwall or noth country has been cut across with the object of finding other parallel reefs, whilst the shafts and crosscuts to the lode have well prospected the hanging wall country.

At 1250 ft. from Hart Shaft a crosscut has been driven 655 ft. Northward.beyond the main reef channel, the end being in clean country. At 370 feet from the main reef, a small irregular reef was cut and followed east for 210 ft., but, though carrying gold, it was too small and too low grade to pay.

Whilst the current work was going on at 1000, 1100, 1250 and 1500 ft., and in other levels above, prospecting drives have been put out at intervals into the footwall and hanging wall country, and, in places small veins have been followed. At the end of the three lowest levels going east the reef channel was kept in the middle of the drive to avoid missing any branch reefs that were likely to "make" off the main reef channel. At times off shoots were met with and followed a reasonable distance until proved poor or perhaps cutting out altogether.

To summarise the general conditions in the east and west end of the workings at 1250 ft. and 1370 ft. it can be said that the payable shoot has been contained within a length of about 1400 ft. and for considerable distances beyond that length the levels have been driven sufficiently far, and cross drives put out, to enable us to say that all reasonable precautions have been taken to thoroughly explore the country beyond the limits of the payable ore shoot.

The whole of the upper workings of the Mine and down to 1000 ft. are inaccessible, but the old records show that down to 715 ft. the levels were continued to the west and north west many hundreds of feet beyond the limits of the payable ore shoot.

In the early stages under the present management, a number of the upper levels were open and wherever possible the old stopes were closely examined and in places further tested with a view to ascertaining whether anything of value had been left. In some instances small blocks of ground were found containing payable quartz which was taken out as opportunity offered. These portions were, as a rule, near the east or west ends of the payable ore shoot, and, consequently did not offer much inducement to follow them beyond a reasonable distance when the ore became of low grade. As there is considerable movement and settlement always going on the keeping open of the workings requires constant attention and in the upper portions of the Mine it is practically impossible to keep them open. The existence of interbedded limestone at each end of the Mine beyond the payable ore has been proved and unnecessary extension of levels into this, and the consequent liberation of carbonic acid gas, has had to be avoided.

CONDITIONS OF WORKINGS AND PLANT.

As stated under the head of Exploratory work, the workings above the 1000 ft. level are now inaccessible, and, owing, to movements in the country due in a measure, to the extensive openings made and to the drainage of the country, they require constant work and attention to keep open the drives, passes and airways.

The conditions of the Old Main Shaft is referred to under the head of ventilation.

Hart Shaft. Is in good condition down to 1370 ft. below which it is connected with the 1500 ft. level by a vertical winze 8 ft. x 4 ft., timbered and centred. This winze has passed through the reef at 1470 ft., so that the shaft from thereon will be on the footwall side of the reef.

Grubb Shaft. Has caused a lot of trouble down to the 500 ft. level having had to be practically retimbered to that level recently and there is still movement going on so that it requires constant attention. The upper portion goes through alluvial drifts, but below 500 ft. the shaft is in good solid country.

The condition of this shaft is unsatisfactory at present in that it does not admit of direct winding from 1500 ft. to the surface, although the capstan gear is available right through. At present the ore is hauled from 1500 ft. to 1370 ft., trucked along a connecting drive to the Hart Shaft, and then hauled to the surface.

All the winding gear, and pumping machinery, are in excellent order and working smoothly.

CONDITIONS OF UNDERGROUND WORKINGS.

The 1100 ft. level is accessible from end to end of the workings, but the western portion in particular has given trouble and needs constant repairs. Near the extreme west end and a boxed rise is open to the 1000 ft., which is the only accessible travelling way to that level. The 1250, 1370 and 1500 ft. levels are all in good repair at the present time, but even these comparitively recent levels require constant attention also.

The stopes are well timbered and the filling is in a fairly well advanced state. The matter of timbering and filling is an important item in this Mine as the movements in the country demand that the filling should be systematically carried out.
The filling material is obtained at a surface quarry adjacent to Hart Shaft - a series of shallow drives being put in and stopes formed which/are held up by old waste stulls. These stopes are allowed to remain open for some time, the timbers are then shot away, and the rock allowed to gravitate into conveniently placed passes. The material is then trucked to the shaft and sent below to the return trip of the cage. The actual filling below is so arranged that the material is sent into the stopes and, by rilling, reaches its place with little or no shovelling.

GENERAL.

OUTPUT. On the formation of the Company in 1903 it was evidently the intention to provide for an out put of 80,000 to 90,000 tons per year. It was with this object that Grubb Shaft was sunk and the Mine equipped with such powerful pumping and winding machinery. The average amount of material treated, however, has been 55, 000 tons. With this output per annum the standing charges for pumping, winding, administration, &c., overweight it and form too large a proportion of the costs per ton.

ORE RESERVES. These were estimated at 30th. September, 1913, at 41, 389 tons since then 20,866 tons have been mined and treated and no further driving has been done at the 1500 ft. level. Net 20,000 tons + 500 x 7 x 30 = 25.000 tons - 55.000 t.

The Mines during the last seven months has been treated as a Mine about to be closed down as far as possible the best stopes have been worked.

Present estimated ore reserves are shown in Schedule J.

SAMPLING & VALUATION OF ORE. A close and systematic method of sampling the reef has been in existence and the results duly recorded on the plans. The samples have been taken at intervals of 4 to 5 ft. apart, and the results recorded for all levels, rises, winzes and stopes. Consequently, a complete record of the width and value of the reef can be seen from the 815 ft. to the 1500 ft. level, with the exception of the last few months. During the latter period sampling has been discontinued, but as the 280 levels and rises had previously been done a fair estimate can be made of quantities and values yet remaining to be stoped above 1500 ft. The battery results for the last few months are also a reliable means of getting at the values of the working stopes.

There is a greater width of ground stoped than is sampled and it is found in practice that the tonnage of stone broken and sent to the Mill is 50% more than the estimated tonnage worked out on the sampled widths and the value per ton 33% less.

The values given in the annual reports are on the sampled widths, and are subject to this deduction of one third, to represent the value per ton of the stone as sent to the Mill.

This is meant to be made clear to the shareholder in such paragraphs as the following, taken from the report for 1913.

"The total distance opened up to the date at the 1500 ft
"level has been 940 ft., average width of quartz 7 ft.
"Average assay value just under 13 dwts., the stoping width being
"about 10½ ft."

but may not always be understood.

In this case, if the whole of the stone broken from the block were accurately represented by this sampling which is the only along one side, the value of the stone sent to the Mill would be 9.3 dwts. per ton instead of 13 dwts.

The quantity of pyrites in the stone has increased with depth and the proportion of the gold recoverable by amalgamation has gradually fallen with a corresponding increase in treatment costs due to concentrat and the treatment of the concentrate by fine grinding, roasting, cyanidin by agitation and filter pressing.

Costs. Some economies could be effected but none of such importance as to effect the main issues being considered at present, viz., values and cost of deeper sinking.

A highly qualified staff would always be necessary to handle and control to the best advantage power and pumping plant of such capacity, and the general conditions call for the very highest skill in management.

The way to economize in administration cost would be to increase the output and so get the very best value out of all standing charges with corresponding reduction per ton of ore.

Labour. Wages, at present, are not high in comparison with other mining fields, but are subject to the conditions and regulations prevailing in Australia and may, therefore, be expected to rise.

In regard to supply of labour, the Mine is at a disadvantage in being isolated, although climate and living conditions are good.

The bulk of the Driving, Rising and Stoping and any other work that can be arranged for is done by contract.

(Schedule K).

DIAMOND DRILLING. A suggestion has been made that the lode channel might be tested below 1500ft. by the use of the Diamond Drill. We consider that little satisfaction would result, as experience shows that the reef frequently varies in width and the values also vary considerably. The boreholes might pierce the reef channel in a number of poor places, or, on the other hand, they might do so in an abnormally rich part, and whatever the results they would be unconvincing as regards the true value of the reef. Boring is not necessary to prove that the reef goes down, that is sure, and it is likely to continue to great depths.

SUMMARY AND CONCLUSION.

The Mine is practically worked out down to 1500 ft.

There have been blocks of good grade ore down to the 1500 ft. level, but the stone as stoped in bulk and sent to the Mill has been unpayable.

The reef is going down strongly, but there is no evidence on which to found expectation of increased values at the next or lower levels, though there is always a possibility of that happening.

On that possibility, and if we were reporting for Shareholders, we would not recommend the expenditure of £50,000 to £60,000 to sink the shafts and develop another lift.

We are,

Yours faithfully,

W. H. Cundy

Fawcett

SCHEDULE " A ".

SUMMARY OF FINANCIAL POSITION.

June. 1903.	The Tasmania Gold Mining Co., Ltd. started with a working Capital of	£180,000
	The Tasmania Gold Mine, Ltd., has called up since reconstruction in 1910.	25,000
		205,000
Mar. 31st., 1914	To Credit (liquids assets)	10,500
	Loss	£194,500

Aug. 1910.	There was to credit	£5,000
	call on reconstruction produced	25,000
		30,000
Mar. 31st. 1914.	To Credit	10,500
	Loss since reconstruction in 1910.	£19,500

This loss is now represented by value of Plant and recoverable in concentrates and slime.

SCHEDULES.

A Summary of Financial position.

B Comparison of Financial position for three years.

C Production and Yields.

D Quartz Treated, Gold Produced, Average Yield &c. from 1877 to 1914.

E Annual Expenditure for Pumping.

F Estimated Cost of Machinery (Underlay Shaft).

G. Cost of Developing Levels to 1625 ft. under new scheme.

G1 Cost of sinking the two Vertical Shafts.

H Retreatment Products in Stock.

I Values along 1370 ft. level west.

J Ore Reserves.

K Contract Prices.

SCHEDULE "B".

COMPARISON OF FINANCIAL RESULTS FOR THREE YEARS.

For year ending 30th September	1911	1912	1913
Tons mined	53.590	52,918	53,764
Assay value to the battery in dwts.	9,436	8,252	9,219
Balance Sheet - loss on Revenue A/c	£3,653	3,736	2,703
Capital Expenditure	2, 113	---	---
Deficit.	£5,766	3,736	2,703
Excess Gold from residue heaps less treatment cost	£2,404	13,500	9,826
Deficit on Mining Operations	£8,170	17,236	12,529
Development @ 4/- per ton	10,718	10,383	10,757
" actually spent.	11,775	9,968	7,636
Difference	£1,057	415	3,121
Deficit on Mining with proper proportion of development	£7,113	17,651	15,650
Pumping with proper development should have been	£18,854	18,033	18,990
actual expenditure	19,867	17,986	14,819
Difference	£1,013	£47	£4,171
Deficit on Mining with proper proportion of development, Pumping, and Pump extension would have been.	£6,100	£17,698	£19,821

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SCHEDULE "C".

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PRODUCTION AND YIELDS.

	Tons.	Ozs.	Per Ton. Dwts.
1904	24238	19600	16.16
1905	43742	30648	14.0
1906	48076	32914	13.66
1907	58339	30354	10.33
1908	70272	30302	8.5
1909	53787	21850	8.08
1910	67113	20718	6.16
1911	53564	23143	8.58
1912	51899	21409	8.25
1913	53812	21205	7.87
To 31/3/14	20866	7973	7.64
	54508	250116 -	9.16

Add recoverable Value of		
Concentrate on hand	9088	
Slime " "	15042	
	274246 -	10.0

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SCHEDULE "D".

Quartz treated, Gold Produced, Average yield and Dividends paid in varying periods from 1877 to March, 1914:-

Period	Tons Treated	Gold Produced ozs.	Average per ton dwts.	Dividend £.
1877 to 1896	299,000	371,408	24.84)	
1896 to 1903	198,850	199,435	20.06)	772,072
1903 to 1913	524,842	242,143	9,22	
Sept. 1913 to Mar. 1914	20,866	7,973	7,64	

SCHEDULE "E".

Showing the annual expenditure for pumping

From 1906 to 31st March 1914:-

1906.	£18,116.
1907.	21,044.
1908.	19,186.
1909.	19,260.
1910.	17,944.
1911.	19,898.
1912.	17,985.
1913.	14,819.
Half-year ending 31st March, 1914.	6,175.

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SCHEDULE "F".

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ESTIMATED COST OF MACHINERY FOR SINKING

UNDERLIE SHAFT.

3	Generator Sets with direct coupled Alternator	£2940.	
	Generating Station Switch Gear	230.	
	Wiring Engine Room	50.	
	Tasmania Cable - 9000 ft.	3510.	
6	End Connecting Boxes	27.	
6	Straight through Connecting Boxes	21.	
	Switch Gear (Underground)	75.	
	Electrically driven Rees Roturbo Pump.	1455.	
	Testing at Works	90.	
	Piping 10" 960ft. with bolts & joints	352.	
	Extras - Bends, Tees, Sluice Valves	105.	
3	Ventura Meters with Recording Charts	550.	
			£9405.
	Duty, Freight, Shipping Charges		£3868.
	Quotation for piping, Cast Steel Fittings, Bands, Tees, Crosses, Stop Valves, Gun Metal Seats		£365.
	Cleats for Cables	£185.	
	Steam Pipes	20.	
	Bolts, Nuts, Joints	50.	
	Foundations 161 cub. yds.	697.	
	Condenser - Moving & Installing	250.	
	Oil Separator	150.	
	Engine Room Wall Cutting	30.	
	" " Flooring & Painting	100.	£1482.
	Erecting Engines, Wiring, erecting Pumps & Pipe Lines	£156.	
	Sundries	78.	£234.
	Contingencies		£244.
	Total -		£15,699.

If Venture Meters be cut out would reduce total by £1290.

If only 2 Generating Sets, two pumping Sets, Switch Gear for three, but only two lines of cable the price would be £11,350.

With cheaper cable and no meters the price would be further reduced by £860.

If only two Generators Sets and three Pumping Sets and Switch Gear for three, but only two cables installed the price would be £12,500.

UNDERLIE SHAFT SCHEME.

Cost of developing Level at 1625 feet.

NEW UNDERLIE SHAFT TO 1750 FT.

Machinery	£12540.
Chamber	400.
Foundations	80.
Sinking 250 ft. @ £26	6500.
Floodgate	350.
Opening out below & contingencies, say	2130.
	£22000.

DEVELOPMENT FROM GRUBB SHAFT.

Crosscut 120ft. @ 50/-	£300.
Floodgate	350.
Main Drive 350ft. @ 40/-	700.
Rises & Winzes 500ft. @ 40/-	1000.
Fluming 400ft. @ 10/-	200.
	£2550.

GRUBB SHAFT TO 1625 FT.

Sinking 125 ft. @ £45	£5625.
Plat	150.
Extending Pump Work	3000.
	£8775

HART SHAFT.

Driving 1500 ft. Level West 300 ft. @ 40/-	600.
Sinking 160 ft. @ £15	2400.
" 125 ft. @ £25	3125.
Extension of Pumps	1700.
	£7825.

VENTILATION.

Clearing Main Shaft	£100.
Rise Connection 1625 ft. to 1100ft. 400 ft. @ 40/-	800.
	£900.

The probable cost of Pumping during 2 years will be £30,000.

The total cost would then be -

New Underlie Shaft to 1750 ft.	£22,000.
Grubb " " 1625 ft.	8,775.
Hart " " "	7,825.
Development	2,550.
Ventilation	900.
Pumping during 2 years	30,000.
Total -	£72,000.

SCHEDULE "G1"

SINKING THE TWO VERTICAL SHAFTS.

GRUBB SHAFT.

Sinking 125 ft. at £45 per foot	£5625.
Plat	150.
Draw Lift Work	1000.
Permanent Pump Work	2000.

HART SHAFT.

Sinking 160 ft. at £15 per foot	2400.
Permanent Pump Work	2300.
Development	2550.
Sinking further 125 ft. at £25 per foot	3125.
Ventilation	900.
Driving west at 1500 ft. 300 ft.	600.
Pumping during 2 years	30000.
Total -	£50650.

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Part Reference 1/6

CHIEF INSPECTOR OF MINES,

HOBART, 31st May, 1923.

Sir,

In compliance with your instruction for a report on the Tasmania Gold Mine, I beg to submit the following:-

The Tasmania Gold Mine is situated at Beaconsfield, twentysix miles from Launceston. It is two miles distant from Beauty Point and connected to it by a light Railway owned by the Company. The Reef consists of Quartz which trends Easterly and Westerly. It is a true fissure vein traversing almost at right angles to their bedding a series of Sandstone and Conglomerates. The reef terminates at the West on a fault, and at the East it dies out in the limestone. The fault appears to have the same dip as the Eastern Limestone, the reef varying very slightly in length at the various levels. The length of the reef may be taken as about 1300'. The reef varies in width - sometimes reaching a width of 20', but thins out Easterly. The average width may be taken as 7'.

The Mine was discovered in the year 1877 by Mr. Dally. In that year the mine was floated into a Company known as "The Tasmania Gold Mining and Quartz Crushing Company." From 1877 to 1896 this Company treated 299,000 tons of Ore for a return of 371,408 ounces of Gold by amalgamation. From 1896 to 1903 this Company treated 198,850 tons for a yield of 152,803 ounces of Gold by amalgamation, and by chlorination obtained 46,622 lbs. of Gold, equal to 4,68 dwts. per ton, being a total of 199,435 ozs. of gold equal to 20.05 dwts. per ton. The total value of Gold was £2,094,833 of which £772.072 was distributed in dividends.

The Tasmanian Gold Mining Company Limited acquired the mine in 1903 and worked it until operations ceased in 1914, except for the few months the mine was worked on tribute by a co-operative party under arrangement with the Tasmanian Government.

From 1903 to 1913 this Company crushed and treated 524,842 (524,832) tons for a yield of 242,143 (251,947) ounces, and no dividends were declared.

The following is a summary of ore treated and gold recovered. The residues are stated to have contained from 4 to 5 dwts. of gold.

PERIOD	TONS	OUNCES	YIELD PER TON.	
			dwts.	grs.
1903- 4	24,238	19,600	16	4
1904- 5	43,742	30,648	14	0
1905- 6	48,076	32,914	13	16
1906- 7	58,339	30,354	10	8
1907- 8	70,272	30,302	8	12
1908- 9	53,787	21,854	8	2
1909-10	67,113	20,718	6	4
1910-11	53,564	23,143	8	14
1911-12	67,113	21,409	8	6
1912-13	53,812	21,005	7	21
	524 832	251,947		

The original Company obtained their yield above the 815 ft. level and the latter Company below that level to the 1500' level, which was the lowest level worked.

It would appear that a poorer zone was entered at 1100' level and that at 1500' the values were improving and it is to be regretted that an additional level was not opened up to demonstrate if the values continued to improve. The history of Quartz mines throughout the world has been that poor and rich zones have been encountered. If mines had been stopped when values became poor some of the best mines of the world would have terminated their existence at shallow depths. I had the opportunity of several visits to this mine during the last year it operated, and I was of the opinion that there was every possibility of a better zone of enrichment. To test this I advised that a winze should be sunk below the 1500' level. This was done about 400' East of Grubb's shaft, and reached a depth of about 20'. The values were very encouraging at the last sampling and gave an assay of 1oz. per ton for a width of 5'. The values in the block immediately above this winze were very low, being less than 10 dwts. per ton. The future of the mine with regard to the reef which the Company worked depends largely on the improvement of values below the 1500' level, and the indications certainly lead one to expect that an improvement is very probable below that level.

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The Company gives the following values over a distance of 296' East of Grubb's Xcut at the 1500' level.

BLOCK.	LENGTH IN FEET.	WIDTH.	AVERAGE VALUE. dwts.
551	94	5	21½
552	23½	7	45¾
552	81½	2½	13¾
553	94	4	13¼

The drive East of the Xcut at the 1370' level for a distance of 292' gave the following values:-

BLOCK.	LENGTH IN FEET.	WIDTH.	AVERAGE VALUE. dwts.
355	70	5¼	5½
355	39	2¼	trace
356	83	3¾	3½
357	72	4	1½
357	28	3½	33

The average values at the 1500' level are given as 13 dwts. over an average width of 7' for a length of 940'. In the 1370' level the same 940' had an average value of 8.73 dwts. over a width of 5', which shows a very marked improvement of values at the 1500' level. This and the improvement in the winze East must be considered most encouraging. The reef averages about 7' in width, and the management estimated an output of 100,000 tons for each 100' of depth. On page 27 of the Company's 1912 report the Superintendent's report states "that to date the total distance driven on the 1500' level is 359' including the hanging wall stone, the average width of quartz is 5½', average assay 22½ dwts, the stoping width being 7'." The values of the hanging wall branch have been as follows:-

BLOCK.	LENGTH IN FEET.	AVERAGE WIDTH.	VALUE DWT.
552	40	1½	72½
553	100	2	24¾
554	42½	4	12½

The working expenses for the years 1911, 1912 and 1913 were as follows:-

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Heads of Expenditure.	1911 Cost per ton. £.	1912 Cost per ton. £.	1913. Cost per ton £.
Administration & Survey	0- 1- 1.30	0- 1- 2.74	0- 1- 2.1[illegible]
Development Work	0- 4- 4.74	0- 3-10.09	0- 2-10.0[illegible]
Mining & Stoping	0-10- 5.64	0-10- 5.28	0-10-11.88
Pumping	0- 7- 4.98	0- 6-11.14	0- 5- 6.1[illegible]
Ventilation	0- 0- 5.50	0- 0- 3.12	0- 0- 1.88
Crushing & Tramming	0- 0- 9.26	0- 0- 9.58	0- 0- 9.72
Milling Cyaniding	0- 9- 6.69	0- 9- 0.25	0- 8- 8.75
Surface Costs	0- 1- 0.88	0- 0-11.70	0- 1- 1.67
Repairs	0- 0- 9.90	0- 1- 2.50	0- 0- 2.35
Railway	0- 0-10.66	0- 0- 9.00	0- 0- 8.5[illegible]
General Expenses	0- 1- 3.26	0- 1- 2.49	0- 1- 9.[illegible]
	£1-18- 3.31	£1-16- 7.89	£1-15- 0.48
Credits Freights etc.	£0- 1- 2.25	£0- 0-11.87	£0- 1- 4.62
	£1-17- 1.06	£1-15- 8.02	£1-13- 7.86

The loss on working for 1911 is not available, but for 1912 the report shows a loss of £4,165, and for the year 1913 - £3,030. The accumulated residues are stated to contain 4 to 5 dwts. of Gold per ton, and a heap of concentrates to contain 10,000 ozs. of Gold. It would appear that with a good milling recovery a profit would have been made in lieu of a loss. The output of ore per man per shift worked is recorded by the Company as follows:-

	1911	1912	1913.
Tons crushed	53,564	51,809	53,812
Tons per shift underground	.813	.875	.979
Tons per shift underground) Surface and Reduction Works)	.388	.405	.453

In June 1914 the Company decided to close the mine, and an arrangement was made by the Tasmanian Government to take over the mine on tribute. Arrangement was made to work the mine co-operatively by the miners, and this method operated until November 1914. The tributors treated 16,556 tons for a yield of £24,739/6/1, being a value of 29.10.627 per ton treated. The Automatic Mine Bin Sample Assays for 16,556 tons returned an average of 12.789 dwts. per ton. The tributors obtained nearly the whole of the ore from blocks developed by the Company, and which were considered unpayable. The last crushing was 2,542 tons for £4,185-3-9. The tribute party extended the West drive at the 1500' level, a distance of 49'. The value at the commencement was 34 dwts. the values decreased as the drive advanced, and when stopped was in values of 4½ dwts. There is no record of the value of the residues as no allowance was made to the tributors for them, and they were not allowed to take samples. This was very unsatisfactory as

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the bin samples each week indicated that the mill returns should have been considerably higher. Apparently one sample was taken as an assay return dated the 14th December shows Battery Sands (Pit) 10.9 dwts. per ton. The large amount of water which had to be pumped from the mine was a very large factor in the cost of production being about 5/- per ton of ore raised. There have been many theories as to the origin of the water supply, most asserted that it was accumulated in the surrounding limestone country while others were of the opinion that the water was a seepage from the Tamar river or other inland sources. The Company made a very thorough investigation into the matter and appear to have reached the conclusion that the large influx was from Blyth's Creek. In 1906 the deviation channel at Blyth's Creek overflowed before repairs could be affected. The effect of the water was felt in the mine twenty-three hours after the occurrence, and several days later flooded the mine from the 1000' l. to the 846' level. The pumps and bailing tanks were removing 5,700, gallons per day. The pumping unit at Grubb's Shaft was started and 8,600,000 gallons per day were raised. A geological survey of the district diclosed that no places had been located where precautionary work would be any advantage to overcome the water difficulty other [illegible] at Blyth's Creek where the water had been deviated by fluming. I[illegible] shown that nearly the whole of the flow from this creek entered the mine by the sandstone beds at the West end of the mine. The pumping records kept by the Company are very complete and are available for many years, and as they are of such importance I am attaching a weekly record from January 1911 to May 1914. These are of interest as they show a gradual decrease in the quantity as depth is obtained. It appears very probable that the Western crosscourse may be a drainage channel from the surrounding country.

The following table shows the quantity of Water raised for the years 1911. 1912 and 1913:-

YEAR.	TOTAL GALLONS.	[illegible]
1911 - Jan1st to Dec.31st.	1,171,883,700	22,536,225
1912 - " "	983,905,520	18,921,260
1913 - " "	741,[illegible]1,016	14,268,481
1914 - " May 11th.	246,203,100	12,958,058.

The pumping cost for the four financial years from October 1910 to May 1914 were as follows:-

October 1910 to September 1911 - £19,898/17/9
October 1911 to September 1912 - £17,985/17/10
October 1912 to September 1913 - £14,819/ 1/ 5
October 1913 to May 1914 - £ 7,849/ 5/ 7.

The water in Hart's shaft on April 3rd 1923 was 704' from the surface.

There are several shafts on the Mine, but the two principle ones are known as Grubb's and Hart's - both shafts are connected with the 1500' level. Hart's shaft is 17' x 7'4" - when the mine stopped this shaft was in good condition. Grubb's shaft is 32' x 8'. The first 400' of this Shaft were in a pug deposit, the swelling nature caused considerable trouble owing to crushing and distorting the timbers. This shaft is in good order below the 500', but above the 400' level was in very bad order and not workable.

During my recent visit to Beaconsfield my attention was drawn to a development which may have a very great influence on the re-opening of the mine. It is stated that in 1887 a Crosscut was driven South, at the 500' level a branch lode was located which yielded 10,000 ozs. gold. The Xcut was continued for a total distance of about 400' where a pyritic lode was cut, having a width of 6 to 8 feet. The lode was trending parallel to the main reef, but it was not driven on owing to gas emanating from it. The drive was clayed up to stop the gas. It is stated that the Bonanza Mine cut a similar reef at 1000'. I made enquiries from several persons who stated that they had seen the reef and each one agreed with regard to its position. Mr. O'Keefe M.H.A. who was employed on the mine at the time, informed me that he personally saw the reef and that he knew that one of the men had an assay made which gave a return of 14 dwts. per ton, others varied with regard to the value from 10 to 15 dwts. per ton. It was stated that the reef where cut in the Bonanza Mine assayed 11 dwts. per ton, and it was asserted that the Bonanza did not develop the reef owing to its being at a corner of the section and that they had a very limited amount of ground where it was discovered. If these statements are correct it would not be a costly matter to develop it as the water is at 704' from surface. It would only require the installation of a small winding and ventilation plant. My attention was drawn to another discovery at the 1500' Level by the tributors in 1914. At about 500' East of Grubb's shaft a hanging wall crosscut cut a reef at 9' which was 6 feet in width. It was stated that no assay was taken from the reef, it was cut as the tributors were abandoning the mine. Possibly this is the reef mentioned in Mr. Heathcote's 1914 report, which states that at 897½' a crosscut into the hanging wall at 22' cut a branch lode 4½' in width which was driven on 5½', the average assay value being 4 dwts. per ton.

In conclusion it can be asserted that the values in the 1500' level show a marked improvement. That a good body of stone for at least a length of 400' carries payable values. That cross-cutting in the hanging-wall above water is likely to develop a payable ore body. That with cheap power, up-to-date mining and milling methods, there is every possibility of the mine again becoming payable below the 1500' level.

I desire to express my thanks to Mr. Nightingale, Agent for the Liquidators for making available the records of the Company and rendering every possible assistance, also to the members of the Tourist and Progress Association for supplying information, and to Mr. O'Keefe, M.H.A. for data as to the mine's workings in its early stages.

I have the honour to be,

Sir,

Your obedient Servant,

(Sgd.) J. C. Hudson,

CHIEF INSPECTOR OF MINES.

The Honourable,
Minister for Mines,
HOBART.

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The following table, constructed from Mr. Heathcote's carefully prepared statistics, will show the average number of men employed daily in the various Departments during the past three years, and the average earnings:-

Particulars.	1911.	1912.	1913.
Underground -			
Number of Men Employed	226.53	205.10	191.11
Wages Paid	£30,302.9	£27,732.5	£26,717.2
Average Per Shift	9s. 2.8d.	9s. 4.2d.	9s. 8.6d.
Surface -			
Number of Men Employed	117.07	113.26	111.79
Wages paid	£13,244.8	£12,753.0	£13,306.0
Average Per Shift.	7s. 0.5d.	7s. 3.8d.	7s. 5.4d.
Reduction Works -			
Number of Men Employed	104.10	97.3	89.8
Wages Paid	£12,316.8	£11,341.3	£11,066.8
Average Per Shift	7s. 1.0d.	6s. 11.6d.	7s. 6.9d.
* Totals			
Number of Men Employed	447.75	415.70	392.68
Wages Paid	£55,864.5	£51,826.8	£51,090

* Table does not include Railway Department, which was run separately.

The output of [illegible] for the years:

	Tons Treated	Ozs.	Value	Men employed.
1913	53,233	19,976	£85,912- 4-10	412
1914	15,280	11,247	£48,397	205
1914 (Govt. Tribute)	16,536	4,078½	£24,739	150
1915 Slimes Treatment		4,688	£19,913	-

[illegible]

TOTAL GALLONS OF WATER PUMPED PER WEEK. 1911.

1911 Week ending	
Jan. 9	22,634,040
16	22,157,460
23	21,527,740
30	21,325,460
Feb. 6	21,320,780
13	21,631,400
20	21,185,060
27	19,890,260
Mch. 6	19,598,[illegible]
13	19,639,880
20	19,466,200
27	18,944,640
Apl. 3	18,986,500
10	18,709,080
17	18,345,600
24	18,623,280
May 1	18,271,240
8	18,307,380
15	17,979,780
22	17,789,720
29	17,882,280
June 5	17,632,680
12	17,519,320
19	17,501,120
26	17,197,180
July 3	16,893,240
10	16,875,040
17	16,803,280
24	16,643,900
31	16,466,340
Aug. 7	21,108,360
Forward	588,900,520

1911 Week ending	
Carried Forw'	588,900,520
Aug. 14	47,09[illegible],160
21	43,766,580
28	37,471,200
Sept. 4	33,424,040
11	31,075,200
18	28,539,160
25	26,586,560
Oct. 2	25,36[illegible],560
9	24,621,740
16	24,047,140
23	23,726,040
30	23,024,300
Nov. 6	22,5[illegible]
13	23,[illegible]
20	27,092,260
27	25,756,900
Dec. 4	25,32[illegible]
11	23,678,720
18	22,7[illegible]
25	19,753,760
Jan. 1 (1912)	23,423,140
	1,171,883,700

Average for 52 Weeks:-

22,536,225.

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THE TASMANIAN GOLD MINE.

TOTAL GALLONS OF WATER PUMPED PER WEEK. 1912.

1912 Week ending		Gallons.
January	8	22,086,220
	15	23,704,720
	22	23,891,400
	29	22,991,540
February	5	22,230,780
	12	21,464,300
	19	19,705,920
	26	21,854,560
March	4	20,789,600
	11	17,522,180
	18	20,724,600
	25	21,008,780
April	1	20,243,600
	8	16,095,560
	15	21,064,940
	22	20,102,420
	29	19,486,220
May	6	15,614,820
	13	19,894,420
	20	19,739,460
	27	18,723,380
June	3	18,601,440
	10	14,410,500
	17	16,824,000
	24	21,611,460
July	1	21,263,320
	8	20,857,460
	15	20,161, 180
	22	19,402,500
Forward	-	582,071,360

1912 Week ending		Gallons.
Carried Fw'd.		582,071,360
July	29	19,174,220
August	5	18,801,900
	12	17,900,480
	19	17,614,480
	26	17,598,880
Sept.	2	17,588,740
	9	17,481,620
	16	17,435,340
	23	17,334,720
	30	17,314,160
October	7	17,421,820
	14	17,530,500
	21	17,262,440
	28	17,121,000
November	4	17,207,060
	11	17,087,460
	18	17,223,700
	25	17,316,520
December	2	17,372,940
	9	17,363,320
	16	17,149,060
	23	16,869,060
	30	16,663,920
		983,905,5[illegible]

Average for 52 weeks -
18,921,26[illegible].

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THE [illegible] GOLD MINE.

TOTAL GALLONS OF WATER [illegible] PER WEEK. 1913.

1913 Week ending.		Gallons.
January	6	16,766,360
	13	16,939,780
	20	16,990,220
	27	17,225,520
February	3	16,964,480
	10	17,062,760
	17	16,569,956
	24	16,336,060
March	3	16,305,640
	10	16,545,100
	17	16,248,180
	24	15,149,160
	31	15,050,620
April	7	14,369,680
	14	14,417,260
	21	14,338,380
	28	14,499,680
May	5	14,456,020
	12	14,275,040
	19	14,166,180
	26	14,139,580
June	2	14,277,1[illegible]
	9	14,556,360
	16	14,443,520
	23	13,839,280
	30	13,679,380
Forward		399,637,316

Average of 26 weeks:

15,370,666

1913 Week ending.		Gallons.
Carried Forw'.		399,637,316
July	7	13,933,400
	14	13,847,080
	21	13,592,020
	28	13,623,480
August	4	13,515,320
	11	13,420,420
	18	13,4[illegible],800
	25	13,252,460
September	1	13,111,[illegible]40
	8	13,224,120
	15	13,295,880
	22	13,015,060
	29	12,914,720
October	6	12,952,940
	13	13,019,760
	20	13,264,840
	27	13,034,300
November	3	13,057,720
	10	12,9[illegible],780
	17	12,949,040
	24	13,1[illegible],840
December	1	12,787,[illegible]
	8	12,654,[illegible]
	15	12,744,780
	22	12,62[illegible]
	29	12,991,[illegible]
		741,961.016

Average of 26 weeks :-

13,166,[illegible].

THE TA[illegible] GOLD [illegible] LIMITED. Quarter ending September 27th 1913. COST OF STEAM. 14.

Boiler House.		Fuel. £ s. d.	Stores £ s. d.	Wages £ s. d.	Water £ s. d.	Battery Tram. £ s. d.	Totals. £ s. d.	Pence per I.H.P. Hour.
B. & [illegible]	W.	3203 4 5		461 2 0			3664 6 5	
	M.		22 5 4	179 2 10	50 16 1		252 4 3	
							3916 10 8	.4280
Winding	W.							
	M.							
TOTAL [illegible]		3203 4 5	22 5 4	640 4 10	50 16 1		3916 10 8	.4280
Reduction Wks.	W.	924 19 6		146 0 11		15 15 4	1086 15 9	
	M.		18 7 5	45 7 4	11 15 8		75 10 5	
							1162 6 2	.5595
TOTALS		4128 3 11	40 12 9	831 13 1	62 11 9	15 15 4	5078 16 10	.4523

Distribution on 2,694,502 I.H.P. Hours.

Cost per ton Crushed 85.71 pence.

Cost per I.H.P. per annum £16-10- 2.14

Average H.P. in constant use 1146.

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THE [illegible] LIMITED.

Steam Costs.

Quarter ending	At the rate of per annum per I.H.P.	Average H.P. in constant use.
June 1911	16. 12. 1.	
Sept. 1911	14. 9. 7.92	
Decr. 1911	13. 14. 3.74	
March 1912	13. 17. 11.52	
June 1912	14. 15. 6.91	1392
Sept. 1912	14. 14. 1.39	1415
Decr. 1912	15. 2. 1.72	1317
March 1913	15. 10. 5.61	1274
June 1913	16. 11. 8.54	1192
Sept. 1913	16. 10. 2.14	1146
Decr. 1913	15. 13. 11.66	1098.

THE TASMANIA GOLD MINE.

TOTAL GALLONS OF WATER PUMPED PER WEEK. 1914.

1914.	Week Ending.		Gallons.
"	January	5	12,840,620
"		12	12,912,640
		19	12,650,820
		26	12,687,740
	February	2	13,031,460
		9	13,174,720
		16	12,927,720
		23	13,166,660
	March	2	13,328,640
		9	13,449,020
		16	13,359,320
		23	13,380,120
		30	13,053,040
	April	6	13,046,540
		13	12,910,820
		20	12,527,580
		27	12,788,100
	May	4	12,606,360
		11	12,361,180
		17	9,543,820 (for 5½ days.).
		Total	246,203,100
Average for Nineteen Weeks		-	12,958,058.

PART. B.
Reference 1/6A.

Tasmania Dept of Mines
Technical Reports
No 8 1963

PART I

☆

GEOLOGICAL REPORTS

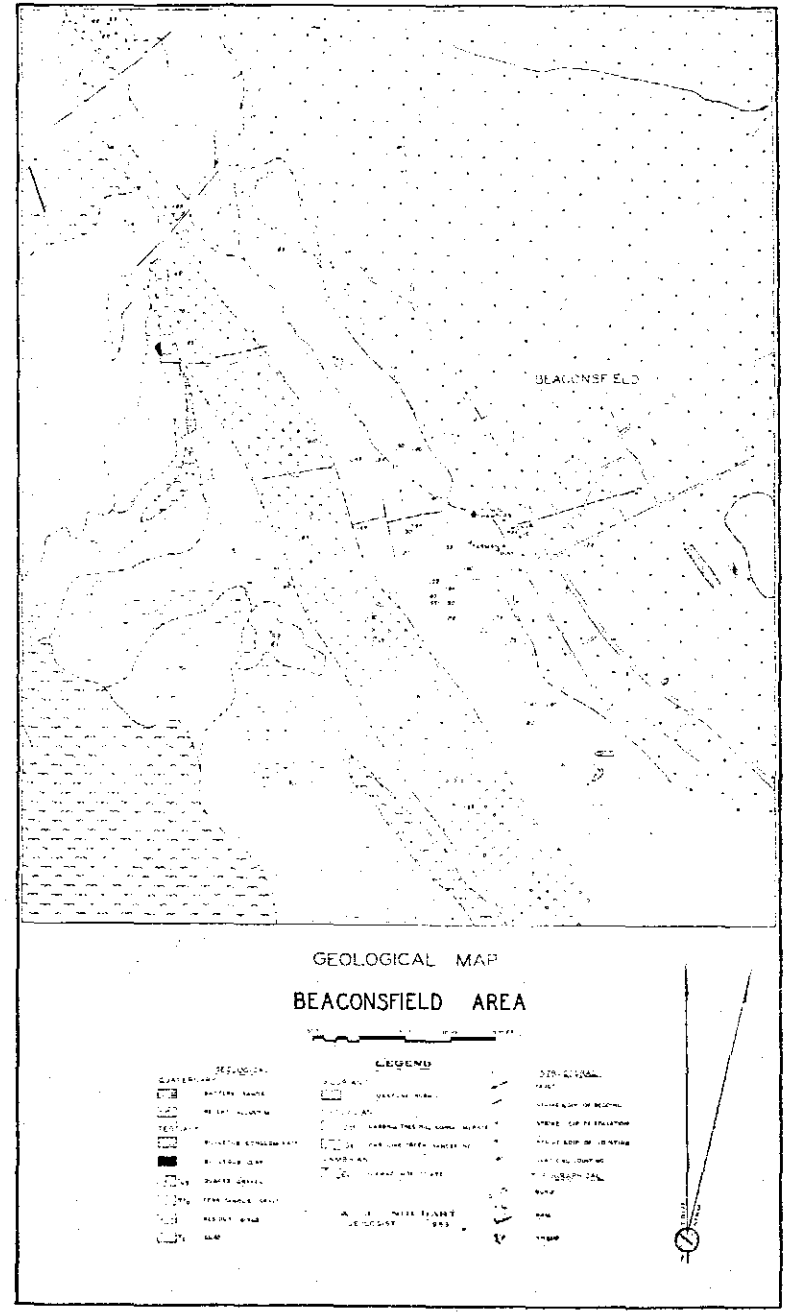

FIGURE 2.

Section 1–Economic and General Geology

1. NOTES ON AURIFEROUS DEPOSITS, BEACONSFIELD GOLDFIELD

By A. J. Noldart.

INTRODUCTION

The Beaconsfield township is situated 26 miles by road northerly of Launceston, 2 miles west of the Tamar estuary. The main auriferous deposits occur immediately west of the township on the crest and eastern flank of the Cabbage Tree Hill with smaller deposits occurring to the north and south along a narrow belt of country centred on the Cabbage Tree Hill-Blue Tier ridge line. The belt is approximately 5 miles long by ½ mile wide and has been subject to small scale mining over the entire length. Only the Tasmania quartz reef proved to be payable to any depth.

Topographically the main feature of the immediate area is the long strike ridge formed by the Blue Tier and Cabbage Tree Hill prominences. The ridge rises to about 650 feet above sea level at the Blue Tier (southern) end descending gradually to about 350 feet at the Cabbage Tree Hill to the north. The township of Beaconsfield is located on the lower slopes of the eastern flank of the Cabbage Tree Hill at an elevation of 100 feet above sea level.

The ridge loses its character rather abruptly at both northern and southern ends where it loses elevation and disappears under younger rock successions. The line of the ridge is broken at its approximate centre by the Blythes Creek water gap and towards the northern end by the less conspicuous Brandy Creek water gap. The creeks cut the ridge along prominent fault lines and similar faulting has caused off-setting of the southern spurs of the Blue Tier ridge.

Easterly of the township a gently undulating plain descends gradually to sea level at Middle Arm. Easterly of Middle Arm and southerly of the township, the plain rises into a series of low hills and ridges. Westerly of the ridge Blythes Creek drains a gently undulating plain which also rises to the south and west into low foothills. To the north this plain is flanked by high level (approximately 200 feet) terrace-like gravel beds. Blue Peaked Hill forms a prominent landmark on the SW edge of the valley.

GEOLOGY

The crest of the Cabbage Tree Hill-Blue Tier ridge is composed of rocks of Cabbage Tree Conglomerate. These beds are conformably overlain by members of the Caroline Creek Sandstone. Correlation of these beds with the type formations of the West Coast areas is based mainly on lithological similarities in conjunction with the reported occurrence of brachiopods, trilobites, and casts of a species of *Orthis* found in a member of the Caroline Creek Sandstone in the workings of the Tasmania mine. These rocks are of Ordovician age.

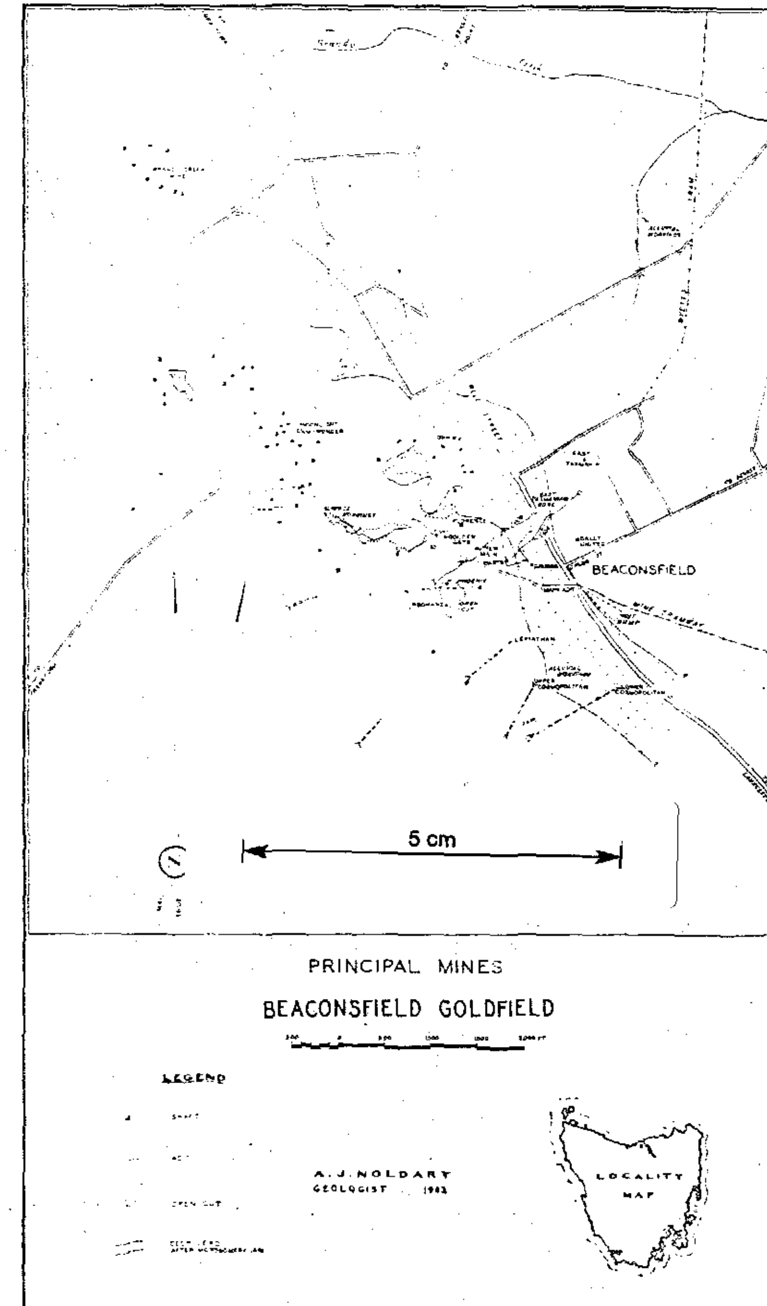

FIGURE 3.

Black slate, greywacke, sub-greywacke, siltstone &c., of the Ilfracombe Slate of Cambrian age underlie Cabbage Tree Conglomerate to the west. The contact is obscured but the two sequences appear to be concordant.

The Cabbage Tree Hill-Blue Tier ridge is a strike ridge, trending about N35°W dipping 50°-60° NE. Local internal folding in the Ordovician succession is indicated by local reversals of dip noted on the northern spur and flanks of Cabbage Tree Hill and reported from mine workings.

Similar rocks of probable Ordovician age form the prominent ridge of Blue Peaked Hill.

SE of the range the lower Palaeozoic rocks are unconformably overlain by gently dipping rocks of Permian age. These rocks are in turn overlain by sand, clay, conglomerate, quartz, gravel, &c. of Tertiary age composing the undulating plain easterly of the township. The higher level terrace-like deposits north and NW of Cabbage Tree Hill are composed of white quartz gravel, also of Tertiary age.

Immediately east of Cabbage Tree Hill, and underlying portion of Beaconsfield township, a deep channel of probably early Tertiary age has been eroded into Gordon Limestone. The channel is filled with varicoloured clay interbedded with sand, semi-consolidated conglomerate, loose gravel, scree &c., to a known depth of more than 400 feet. The upper horizons are reported to be composed of compact, semi-consolidated clay and siltstone with unconsolidated sand, scree and gravel filling the lower portion of the channel. Specimens of fossil fruits, timber, leaves, &c., referred to the older Tertiary, have been recorded from intermediate clay horizons in the channel.

TASMANIA MINE

General

Discovered in 1877, the Tasmania auriferous quartz reef remained in production until 1914 when economics of mining forced closure of the underground workings. Subsequent operations were concentrated on the retreatment of mine tailings dumps until the final plant shutdown in 1924. Overall production from the Tasmania reef (inclusve of tailings retreatment) was 854,600 oz. from 1,067,556 tons of ore, for an average recovery of 16.01 dwts of gold per ton. The gold won was valued at £3,613,000 of which £772,671 was distributed in dividends to the shareholders. No dividends were paid after 1905.

With few exceptions the underground workings are now inaccessable and the following information has been obtained from old literature. A list of the principal references is attached.

Closure of the mine was due to a combination of factors involving fall in grade, economics associated with depth, excessive water intake, &c. It was reported in 1912 that about 17,275,000 gallons was pumped from the mine per week, with an additional 21,000,000 gallons storage encountered for each foot of extra sinking undertaken.

In the latter stages of the life of the mine employer-employee relations deteriorated rapidly and, with final closure threatened, the then Government of Tasmania assumed control of the mine

and plant in an endeavour to avoid shut down. On the employees' assurance that mismanagement was mainly responsible for the projected closure the Tasmanian Government then subsidized a tributing party of about 150 mine employees to carry on mining activities. The venture proved unsuccessful and the mine closed after a short period.

It is apparent that irrespective of gold values still obtaining in the bottom (1500 foot) level, the then prevailing economic limits of mining, as applicable to the circumstances peculiar to the Tasmania mine, had been reached.

THE WORKINGS.

Underground mine plans, cross sections and longitudinal sections of the mine are available at the Department of Mines and a full description of the underground workings is not considered warranted here. Briefly, the Tasmania reef was worked from the surface to about 480 feet from the "Golden Gate" and "Florence Nightingale" shafts. Subsequent to the amalgamation of the original companies, mining activities were extended to 1500 feet by a succession of deeper shafts: namely the "New Main shaft" (1,000 feet) "Hart's Shaft" (1,375 feet), and "Grubb's Shaft" (1,500 feet). The last named shaft was sunk partly in the deep lead and, due to movement in this section, was later restricted to use as a general service shaft. All heavy haulage operations were directed through the more stable "Hart's Shaft".

Stoping was consistent over the full 1,300 feet of the ore body to a depth of 1250 feet but below that level the stoping limits, as indicated on the longitudinal sections, were progressively reduced on the 1375 foot and 1500 foot levels to a final stope length of 940 feet. This sharp reduction in stope length is probably due to the economics of mining at depth and does not necessarily represent an actual decrease in the size of the ore body.

THE OREBODY.

The Tasmania reef is a fissure reef striking about N50°E with the quartz emplaced in a pre-existent fault zone. The movement on the fault is shown on old plans as about 100 feet north side east. The reef has been itself displaced by two major fault zones and numerous smaller movements. The major faults were termed the "main cross course" (easternmost fault) with a strike of N30°W dipping steeply SW and No. 2 fault striking N45°W also dipping steeply SW. Easterly of the "main cross course" the reef maintains a fairly constant strike of about N50°E with a slight swing to N45°E westerly of the fault. The reef in these sections of the mine transgresses almost all members of the Caroline Creek Sandstone and lies entirely within that succession. The overall dip of the reef here is about 50°-55°SE and the two sections are obviously dislocated portions of the same reef.

Westerly of the No. 2 fault, however, the reef as mined shows a marked swing northerly to about N55°W with a dip to the SW. This section of the mine lies entirely within Cabbage Tree Conglomerate. The sharp swing in strike together with a marked change in the mineralization pattern and strength has raised strong doubts as to this section actually being the western continuation of the Tasmania reef.

Movement on the "main cross course" appears to be west side north with a displacement of about 240 feet on the ore body. However, a simple lateral movement cannot fully explain the displacement as the enclosing strata appear to have been displaced by some 1,000 feet as distinct from the smaller movement in the ore body. It is apparent that a considerable west side up movement has occurred with a lateral component of about 500-600 feet. A similar type movement appears to have occurred in the "No. 2 fault" but the displacement here cannot be ascertained due to the doubtful identity of the reef mined westerly of this fault.

A comprehensive discussion of the effects of the faulting on the Tasmania reef is given by Montgomery (1891) together with an outline of the possible positions that the reef could have assumed westerly of the "No. 2 fault".

The main Tasmania reef has an overall length of about 1300 feet. The strike averages N50°E with a dip of 50°-60° to the SE. Stoping outlines as shown on the mine longitudinal sections and plans indicate an overall plunge of the ore body to the NE at 55° with individual richer "shoots" within the ore body also trending NE but with shallower plunges varying from 35° to 50°. With the overall plunge indicated, the 300 feet of main reef lying to the west of the "main cross course" at the surface thus becomes progressively shorter with depth, finally plunging away from the between faults block at about the 600 foot level. Below this level the entire stope length of the ore body lies to the east of the "main cross course".

The reef varies in width from a few inches to upwards of 25 feet in some lenses with an overall stoping average of about 7-8 feet. Gold values in the reef are reported to have been fairly consistent along the length of each individual level but varied considerably with depth. From the surface to about the 400 foot level an average grade of 25 dwts. per ton was maintained but average grades over the next 300 feet dropped to about 16-17 dwts. per ton. Still further reductions in grade occurred with greater depth dropping to as low as 2½ dwts. per ton at the 1370 foot level. A considerable improvement to an average grade of up to 13 dwts. per ton over a stoping length of 940 feet was reported from the bottom (1500 foot) level.

Gold quality is also reported to have changed with depth, the gold obtained from the richer upper levels consisting mainly of free milling, auriferous quartz, readily amalgamated. Changes in mineralization below about the 400 foot level showed the presence of pyrite, chalcopyrite, sphalerite, galena, &c., in increasing amounts with a considerable proportion of the gold intimately associated with the sulphides necessitating more specialized and expensive treatment methods.

Westerly of the "No. 2 fault" mining operations were only continued to about the 100 foot level. The ore body in this portion of the mine proved to be inconsistent and very "bunchy" varying in size from mere threads to lenses up to 3 feet in width. Mineralization was weak with values considerably lower than in the main Tasmania reef.

OTHER MINES

Moonlight-cum-Wonder Mine

This mine comprises the old "Moonlight", "Little Wonder", "Olive Branch" and "Amalgamated West Tasmania" mines, all of which operated to some extent on the same line of lode.

Situated near the crest of the Cabbage Tree Hill, the mine was developed entirely in Cabbage Tree Conglomerate. The overall strike of the ore body is N55°-60°W. The general dip is to the SW although a reversal of dip is reported from the deeper levels in the northern section.

The auriferous quartz in these workings was not confined to one ore channel with occasional splitting of the reef, as was the case in the Tasmania reef, but was reported to have been distributed in a number of parallel or sub-parallel veins, often in broken ground and subject to rapid variation in size both along strike and down dip. In some areas, as in the "Olive Branch" section, the veins were too small to be mined individually but were rich enough and numerous enough to encourage attempts at bulk open cut mining.

Generally good values were obtained in the older shallow workings to depths of about 250 feet but values diminished rapidly below this depth. The ore body was tested to the 800 foot level with exploration drives at the 400, 500, 600 and 800 foot levels without success. The ore channels are reported as varying from threads up to 18 inches in width with occasional lenses up to 9 feet in thickness. Values did not improve with size of the reefs and were often reported as richer in the narrower zones. Thureau (1883) recorded: "very rich 'shoots' of gold in the reef dip as from a common centre both east and west . . ." in the 130 foot level of the "Little Wonder" mine. It is probable that these are small saddle reefs reflecting one of the minor flexures in the strata of Cabbage Tree Hill.

The mineralization pattern in these reefs is almost identical with that in the western section of the Tasmania mine and it is probable that the two reef systems are located on the same fissure zone.

Tonnage and grades of ore from these reefs are not available but records of the Department of Mines show a gross recovery of 1,044 oz. of gold from these mines.

Ophir Mine

The deep lead along the eastern flank of Cabbage Tree Hill has been investigated by several shafts and drill holes. Most of the workings were concentrated on the western slope of the lead, or at intermediate levels in the lead. No records are available of any testing having been carried out on the true bottom of the lead except by drill holes which penetrated to close proximity of the bottom. Of these prospect shafts the "Ophir" mine was the only one in which any attempt was made to test the bottom of the lead.

The last information available on this mine was that a shaft had been sunk to a depth of 405 feet with the upper 275 feet (about) sunk through the material of the deep lead. Levels were driven easterly into the deep lead at depths of 300 feet and 400 feet. From the 400 foot level a winze was sunk in the wall of the lead to a depth of 60 feet (460 feet from the surface) and a level driven east below the deep lead. Future intentions to rise from this 460 foot level into the bottom of the lead did not eventuate.

Other portions of the lead were tested in the Tasmania mine adit (300 feet of drive); the lower "Cosmopolitan" adit (412 feet of drive) and from the Nos. 4 and 5 levels of the old "Florence Nightingale" workings at depths of 270 feet and 330 feet respectively. No payable values are reported from any of these workings but it is probable that any enrichment from southerly of the Tasmania reef would necessarily be weak, and that the drives would be too far south to intersect any enrichment from the main Tasmania reef.

Salisbury Mining Centre

The Salisbury mining centre is located at the southern end of the Blue Tier ridge. The main workings were the "Salisbury", also known as the "Victoria"; and the "Duchess of York", also called the "Gladstone" and "Santa Claus", mines. Both mines were explored by a combination of shafts and adits from the eastern flank of the ridge. The workings are roughly in the same position respective to the strata as the "Cosmopolitan" mine on the eastern flank of Cabbage Tree Hill. Some open cut, hydraulic sluicing workings are located in the nose of the southern spur of the ridge.

The mineralization in these mines differs markedly from that of the Cabbage Tree Hill mines in that the majority of gold occurrences in the near surface workings occurred as "coarse lumps of gold" and "patches of free gold met with in sugary quartz and soft seams of pug". In several instances the gold had a superficial coating of black manganiferous oxides giving rise to the so called "black gold" of the locality.

In the deeper levels of the adits all the gold was reported to be intimately associated with sulphide mineralization. Occurrences of nickel and chromium minerals are recorded from the main adits closely associated with an intrusive body of basic rock.

High grade concentrations or "pockets" of eluvial/alluvial gold occurred in the talus on the crest of the south spur and in the talus/alluvium admixture at the foot of the spur. Gold values in these deposits was also reported to be extremely "patchy" with high grade pockets interspersed with large areas of almost barren material.

No production figures can be given from the centre.

Smaller Mines

Very little is known about the smaller mining operations in the district. Innumerable small shafts and costeans cover the eastern slopes of Cabbage Tree Hill but only a very small proportion of the smaller workings encountered payable reefs.

Travelling north along Cabbage Tree Hill from the Blythes Creek water gap the more significant of these workings southerly of the Tasmania reef are:—The "Rising Sun" mine immediately above Blythes Creek, and the "Cosmopolitan", "Leviathan", "Bonanza", "Star" and "Phoenix" mines, all located on the eastern flank of the ridge. The "Garfield" mine is also located on the eastern flank of the ridge but is northerly of the Tasmania reef towards the northern spur of Cabbage Tree Hill.

All these mines were designed to test possible occurrences of the "Moonlight-cum-Wonder" type reef formations with but little success. Some minor copper/silver type mineralization was encountered in the "Rising Sun" mine, and small irregular auriferous reefs were encountered in the other mines. The gold values in each case were insufficient to encourage further exploration. The "Phoenix" mine, though originally worked on a reef similar to the other small mines, was ultimately deepened to intersect the Tasmania reef to become part of the main workings.

Immediately to the north of Brandy Creek, on a low ridge extension of Cabbage Tree Hill, moderately payable gold reefs of the "Moonlight-cum-Wonder" type were worked in the "Brandy Creek" mine but again values did not persist with depth. A similar type mineralization also occurred in the "North Tasmania" mine located some 1300 feet further north along the strike.

These two mines appear to have been the only ones other than the Tasmania and Moonlight-cum-Wonder mines where payable gold mineralization was encountered. Full records of production are not available but the North Tasmania is recorded as having produced 987 oz. of gold.

As far as can be determined the mineralization in all of the smaller mines was similar in all respects to that of the Moonlight-cum-Wonder reefs; i.e. surface enrichment in narrow, irregular quartz veins, rapidly diminishing in value with depth.

Alluvial Workings

With the exception of the Salisbury locality, auriferous deposits of an alluvial nature have not been a significant feature of the district. Occasional small patches have been worked but the majority of the so called "alluvial" deposits were in fact remnant pockets of Tertiary gravels of a semi-deep lead nature partially uncovered by subsequent erosion.

ORE PROSPECTS

Tasmania Reef

Unfortunately no geological information is available on the Tasmania reef after 1903 so that nothing is known of the limiting factors controlling the extremities of the ore body at depths below about 700 feet. A summary of such information as is available to that depth is given below.

On the eastern end the Tasmania reef is reported to have feathered out into a series of thin stringers on entering brecciated zones in Caroline Creek Sandstone close to the footwall of Gordon Limestone. Twelvetrees (1903), discussing the 700 foot level, wrote as follows:—

> "Behind the limestone, conformable with it and underlying it, the level passed through a bed of dense, tenacious clay . . . . This clay band is known in the mine as 'the dyke'. Westwards it merges gradually into a zone of what can best be described by the term 'broken formation', or 'broken country'. This consists of sandy material showing lines of false deposition, and containing angular fragments of sandstone, giving place to the west to more solid shattering and disintegration in *situ*. Hard blocks of sand-

stone are met with, having the sandy material between them for a length of about 60 ft. It is noteworthy that the reef in this section of the level became irregular, splitting and jumping up and down. The reef tails out just where the broken formation begins; its track goes into the broken (*sic*) for a little way and then disappears.

> "In the level above the 600 feet, the reef behaves in the same way when the broken country is entered".

And further with reference to the 700 foot level, the deepest then being worked, he recorded the following:—

> "The actual appearance of the reef in the east end of the 700 foot level is sufficient to cause anxiety. It feathers out when entering the broken country. It has no appearance of having been sheared off by a fault, and there is no track or channel in the limestone".

The limiting factor on the eastern end of the ore body down to the 700 foot level is evidently lack of continuity of the reef through zones of brecciation and it is probable that similar conditions restrict the ore body at depths below that level. Longitudinal sections of the mine do in fact show a marked steepening of the eastern stope limits between the 700 foot and 1250 foot levels suggesting that the bounding control at this end of the ore body is structural and not lithological.

On the western margin of the ore body a different set of conditions exists. As mentioned previously the western extension of the Tasmania reef past the "No. 2 fault" is questionable, but from the information available it is doubtful if the main Tasmania reef as such ever extended any distance into the up faulted members of the Cabbage Tree Conglomerate west of the fault. It is apparent from the longitudinal sections that the western limit of the Tasmania ore body fairly closely follows the attitude of the bedding planes of the country rocks. At no stage were the workings continued into the underlying Cabbage Tree Conglomerate, the mineralization dying out on all levels at a point where it could be expected to approach these beds.

It would appear that the members of the Cabbage Tree Conglomerate are not in themselves very favourable to ore deposition and that they have acted as a bounding influence on the western limits of the ore body.

Montgomery (1891), with reference to the country rocks, made the following observations:—

> "The Tasmania reef has been auriferous throughout all the strata traversed by it. The richest stone is found in a number of distinct 'shoots' or 'chutes', ... Ouside of the 'shoots' however, the quartz has been generally payable . . . . The strata that have proved 'favourable country' for gold in the mine may be said to be all those between the lower beds of grits and conglomerates and the main limestone bed. . . . In the mines on the Moonlight line of reef rich stone has been got in the upper levels of all, and as long as the quartz was found in the light coloured grits and sandstone, but on getting down into the black country the stone has become unpayable in every case . . ."

The inconsistency of the reefs in Cabbage Tree Conglomerate is due in a large measure to the relative competencies of the beds involved, resulting in poorly defined fissures in the harder beds with the development of multiple fracture patterns and consequent dispersal of mineralization along a number of more or less poorly defined channels.

These rocks then cannot be considered as offering good prospects for extensive gold mineralization.

With regard to the Tasmania mine at depth, it is known that good values recurred on the bottom (1500 foot) level over a stoping length of about 940 feet. Whether or not this represents the true length of the ore body at this depth or whether the ore body continues to maintain an overall length of about 1300 feet is questionable, but the possibility must be considered that the restriction indicated on the mine plans is due to economic limits rather than mineralization limits. If so, a good gold prospect lies below the present known workings with good chances of permanency with further depth.

If, however, the bounding controls of lithology on the western limits and country fracturing on the eastern limits, continue to control the ore body at depth, then the limits of the ore body below the 1500 foot level could be expected to contract fairly quickly with depth. Any such contraction would greatly reduce the potential of the ore body, possibly to the extent of unpayability.

The Deep Lead

All payable material obtained before 1903 appears to have come from the western wall or "high reef" zone of the lead, and from minor workings on false bottoms at intermediate levels. Both the western wall zone and a false bottom of "black ligneous clay" at 112 feet depth were reported as "fairly payable".

Two bores sunk to bedrock through the deep lead by the Ophir Company were reported to give good values. This report, submitted to the Company by one of the Directors, stated that the first bore, to a depth of 375 feet on the western wall, encountered:—"gravel containing gold at two ounces to load" at 240 feet, and "9 feet of wash with gold at the rate of 4 ounces to the load" at the bottom of the bore. In the second bore to a depth of 286 feet in the eastern wall, "about 12 feet wash, giving returns at 2 ounces to load" was reported from the bottom. The term "load" as used is not defined in the reports available.

Very little core was obtained in the course of the drilling and the above reported values can be taken only as indicative that the deep lead is auriferous in part, particularly in the lower levels. It is pointed out that the 270 foot and 330 foot levels of the older workings on the Tasmania reef penetrated the deep lead but no values were reported. These penetrations would be too far south to encounter any enrichment from the surface outcrop of the Tasmania reef.

Since the proven depth of the deep lead in the vicinity of the Ophir bores is in excess of 400 feet as shown by the Ophir mine workings, the actual bottom of the deep lead has not been tested and must be considered to be a reasonable gold prospect.

Other Mines

The prospects of the smaller mines on the field are not promising. The Moonlight-cum-Wonder reef system was fairly extensively prospected during the life of the mine, particularly at depth, with no success. Too little is known about other mines such as the North Tasmania and Brandy Creek mines, to be able to suggest any exploration programme, and the still smaller mines such as the Leviathan, Cosmopolitan, &c, are too small to warrant testing Any mineralization on this belt would be small and patchy and restricted to near surface depths. The best exploration for this type of mineralization would be surface costeaning over large areas, a method commonly used by early prospectors in the district.

Mineralization at the Salisbury end of the Blue Tier ridge is also too irregular for any deep exploration programme to be successful.

Two mines not previously mentioned due to their position are the "East Tasmania" mine and "Dally's United" mine. Neither of these mines were active producers but were sunk as prospecting ventures attempting to intersect any extension of the Tasmania reef easterly of the Gordon Limestone. Should the line of fissure persist east of the Gordon Limestone then comparatively small "shoots" of ore could occur in the sandstone members of the succession overlying the Gordon Limestone, but any mineralization found would be repetition on a small scale and not a continuation of the Tasmania ore body as such.

Some small quantities of "alluvial" gold may still be won from the thin Tertiary gravel beds on the lower slopes of Cabbage Tree Hill, and on the plain easterly of the Beaconsfield township, but large accumulations cannot be expected.

CONCLUSIONS

I. The only auriferous prospect of any size in the field is the continuation of the Tasmania reef at depth below the abandoned workings. Dependent on the bounding controls at the eastern end of the ore body, a large tonnage of medium grade ore could exist. A diamond drilling exploration programme to test this prospect is warranted.

II. Testing of the bottom of the deep lead may give favourable results. All gravel horizons in the deep lead northerly of the strike of the Tasmania reef are potential zones of enrichment. Accurate assessment of the deep lead would require a maximum recovery of the unconsolidated strata penetrated in any drilling programme attempted.

III. A limited amount of "wild cat" drilling could, if necessary, be carried out on the western flank of Cabbage Tree Hill in the vicinity of the old Britannia Shaft, and easterly of the Gordon Limestone in the vicinity of the East Tasmania-Dally's United mines.

IV. No other prospecting can be recommended in the area with the exception of small localized surface activities by small parties.

It should be noted that even though the testing of III above may be only a case of "proving the absence" of payable mineralization, the deflections occurring in shallow diamond drill holes in these localities could be used to assess the influence of the strata on any deeper penetration attempted on the deeper Tasmania reef prospect should such drilling be attempted.

RECOMMENDATIONS

1. The Tasmania auriferous quartz ore body should be tested for continuity at a depth of 500 feet (vertical) below the old workings. Testing at shallower depths would not be satisfactory as considerable ore reserves would be a prerequisite to any attempt at future mining.

Testing should be initially by two diamond drill holes designed to intersect the ore body at the 2000 foot level (vertical) below the following surface datum points:—

A. A point 382 feet from the centre of the Grubb Shaft on a bearing of N33°E (mag.).
B. A point 762 feet from the centre of the Grubb Shaft on a bearing of N32½°E (mag.).

2. Subsidiary testing of the deep lead would be most effective by exploratory drilling at a point 150 feet on a bearing of S55°E from the old Ophir Shaft.

3. Diamond drilling is not recommended on the "wild cat" prospects on the western flank of Cabbage Tree Hill or in the vicinity of the East Tasmania-Dally's United mines at this juncture unless drill deflection data from these holes would have a bearing on the collar locations of the deeper holes.

Geological plans of the Beaconsfield area (Figure 2) and the deep lead (Figure 3) showing recommended prospects accompany this report. Subsurface mine level plans, mine cross sections, mine longitudinal sections, &c., of the Tasmania mine may be inspected at the Department of Mines, Hobart.

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PART B
Reference 1/7

5. EXPLORATORY DIAMOND DRILLING, TASMANIA GOLD MINE, BEACONSFIELD GOLDFIELD

by A. J. Noldart

INTRODUCTION

After extended investigations into auriferous mineralisation in the Beaconsfield Goldfield it was decided to test by diamond drilling the deeps of the 'Tasmania' gold reef, Beaconsfield. A Joy-Sullivan H.D. 30 diamond drilling plant was purchased and drilling commenced on 22 June 1964. One parent hole and two diversions were drilled for a total of 2,609 feet (see fig. 36).

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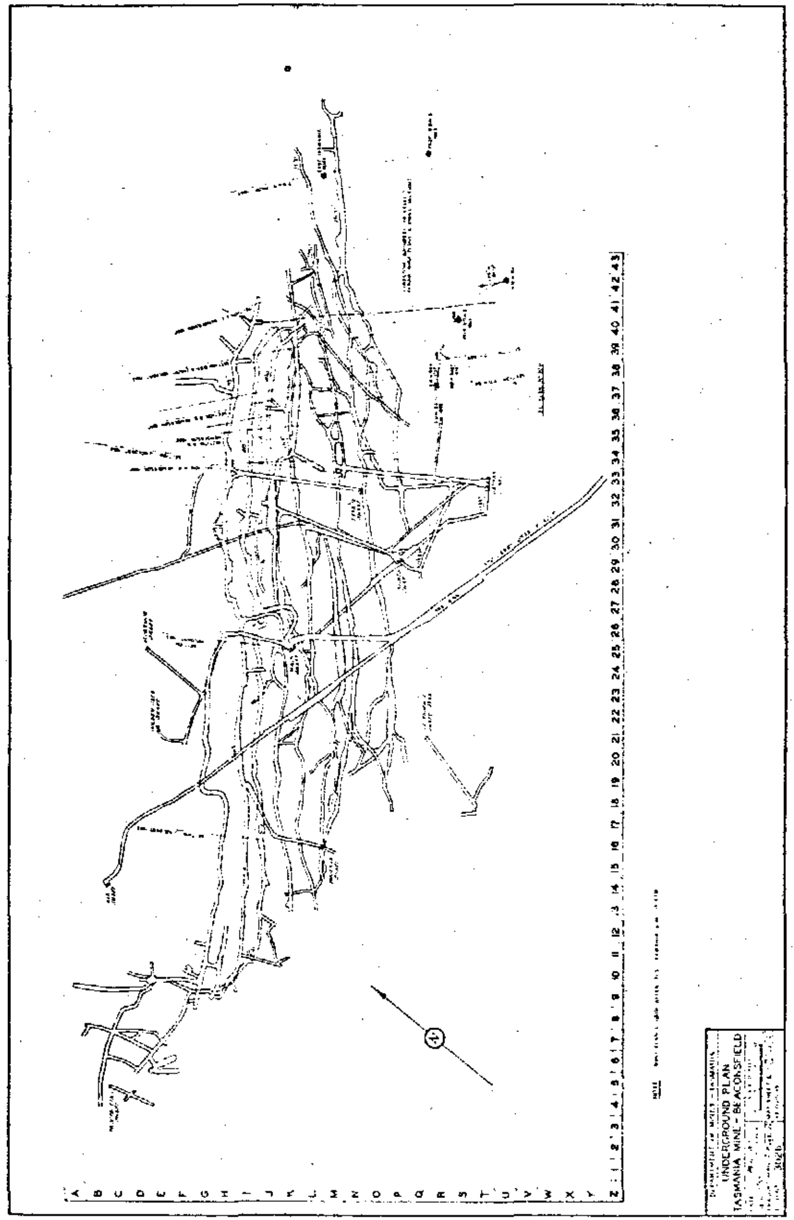

FIGURE 8

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Drilling was completed on 24 April 1967, and further drilling was suspended in favour of possible company activities on the reef. The plant was dismantled and removed for overhaul.

LOCATION AND TARGET

Diamond drill hole No. B4 (D.D.H.B.4) was collared close to the old Daly's United mine shaft at a point 475 feet distant, bearing 45° magnetic from the centre of Grubb's Shaft, Tasmania Goldmine. The hole was commenced at a depressed angle of 85° on a bearing of 299° magnetic.

The proposed target was approximately midway along the projected position of the known ore body at a vertical depth of 2,000 feet below the collar of the Main Shaft, Tasmania Mine, datum 00 feet R.L., and some 550 feet below the deepest (1,500 feet) mine level in the plane of the ore body. Secondary targets were located 400 feet E and W of the first objective.

DRILLING

No information was available on the influence that the bedding plane and cleavage characteristics of the rocks to be penetrated would have on the course of the hole. Although it was anticipated that a marked westerly drift could be expected, a reliable assessment of the intensity could not be made.

The hole was commenced on a bearing approximately parallel to the strike of the country rocks (300° magnetic) and depressed at an angle of 85° to the dip of the bedding planes (55°-65°). Little trouble was anticipated with excessive variation in the inclination of the hole on this course, and it was considered that any excessive westerly azimuth drift could be corrected by normal wedging.

Drill hole surveys at 100 feet intervals during the course of drilling indicated that the inclination variation was not excessive (5° lift from 1-1,000 feet) but with depth a 'corkscrew' effect to the W became more pronounced. Drill hole plots and calculations indicated that an intersection would still be made, although at a higher level than planned, and it was decided to continue the hole as far as possible without mechanical correction.

An intersection was eventually made at a depth of 1,720 feet vertical (R.L. 00 feet, collar Main Shaft, Tasmania Mine) some 260 feet below the bottom level (1,500 feet) of the old workings in the plane of the ore body and approximately midway along the reef.

A short diversion was then effected close to the initial intersection to obtain a check assay of the reef. A second diversion was commenced at a point 1,314 feet down the parent hole; ten wedgings were made using Clappison wedges and the diversion hole was steepened to 86° and turned S to attempt a deeper penetration. The third intersection on the ore body was made at a depth of 1,760 feet vertical, approximately eighty feet distant from D.D.B.H.4 intersection, in the plane of the lode.

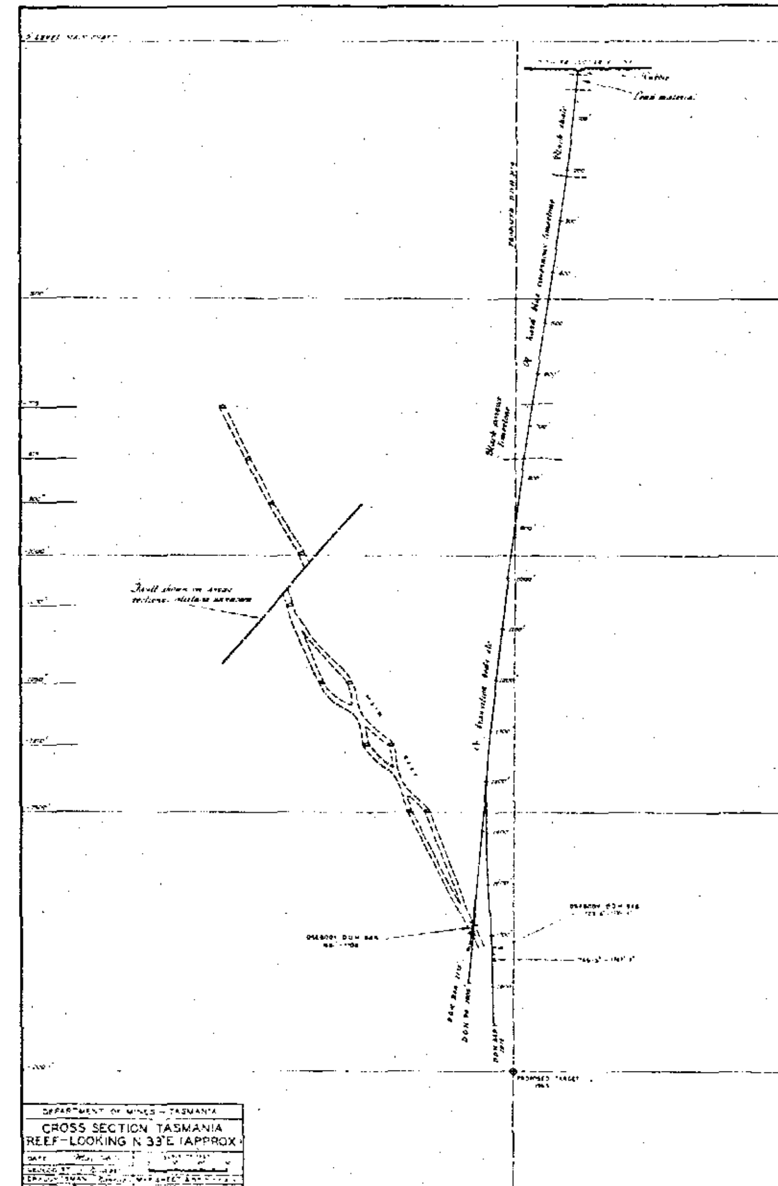

FIGURE 9

GEOLOGY

Description of surface and underground geology and mineralisation is given in previous reports and need not be repeated here. Brief summaries of the geological information from the drilling is given in the following core logs.

D.D.H.B.4

Depth From	Depth To	Description
feet	*feet*	
0	13	Rubble and old mine talus.
13	36	Clay, sand and gravel representing the Eastern lip of the Beaconsfield deep lead.
36	213	Black carbonaceous shale.
213	660	Hard blue cavernous Gordon Limestone, Ordovician age.
660	764	Interbedded blue limestone and black, porous carbonate rock.
764	1,689	Transition beds comprising grey massive lime-rich sandstone, impure limestone, siltstone and occasional horizons of hematite-stained dark brown to pink limestone containing numerous crinoid fragments. Below 1,200 feet the beds have progressively less carbonate content and more silica, grading to sandstone and quartzite of light to medium grey colour.
1,689	1,708	ORE ZONE.
1,708	1,716	Medium grey quartzite.
1,716	1,753	Dark grey to black quartzite containing minor grit and pebble bands.
1,753	1,805	Predominantly light coloured pebble conglomerate of the Cabbage Tree Conglomerate type.
		END OF HOLE.

D.D.H.B.4 A (First Diversion)

Depth From	Depth To	Description
feet	*feet*	
1,490	1,681	Mainly light to medium grey quartzite and sandstone.
1,681	1,704½	ORE ZONE
1,704½	1,714	Quartzite as above.
1,714	1,733	Dark grey to black quartzite.
		END OF HOLE.

D.D.H.B.4 B (Second Diversion)

Depth From	Depth To	Description
feet	*feet*	
1,314	1,660	No core due mainly to intensive wedging.
1,660	1,723½	Medium grey quartzite—Lower Ordovician Orthid, Tritoechia(?) *careyi* at 1,701 feet.
1,723½	1,747½	ORE ZONE.
1,747½	1,787	Medium grey quartzite.
1,787	1,852	Dark grey to black quartzite.
1,852	1,874	Predominantly light coloured pebble conglomerate of the Cabbage Tree Conglomerate type.
		END OF HOLE.

THE ORE BODY

The ore body in each intersection is composed of a quartz reef, impregnated with sulphides in variable concentrations and containing visible gold in some sections. Sulphide mineralisation includes pyrite, chalcopyrite, galena, sphalerite and arsenopyrite. Tetrahedrite has been recorded from the ore body by previous workers but was not seen in the core. The main gangue is siderite.

Detailed logs of the ore intersections are given below. Ore body recovery was 100% in all intersections.

D.D.H.B.4

Depth From		Depth To		Description
feet	*in.*	*feet*	*in.*	
1,689	0	1,693	0	Quartz/siderite reef heavily impregnated with sulphides—progressively less sulphide.
1,693	0	1,695	9	As above—low sulphide content.
1,695	9	1,697	9	Quartz/siderite/sulphide as above.
1,697	9	1,698	9	Quartz with coarse flecks of gold and minor sulphides.
1,698	9	1,702	0	Quartz/siderite with moderate sulphide content—leached over last six inches.
1,702	0	1,704	0	Quartz with coarse flecks of gold and minor sulphides.
1,704	0	1,705	9	Slightly mineralised grey quartzite.
1,702	0	1,704	0	Quartz with coarse fleck of gold and minor sulphides.
1,706	11	1,707	8	Mineralised chert.
				END OF ORE ZONE.

D.D.H.B.4 A

Depth From feet	in.	Depth To feet	in.	Description
1,680	10	1,683	5	Quartz/siderite/sulphide — sulphide prominent.
1,683	5	1,688	2	Quartz with small flecks of gold.
1,688	2	1,688	10	Quartz—strongly leached and honeycombed.
1,688	10	1,690	10	Quartz with small flecks of gold—minor sulphide.
1,690	10	1,696	5	Quartz/siderite reef with moderate sulphides.
1,696	5	1,702	7	Quartz with minor sulphides and occasional small flecks of gold—some assimilated country rock.
				END OF ORE ZONE.

D.D.H.B.4 B

Depth From feet	in.	Depth To feet	in.	Description
1,723	6	1,727	2	Quartz/siderite reef with moderate sulphides and occasional fine flecks of gold.
1,727	2	1,730	0	As above but no visible gold.
1,730	0	1,731	2	Siderite vein along core.
1,731	2	1,731	8	Quartz/carbonate with massive sulphides.
1,731	8	1,735	3	Quartz/siderite with moderate sulphides and occasional fine flecks of gold.
1,735	3	1,744	6	Gangue—massive siderite with occasional blebs of pyrite and quartz.
1,744	6	1,745	3	Leached massive siderite.
1,745	3	1,747	1	Quartz/carbonate with moderate sulphides.
1,747	1	1,747	7	Quartz/carbonate with massive sulphides—quartz minor.
				END OF ORE ZONE.

ASSAY RESULTS

Assay sections and values are given below.

D.D.H.B.4

Depth From feet	in.	Depth To feet	in.	Au dwts.	Ag dwts.	Cu %	As %	Pb %	Zn %	Mn %	S %
1,689	0	1,691	0	23.0							
1,691	0	1,693	0	10.7							
1,693	0	1,695	0	1.7							
1,695	0	1,697	0	7.5							
1,697	0	1,699	6	18.3							
1,699	6	1,702	0	35.3							
1,702	0	1,704	0	591.0							
1,704	0	1,706	0	2.0							
1,706	0	1,707	8	19.1							
Composite											
1,689	0	1,707	8	60.20	4.7	1.06	1.49	0.10	0.80	0.49	7.5

D.D.H.B.4 A

Depth From feet	in.	Depth To feet	in.	Au dwts.	Ag dwts.	Cu %	As %	Pb %	Zn %	Mn %	S %
1,680	10	1,683	10	15.4							
1,683	10	1,686	10	71.0							
1,686	10	1,689	10	75.0							
1,689	10	1,692	10	45.0							
1,692	10	1,695	10	11.0							
1,695	10	1,698	10	79.2							
1,698	10	1,701	10	16.0							
1,701	10	1,704	6	34.2							
Composite											
1,680	10	1,704	6	42.1	6.6	0.91	0.43	0.03	0.16	0.36	4.9

D.D.H.B.4 B

Depth From feet	in.	Depth To feet	in.	Au dwts.	Ag dwts.	Cu %	As %	Pb %	Zn %	Mn %	S %
1,723	6	1,726	6	21.7							
1,726	6	1,729	6	17.4							
1,729	6	1,732	6	41.8							
1,732	6	1,735	3	39.8							
1,735	3	1,739	3	0.8							
1,739	3	1,743	3	1.2							
1,743	3	1,745	3	3.4							
1,745	3	1,747	7	16.4							
Composite											
1,723	6	1,735	3	26.8	11.0	1.10	0.05	0.10	0.03	0.39	3.58

CONCLUSIONS

Exploratory work to date indicates that the character of the Tasmania reef has not altered with depth but is constant throughout, the only apparent variable being the gold content. Assay sections of the mine indicate sporadic gold values throughout but consistently higher towards the central and E sections of the ore body.

Previous estimates of the average width of the reef vary from 5 to 8 feet with probable average of 5 to 6 feet. The apparent widening of the lode where intersected in the current programme is probably due to the intersection coinciding with the splitting of the lode as indicated in the mine plans on the 1,250, 1,370 and 1,500 feet levels. There is an inclusion of country rock, in the D.D.H.B.4 intersection suggesting a strong hangingwall lode and a weaker footwall lode as indicated by old assay sections, and a similar structure occurs in the D.D.H.4 B intersection with massive gangue material replacing country rock.

The pebble conglomerate beds intersected in the footwall of the ore body in each intersection appear to be extensions of the 'auriferous conglomerate' (Twelvetrees, 1903) reported from the 600 and 718 feet levels in the mine, and, as the western limits of the ore body appear to be lithologically controlled by the 'lower' conglomerate and black sandstone, the occurrence of these beds suggests no great change in the length of the ore body at this depth as would be the case if faulting or flattening of the strata had occurred.

Where intersected the sulphide content of the ore body is not a major cost item, as has been suggested by earlier investigators, indeed the copper values obtained, if consistent throughout, would be a considerable factor in mining economics.

RECOMMENDATIONS

Deeper exploration is necessary both to prove continuity of ore body length and grade and to determine controlling factors of the ore body in the deeper levels.

Further drilling should be carried out between the 2,000 feet and 3,000 feet levels as a primary programme with extended drilling possibly to the 4,000 feet level as a follow-up. With drill hole deviation information now available, careful planning would permit several widely-spaced intersections from each parent hole.

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Part B.
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TASMANIA DEPARTMENT OF MINES

1974

GEOLOGICAL SURVEY EXPLANATORY REPORT

GEOLOGICAL ATLAS 1 MILE SERIES

ZONE 7 SHEET No. 30 (8215N)

BEACONSFIELD

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ECONOMIC GEOLOGY

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The area covered by the Beaconsfield map sheet contains a variety of mineral occurrences of which the most significant have been the goldfields of Beaconsfield and Lefroy. At Beaconsfield the Tasmania mine, closed since 1914, remains the largest single producer of gold in Tasmania with a recorded production of some 26 580 kg. At Lefroy the largest producer was the New Pinafore mine with a production of 1570 kg of a total of nearly 5200 kg from this area.

Comparisons with other gold producing districts in Tasmania show productions of over 8400 kg from Mathinna, 19 900 kg from the Mt Lyell copper mines and 16 080 kg from the Rosebery-Mt Reid lead/zinc complex. Gold production from the last two areas is as a by-product from base metal mining.

Some of the earliest known mining activities in Tasmania, with the exception of some coal mining and the extraction of stone for building purposes, were based on the pisolitic hematitic iron ores at Andersons Creek, and the production of burnt lime for agricultural requirements.

Current mining activities are restricted to the extraction of detrital silica from the western slopes of Cabbage Tree Hill for the manufacture of silico-manganese and ferro-silicon, the exploitation of sand and gravel for construction purposes, and the treatment of small quantities of silica sand from Port Sorell for the moulding industry.

Prospecting and exploration is currently being carried out to determine the feasibility of reopening the Tasmania gold mine; to determine the potential of the asbestos deposits at Andersons Creek; and to treat detrital chromite deposits in the same locality.

HISTORY AND PREVIOUS INVESTIGATIONS

The earliest reference to mineral occurrences in the Beaconsfield Quadrangle was by the then Surveyor General G.W. Evans (1822) when he reported:

'Within a few miles of Launceston there is a most surprising abundance of iron. Literally speaking there are entire mountains of this ore, which is so remarkably rich that it has been found to yield 70 per cent of pure metal. These mines have not been worked;...'

The occurrence of the iron and other mineral deposits however, were known almost from the start of the settlement at York Town in 1804. F.G. Duff (1888), in an article reprinted from the Launceston Examiner reported:

'...in 1804 Colonel Paterson...formed the first settlement in northern Tasmania, and founded the township of York Town...The deposits of iron ore, sandstone, asbestos, and serpentine in the vicinity of York Town were soon discovered...'

Later attempts (1872-1875) to produce commercial iron from these and other deposits were not successful.

The utilisation of limestone and other calcareous deposits in the district was apparently one of the earlier mining activities, but there is no information available on these activities except comments by Gould (1866) to the effect that '...the lime which has been for many years past an article of export from the district.'

'The blue limestone has been worked for many years past, and in several localities.'

'The calcareous bands found by the existence of these fossils are in places sufficiently important to be working as sources of lime; and at various periods kilns have been in operation in the district for that purpose.'

The earliest reference to gold discoveries is also by Gould (1864) when he stated:

'Gold has, indeed, been found in small quantities at many points throughout the district, and in some instances the character of the gold has been such as to indicate its source near at hand...Among the places in which gold has already been obtained may be enumerated..., Nine Mile Springs,...'

Nine Mile Springs was the early name for a small township near Lefroy. The major producer in the district, the Tasmania mine, was not discovered until 1877.

Other mineral occurrences to be exploited on a limited scale are the asbestos deposits at Andersons Creek; serpentinites from Andersons Creek for terrazzo requirements; ochre from Andersons Creek for paint manufacture and gas purification; leached sandstone from Brandy Creek for the manufacture of sandsoap, etc., and small amounts of white clay for pottery manufacture.

In more recent years attention has been focused on the possibility of utilising pyritic shales at Port Sorell as a source of sulphur; residual clays overlying the serpentinite at Andersons Creek for nickel and cobalt; and on the search for heavy mineral concentrations in Recent coastal deposits.

Gravel and sand deposits have been worked for many years, mainly for road repairs, but construction demands in recent years have created a major industry operating over large areas.

All old mine openings are collapsed or in a general state of disrepair, most other workings obscured by scree and regrowth, and reports by early workers form the only source of information on these workings. Miscellaneous reports by Thureau (1882, 1883*a*, 1883*b*, 1884), Montgomery (1891, 1896, 1897) and Twelvetrees (1900, 1902, 1903*a*, 1903*b*) contain the basic data used by later investigators on the Beaconsfield, Salisbury and Lefroy goldfields; and a report by Just (1891) outlines the history of the iron deposits.

Examinations were later carried out on the asbestos deposits by Twelvetrees (1917) and Reid (1919) during a resurgence of interest in these deposits and by Broadhurst (1935) in the Lefroy district. There are numerous reports by other workers on specific mines or prospects, by officers of the Department of Mines made in the course of mineral resources investigations, and by mining and exploration company personnel.

With respect to the older mining ventures the authors have relied entirely on the earlier reports. Some sections of these are quoted without change in order to retain the original writer's concepts on specific aspects as pertaining at the time they were written.

Metallic minerals

GOLD

Gold has been the only mineral produced from the area in significant economic quantities. By far the greatest production has been from the Tasmania line of lode in the Beaconsfield goldfield, with moderate production from some of the mines in the Lefroy goldfield notably the Pinafore, Chum, Volunteer and Native Youth lines of lode. Minor production is recorded from other lodes in these districts and from the Salisbury area. Minor amounts of gold have also been recovered from small Tertiary to Recent detrital deposits in the three districts.

Production

There is no complete record of production during the early days of mining in Tasmania, particularly where detrital gold deposits were exploited. This is marked in the goldfields lying within the Beaconsfield map sheet, where the only information available is a comment by Commissioner Bernard Shaw (1873) where he stated, in reference to the Lefroy field, that 'the value of the gold found in alluvial deposit during the year 1872 was £8000.'

Nothing is known of production from other centres but although the deposits were significant in holding prospectors in the fields the deposits were too small to have contributed anything but a very small proportion of the overall production.

Production from lode mining was better documented through mine managers' reports etc., particularly for the larger mines, but many of the smaller mines of the early period have no recorded production. From descriptions of the underground workings by early investigators it is unlikely that any significant amount of gold was recovered from these workings.

The following table lists gold production data from the main reefs and lode formations in the Beaconsfield and Lefroy goldfields.

Goldfield	*Reef or Lode Formation*	*Production kg*	*Mining Commenced*	*Mining Ceased*
Beaconsfield	Tasmania	26 580	1877	1914
Lefroy	New Pinafore	1 712	1890	1896
Lefroy	Chum	1 313	1881?	1896
Lefroy	Volunteer	1 277	1891	1904
Lefroy	Native Youth	749	1877?	1888
Lefroy	New Golden Point	60	1881?	1903
Lefroy	Morning Star	37	1883	1903
Beaconsfield	Moonlight-Cum-Wonder	32	1898	1903
Beaconsfield	North Tasmania	31	1898	1911

The production figures for the Tasmania and New Pinafore lodes include production from retreatment of battery sands etc., after the cessation of mining activities.

There is a total recorded production of 49 kg gold from a number of other small mines in the Lefroy field.

BEACONSFIELD GOLDFIELD

The Beaconsfield goldfield lies 39 km by road northerly from Launceston and 3 km west of the Tamar estuary. The main deposits were found immediately west of the Beaconsfield town site on the eastern flank of the Cabbage Tree Hill. Several smaller deposits were found to the north-west and south-east along a narrow belt centred on the crest of the ridge line of the Cabbage Tree Hill. Further to the south the extension of this ridge forms the Blue Tier ridge with the Salisbury goldfield located at its southern extremity.

All the workings are inaccessible with the exception of the Garfield adit on the west flank of the hill. Remnants of the surface works of the Tasmania mine such as the remains of chimney stacks, massive concrete surrounds of the Harts and Grubbs shafts, and the still standing brickwork of the winding houses and boiler rooms servicing the shafts form a prominent landmark in the town and have become a focal point for tourists.

History

Reef gold was first discovered high on the eastern flank of the Cabbage Tree Hill by the Dally brothers in 1877. This find stimulated interest in the area resulting in the tracing of the initial discovery downslope, and the location of other small occurrences along the crest of the ridge to the north and south.

Numerous mining companies sprang into existence including the Tasmania Gold Mining Co. (on the original discovery), the Golden Gate Gold Mining Co., and the Florence Nightingale Gold Mining Co., on the line of the Tasmania reef and the Lefroy Gold Mining Co., exploring immediately to the east of the Florence holdings.

Along the crest of the ridge discoveries were made by the Moonlight and Little Wonder Gold Mining Companies to the north of the Tasmania find, the Garfield further north, and the Dundee and Excelsior (Brandy Creek mines) to the north of Brandy Creek. To the south prospecting by the Phoenix, Leviathan, Cosmopolitan and Rising Sun companies disclosed further small auriferous quartz reefs.

The Tasmania and Golden Gate companies merged shortly after the commencement of operations followed in 1888 by the further amalgamation of these mines with the Florence Nightingale and Lefroy companies with the object of a combined effort to control water intake into the mines. A later reorganisation of the group resulted in the formation of the Tasmania Gold Mining and Quartz Crushing Co., under which name the mine operated until its closure.

During this period surface prospecting and underground development of the Florence mine indicated the presence of a deep extensive channel infilled with clays and gravels occurring to the east of the mines. The potential of this channel was recognised early and several efforts were made to explore and mine detrital gold contained in the sediments. The Ophir, Denmark and later the Ballarat companies were the main groups but extremely hazardous mining conditions and heavy water intake resulted in the failure of all attempts at deep mining.

Exploration and mining on the smaller properties continued sporadically but by 1891 virtually all operations were dormant with the exception of the Moonlight-cum-Wonder amalgamation. The discovery in 1898 of the North Tasmania mine revived some interest but closure of these two mines came in 1903 and 1911 respectively.

The end of the mining industry at Beaconsfield came with the closure of the Tasmania mine in 1914. Retreatment of battery sands and tailings from the Tasmania Gold Mining and Quartz Crushing Co. treatment plant continued for several years after the cessation of mining.

Lode Deposits

Tasmania Mine

Discovered in 1877, the Tasmania auriferous quartz reef remained in production until 1914 at which time the economics of mining forced closure of the underground workings. Subsequent operations were concentrated on the retreatment of mine tailings dumps until the final plant shutdown in 1924. Overall production from the Tasmania reef, inclusive of tailings retreatment, was 26 580 kg of gold from 1 084 690 t of ore, for an average gold recovery of 24.7 g/t. The gold won was valued at £3,613,000 of which £772,072 was distributed in dividends to shareholders. No dividends were paid after 1903 when the mine was taken over by a company based in England.

The underground workings are now inaccessible and the following information has been obtained from old literature. The main references are those by Thureau (1883*a*), Montgomery (1891), Anon. (1898), Twelvetrees (1903*b*), and Cundy and Fawcett (1914).

Closure of the mine was due to a combination of factors involving fall in grade, economics associated with depth, increased metallurgical problems, excessive water intake etc. It was reported (Hudson, 1923) that in 1913, the last year of normal mining and development, almost 86 400 *kl* of water was raised from the mine and it was estimated that an additional 313 300 *kl* would be encountered for each metre of extra sinking undertaken.

In the latter stages of the life of the mine employer-employee relations deteriorated rapidly and, with closure of the mine in April 1914 by the management, the then Government of Tasmania assumed control of the mine in an endeavour to avoid a shutdown. On the employees' assurance that mismanagement was mainly responsible for the projected closure the Government then subsidised a tributing party of about 150 mine employees to continue mining activities. The mine was reopened in June 1914, but the venture proved unsuccessful and final closure occurred on 21 November 1914.

The workings. Underground mine plans, cross sections and longitudinal sections of the mine are available at the Department of Mines and a full description of the underground workings is not considered warranted here. Briefly, the Tasmania reef was worked from the surface to about 145 m from the 'Golden Gate' and 'Florence Nightingale' shafts. Subsequent to the amalgamation of the original companies, mining activities were extended to 450 m by a succession of deeper shafts: namely the 'New Main Shaft' (300 m) 'Hart's Shaft' (420 m), and 'Grubb's Shaft' (450 m). The last named shaft was sunk partly in the deep lead and, due to movement in this section, was later restricted to use as a general service shaft. All heavy haulage operations were directed through the more stable 'Hart's Shaft'.

Stoping was consistent over the full 395 m of the orebody to a depth of 380 m but below that level the stoping limits, as indicated on the longitudinal sections, were progressively reduced on the 1375 foot (420 m) and 1500 foot (450 m) levels to a final stope length of 290 m. This sharp reduction in stope length is due to the economics of mining at depth and does not represent an actual decrease in the size of the orebody (Cundy and Fawcett, 1914).

The orebody. The Tasmania reef is a fissure reef striking about N50°E with the quartz emplaced on a pre-existent fault zone. The movement on the fault is shown on old plans as about 30 m north side east. The reef has been itself displaced by two major fault zones and numerous smaller movements.

The major faults were termed the 'main cross course' (easternmost fault) with a strike of N30°W dipping steeply south-west and No. 2 fault striking N45°W also dipping steeply south-west. East of the 'main cross course' the reef maintains a fairly constant strike of about N50°E with a slight swing to N45°E west of the fault. The reef in these sections of the mine transgresses almost all units of the previously termed Caroline Creek Sandstone sequence and lies entirely within that succession. The overall dip of the reef here is about 50-55° SE and the two sections are obviously dislocated portions of the same reef.

West of the No. 2 fault, however, the reef as mined shows a marked swing northerly to about N55°W with a dip to the south-west. This section of the mine lies entirely within the massive conglomeratic quartzites underlying the above sandstone sequence of the Cabbage Tree Formation. The sharp swing in strike together with a marked change in the mineralisation pattern and strength has raised strong doubts as to this section actually being the western continuation of the Tasmania reef.

Movement on the 'main cross course' appears to be west side north with a displacement of about 70 m on the orebody. However, a simple lateral movement cannot fully explain the displacement as the enclosing strata appear to have been displaced by some 300 m as distinct from the smaller movement in the orebody. It is apparent that a considerable west side up movement has occurred with a lateral component of about 150-180 m. A similar type of movement appears to have occurred in the 'No. 2 fault' but the displacement here cannot be ascertained due to the doubtful identity of the reef mined west of this fault.

A comprehensive discussion of the effects of the faulting on the Tasmania reef and an outline of the possible positions that the reef could have assumed west of the 'No. 2 fault' are given by Montgomery (1891).

The main Tasmania reef has an overall length of about 395 m. The strike averages N50°E with a dip of 50-60° to the south-east. Stoping outlines as shown on the mine longitudinal sections and plans indicate an overall plunge of the orebody to the north-east at 55° with individual shoots within the orebody also trending north-east but with shallower plunges ranging from 35° to 50°. With the overall plunge indicated, the 90 m of main reef lying to the west of the 'main cross course' at the surface thus becomes progressively shorter with depth, finally plunging away from the between faults block at about the 180 m level. Below this level the entire stope length of the orebody lies to the east of the 'main cross course'.

The reef varies in width from several centimetres to more than 8 m in some lenses with an overall stoping average of about 2-2.5 m. Gold values in the reef are reported to have been fairly consistent along the length of each individual level but varied considerably with depth. From the surface to about the 400 foot (120 m) level an average grade of 38 g/t was maintained but average grades over the next 90 m dropped to about 25 g/t. Still further reductions in grade occurred with greater depth dropping to as low as 3.8 g/t at the 1370 foot (415 m) level. A considerable improvement to an average grade of up to 20 g/t over a stoping length of 285 m was reported from the bottom, 1500 foot (450 m), level.

Gold quality is also reported to have changed with depth, the gold obtained from the richer upper levels consisting mainly of free milling auriferous quartz, readily amalgamated. Changes in mineralisation below about the 400 foot (120 m) level showed the presence of pyrite, chalcopyrite, sphalerite, galena, etc., in increasing amounts with a considerable proportion of the gold intimately associated with the sulphides necessitating more specialised and expensive treatment methods.

West of the 'No. 2 fault' mining operations were only continued to about the 100 foot (30 m) level. The orebody in this portion of the mine proved to be inconsistent and very 'bunchy' ranging in size from mere threads to lenses up to one metre in width. Mineralisation was weak with values considerably lower than in the main Tasmania reef.

Other mines

Moonlight-cum-Wonder mine. This mine comprises the old 'Moonlight', 'Little Wonder', 'Olive Branch' and 'Amalgamated West Tasmania' mines, all of which operated to some extent on the same line of lode.

Situated near the crest of the Cabbage Tree Hill, the mine was developed entirely in the massive conglomeratic black quartzites of the lower sections of the Cabbage Tree Formation. The overall strike of the orebody is N55°-60°W. The general dip is to the south-west although a reversal of dip is reported from the deeper levels in the northern section.

The auriferous quartz in these workings was not confined to one ore channel with occasional splitting of the reef, as was the case in the Tasmania reef, but was reported to have been distributed in a number of parallel or sub-parallel veins, often in broken ground and subject to rapid variation in size both along strike and down dip. In some areas, as in the 'Olive Branch' section, the veins were too small to be mined individually but were rich enough and numerous enough to encourage attempts at bulk open cut mining.

Generally good values were obtained in the older shallow workings to depths of about 75 m but values diminished rapidly below this depth. The orebody was tested to the 800 foot (245 m) level with exploration drives at the 400, 500, 600 and 800 foot (120, 150, 180 and 245 m) levels without success. The ore channels are reported as varying from threads up to 0.5 m in width with occasional lenses up to 2.75 m in thickness. Values did not improve with size of the reefs and were often reported as richer in the narrower zones. Thureau (1883a) recorded: "very rich 'shoots' of gold in the reef dip as from a common centre both east and west..." in the 130 foot (40 m) level of the 'Little Wonder' mine. It is probable that these are small saddle reefs reflecting one of the minor flexures in the strata of Cabbage Tree Hill.

The mineralisation pattern in these reefs is almost identical with that in the western section of the Tasmania mine and it is probable that the two reef systems are located on the same fissure zone.

Tonnage and grades of ore from these reefs are not available but records of the Department of Mines show a gross recovery of 32 kg of gold from these mines.

Smaller mines. Very little is known about the smaller mining operations in the district. Innumerable small shafts and costeans cover the eastern slopes of Cabbage Tree Hill but only a very small proportion of the smaller workings encountered payable reefs.

Travelling north along Cabbage Tree Hill from the Middle Arm Creek water gap the more significant of these workings south of the Tasmania reef are: The 'Rising Sun; mine immediately above Middle Arm Creek, and the 'Cosmopolitan', 'Leviathan', 'Bonanza', 'Star' and 'Phoenix' mines, all located on the eastern flank of the ridge. The 'Garfield' mine is also located on the eastern flank of the ridge but is north of the Tasmania reef towards the northern spur of Cabbage Tree Hill.

All these mines were designed to test possible occurrences of the 'Moonlight-cum-Wonder' type reef formations with but little success. Some minor copper/silver type mineralisation was encountered in the 'Rising Sun' mine, and small irregular auriferous reefs were encountered in the other mines. The gold values in each case were insufficient to encourage further exploration. The 'Phoenix' mine, although originally worked on a reef similar to the other small mines, was ultimately deepened to intersect the Tasmania reef to become part of the main workings.

Immediately to the north of Brandy Creek, on a low ridge extension of Cabbage Tree Hill, moderately payable gold reefs of the 'Moonlight-cum-Wonder' type were worked in the 'Brandy Creek' mine (Dundee and Excelsior mines) but again values did not persist with depth. A similar type mineralisation also occurred in the 'North Tasmania' mine located some 395 m further north along the strike.

These two mines appear to have been the only ones other than the Tasmania and Moonlight-cum-Wonder mines where payable gold mineralisation was encountered. Full records of production are not available but the North Tasmania is recorded as having produced 31 kg of gold.

As far as can be determined the mineralisation in all of the smaller mines was similar in all respects to that of the Moonlight-cum-Wonder reefs; *i.e.* surface enrichment in narrow, irregular quartz veins, rapidly diminishing in value with depth.

Detrital Deposits

The Deep Lead

The deep lead running along the eastern flank of Cabbage Tree Hill has been investigated by several shafts and drill holes. No records are available of any testing having been carried out on the true bottom of the lead although an attempt was at last report being made by the Ophir Gold Mining Company.

The last information available on this mine, Twelvetrees (1903a), was that a shaft had been sunk to a depth of 123 m with the upper *c*.84 m sunk through the material of the deep lead. Levels were driven eastwards into the lead at depths of 90 m and 120 m. From the 400 foot (120 m) level a winze was sunk in the west wall of the lead to a depth of 18 m (138 m from the surface). A level was then driven eastwards from the bottom of the winze through 56 m of sandstone followed by some broken ground and finally limestone. This drive appears to be below the bottom of the lead.

Future plans to rise from this level into the bottom of the lead do not appear to have been put into effect.

All payable material obtained before 1903 appears to have come from the western wall of the lead, known locally as the 'high reef' zone, and from minor workings on false bottoms at intermediate levels. Both the western wall zone and a false bottom of 'black ligneous clay' at a depth of 34 m were reported to be 'fairly payable'.

Two bores sunk to bedrock through the deep lead by the Ophir Company were reported to give good values. Montgomery (1891) quotes a report submitted to the Company by one of the Directors, where he stated that the first bore to a depth of 114 m on the western wall, encountered:'...gravel containing gold at two ounces to load,...' at 73 m, and '...9 feet [2.7 m] of wash with gold at the rate of 4 ounces to the load.' In the second bore to a depth of 87 m in the eastern wall, 'about 12 feet [3.7 m] wash, giving returns at 2 ounces to load.' Montgomery then commented: 'If these results are reliable the richness of the lead would be phenomenal', but the term 'load' is not defined in the reports available.

Very little core was obtained in the course of the drilling and the above reported values can only be taken as indicative that the deep lead is auriferous in part, particularly in the lower levels.

Other portions of the lead were tested in the Tasmania mine adit (No. 2 level, 90 m of drive); the lower Cosmopolitan adit (125 m of drive); and from the No. 4 and 5 levels of the old Florence Nightingale workings at depths of 82 m and 100 m respectively. No payable values are reported from any of these workings but it is probable that any enrichment from south of the Tasmania workings would necessarily be weak, and that the level drives would be too far south to intersect any enrichment from the main Tasmania reef.

From the evidence available it appears that the main enrichment has been from the flanks of the Cabbage Tree Hill and north of the surface expression of the Tasmania reef. The deposits worked on the western wall required crushing indicating that the bulk of this material is merely eluvial detritus shed from the auriferous quartz reefs. This would suggest a fairly localised concentration of this type of material with finer alluvial gold dispersed throughout wash horizons.

Other alluvial deposits

Small deposits of alluvial gold have been worked along Brandy Creek downstream of the Brandy Creek mines and on the flats to the east of the townsite but all occurrences are small and of little economic value.

Other deposits worked on the east flank of Cabbage Tree Hill north of the surface outcrop of the Tasmania reef occur as fillings in depressions and embayments in the hill slope and are probably perched remnants of a higher level of deep lead fill most of which has since been removed by erosion.

Ore Prospects

Tasmania Reef

Unfortunately no geological information is available on the Tasmania reef after 1903 so that nothing is known of the limiting factors controlling the extremities of the orebody at depths below about 210 m. A summary of such information as is available to that depth is given below.

On the eastern end the Tasmania reef is reported to have feathered out into a series of thin stringers on entering brecciated zones in the previously titled Caroline Creek Sandstone close to the footwall of Gordon Limestone Correlate. Twelvetrees (1903*b*) discussing the 700 foot (210 m) level, wrote as follows:

'Behind the limestone, conformable with it and underlying it, the level passed through a bed of dense, tenacious clay....This clay band is known in the mine as 'the dyke'. Westwards it merges gradually into a zone of what can best be described by the term 'broken formation', or 'broken country'. This consists of sandy material showing lines of false deposition, and containing angular fragments of sandstone, giving place to the west to more solid shattering and disintegration *in situ*. Hard blocks of sandstone are met with, having the sandy material between them for a length of about 60 ft [18 m]. It is noteworthy that the reef in this section of the level became irregular, splitting and jumping up and down. The reef tails out just where the broken formation begins; its track goes into the broken *[sic]* for a little way and then disappears.

In the level above the 600 feet [180 m], the reef behaves in the same way when the broken country is entered'.

And further with reference to the 700 foot (210 m) level, the deepest then being worked, he recorded the following:

'The actual appearance of the reef in the east end of the 700 foot [210 m] level is sufficient to cause anxiety. It feathers out when entering the broken country. It has no appearance of having been sheared off by a fault, and there is no track or channel in the limestone'.

The limiting factor on the eastern end of the orebody down to the 700 foot (210 m) level is evidently lack of continuity of the reef through zones of brecciation and it is probable that similar conditions restrict the orebody at depths below that level. Longitudinal sections of the mine do in fact show a marked steepening of the eastern stope limits between the 700 foot (210 m) and 1250 foot (810 m) levels suggesting that the bounding control at this end of the orebody is structural and not lithological.

On the western margin of the orebody a different set of conditions exists. As mentioned previously the western extension of the Tasmania reef past the 'No. 2 fault' is questionable, but from the information available it is doubtful if the main Tasmania reef as such ever extended any distance into the up faulted members of the conglomerates and black quartzites west of the fault. It is apparent from the longitudinal sections that the western limit of the Tasmania orebody fairly closely follows the attitude of the bedding planes of the country rocks. At no stage were the workings continued into the underlying conglomerates and black quartzites, the mineralisation dying out on all levels at a point where it could be expected to approach these beds.

It would appear that conglomerates and black quartzites are not in themselves very favourable to ore deposition and that they have acted as a bounding influence on the western limits of the orebody.

Montgomery (1891), with reference to the country rocks, made the following observations:

'The Tasmania reef has been auriferous throughout all the strata traversed by it. The richest stone is found in a number of distinct 'shoots' or 'chutes',....Outside of the 'shoots' however, the quartz has been generally payable...The strata that have proved 'favourable country' for gold in the mine may be said to be all those between the lower beds of grits and conglomerates and the main limestone bed...In the mines on the Moonlight line of reef rich stone has been got in the upper levels of all, and as long as the quartz was found in the light coloured grits and sandstone, but on getting down into the black country the stone has become unpayable in every case...'

The inconsistency of the reefs in black quartzites and conglomerates is due in a large measure to the relative competencies of the beds involved, resulting in poorly defined fissures in the harder beds with the development of multiple fracture patterns and consequent dispersal of mineralisation along a number of more or less poorly defined channels.

These rocks then cannot be considered as offering good prospects for extensive gold mineralisation.

With regard to the Tasmania mine at depth, it is known that good values recurred on the bottom 1500 foot (460 m) level, over a stoping length of about 285 m. Whether or not this represents the true length of the orebody at this depth or whether the orebody continues to maintain an overall length of about 400 m is questionable, but the possibility must be considered that the restriction indicated on the mine plans is due to economic limits rather than mineralisation limits. If so, a good gold prospect lies below the present known workings with good chances of permanency with further depth.

If, however, the bounding controls of lithology on the western limits and country fracturing on the eastern limits, continue to control the orebody at depth, then the limits of the orebody below the 1500 foot (460 m) level could be expected to contract fairly quickly with depth. Any such contraction would greatly reduce the potential of the orebody, possibly to the extent of unpayability.

Other mines

The prospects of the smaller mines on the field are not promising. The *Moonlight-cum-Wonder* reef system was fairly extensively prospected during the life of the mine, particularly at depth, with no success. Too little is known about other mines such as the North Tasmania and Brandy Creek mines, to be able to suggest any exploration programme, and the still smaller mines such as the Leviathan, Cosmopolitan, etc., are too small to warrant testing. Any mineralisation on this belt would be small and patchy and restricted to near surface depths. The best exploration for this type of mineralisation would be surface costeaning over large areas, a method commonly used by early prospectors in the district.

Two mines not previously mentioned due to their position are the 'East Tasmania' mine and 'Dally's United' mine. Neither of these mines were active producers but were sunk as prospecting ventures attempting to intersect any extension of the Tasmania reef east of the Gordon Limestone correlate. Should the line of fissure persist east of the Gordon Limestone correlate, then comparatively small 'shoots' of ore could occur in the sandstone members of the succession overlying the Gordon Limestone correlate, but any mineralisation found would be repetition on a small scale and not a continuation of the Tasmania orebody as such.

Some small quantities of 'alluvial' gold may still be won from the thin Tertiary gravel beds on the lower slopes of Cabbage Tree Hill, and on the plain east of the Beaconsfield township, but large accumulations cannot be expected.

SALISBURY DISTRICT

The Salisbury goldfield is situated at the southern end of Salisbury Hill 6 km SSW of Beaconsfield and is in effect an extension of the Beaconsfield goldfield. There is no information on the discovery of the field except for the comments by Duff (1888, p.3) suggesting that very little time elapsed between the discovery of the Tasmania reef and the deposits at Salisbury.

By 1883 the Victoria workings had been completed after driving an adit 185 m into the east flank of the hill and a further 180 m northerly along a lode formation and all mining had ceased. A short revival of interest occurred in 1893-96 with further work in the Victoria mine and on sluicing operations at the nose of the southern spur but the field again became dormant until the sinking of the Salisbury shaft in 1903.

The main workings were the Salisbury, also known as the Victoria; and the Duchess of York, also called the Gladstone and Santa Claus, mines. Both mines were explored by a combination of shafts and adits from the eastern flank of the ridge. The workings are roughly in the same position respective to the strata as the Cosmopolitan mine on the eastern flank of Cabbage Tree Hill. Some open cut, hydraulic sluicing workings are located in the nose of the southern spur of the ridge.

The mineralisation in these mines differs markedly from that of the Cabbage Tree Hill mines in that the majority of gold occurrences in the near surface workings occurred as 'coarse lumps of gold' and 'patches of free gold met with in sugary quartz and soft seams of pug'. In several instances the gold had a superficial coating of black manganiferous oxides giving rise to the so called 'black gold' of the locality.

In the deeper levels of the adits all the gold was reported to be intimately associated with sulphide mineralisation. Occurrences of nickel and chromium minerals are recorded from the main adits closely associated with an intrusive body of basic rock.

High grade concentrations or 'pockets' of eluvial/alluvial gold occurred in the talus on the cresh of the south spur and in the talus/alluvium admixture at the foot of the spur. Gold values in these deposits was also reported to be extremely 'patchy' with high grade pockets interspersed with large areas of almost barren material.

No production figures for the Salisbury district are available.

LEFROY GOLDFIELD

The Lefroy goldfield is situated 40 km north of Launceston, and 15 km east of George Town. Some 30 auriferous formations occur in the field the majority occurring en echelon in a NNW-trending zone 4 km long centred on the township of Lefroy.

All the workings are inaccessible and the only evidence of mining is the old dumps, remains of foundations and collapsed shafts.

History

Gold was known to occur in the Lefroy district prior to 1864 (Gould, 1864) and possibly as early as 1853. Reef gold was first discovered at Specimen Hill by S. Richards and party in 1869 resulting in the opening of the Reward mine. Further prospecting soon located the eastern extensions of the Land-O'-Cakes and Volunteer reef systems and the township of Nine Mile Springs sprang up centred on these workings. Later prospecting to the west and north resulted in the discovery of the Golden Point and Native Youth orebodies and the township of Lefroy grew around these mines.

The field had a history of sporadic activity as the gold on each new discovery was exhausted. The early finds were soon worked out and the field lapsed until the discovery of the Chum, Golden Era and extensions of the Land-O'-Cakes orebodies in 1880 but again gold values declined sharply at

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Part B.
Reference 1/9.

SUMMARY REPORT ON DIAMOND DRILLING ACTIVITY

ON THE TASMANIA LODE, BEACONSFIELD, TASMANIA

1964 - 1974

T.W. MIDDLETON

December, 1974

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511290

CONTENTS

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SUMMARY

Two stages of diamond drilling on the Tasmania Lode at Beaconsfield have indicated the continuity of medium to high grade gold mineralisation, over mineable widths, down to a depth of about 400 ft below the limit of the old workings, ie 1900 ft below mine datum. A third hole, DDH A3, would appear to have been sited too far to the north east, considering the ultimate course, to intersect the lode in a favourable lithological horizon.

The so called "Transition Beds", a series of conglomerates, pebbly sandstones, sandstones, calcareous sandstones, siltstones and shales, which act as an effective host for the Tasmania Lode, would appear to continue below at least -3,000 ft R.L. without major dislocation. There is an apparent major shear indicated in DDH A3, to the east of the workings, which may have a terminating effect on the lode, but not above -3,000 to -4,000 ft R.L. The Tasmania Lode fissure may in fact have been resultant from overthrust movement on this feature.

INTRODUCTION

This report is presented as a brief summary of drilling activities and findings on the Tasmania Lode for the purpose of the record and with a view to examining the ultimate lode continuity potential.

The drilling was carried out from three sites: DDHB4 by the Tasmanian Mines Department (1964-1967); DDHA3 by Allstate Exploration N.L. (1969-1972); and DDHA6 by Allstate Explorations N.L./Tricentrol Australia Limited (1973-1974).

Unfortunately there is a somewhat variable quality of documentation associated with the activities, however, an attempt has been made to construct a composite cross section showing the drillhole plots in relation to the known lode position as indicated by the original workings.

The views with regard to interpretation expressed herein are those of the writer, drawing on a compilation of the available data.

DIAMOND DRILLING

(a) Mines Department 1964-1967

This programme has been well documentated (Noldart, 1968) with several plans and sections presented showing the relationships of the old mine survey data with the diamond drill hole. Although intended to intersect the lode at -2,000 feet R.L., variable azimuth and dip deviation resulted in an initial intersection centred at about -1,700 feet R.L. with first wedge hole (B4A) making an intersection close by and a second wedge hole (B4B) making an intersection centered at about -1,745 feet R.L.

Down the hole surveying was carried out by tropari and this data and the compilations based on it are accepted without question.

(b) Allstate 1969-1972

This programme is unfortunately not very well documented. The only summary (Sheehan, 1972) is incomplete and the diamond drill logging, which was carried out by several authors, reflects lack of continuity in documentation.

Apparently the original hole A3(1) was cored from 552 feet down to an unknown depth (about -2,760 feet according to the only available section). This hole was reamed to 898 feet and a second hole commenced and carried down to 3,530 feet. A third hole, A3(iii) was wedged off A3(ii) at 2,156 feet 2 inches and carried down to 3,790 feet. ? A3

Several quartz veined zones and shears were encountered however no assays were made.

The only systematic survey data was obtained by a "Piko" camera instrument, recording at mostly 100 ft intervals from the surface to the bottom of hole A3(iii). Records of surveys in the original holes are not available and doubt has been expressed on the original Allstate cross section as to the reliability of same.

(c) Allstate/Tricentrol 1973-1974

This hole was originally sited as DDHA5 at about 110 feet north east of DDHB4 and collared with a Failing rotary rig down to 690 feet whereupon the hole was abandoned due to casing difficulties.

* A new hole, DDHA6, was sited 36 feet to the east and taken down to 970 feet with the Failing rig before diamond drilling was commenced at a wedge off point about 570 feet down the hole, due to casing difficulties and excessive flattening. Eventually a lode intersection was obtained at 1902.5 ft (about -1910 ft R.L.). This hole was carried through the lode and terminated at 2040 feet.

A branch hole, DDHA7 was wedged off A6 at 1534 feet, this intersecting the lode at 1891.75 feet down the hole (about -1870 ft R.L.). The hole was terminated at 1929 feet.

The lode intersections were halved and sampled in one foot intervals, these being submitted to the Mines Department Laboratories in Launceston for precious metal fire assays and copper A.A.S. assay. The residues after assay were used for intensive bench scale metallurgical studies.

/....4

Down the hole surveys were conducted regularly throughout the two holes using a "Pajari" instrument which appears to have been quite reliable.

Cross Section Construction

The longitudinal projection as constructed by Noldart (1967) based on the original mine plans and showing the projection of the Mines Department DDHB4 has been taken as the basis of the section. This plane strikes 37^0 -217^0 magnetic and the section is arbitrarily placed through the collar of DDHA6.

The other drill holes are projected onto this plane, adequate control being available in the case of DDHA6/A7, however, in the case of DDHA3, the survey data available for A3(iii) is all that can be accepted and the other two branches are omitted.

The vertical projection shows the relationship of the drill holes with the projected lower levels of the old workings. Considering an average lode strike of 24^0 - 204^0 for the 1500 foot level and an average dip of 60^0 the lode mean position at -2,000, -2,500 and -2,700 feet R.L. has been plotted on the vertical projection. These are laterally limited by arbitrary projections based on the limits of stoping on the upper levels. It should be stressed that there is considerable divergence in strike and dip evident in the lode down to the 1500 foot level, therefore the lode contours indicated purely suggest mean position.

The stratigraphic continuity as represented on the cross section will only be free from distortion if the strike is more or less normal to the plane of section.

GEOLOGY

(a) Lithology

Good correlation of lithologies is possible between DDHB4 and DDHA6, each intersecting the complete Ordovician section of (from the top) black shale, massive limestone; argillites and arenites with diminishing calcareous content; black quartzite and the upper conglomerates. This same sequence is met with in DDHA3(iii) below about -1,850 feet R.L. and correlation with the other two holes indicates an overall average dip of about 55^{o} which is quite on a par with the available information from old reports.

A major shear - breccia zone was apparently intersected from about 1585 - 1774 feet down the hole in DDHA3(iii) with often poor core recovery of limestone and quartzite fragments in a calcareous matrix. Above the shear zone, logging continuity was disrupted, however the descriptions indicate a strongly foliated, distorted section of phyllites, calc-silicate rocks and limestones, suggestive of a sheared upper "Transition Beds" section with overlying limestones. This apparent repetition would appear to be indicative of an overthrust movement, east block up.

The stratigraphy as indicated on the accompanying cross section is based on Gee and Legge (1974) with the addition of terms such as "Transition Beds", which are not necessarily to be considered as strict stratigraphic units.

(b) Structure

The evidence of a major thrust fits in well with Gee and Legge's (1974) structural interpretation of the Beaconsfield area with a series of thrust slices 'riding up' against the Precambrian massif of the Asbestos Ranges.

Certainly an upthrusting is necessary to explain the presence of known Cambrian metasediments and volcanics about two miles south-east of Beaconsfield. If the same north-west trending thrust plane is that intersected in DDHA3 then an anti-clockwise rotation is indicated, with the hinge point just north of Beaconsfield.

The fault zone occupied by the Tasmania Lode may be a tensional feature associated with movement on the main thrust plane. The possibility of other auriferous quartz filled tensional faults on the west or east side of the thrust cannot be discounted.

It should be noted here that a degree of dextral transcurrent movement has taken place on the Tasmania Lode fault, ie the footwall country is displaced about 30 m to the north-east (Noldart and Threader in Gee & Legge, 1974).

From the vertical projection presented herewith it can be seen that DDHA3 should have intersected the eastern part of the Tasmania Lode at -2,700 feet R.L. The log of DDHA3(iii) indicates a section between 2379 - 2393 feet down the hole (about -2,425 R.L.) of brecciated light grey "quartzite" with calcite and siderite veining and minor pyrite in the veins. This breccia, in predominantly limestone country, could conceivably be the eastern manifestation of the Tasmania Lode.

Other shear zones occur further down this hole, however their significance is not known.

(c) Lode Characteristics and Mineralisation

The intersections in DDHB4 and A6 indicated a quartz-carbonate lode with variable sulphides and some visible free gold. Mineragraphic studies have indicated the Paragenesis as: an original quartz-carbonate vein (carbonate probably ankerite), brecciated and invaded by pyrite with further brecciation and an invasion by arsenopyrite, chalcopyrite, galena and sphalerite (Whitehead, 1974).

Some fine to coarse specks of free gold were identified in the DDHB4 intersections (Noldart, 1968) and also some very fine particles as inclusions in pyrite from the A7 interval (Whitehead, 1974), however, microscopic examination of core from A7 indicates a substantial amount of free gold.

Chalcopyrite would appear to be present in concentration of 2-3% overall making a possible by-product production, while silver occurs in lower concentrations than gold but with apparently no direct relationship. Of the elements analysed for in DDHA6, only arsenic shows a direct proportional relationship with gold, however the same ratio is not observed from hole to hole.

A tendency for the lode to split has been noted in the lower mine workings and this behaviour has also been emulated in DDHB4, B4B and A6. In the latter, a weakly auriferous, thin, footwall pyritic quartz vein was separated from the main (hanging wall) lode by a "horse" possibly three feet in thickness (6 feet along the hole).

The ultimate points of intersection from the DDHB4 and DDHA6 collars are situated rather close together and no information is available regarding the horizontal length of the ore shoot. The stoped length on the 1500 foot level was about 840 feet compared with in excess of 1200 feet in the levels above, the cessation of development possibly accounting for the difference, rather than shortening of the (now) economic length.

Lode Continuity Possibilities

The drilling has shown the lode in many respects to be more or less similar at the lowest point intersected to the higher levels. There is possibly an overall increase in sulphides and a decrease in the proportion of free milling gold however it must be remembered that a degree of deep weathering has undoubtedly taken place, considering the presence of Tertiary "Deep Lead" Channel which is possibly 400-500 feet deep. Solution action in lime rich beds has probably led to considerably deeper partial oxidation effect thus creating an apparent deep gold enrichment effect. Not enough is known about the sulphides make-up in the lower workings to be able to suggest a zonation effect which might suggest a lowering concentration of gold.

If the postulated thrust plane exists as shown on the cross section, then a structural limit to the Tasmania Lode as such is implied, but not above -3000-4000 ft R.L.

The "Transition Beds" host rock strata would appear to show no major dislocation between DDHA6 and DDHA3, suggesting that the structure, regardless of metal concentration, should be continuous to at least 3500 feet.

T.W. MIDDLETON
18th December, 1974

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REFERENCES

Gee, R.D., and Legge, P.J.	1974	Geological Survey Explanatory Report 1 Mile Series: Beaconsfield <u>Tas. Mines Dept.</u>
Noldart, A.J.	1968	Exploratory Diamond Drilling, Tasmanian Gold Mine, Beaconsfield Goldfield. <u>Technical Repts., Tas. Mines Dept.</u>
Sheehan, M.	1972	Beaconsfield Gold Project Final Report Private Report to Allstate Explorations N.L.
Whitehead, S.	1974	Reports MP 3634/74 and MP 3929/74 Mineragraphic & petrographic descriptions reported to Allstate Explorations N.L. by The Australian Mineral Development Laboratories.

APPENDIX

SUMMARIES OF DRILL HOLE ASSAYS

DDH B4, B4A, B4B, A6,A7

DIAMOND DRILL HOLE	INTERVAL	INTERSECTION WIDTH	WEIGHTED ASSAY Au dwt/1 ton	Ag dwt/1 ton	Cu%	As%	S%
B4	1689 - 1707.67 ft.	18.67 ft.	59.3				
B4 composite	1689 - 1707.67 ft.	18.67 ft.	60.2	4.7	1.06	1.49	7.5
B4A	1680.83 - 1704.5 ft.	23.67 ft.	41.4				
B4A composite	1680.83 - 1704.5 ft.	23.67 ft.	42.1	6.6	0.91	0.43	4.9
B4B	1723.5 - 1747.58 ft.	24.08 ft.	16.8				
B4B composite	1723.5 - 1735.25 ft.	11.75 ft.	26.8	11.0	1.10	0.05	3.6
A6	1903.5 - 1911.67 ft.	8.17 ft.	8.48	Nil ?	0.71	1.23	
A6	1918.33 - 1919 ft.	0.67 ft.	3.60	Nil ?			
A7	1891.75 - 1905.75 ft.	14.00 ft.	23.1	7.2			
A7 composite	1891.75 - 1905.75 ft.	14.00 ft.	24.4	7.1	0.54	0.67	

All assays carried out by Department of Mines Laboratories, Launceston.

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TASMANIA 511302 PART B. Reference 2/1

No. 17/73 (Regulation 6A)

THE MINING ACT 1929

EXPLORATION LICENCE UNDER SECTION 15B

ISSUED to Allstate Explorations N.L. of Suite 3109, Tower Building, Australia Square, SYDNEY, N.S.W. 2000

in respect of Five Thousand Three Hundred ~~square miles~~ Acres (5300) (more or less) of land in the Land District of DEVON

vicinity of Beaconsfield as described in the schedule hereto.

This licence shall remain in force until the Twelfth day of January, 1974.

This licence is subject to the following conditions:—

1. That the licensee shall immediately on the issue of this licence take steps to commence preliminary works necessary for the investigation of the area.
2. That the licensee shall carry out investigations as may be necessary to determine the potential of the area and in particular will commence a new diamond drill hole to intersect the lode about the 2500 ft. level. If the lode is intersected offsets will be wedged from the parent hole.
3. That the licensee shall employ such technical and other staff and equipment as may be necessary effectively to carry out such investigations.
4. That the licensee shall furnish the Director of Mines, Hobart with complete records including plans of drilling and other work within the compass of the programme of exploration. Such records and plans shall be held for official purposes during such time as the areas involved are lawfully held by the licensee or as otherwise agreed to.
5. That the licensee shall observe the provisions of Section 35 of the Mines Inspection Act, 1968, with regard to notification of boreholes preservation of cores and disposal thereof.
6. If required by the Director of Mines, the licence holder will forward duplicate samples of rock and mineral samples obtained in the licence area to a place approved by the Director of Mines.
7. That a Statement of Expenditure verified by statutory declaration shall be lodged with the Director of Mines, Hobart, at the end of each calendar month from the date of this licence.
8. That such Statement shall be accompanied by a progress report of operations.
9. This licence shall apply to all minerals
10. When no longer required, all large or deep excavation, particularly those made by bulldozer or other earth moving equipment, shall be filled in, or otherwise made safe, in accordance with the Mines Inspection Act, 1968, and reasonable rehabilitation measures taken to the satisfaction of the Director of Mines.
11. The licensee shall conduct operations so as not to disturb the environment except in so far as this may be necessary to undertake the programme of exploration required by this licence:

    and in particular shall -

    (a) not clear any natural vegetation or make excavations which may be visible from populated areas;

    (b) discuss with the local Municipal Council any proposal to clear areas of natural vegetation and shall comply with the reasonable requirements of such Council.

330 The licensee shall observe any instructions which may be given by the Director of Mines with a view of minimising or preventing damage to public or private property.

13. If it is found, that the operations hereby authorised, are causing an undue damage to, or erosion of, the subject land or other land in the vicinity thereof or are unnecessarily disturbing the environment, the Minister may cancel the licence without compensation to the licensee giving seven days' notice in writing of his intention so to do.

14. The licensee shall not light any fires without the approval of the Ru Fires Board or other relevant authority.

15. Where any aboriginal relic or objects of historic interest are discov operations shall be conducted so as not to damage or interfere with s site or object and details of such discovery must be reported to the Director of Mines.

16. The licensee shall notify the local representative of the Forestry Commission before entering on a State Forest and shall comply with th reasonable requirements of such officer in operations on any such State Forest.

17. The licensee shall not interfere in any way with native fauna or bird life.

18. The licensee shall notify the owner and occupier of private land befo entering such land.

19. That the security provided by Section 15B (5) of the Mining Act, 1929 (see below) shall be lodged with the Director of Mines before enterin private land.

20. The licensee shall deposit an amount of $500 as security that the conditions contained herein shall be observed. Upon expiry or soone determination of the licence, if the licensee satisfies the Director of Mines that such conditions have been complied with, the Director s refund such deposit, or such portion thereof, as he may determine.

SCHEDULE

Commencing at the south-east corner of land demised and being the nor east corner of 20 acres purchased by G. Blythe thence 2 miles 78 chain westerly 12 miles 87 chains northerly to the south-west corner of 100 acres 11 perches purchased by P.E. Thomas 2 miles 27 chains easterly to a point at high-water mark at Bowen's Jetty on the foreshore of Middle Arm Bay and thence in a general southerly direction by the for shore at high-water mark of Middle Arm Bay aforesaid including the estuary of Brandy Creek and by Middle Arm Creek and Salisbury Creek t the point of commencement. Excluding that part of E.L. 21/72 contain within the area described.

EXCLUSIONS

The area embraced by this licence shall beexclusive of:-

(a) All municipal and public reserves and roadways.

(b) All forms of mining tenements and water licences including leas water licences, easement licences, special and exploration lice prospectors licences, miners rights, permits to enter, owners consents and owners rights which were in lawful possession or marked out prior to the date of marking out of this licence.

(c) Land exempt from the provisions of the Mining Act, 1929.

(d) Land under the National Parks and Wildlife Act, 1970 not subjec the Mining Act, 1929.

(e) Any Crown reservations or other land set apart or dedicated for any public purposes.

Eric Reece
MINISTER FOR MINES
14 July, 1973.

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DEPARTMENT OF MINES - TASMANIA 511304

SCHEDULE 'A'

Conditions of Special Prospectors' Licences and Exploration Licences under the Mining Act, 1929.

Operational:

1. That the licensee shall observe the provisions of Section 35 of the Mines Inspection Act 1968 with regard to notification of bore holes preservation of cores and disposal thereof.

2. At the termination of the licence or at any time at the option of the licensee all drill core and samples required by the Director of Mines shall be delivered in approved containers into the Department of Mines Drill Store at Hobart at the cost of the licensee unless the Director of Mines notifies the licensee in writing that such core or samples are not required.

3. When no longer required, all large or deep excavations, particularly those made by bulldozer or other earth moving equipment shall be filled in, or otherwise made safe, in accordance with the Mines Inspection Act, 1968 and reasonable rehabilitation measures taken to the satisfaction of the Director of Mines.

4. The licensee shall conduct operations so as not to disturb the environment except in so far as this may be necessary to undertake the programme of exploration required by this licence; and in particular shall -

   (a) not clear any natural vegetation or make excavations beyond that necessary for actual sampling;
   (b) discuss with the local Municipal Council any proposal to clear areas of natural vegetation and shall comply with the reasonable requirements of such Council.

5. The licensee shall observe any instructions which may be given by the Director of Mines with a view of minimising or preventing damage to public or private property.

6. The licensee shall not light any fires without the approval of the Rural Fires Board or other relevant authority. In the case of Crown land the approval of the Director of Lands is required.

7. The licensee shall notify the local representative of the Forestry Commission before entering on a State Forest and shall comply with the reasonable requirements of such officer in operations on any such State Forest.

8. Where any aboriginal relic or objects of historic interest are discovered, operations shall be conducted so as not to damage or interfere with such site or object and details of such discovery must be reported to the Director of Mines.

9. The licensee shall not interfere in any way with native fauna or bird life.

10. Where investigations are undertaken near the coastline or in sand dune areas or where erosion is likely the following conditions shall apply in addition to the foregoing conditions.

   (a) Earth moving equipment comprising bulldozer, back-hoes, or similar mechanical equipment is not to be used and operations are to be conducted by means of drilling or by the use of hand tools or equipment.
   (b) The licensee shall conduct operations as not to cause any damage to shrubs, trees, or other native flora growing on the area demised.

DEPARTMENT OF MINES - TASMANIA

SCHEDULE 'A'

10. (c) Where it is necessary to remove native or other grasses to enable prospecting operations by boring or otherwise the licensee shall keep to a minimum the area of surface to be disturbed.
(d) All surface soil and grasses shall be stacked separately for replacement.
(e) All excavations shall be filled in as soon as practicable and surface soils and grasses replaced.

11. Before commencing the construction of access routes or other works likely to have any significant effect on the environment the licensee shall forward details with plans to the Director of Environmental Control at Hobart and shall forward a copy to the Director of Mines.

Reporting:

1. A Statement of Expenditure verified by statutory declaration shall be lodged with the Director of Mines, Hobart at the end of each calendar month from the date of this licence.

2. This Statement shall be accompanied by a progress report of operations.

3. The licensee shall furnish the Director of Mines, Hobart, with complete records of all investigations undertaken during the term of the licence. These shall include detailed reports, plans, sections, analyses, metallurgical investigations and feasibility and other studies. All plans must include transparencies unless the Director of Mines advises in writing that such are not required.

4. All information furnished to the Director of Mines under this licence will be held for official purposes:

   (a) during a period of five years from the date on which such information was furnished to the Director of Mines; or
   (b) until the areas to which the reports relate are no longer lawfully held under the Mining Act, 1929

   whichever shall occur first.

5. Upon relinquishment of any part of the area described in the schedule hereto the licensee shall furnish a report containing all information relating to such area unless the Director of Mines advises in writing that such report is not required.

SCHEDULE 'A'

DEPARTMENT OF MINES - TASMANIA

Other:

1. That the licensee shall carry out investigations as may be necessary to determine the mineral potential of the area.

2. That the licensee shall employ such technical and other staff and equipment as may be necessary effectively to carry out such investigations.

3. This licence shall apply to ..ALL MINERALS EXCLUDING COAL... ....AND OIL..........................

4. The licensee shall notify the owner and occupier of private land before entering such land.

5. That the security provided by Section 15B (5) of the Mining Act, 1929 (see below) shall be lodged with the Director of Mines before entering such land.

6. The licensee shall observe, perform and fulfill the conditions as set forth in Schedule 'A' attached hereto.

7. The lessee shall be liable to pay the cost of any work carried out to remedy any damage arising from any breach of the conditions of this licence.

8. The licensee shall deposit an amount of \$ 500 as security that the conditions contained herein shall be observed. Upon expiry or sooner determination of the licence, if the licensee satisfies the Director of Mines that such conditions have been complied with, the Director of Mines shall refund such deposit, or such portion thereof, as he may determine.

9. If it is found that the operations hereby authorised, are causing any undue damage to, or erosion of, the subject land or other land in the vicinity thereof or are unnecessarily disturbing the environment, the Minister may cancel the licence without compensation to the licensee by giving seven days' notice in writing of his intention so to do.

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ENDORSEMENTS

TRANSFER

Transferred under Section 15C (10) of the Mining Act, 1929, to ALLSTATE TASMANIA PTY.LTD. of 291 George Street, Sydney, N.S.W. 2000. Dated this 23rd day of August, 1973.

MINISTER FOR MINES

24 August, 1973.

EXTENSION

This licence is extended under the provisions of Section 15C (6) of the Mining Act, 1929, until the 12th July, 1974.

MINISTER FOR MINES.

17 January, 1974.

EXTENSION

This licence is extended under the provisions of Section 15C (6) of the Mining Act, 1929, until the 12th January, 1975.

MINISTER FOR MINES

18 June, 1974.

EXTENSION & ALTERATION OF CONDITIONS

This licence is extended under the provisions of Section 15C (6) of the Mining Act, 1929, until the 12th July, 1975. The current conditions of the licence are rescinded and replaced by those on Schedule 'A' attached.

MINISTER FOR MINES

18 December, 1974

EXTENSION

This licence is extended under the provisions of Section 15C (6) of the Mining Act, 1929, until the 12th January, 1976.

Acting MINISTER FOR MINES

June, 1975.

335

511308

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- 4 JUL 1974
Ans'd.........

EXTRACTS FROM THE PROVISIONS OF THE MINING ACT 1929

Exploration Licences

SECTION 15B (3) An exploration licence—

I Has effect in relation to such area, and for such period, as the Minister may determine:

II Shall be granted upon and subject to such terms and conditions as are prescribed in this section and such other terms and conditions (including conditions as to the fees and rent to be paid by the holder thereof) as the Minister may determine:

and

III While in force, has effect to authorize the holder thereof, subject to the observance of the terms and conditions thereof and to the provisions of this Act, to enter upon and pass over or across Crown Lands and, subject to subsection (5) of this section, private lands, within the area to which it relates, and to prospect and search thereon for such mining products as may be specified in the licence and to do all such other acts and things as may be reasonably necessary to enable the holder thereof to engage in large scale exploration work.

(4) The holder of an exploration licence shall, subject to this Act—

I Engage, to the satisfaction of the Minister, in such aerial, geological, or geophysical surveys and exploration as the Minister may direct or approve:

II Furnish the Director with such periodical reports and returns as the Minister may direct or approve:

and

III Keep an adequate record of all operations conducted under the authority of the licence, and at all reasonable times permit the Director, or any officer authorized by the Director so to do, to examine those records and inspect any specimens or materials obtained in the course of those operations.

(5) The holder of an exploration licence shall not enter on private land thereunder unless he has given security as provided in subsection (2) of section seventy, and, upon entering on private land, is subect to sections seventy-one and seventy two, as if his exploration licence were a permit under section seventy.

15C (6) Upon application made in that behalf by the holder of a special prospector's licence or of an exploration licence before the expiration of the period for which it is granted to have effect the Minister may extend the licence for such further period or periods, and upon such conditions, as he thinks fit but so that in the case of a special prospector's licence, the aggregate period for which the licence is, and all extensions thereof are, granted, does not exceed twelve months.

(7) Where a licence is extended pursuant to subsection (6) of this section, the Minister may—

I On the recommendation of the Director, add to: or

II Reduce the area of land comprised in the licence.

(8) If the holder of a special prospector's licence or of an exploration licence contravenes or fails to comply with any of the provisions of this Act or any of the terms and conditions to which the licence is subject, the Minister may, by notice in writing to the holder, revoke the licence.

(9) With the consent of the holder of a special prospector's licence or an exploration licence—

I A prospector's licence, mining lease, water licence, or easement licence may be granted in respect of land comprised therein as if the special prospector's licence or exploration licence did not exist: and

II A Miner's right may be exercised as if the land comprised in the special prospector's licence or exploration licence were unoccupied land.

(10) Special prospector's licences and exploration licences may, with the consent of the Minister, be transferred as prescribed on payment of the prescribed fee.

REGULATION 4 (5) Where application is made for a special prospector's licence or an exploration licence, and a licence is granted thereon in respect of an area less than that comprised in the application, the holder, within seven days after the issue of such licence, shall affix to his datum-post an amended notice, showing the area which he is authorized to prospect.

337

511309 PART B.
Reference 3/1

15th October, 1974.

R684

Gold Recovery Tests - Beaconsfield

Allstate Exploration N.L.

Introduction

Gold recovery tests were required by Allstate Exploration N.L. on five samples of diamond drill taken from intersections of the Beaconsfield orebody. Three of the samples were from holes put down by the Tasmanian Government Department of Mines and two samples were from Allstate drilling.

Sample Preparation

All the available individual core samples were used to make the intersection composites for the recovery tests, and therefore were not weighted proportionately. The following table shows the assays for each intersection and the calculated heads of the samples used in the recovery tests.

D.D.H. No.	B4		B4A		B4B		A601-11		H6WL	
SAMPLE	660984	TEST	662821	TEST	671015	TEST	740911	TEST	740852	TEST
Au g/t	90	167	60	56.5	40	61	11	14.6	38	35.3
Ag g/t	7	18.1	9	3.8	16	-	-	2.9	11	6.2
As %	1.5	0.95	0.4	0.51	0.05	-	1	0.74	0.7	0.72
Cu %	1.1	1.43	0.9	0.98	1.1	1.3	0.8	0.89	0.5	0.50
S%	7.5	8.4	4.9	4.4	3.6	-	5	5.0	-	13.2

Each sample was roll crushed in closed circuit with a 1 mm. screen until all the material passed through the screen.

Test Work and Results

1. **Test N1 - sample Reg. No. 671015 - D.D.H. B4B**

Core from diamond drill hole B4B was selected for a preliminary test, beacause of its low arsenic content, and was in this aspect different from the remaining samples.

The whole sample was passed over a Denver laboratory mineral jig (No. 1M). The jig tailing was ground in the 8" diam. x 8" Warman laboratory ball mill, and was then floated in the Denver D1 laboratory flotation cell, using 3 kg/tonne of sodium carbonate and 0.2 kg/tonne of sodium ethyl xanthate, and pine oil as frother to produce a bulk sulphide concentrate. This concentrate was re-floated in a cleaning operation. The cleaned concentrate was re-floated using sodium cyanide at the rate of 0.1 kg/tonne of rougher feed as a pyrite depressant to form a copper concentrate. The tail thus formed was a pyrite concentrate.

The initial flotation test to produce the bulk sulphide concentr- was very active and at the time was thought to be due to the choice of frother.

At this stage of the investigation, copies of Amdel reports MP 3634/74 and MP 3929/74 were received and it was noted that no gold was found by visual examination. The jig concentrate was then examine under the microscope and free gold, gold/ pyrite composites and gold/quartz composites were found and were removed by handpicking.

The remainder of the jig concentrate was ground with a pestle and mortar to pass through a 250 µm screen. The ground product was then floated using Aerofloat 238 as collector and Teric 401 as frother to produce a copper concentrate. A pyrite concentrate was then floated off using sulphuric acid and potassium amyl xanthate. In the flotation of the jig concentrate there was no sign of over-activity in flotation as had been noticed in the bulk-sulphide flotation of the ground jig tail.

The results of the bulk-sulphide flotation test were as follows:-

Product	Mass		Assays		Distribution %	
	%	Au g/t	Cu %	As %	Au	Cu
F7C	1.82	339	24.9	0.28	10.1	34.5
F7T	6.35	178	7.9		18.5	38.1
F6C	8.17	214	11.7		28.6	72.6
F6T	4.51	26	0.87		2.0	3.0
F5C	12.68	147	7.8		30.6	75.6
F5T	84.48	9.5	0.07		12.9	4.5
J1T	97.16	27	1.08		43.5	80.1
J1C	2.84	1210	9.2		56.5	19.9
H	100.0	61	1.3		100.0	100.0

Flotation of the jig concentrate after hand-picking gold from it, gave the following results.

Product	Mass	Assays		Distribution %	
	%	Au g/t	Cu %	Au	Cu
H/P2C	0.0	**		27.1	-
F3C	0.71	250	22.5	2.8	12.5
F4C	2.08	800	4.5	26.6	7.4
F4T	0.05	-	-	-	-
J1C	2.84	1240	8.9	56.5	19.9

** Clean metallic gold, not assayed for silver. Assumed 100% Au for calculation purposes.

2. Tests N2 and N3 - sample Reg. No. 660984 D.D.H. N4

Before treatment of this sample, the depth of ragging in the jig was altered so that less sulphide material would be pulled into the concentrate and a higher grade of concentrate would be produced.

The sample from diamond drill hole N4 was passed over the jig and the weight of concentrate was greatly reduced. The free and composited gold was hand-picked out of the jig concentrate and the jig bed in the jig at the end of the test, and the remaining concentrate and jig bed material was put with the jig tail.

The jig tail was then halved by riffling, and one half was ground for five minutes in the ball mill, and the other half was ground for fifteen minutes in the ball mill.

After 15 minutes conditioning with 1 kg/tonne of sodium sulphite, flotation, with 0.05 kg/tonne of Aerofloat 238 and Teric 401 as a frother, produced a copper concentrate, and 0.1 kg/tonne

..../ 3

of potassium amyl xanthate was used to produce the pyrite concentrate.

..../3

However, the over-active condition was again present in the copper flotation as in the bulk sulphide flotation in test N1. This overactivity resulted in the copper concentrates being low grade.

The results of the two tests with coarse and fine grinding are as follows:-

		Mass %	Assays Cu%	Au g/t	Ag g/t	S%	As%	Distribution % Cu	Au	Ag	S	As
	Free gold			*	*				65.1			
N2	F1C	16.4	5.7	192	22	34.7	3.32	77.7	19.6	23.2	68.5	56.7
Coarse Grind	F2C	12.2	2.0	75	34	16.6	2.47	17.3	5.7	26.5	24.4	31.4
	F2T	71.4	0.10	22	11	0.82	0.16	5.0	9.6	50.3	7.1	11.9
	calculated head	100.0	1.41	160	15.6	8.30	0.96	100.0	100.0	100.0	100.0	100.0
	Free gold			*	*				60.4			
N3	F1C	15.2	7.9	203	29	28.8	2.15	82.8	17.8	21.5	51.4	35.2
Fine Grind	F2C	19.1	1.2	153	29	20.4	2.98	15.8	17.0	27.2	45.8	61.3
	F2T	65.7	0.03	13	16	0.36	0.05	1.4	4.8	51.3	2.8	3.5
	calculated head	100.0	1.45	173	20.5	8.51	0.93	100.0	100.0	100.0	100.0	100.0

* Clean metallic gold, not assayed for silver.
Assumed 100% Au for calculation purposes.

The results of tests N2 and N3 confirm that the over-active condition during copper flotation produced a low grade of concentrate. The results also indicate that the fine grind has produced better recoveries of sulphur and all the metals except silver.

3. Tests N4 & N5 - sample Reg. No.662821 - D.D.H. B4A
   Tests N6 & N7 - sample Reg. No.740911 - A601 - 11
   Tests N8 & N9 - sample Reg. No.740852 - H6WL 1 - 14

Each of the above samples were treated similarly in the following manner. The samples were passed over the jig, and the free and composited gold was hand-picked out of the concentrate and the jig bed, and the remaining material from these products was put with the jig tail. The jig tail was then riffled in half, and each half was ground in the laboratory ball mill for 15 minutes.

Because of the over-active condition of the flotation tests on the first two samples, it was decided to try pre-flotation. The ground sample was conditioned for fifteen minutes with 1 kg/tonne of sodium sulphite, and then floated without any collector type flotation reagent and with only sufficient frother, Teric 401, to maintain a froth.

During five minutes flotation, a flotation concentrate was made which was dirty in appearance. This concentrate may contain a low density free-floating non-metallic mineral such as talc, sericite, or similar material.

A copper concentrate was then produced with the addition of 0.05 kg/tonne of Aerofloat 238 and five minutes flotation, and a pyrite concentrate was produced with the addition of 0.1 kg/tonne of potassium amyl xanthate and ten minutes flotation.

The results of these tests are as follows:-

		Mass	Assays					Distribution %				
		%	Cu %	Au g/t	Ag g/t	S %	As %	Cu	Au	Ag	S	As
	Free Gold			*	*				57.7			
	F1C	2.79	10.5	8	13	12.8	0.34	30.2	0.4	8.7	8.9	1.8
N4	F2C	2.51	15.6	520	61	22.8	3.3	40.4	22.6	36.9	14.2	16.1
	F3C	10.47	2.0	100	11.5	26.0	3.4	21.6	18.2	29.0	67.5	69.0
	F3T	84.23	0.09	0.75	1.25	0.45	0.08	7.8	1.1	25.4	9.4	13.1
	Calc. head	100.0	0.97	57.7	4.2	4.0	0.52	100.0	100.0	100.0	100.0	100.0
	Free Gold			*	*				60.2			
	F1C	2.15	9.1	18	2	12.1	0.38	19.8	0.7	1.3	5.5	1.7
N5	F2C	2.76	18.0	372	53	20.2	2.9	50.1	18.6	42.5	11.8	16.1
	F3C	10.05	2.2	102	8.7	35.0	3.3	22.4	18.6	25.3	74.3	66.8
	F3T	85.04	0.09	1.25	1.25	0.47	0.09	7.7	1.9	30.9	8.4	15.4
	Calc. head	100.0	0.99	55.2	3.4	4.7	0.50	100.0	100.0	100.0	100.0	100.0
	Free Gold			*	*				15.1			
	F1C	2.70	7.6	5	4	7.3	0.48	22.5	0.9	4.0	4.0	1.8
	F2C	3.34	14.8	85	24	23.6	1.2	54.0	19.0	29.5	16.2	5.4
N6	F3C	11.01	1.5	65.3	1.3	28.3	3.3	18.1	48.3	5.4	64.1	49.7
	F3T	82.95	0.06	3	2	0.92	0.38	5.4	16.7	61.1	15.7	43.1
	Calc. head	100.0	0.91	14.9	2.7	4.9	0.73	100.0	100.0	100.0	100.0	100.0
	Free Gold			*	*				15.7			
	F1C	3.55	5.7	6.7	2.7	7.4	0.50	23.4	1.6	3.1	5.1	2.4
	F2C	2.30	19.2	88	50	24.8	1.3	51.0	14.2	37.0	11.1	4.0
N7	F3C	14.63	1.3	57	10	26.3	3.3	21.9	53.2	47.1	74.8	64.8
	F3T	79.52	0.04	2.75	0.5	0.58	0.27	3.7	15.3	12.8	9.0	28.8
	Calc. head	100.0	0.87	14.3	3.1	5.1	0.75	100.0	100.0	100.0	100.0	100.0
	Free Gold			*	*				17.8			
	F1C	1.41	2.3	9.7	17.5	9.9	0.37	6.5	0.5	4.0	1.1	0.7
N8	F2C	2.56	13.7	428	102	37.3	0.74	71.0	37.7	42.4	7.8	2.6
	F3C	29.71	0.33	36.5	10	36.9	2.2	19.8	37.2	48.2	87.8	89.4
	F3T	66.32	0.02	3	0.5	0.60	0.08	2.7	6.8	5.4	3.3	7.3
	Calc. head	100.0	0.49	29.1	6.2	12.4	0.73	100.0	100.0	100.0	100.0	100.0
	Free Gold			*	*				12.5			
	F1C	1.36	2.4	4.8	18.3	9.8	0.35	6.5	0.1	4.0	1.0	0.7
N9	F2C	2.80	12.4	412	79	43.6	0.85	69.6	27.9	35.9	8.7	3.4
	F3C	26.62	0.37	70.5	10	44.9	1.9	19.7	45.3	43.2	85.4	71.5
	F3T	69.22	0.03	8.5	1.5	1.0	0.25	4.2	14.2	16.9	4.9	24.4
	Calc. head	100.0	0.50	41.4	6.2	14.1	0.71	100.0	100.0	100.0	100.0	100.0

* Clean metallic gold not assayed for silver. Assumed to be 100% Au for calculation purposes.

Subsequently the free gold taken by treatment of the sample H6WL1 - 14 was assayed to find the silver content, and was found to be 99.7% Au and 0.3% Ag. Because of this assay no adjustments were made to the metal balances where the free gold had been assumed to be 100% gold.

The results show that the pre-flotation concentrate contains between 20% and 30% of the copper for the diamond drill holes B4B and A601 - 11 and 6.5% of the copper for diamond drill hole W6WL1 - 14 Less than 1% of the gold present in the drill holes tested, reports in the pre-flotation concentrate.

After having taken off the pre-flotation concentrate, there was no trouble in producing a copper concentrate of reasonable grade, ranging from 12.4% to 19.2% in all the tests. The concentrate produced was a rougher concentrate, and there should be no problem in upgrading this to a concentrate of saleable grade by cleaning and recleaning flotation. Over 70% of the copper has been received in the pre-flotation concentrate plus the copper concentrate. A further 20% of the copper is recovered in the pyrite concentrate.

Gold recovery is a little irregular varying from 83% to 99% in the jig and flotation concentrates, depending largely on the head grade. It may be more useful to examine the tailing assays. In al tests the gold assay of the tailing was 3 grams/tonne or less except test N9 in which the assay was 8.5 grams/tonne. The calculated gold head assays of tests N8 and N9 do not duplicate very well, and re-assaying of the products in the tests did not make any appreciable difference. The calculated sulphur head assays in these tests did not duplicate very well either.

Silver recoveries were also erratic varying from 69% to 94% in the flotation concentrate except in test N6 where the recovery was 39%.

The major proportion of the arsenic reported in the pyrite concentrate.

4. <u>Cyanidation Tests</u>.

Cyanidation tests on roasted pyrite concentrates were then undertaken to determine the recovery of gold from these concentrates.

Composite pyrite concentrates for each diamond drill hole were made by taking 50 grams each of pyrite concentrates from the duplicate tests and mixing, viz., from N4 and N5, N6 and N7, and N8 and N9.

The concentrates were then roasted at about 900°C until sulphur dioxide could no longer be detected.

The calcines were then agitated for 16 hours in beakers by means of magnetic stirrers, in 600 mls of solution containing 0.2% potassium cyanide and 0.1% lime.

The results of the cyanidation tests on calcine are as follows:-

Pyrite Conc.	Calcine weight as % of pyrite conc.	Calcine Assay % As	As Distribution in calcine	As Distribution in exhaust	% Recovery in solution Au	% Recovery in solution Ag
B4A (N4 & N5)	72.8	1.7	36.9	63.1	82.8	nil
A601-11 (N6 & N7)	81.9	3.2	79.4	20.6	48.4	nil
N6WL1 - 14 (N8 &N9)	69.1	0.82	27.6	72.4	79.6	10.0

Cyanidation tests were also done on the flotation tailings.

Composite flotation tailings for each diamond drill hole were made by taking 500 grams each of flotation tailings from the duplicate tests and mixing, viz., from N4 and N5, N6 and N7, and N8 and N9.

The tailings were agitated for 24hours in a litre of solution containing 0.5% sodium cyanide, and lime to give a pH of 11 to 12.

The results of the cyanidation tests on flotation tailings are as follows:-

Flotation Tailing	Cyanide Consumption Kg/tonne of tailing	Gold recovery in solution gms/tonne	Gold recovery in solution %
B4A (N4 & N5)	1.6	0.55	54.5
A601 - 11 (N6 & N7)	0.2	0.66	22.9
N6WL1 - 14 (N8 & N9)	0.2	1.80	31.3

Conclusions

Free gold can be recovered by jigging. The amount recovered in this manner is largely dependent on the head value of the ore.

There is a free-floating gangue mineral present in the ore. It should be advantageous to remove this material by pre-flotation. Some copper tends to float with this material, but gold does not. When ore becomes available, more extensive flotation test work should be carried out in order to determine the best method of coping with this free-floating material and endeavouring to plawe more copper into the copper concentrate.

Copper flotation yielded concentrates assaying about 400 grams of gold per tonne and about 15% copper. Cleaning and recleaning of the copper concentrate should appreciably raise these assays. About 20% of the gold present is recovered in the copper concentrate and this figure is somewhat dependent on the head grade of the ore.

The pyrite concentrate carried appreciable quantities of gold, After calcining the pyrite concentrates and cyanidation, about 80% of the gold in the pyrite concentrates can be recovered.

There is little point in cyanidation of the flotation tailings as there is not much gold present, and recovery of this gold is low.

Diamond drill hole A601-11 was the lowest in grade, and it was also the most difficult to treat. About 16% of the gold present in this ore found its way into the flotation tailing. Cyanidation of the tailing gave the lowest recovery - 22.9%, and cyanidation of th calcine of the pyrite concentrate also gave the lowest recovery - 48.4%.

It is planned to try cyanidation of the pyrite concentrate without roasting to compare recoveries. The result of this test will be reported separately.

Senior Metallurgist. Chief Chemist & Metallurgist.

91460 344

TELEPHONES:

Metallurgical Research | 4 2431-2
Laboratory | 2 Lines
Mines Inspection |
Explosives and Inflammable Liquids |

Registrar of Mines 2 2457

Tasmania 511316

Department of Mines,
Launceston Offices,
287 Wellington Street,
South Launceston

19th November, 1974.

Allstate Exploration NL.,
Suite 3107,
Australia Square,
SYDNEY, 2000 N.S.W.

Dear Sirs,

R 684

Please find attached results of further test work on above project.

Yours faithfully,

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

Encls.

R 684 Part (ii)

Gold Recovery Tests Beaconsfield

Allstate Exploration N.L.

Cyanidation Tests

A cyanidation test was carried out on pyrite concentrate made in tests N8 and N9 from diamond drill core from sample 740852. A composite of 50 grams from each of these tests was agitated for 16 hours in a beaker by means of a magnetic stirrer in 600 mls of solution containing 0.2% potassium cyanide and 0.1% lime. The results are as follows together with the results of the cyanidation of the calcine for comparison.

	% Recovery in solution	
740852(H6WL 1-14)	Au	Ag
calcine	79.6	10.0
pyrite concentrate	86.0	14.0

There was insufficient pyrite concentrate from the other bore holes to compare direct cyanidation of the pyrite concentrate with cyanidation of the calcine.

The result obtained on 740852 indicates that on this sample of ore there is no benefit gained in calcining the pyrite concentrate prior to cyanidation.

A sizing analysis on the flotation tail from test N3 on 662821 (B4A) is as follows:-

Screen Aperture(µm)	% mass	% mass cum.
+212	0.2	0.2
+150	1.5	1.7
+106	8.9	10.6
+75	13.1	23.7
+53	13.8	37.5
+38	13.5	51.0
-38	49.0	100.0

(L.J. Rhodes)
Senior Metallurgist

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

347

511318

PART 'B'
Reference 3/2

50 Bong Bong Street,
Kiama
N.S.W. 2533.

11.3.75

Mr. A. Silver,
General Manager,
Allstate Explorations N.L.,
Suite 3910, Tower Building,
Australia Square,
Sydney.

Dear Sir,

Beaconsfield Project.

1. Introduction

Based on a study, firstly of the following reports:

a. by H.W. Cundy and D. Fawcett of May 5th, 1914 to Hon. J.E. Ogden, Minister for Mines, Hobart.

b. Special Report of Mr. Arthur Llewellyn "Tasmania Gold Mine, Beaconsfield, Tasmania 17th January 1914" to Messrs. John Taylor and Sons,

c. Australian Mineral Development Laboratories Reports MP 3634/74 of 3rd April, 1974 and MP 3929/74 of 22nd May, 1974, and

d. Report 684 of the Launceston Office of the Department of Mines - Tasmania 17th October, 1974, with following letters of 19th and 20th of November

and secondly, of a survey of the literature on the treatment of ores with similarities to that of Beaconsfield, and comparisons with the methods and results of the treatment of Australian ores which have points in common with Beaconsfield, viz. Kalgoorlie and Wattle Gully, the following is a summary of the information gained.

2. Summary

2.1 General

The information is largely based on the results of the averaged tests made on three drill core samples from Beaconsfield. It is not known how representative the samples are of the ore to be mined.

The assay values of the three samples and the response of the samples to treatment varied considerably. The averaged assays of the samples were

Cu %	0.79
Au grammes/tonne	35.6
Ag " "	4.3
S %	7.5
AS %	0.66

2.2 Gold Recovery

2.2.1 Total Gold

This includes free gold, gold in copper concentrate and gold in cyanide solution.

The laboratory recovery from a feed containing 35.6 grammes gold per tonne was 86.6 per cent.

Using the same tailing assay and reducing the feed assay to that of the ore reserve figure of 23.9 grms/tonne results in a laboratory recovery figure of 80.1 per cent.

The tests made did not indicate that the tailing assay would decrease with a decrease in the feed assay.

2.2.2 Free Gold

This is the gold recovered by gravity - amalgamation methods.

In the laboratory tests the recovery varied from 14.7 to 58.9 per cent with an average of 38.2 per cent from a feed of 35.6 grms Au/tonne.

It is not possible to predict from the laboratory results what free gold recovery would be obtained from a feed containing 23.9 grms Au/tonne.

Past records (1914) indicate that the free gold recovery, when carrying the gravity - amalgamation treatment down to finer sizes than in the laboratory tests, would be of the order of 40 per cent with the possibility of this figure decreasing as the sulphide contents increased with depth.

2.3 Copper Recovery

The copper recovered as copper flotation concentrate was 53.3 per cent of that in the original feed or 66.9 per cent of that in the copper flotation feed.

This figure needs to be improved to suit subsequent cyanidation operations.

2.4 Gold in Copper Concentrate

The copper concentrate averaged 2.7 per cent weight of the feed and from an original feed of 35.6 grms Au/tonne assayed 311.7 grms Au/tonne.

It has not been proved, but is probable, that some of this gol could be recovered as free gold by improvement on the laboratory gravity treatment. The gold in the copper concentrate would decrease correspondingly.

2.5 Arsenic in Copper Concentrate

This arsenic assay averaged 1.69 per cent

2.6 Pyrite Flotation

The pyrite flotation concentrate contained 79.8 per cent of the sulphur and 30.8% of the gold in the ore samples.

Expressed as percentages of the sulphur and gold in the pyrite flotation feed the figures become 92.0 and 82.1 respectively.

The copper assay of the pyrite concentrate was 0.96 per cent.

Increases in the recoveries of sulphur and gold in, and a decrease in the copper content of the pyrite concentrate are very desirable if pyrite flotation is adopted in the treatment method.

2.7 Gangue Slime Flotation

This was a preliminary to copper flotation.

The froth contained 20.3 per cent of the copper in the feed in a product assaying 6.9 per cent copper and consisting of 2.33 per cent of the weight of the feed.

This froth is a candidate for further treatment.

2.8 Cyanidation

2.8.1 Cyanidation of Calcine from the Roasting of Pyrite Flotation Concentrate.

The averaged figures of the laboratory tests were:

gold in solution per cent of gold in conc.	73.3
gold in solution per cent of gold in head	22.6
gold in Cy. tail per cent of gold in head	8.2
Cy. tail assay gold grms/tonne	23.5

The assay value of the cyanide tails was high.

It has not been demonstrated but, compared with practice elsewhere, the laboratory roasting temperature may have been too high for the best results in the subsequent cyanide leach.

2.8.2 Direct Cyanidation of Pyrite Concentrate

The result of the only test made by cyaniding the pyrite concentrate without roasting gave a better gold extraction than obtained from the same material by roasting and cyaniding the calcine.

The test was made on one sample only - pyrite concentrate from tests N8 and N9.

The gold obtained in solution compared thus

	Distribution % of Au From	
Treatment	Pyrite Conc.	Original Feed
Calcine cyanidation	79.6	33.4
Conc. cyanidation	86.0	36.1

The literature agrees that clean arsenopyrite is inert in cyanide solution, the same applies to pyrite.

Roasting should therefore not be required unless the gold is so intimately associated with the sulphides that roasting is needed to expose the gold to cyanide solution, or in cases where oxidation to arsenious compounds has occurred.

Also roasting of Beaconsfield products would be very objectionable because of the arsenic content.

2.8.3 Cyanidation of Flotation Tailing

The averaged figures for the cyanidation of pyrite flotation tailing were:

Gold in Solution Per Cent of Gold In		Gold Assay Grms/tonne		Tail Wt. %
Head	Flotation Tail	Flotn. Tail	Cy.Tail	Original
2.1	31.3	3.0	2.1	77.71

2.9 Comparison of Beaconsfield, Kalgoorlie and Wattle Gully Ores and Treatment

2.9.1 Minerals Present

Mineral	Beaconsfield	Kalgoorlie	Wattle Gully
Quartz	x	x	x
Ankerite	x	x	-
Sericite	-	x	-
Slate	-	-	x
Pyrite	x	x	x
Arsenopyrite	x	minor	x
Chalcopyrite	x	minor	minor
Galena	minor	minor	minor
Sphalerite	minor	minor	minor
Telluride	-	minor	-

Assays

Cu %	0.79	0.03	low
As %	0.66	0.013	0.2-0.4
S %	7.5	1.8-2.9	0.3-0.6
Au Grms/tonne	23.9*	7.7-9.0	9.95

* Ore reserve figure

Beaconsfield has the advantage of a high grade gold ore but with much greater amounts of sulphur, copper and arsenic.

2.9.2 Gold Occurrence

Associations	Beaconsfield	Kalgoorlie	Wattle Gully
Indicated free	40% (1914)	Up to 29%	76%
Telluride	No	Yes	No
Galena	-	-	Yes
Pyrite	Yes	Yes	Yes
Chalcopyrite	?	No	No

In all cases some gold in pyrite was in the form of minute inclusions within the pyrite. Owing to the greater amount of pyrite in Beaconsfield ore this factor may be of greater significance for Beaconsfield.

2.9.3 Methods of Treatment

	Beaconsfield	Kalgoorlie	Wattle Gully
Gravity Concentration	Jig	Strakes	Jig
Grinding % - 200	75	70-80	65
Flotation Stages	Gangue	Sulphides	Sulphides
	Copper	-	-
	Pyrite	-	-
Roasting	Yes	Yes	No
Cyanidation feed	Calcine Flotn.Tail	Ore Sulphide Conc. Calcine Flotn.Tail	Sulphide conc.
Gold recovery %	*80.1	90.8-92.5	95.3
Bulk reject assay Au	5.1	0.61-0.83	0.47

* Estimated recovery from feed of ore reserve grade.

The Kalgoorlie treatment varies on the properties. All of the cyanide feeds listed are not treated on any one mine but various combinations are used.

The Beaconsfield data refers to the Launceston tests and the gold recovered as free gold, gold in solution and gold in copper concentrate.

As could be expected from the composition of the ores the Beaconsfield recovery does not match that of the other fields.

2.9.4 Sulphide Flotation Results

	Beaconsfield	Kalgoorlie	Wattle Gully
Flotation feed assay			
Au grms/tonne	14.1	5.4 - 8.7	2.42
S %	6.9	1.8 - 2.9	-
Flotn.Conc.Assay			
Au g/t	64.3	99.5 -110.8	98.2
S %	35.2	29.9 - 42.0	-
Flotn.Tailing assay			
Au g/t	3.05	0.69- 0.86	0.21
S %	0.66	0.15- 0.30	-
Tailing distribn. % Au	17.8	9.1 -14.6	8.7
Tailing distribn. % S	7.9	6.3 -16.4	-

Note the distribution per cent figures apply to the flotation feed as 100 per cent.

The gold and sulphur assays of the flotation tailing are much higher than those of Kalgoorlie and Wattle Gully.

2.9.5 Cyanidation Results

	Beaconsfield		Kalgoorlie		Wattle Gully	
	Gold Assays					
	Feed	Tail	Feed	Tail	Feed	Tail
Pyrite conc.	-	-	100-103	25-31*	98	11.5
Calcine	88	23.5	31-124	2.3-6	-	-

* These products are retreated.

In some cases the tailing from the cyanide treatment of the calcine is also retreated.

The tailing from the cyanidation of Beaconsfield calcine was very high in gold assay. It is suspected, but not proven, that control of the roasting temperature may have been part of the reasons for this.

2.9.6 Gold Recovery

Considering the poorer results shown at each stage of treatment it appears that gold recovery from Beaconsfield ores will not match the Kalgoorlie, Wattle Gully figures.

2.10 Development of a Process for the Treatment of Beaconsfield Ore.

If it is accepted that all Beaconsfield ore which has been treated for the removal of copper minerals can be treated by cyanidation without roasting various combinations of treatment may be considered.

For the treatment of copper flotation tailing these could be

a. direct cyanidation of the total copper flotation tailing,

b. flotation of the copper flotation tailing to produce a pyrite-gold concentrate,

c. intensive treatment of the pyrite concentrate thus produced, with or without

d. cyanidation of the pyrite flotation tailing.

To choose the best of these methods further information from a sample of ore representative of the ore to be mined is required. Factors to be investigated would be:

a. methods to increase the amount of gold recovered by gravity amalgamation methods,

b. means of improving the recovery of the copper from the copper flotation feed in the copper flotation concentrate, this is essential for good cyanide practice,

c. the improvement of the sulphur and gold recoveries from the pyrite flotation feed in the pyrite concentrate,

d. cyanidation of the pyrite concentrate,

e. cyanidation of the pyrite flotation tailing,

f. cyanidation of the copper flotation tailing,

g. the optimum degree of grinding for all systems and the power required for such grinding,

h. environmental effects of the storage of cyanide tailing and possibly some discard of fouled cyanide solution.

The knowledge thus gained should enable a sound choice of the treatment method to be made.

With present knowledge intensive treatment of the comparatively small bulk of pyrite flotation concentrate appears the most attractive of the available methods.

3. Conclusions

3.1 Total Gold Recovery

a. From a head value of 23.9 grammes of gold per tonne the laboratory indications are that the recovery of gold as free gold, gold in cyanide solution, and gold in copper concentrate should total eighty per cent.

b. The laboratory tests covered a sequence of intensive treatments. The finally adopted plant treatment practice may improve on the laboratory results but this remains to be proved.

c. Owing to the greater complexity of the ore it is not to be anticipated that the percentage gold recovery will equal that obtained in some well established Australian plants.

3.2 Recovery of Gold as Free Gold

This cannot be predicted from the Launceston series of tests

In 1914, when recovery of free gold was pursued to a finer grind that used for the Launceston tests, the recovery obtained was given as 40 per cent, with a tendency to decrease as the sulphide content of the ore increased.

The sulphide content of the ore in 1914 was not stated in the reports available.

3.3 Recovery of Gold in Copper Concentrate

The tests gave a recovery of 23.7 per cent of the total gold in a flotation concentrate assaying 15.5 per cent copper and 311.7 grms Au/tonne.

It is probable that part of the gold in the copper concentrate could be recovered as free gold by carrying the gravity concentration to finer sizes than in the laboratory tests. The gold content of the copper concentrate would be correspondingly reduced.

3.4 Recovery of Gold in Cyanide Solution

a. From calcine

The roasted pyrite flotation concentrate gave the high cyanide tailing assay of 23.5 grms Au/tonne.

b. From flotation tailing

The cyanidation treatment of flotation tailing also gave a residue with the high assay of 2.1 grms Au/tonne.

c. From pyrite flotation concentrate

Direct cyanidation of one sample of concentrate gave a better result than cyanidation of the same concentrate after roasting.

3.5 Copper Recovery

The copper flotation concentrate contained 53.3 per cent of the copper from a head sample assaying 0.79 per cent copper, concentrate grade was 15.5 per cent copper. A further 20.3 per cent of the copper was contained in a slime flotation froth assaying 6.9 per cent copper.

It is extremely desirable for the sake of the subsequent cyanidation stages of treatment that the copper recovery be largely increased.

3.6 Arsenic

The presence of 0.66 per cent arsenic in the head sample is deleterious to the production of a readily saleable copper concentrate.

The arsenic content of the copper flotation tailing or flotation pyrite concentrate also makes a roasting process undesirable.

Arsenic as clean arsenopyrite is inert in cyanide solution.

3.7 Pyrite

The presence of gold as minute inclusions within pyrite has been noted.

Such gold is difficult to reach by cyanide solutions and would be a contributing cause to the high gold assays of cyanidation tailings.

3.8 Choice of Treatment Methods

Based on the one test on a sample of pyrite flotation concentrate from one diamond drill core sample, and experience elsewhere, it appears that once the copper mineral has been removed, Beaconsfield ore should be amenable to cyanidation without roasting.

Choice of the best of the alternative ways of carrying out the treatment needs to be guided by further work on a representative sample of the ore to be treated.

3.9 Mining Method

Beaconsfield ore contains a variety of sulphide minerals the successful separation of which is made difficult if oxidation is allowed to occur. The mining method used should preferably guard against broken ore remaining untreated for more than short periods of time.

Details of the test results and other information obtained are given in the sections following.

4. Launceston Tests on Drill Core Samples

4.1 General

The Launceston laboratory of the Department of Mines Tasmania - see their Launceston Office Report of Investigation R 684 of 17th October, 1974, and subsequent communications of November 19th and 20th - cover the results of series of tests made on samples from three Beaconsfield diamond drill cores. The samples were small in amount and the work done was therefore limited to scouting tests along lines likely to be successful on such an ore.

The three samples varied widely in their head values and in their response to the method of treatment. It is not known whether the combination of the three samples is representative of the ore to be mined and further complications occur because two of the three samples were considerably higher in gold assay than the reported ore reserves while one was lower.

In the following sections the results from each sample and the average of all tests are first given. Comment follows with more detailed figures for the average of the results of each stage of treatment.

The tests concerned are the duplicate tests N4 and N5, N6 and N7, N8 and N9 and the weighted average of all tests. Many derived figures are given on the basis of the Launceston reported figures.

Gold and silver assays are given in grammes per tonne. All other assays are in terms of per cent of the element concerned.

When estimating the gold recovered in solution by cyanide treatment of calcines it was assumed that no gold was lost during the roasting operation.

4.2 Laboratory Method of Treatment

For the tests concerned the full treatment given involved the following steps:

Crush to pass 1 mm,
Jig concentration and hand picking to recover free gold,

Grinding to approximately 75% minus 200 mesh.

Flotation Stage 1. To remove a fast floating gangue slime,
Stage 2. To recover a copper concentrate,
Stage 3. To recover a pyrite concentrate,

Roasting of the pyrite concentrate,

Cyanidation of the calcine from roasting,

Cyanidation of the tailing from Stage 3 flotation.

4.3 Test Results

4.3.1 Head Samples

		Assays				
Test No.	Sample No.	Cu.	Au.	Ag.	S.	As.
N4 & 5	662821 - DDH B4A	0.98	56.4	3.8	4.4	0.51
N6 & 7	740911 - A 601-11	0.87	15.0	2.9	5.0	0.74
N8 & 9	740852 - H6 WL 1-14	0.50	35.3	6.1	13.2	0.72
Average		0.79	35.6	4.3	7.5	0.66

Note a. the assays were derived from the weights and assays of the products.

Note b. the expected ore reserve gold content has been given as 24.3 grms/ton = 23.9 grms/tonne.

4.3.2 Free Gold Recovered Distribution Per Cent.

Test No.	Gold Distribution %	Gold Assay Head	Gold Assay Jig Tailing
N4 & %	58.9	56.4	23.2
N6 & 7	15.4	15.0	12.7
N8 & 9	14.7	35.3	30.1
Average	38.2	35.6	22.0

4.3.3 Stage 1. Flotation Froth (Gangue Slime)

Test No.	Weight %	Assays					Distribution %				
		Cu.	Au.	Ag.	S.	As.	Cu.	Au.	Ag.	S.	As.
N4 & 5	2.47	9.0	12.4	8.2	12.5	0.36	24.9	0.5	5.3	7.0	1.7
N6 & 7	3.13	6.5	6.0	3.3	7.4	0.49	22.9	1.2	3.5	4.6	2.1
N8 & 9	1.38	2.3	7.3	17.9	9.9	0.36	6.5	0.3	4.0	1.0	0.7
Average	2.33	6.9	8.5	7.9	9.7	0.42	20.3	0.6	4.3	3.0	1.5

4.3.4 Copper Concentrate

Test No.	Weight %	Assays					Distribution %				
		Cu.	Au.	Ag.	S.	As.	Cu.	Au.	Ag.	S.	As.
N4 & 5	2.64	16.8	441.7	56.3	21.4	3.09	45.3	20.7	39.4	12.9	16.1
N6 & 7	2.82	16.6	86.2	34.6	24.1	1.24	52.6	16.0	33.6	13.6	4.7
N8 & 9	2.68	13.0	419.6	90.0	40.8	0.80	70.3	31.9	39.4	8.3	3.0
Average	2.71	15.5	311.7	60.1	28.7	1.69	53.3	23.7	38.1	10.3	7.0

4.3.5 Copper Flotation Tailing (Pyrite Flotation Feed)

Test No.	Weight %	Assays					Distribution %				
		Cu.	Au.	Ag.	S.	As.	Cu.	Au.	Ag.	S.	As.
N4 & 5	94.89	0.31	11.8	2.2	3.7	0.44	29.8	19.9	55.3	80.1	82.2
N6 & 7	94.05	0.23	10.7	1.9	4.4	0.73	24.5	67.2	62.9	81.8	93.2
N8 & 9	95.74	0.12	19.5	3.6	12.5	0.72	23.2	53.1	56.6	90.7	96.3
Average	94.96	0.22	14.1	2.6	6.9	0.64	26.4	37.5	57.6	86.7	91.5

4.3.6 Pyrite Flotation Concentrate

Test No.	Weight %	Assays					Distribution %				
		Cu.	Au.	Ag.	S.	As.	Cu.	Au.	Ag.	S.	As.
N4 & 5	10.26	2.1	101.0	10.1	30.4	3.35	22.0	18.4	27.4	71.2	67.9
N6 & 7	12.82	1.4	60.6	6.3	27.2	3.30	20.0	51.6	27.6	69.6	57.3
N8 & 9	28.17	0.35	52.6	10.0	40.7	2.06	19.8	42.0	46.0	86.6	80.6
Average	17.08	0.96	64.3	9.1	35.2	2.66	20.7	30.8	36.3	79.8	68.8

4.3.7 Pyrite Flotation Tailing

Test No.	Weight %	Assays Cu.	Au.	Ag.	S.	As.	Distribution % Cu.	Au.	Ag.	S.	As.
N4 & 5	84.63	0.09	1.0	1.3	0.5	0.09	7.8	1.5	27.9	8.9	14.3
N6 & 7	81.23	0.050	2.9	1.3	0.8	0.33	4.5	15.6	35.3	12.2	35.9
N8 & 9	67.77	0.025	5.8	1.0	0.8	0.17	3.4	11.1	10.6	4.1	15.7
Average	77.88	0.057	3.0	1.2	0.7	0.19	5.7	6.7	21.3	6.9	22.7

4.3.8 Gold in Cyanide Solution From Calcines

Test No.	Distribn. % of Gold In. Head	Pyrite Conc.	Gold Assay Pyrite Conc.	Cyanide Tail	Tail Wt. % Orig.
N 4 & 5	15.2	82.8	101.0	23.9	7.47
N6 & 7	25.0	48.4	60.6	38.2	10.50
N8 & 9	33.4	79.6	52.6	15.5	19.47
Average	22.6	70.0	64.3	23.5	12.48

4.3.9 Gold in Cyanide Solution from Flotation Tailing

Test No.	Distribn. % of Gold In Head	Flotn. Tail	Gold Assay Flotn. Tail	Cyanide Tail	Tail. Wt. % Orig.
N4 & 5	0.8	54.5	1.0	0.45	84.6
N6 & 7	3.6	22.9	2.9	2.2	81.23
N8 & 9	3.5	31.3	5.8	4.0	67.77
Average	2.1	31.3	3.0	2.1	77.71

4.3.10 Total Potential Bullion (Free Gold Plus Cyanide Solutions)

Test No.	Gold Distribution % in Potential Bullion
N4 & 5	74.9
N6 & 7	44.0
N8 & 9	51.6
Average	62.9

4.3.11 Distribution of Gold Other Than the Free Gold

Neglecting the free gold the distribution of the remaining gold is as shown:

Test No.

Product	N4 & 5	N6 & 7	N8 & 9	Average
Head	100.0	100.0	100.0	100.0
Cyanide solutions	38.9	33.8	43.3	40.0

Test No.				
Product	N4 & 5	N6 & 7	N8 & 9	Average
Copper Concentrate	50.4	19.2	37.4	38.3
Solutions Plus Copper Conc.	89.3	53.0	80.7	78.3
Bulk cyanide tail	9.5	45.6	19.0	20.7
Stage 1. Flotation Froth	1.2	1.4	0.3	1.0
Head Assay Au	23.2	12.7	30.1	22.0

4.3.12 Direct Cyanidation of Pyrite Concentrate

Pyrite concentrate from tests N8 & 9 was subjected to direct cyanide treatment. The results compared with those obtained by treatment of calcine from the roasting of the pyrite concentrate as follows:

Cyanidation Feed	Weight % Orig.	Assays				Dist. % Au. in Solution		Tailing Assay Au
		Cu.	Au.	S.	As.	Orig. Head	Pyrite Conc.	
Pyrite Conc.	28.17	0.35	52.6	40.7	2.06	36.1	86.0	7.4
Calcine	19.47					33.4	79.6	15.5

Direct cyanidation of the pyrite concentrate gave better results than roasting and cyanidation of the calcine in the one test made.

5. Comments on Test Results

5.1 Gold Recovery

The total potential gold recovery on the averaged figures was

	Distribution % of Gold
Free gold	38.2
Gold in cyanide solutions	24.7
Gold in copper conc.	23.7
Total	86.6

Head assay Au grammes/tonne 35.6

The total gold assay of the averaged head sample, 35.6 grammes/tonne, as against the ore reserve figure of 23.9 grammes/tonne raises queries as to what recovery figure would apply to ore reserve grade figures. The possibilities are illustrated in the following sets of figures.

Averaged Figures for Tests 4 - 9

Product	Weight %	Au Assay	Au Units	Distr. % Au.
Head	100.00	35.6	3557.9	100.0

Product	Weight %	Au Assay	Au Units	Distr. % Au
Calcine cyanide tail	12.48	23.5	293.6	8.2
Flotation cyanide tail	77.88	2.1	163.1	4.6
Total cyanide tail	90.36	5.054	456.7	12.8
Stage 1 flotation froth	2.33	8.5	19.7	0.6
Total rejects	92.69	5.14	476.4	13.4
Potential bullion plus copper conc.			3081.5	86.6

If the head assay varies and the total cyanide tail and Stage 1 flotation froth remain the same the following figures apply

Effect of Variation in Head Value Other Figures Constant

Product	Weight %	Au Assay	Au Units	Distr. % Au
Head	100.0	23.9	2390.0	100.0
Total rejects	92.69	5.14	476.4	19.9
Potential Bullion plus copper concentrate			1913.6	80.1

Thus with the only variation being the gold assay of the head sample a decrease from 35.6 grammes/tonne assay of the head to 23.9 grammes/tonne causes a fall of 6.5% in the recovery of gold. This is purely a mathematical relationship.

If it were possible to maintain the gold recovery despite the fall in the head assay the following figures would be required:

Product	Weight %	Au Assay	Au Units	Distr. % Au
Head	100.00	23.9	2390.0	100.0
Total rejects	92.69	3.46	320.3	13.4
Stage 1 flotation froth	2.33	8.5	19.7	0.8
Total cyanide tail	90.36	3:33	300.6	12.6

To maintain gold recovery constant with the given fall in head value the assay of the total cyanide tailing would need to be reduced from 5.054 gramme/tonne to 3.33 grammes/tonne.

There is no evidence to suggest that such a fall in cyanide tailing assay would follow a reduction in gold head assay. The figures for the various tests were:

Test No.	Head Assay Au	Total Cyanide Tailing Assay
N4 & 5	56.4	2.4
N6 & 7	15.0	6.3
N8 & 9	35.3	6.6

Therefore the laboratory indicated recovery of gold in copper concentrate and cyanide solution from ore of ore reserve grade was 80 per cent.

Gold in solution is not fully recovered as there is loss in separating the solids from the solution and in smelting of the precipitate.

5.2 Recovery of Gold by Gravity - Amalgamation

Past records show that a high proportion of the gold was won by amalgamation with this proportion decreasing to 40 per cent of the gold recovered by 1914, see p 11 of the "Special Report of Mr. Arthur Llewellyn 17th January 1914". At that time the free gold was recovered by means of amalgamation tables after stamp batteries followed by amalgamation in grinding pans which were grinding a concentrating table concentrate for further treatment.

For the Launceston laboratory tests the recovery of the free gold varied from 14.7% to 58.9% with the average 38.2%.

These recoveries were obtained from samples of average assay of 35.6 grammes gold per tonne.

The method of laboratory treatment used, jigging the samples crushed to pass 1.0 m.m. with hand picking of the concentrate to recover the free gold, would not be as effective in the recovery of free fine gold as the old plant practice of amalgamation in grinding pans where the ground product was probably of the order of passing 0.3 m.m.

Any additional gold recovered by gravity concentration and amalgamation at sizes finer than treated by jigging in the laboratory tests would give a corresponding reduction in the gold content of the copper flotation concentrate.

In any future test work considerable attention should be given the problem of maximising the recovery of gold before flotation is given, or even by gravity treatment of the copper flotation concentrate which is relatively small in tonnage rate.

With present knowledge the recovery of gold from ore of ore reserve grade by gravity - amalgamation methods cannot be predicted. The indications from past records are that the figure will be about 40 per cent with lower figures likely if the sulphide contents increase.

5.3 Copper Concentrate

The averaged figures for the copper flotation stage were:

		Assays					Distribution %				
Product	Wt.%	Cu.	Au.	Ag.	S.	As.	Cu.	Au.	Ag.	S.	As.
Feed	97.67	0.64	22.3	4.2	7.5	0.67	79.7	61.2	95.7	97.0	98.5
Conc.	2.71	15.5	311.7	60.1	28.7	1.69	53.3	23.7	38.1	10.3	7.0
Tail	94.96	0.22	14.1	2.6	6.9	0.64	26.4	37.5	57.6	86.7	91.5

Note. The weight per cent and metal distribution are in terms of the original feed.

The copper distribution in the concentrate expressed in terms of copper flotation feed was 66.9%. This is a low figure for a chalcopyrite flotation and means of improving this would probably be found.

Any such increase in copper recovery would be beneficial in any subsequent cyanidation treatment as copper minerals are very undesirable in a cyanide circuit.

The desire to improve the copper recovery may clash with the desire to also reduce the arsenic content of the copper concentrate but arsenopyrite is normally one of the minerals slow in response to flotation.

The desirability of reducing the high gold content of the copper concentrate by improving the recovery of gold by gravity - amalgamation methods has been mentioned previously.

5.4 Pyrite Flotation

The averaged figures were:

	Assay						Distribution %				
Product	Wt.%	Cu.	Au.	Ag.	S.	As.	Cu.	Au.	Ag.	S.	As.
Feed	94.96	0.22	14.1	2.6	6.9	0.64	26.4	37.5	57.6	86.7	91.5
Conc.	17.08	0.96	64.3	9.1	35.2	2.66	20.7	30.8	36.3	79.8	68.8
Tail	77.88	0.057	3.05	1.2	0.7	0.19	5.7	6.7	21.3	6.9	22.7

As pyrite concentrate is a potential cyanide circuit feed the importance of improving the recovery of copper in the copper concentrate is apparent. A copper content of 0.96 per cent in a cyanide circuit feed would be a high figure.

The recovery of sulphur from the pyrite flotation feed was 92.0% but the tailing content of 0.7% sulphur is high compared with that obtained from flotation of sulphides from other Australian gold ores. If the sulphide concentrate is to be treated separately for the recovery of the contained gold it is important to obtain the maximum recovery of the sulphides and gold in the concentrate.

5.5 Stage 1 Flotation - Gangue Slime

The average figures were:

	Assays						Distribution %				
Product	Wt.%	Cu.	Au.	Ag.	S.	As.	Cu.	Au.	Ag.	S.	As.
Feed	100.0	0.79	22.0	4.3	7.5	0.66	100.0	61.8	100.0	100.0	100.0
Conc.	2.33	6.9	8.5	7.9	9.7	0.42	20.3	0.6	4.3	3.0	1.5
Tail	97.67	0.64	22.3	4.2	7.5	0.67	79.7	61.2	95.7	97.0	98.5

This flotation concentrate is a potential source of an increase in the total copper recovery.

5.6 Gold in Cyanide Solution

The averaged figures follow:

	Gold in Solution % of Gold In			Cyanide Tail	
Source	Head	Pyrite Conc.	Flotn. Tail	Assay	Distr. % * Au.
Calcine	22.6	73.3	-	23.5	8.2
Flotn. Tail	2.1	-	31.3	2.1	4.6
Total	24.7	-	-	5.1	12.8

* Distribution % of gold in the original feed.

The assays of the cyanide tailings were high.

The one test made by direct cyanidation of a pyrite concentrate is discussed at a later stage.

As Beaconsfield has some similarities with other Australian ores a comparison has been made with such ores, the practices used, and the results obtained. These are detailed in the following section.

6. Comparison of Beaconsfield, Kalgoorlie and Wattle Gully Ores

6.1 Beaconsfield Ore

See reports by Amdel MP 3634/74 and MP 3929/74.

6.1.1 Minerals Present

The minerals noted were:

quartz,
ankerite,
pyrite,
arsenopyrite,
chalcopyrite,
galena,
sphalerite, and
gold.

6.1.2 Gold Occurrence

MP 3634/74

Sample A602 - no gold was found by visual examination of the section.

Sample A603 - no gold was found by visual examination.

Sample A609 - two grains of metallic gold were detected in the polished section. Both as inclusions within pyrite, one grain measuring approximately 12 x 5 microns and the other 5 x 3 microns. The metallic gold is a pale yellow colour and probably contains some silver.

The gold values for the samples were quoted as:

A602 26.8 dwt = 41.7 grms/ton

A603 16.1 dwt = 25.0 grms/ton

A609 15.4 dwt = 23.9 grms/ton

MP 3929/74

Sample No.1 - no free gold was found and only very small inclusions were found in some pyrite.

Sample No. 2 - gold was found only as minute inclusions in pyrite.

6.1.3 Comment

Past records show the importance of the amount of free gold in the Beaconsfield ore but the Amdel reports show that the gold found as minute inclusions within pyrite could also be a feature as the amount of sulphides in the ore increases.

The samples examined by Amdel are stated to have been from sections deliberately selected because of their high sulphide content.

6.2 Kalgoorlie Ores

See Volume 3 p. 194 of the Eighth Commonwealth Congress 1965.

6.2.1 Minerals Present

These were given as:

quartz,
ankerite,
sericite,
pyrite, and
minor amounts of various other sulphides, tellurides and gold.

6.2.2. Gold Occurrence

"From a treatment point of view gold occurs in four forms. These are:

1. coarse gold, which is concentrated by gravity means and amalgamated,

2. fine free gold and fine gold associated with gangue minerals which is recovered by cyanidation,

3. gold tellurides from which the gold is recovered either by prolonged cyanidation under special conditions, or by cyanidation after roasting, and

4. gold intimately associated with pyrite which is recovered by cyanidation after roasting."

6.3 Wattle Gully Ore

See A.I.M.M. Proceedings No. 191, 1959 "Investigations for a New Flow Sheet and Plant at Wattle Gully Gold Mines N.L." by Clarke, Jackson and Woodcock, also p. 202 of the volume 3 of the 1965 Congress.

6.3.1 Minerals Present

Essentially quartz

Slate 10 - 15%

Sulphides

0.3 to 0.6% sulphur
0.2 to 0.4% arsenic

with lesser amounts of sphalerite and galena and minor amounts of chalcopyrite and boulangerite.

6.3.2 Gold Occurrence

Mostly free and coarse.

Specimens from 7, 8 and 9 levels show a tendency to associate with galena, although some gold is associated with arsenopyrite, pyrite and sphalerite. The gold in pyrite, in contrast to free gold, occurs as particles only a few microns across.

7. Comparison of the Ores

The major minerals in Kalgoorlie and Beaconsfield ores are similar but vary of course in their proportions.

The Wattle Gully ore consisted of quartz and slate with no carbonates mentioned.

Other figures compare as shown.

	Kalgoorlie	Beaconsfield	Wattle Gully
Sulphur %	1.76-2.9	7.5	0.3 - 0.6
Copper %	0.03	0.79	small
Arsenic %	0.013	0.66	0.2 - 0.4

(The Kalgoorlie figures for copper and arsenic are those given by Penrose for G.M.K. ore in "Treatment Methods and Plant of Gold Mines of Kalgoorlie" A.I.M.M. Proceedings.)

The proportions of gold recovered by gravity - amalgamation processes were:

	Head Assay Au grms/tonne	Recovery Au %
Kalgoorlie	7.7 - 9.0	nil - 28.9
Beaconsfield		40 (1914 figure)
Wattle Gully	9.95	75.7

(The Beaconsfield figure is that given by Llewellyn).

The Beaconsfield ore from the drill cores tested thus contains much greater amounts of sulphur, copper and arsenic than the other ores.

In 1914 the figure for recovery by amalgamation was estimated at 40% at Beaconsfield which is between the other figures. It was stated in the case of Beaconsfield when considering high treatment costs, "the reason for this is the intractable character of the ore, which increases with the growing depth of the mine."

The ore from each of the three properties contained gold which was very intimately associated with pyrite.

8. Treatment Practice

8.1 Beaconsfield - Launceston tests.

After removal of coarse free gold by jigging the ore was ground to approximately 75% passing 75 microns followed by:

Stage 1 flotation to remove fast floating gangue slime,

Stage 2 flotation to recover a copper-gold concentrate,

Stage 3 flotation to recover a pyrite-gold concentrate,

Roasting of Stage 3 concentrate,

Cyanidation of the calcine from the roasting stage, and

Cyanidation of the Stage 3 flotation tailing.

8.2 Kalgoorlie

8.2.1 Gravity Concentration by Straking

G.M.K.

Recovered 5 - 7% of the total gold.

Great Boulder.

Did not use.

Lake View and Star.

Recovered 11.3% of the total gold in the grinding circuit plus 17.6% from the calcines.

North Kalgoorlie.

Recovered 12.7% by straking the concentrate from a plane table treating the rod mill discharge plus the cleaner flotation tailing.

8.2.2 Grinding

The overall grind varied from 70 - 80% passing 200 mesh.

8.2.3 Flotation and Cyanidation

G.M.K.

Flotation was given before any cyanide treatment.

High grade concentrate (telluride) was cyanided, the cyanide tailing was roasted and the calcines again cyanided.

Pyrite flotation concentrate was cyanided, and then split into three size fractions. The coarsest fraction was despatched to Perth for roasting followed by cyanidation. The minus 10 micron fraction was discarded as tailing while the intermediate size was roasted and cyanided with the tailing from the cyanidation of the high grade concentrate.

Great Boulder

The grinding section product was cyanided before flotation.

Tailing from the cyanidation step was gassed with sulphur dioxide and treated by flotation.

Flotation concentrate was roasted and the calcine cyanided.

/.

Flotation tailing was discarded.

Lake View and Star

Flotation preceeded any cyanide treatment. Flotation concentrate was roasted, straked and cyanided. The tailing from this treatment was retreated by cyanidation with the flotation tailing.

The flotation tailing plus calcine cyanidation tail were cyanided.

North Kalgoorlie

Flotation preceeded any cyanide treatment.

Flotation concentrate was given a two stage treatment by cyanidation followed by roasting of the tailing from this treatment. The calcine was also given a two stage treatment by cyanide. The tailing from the second of these stages was rejected.

The flotation tailing was given a cyanidation treatment before rejection.

8.2.4 General

It is seen that the pyrite received intensive treatment in that it received at least two stages of treatment by cyanide. In some cases this was given by treatment in the sulphide state followed by cyanidation of the roasted pyrite.

In other cases the calcine received two stages of cyanidation, the first in the calcine section and the second with the flotation tailing.

Flotation of the pyrite was practiced in all cases.

Roasting of the flotation concentrate followed by cyanidation of the calcine was always given.

Flotation tailing was treated by cyanidation in two cases but not in the other two.

8.3 Wattle Gully

8.3.1 Grinding and Jigging

Minus ½ in. feed was ground to 65% minus 200 mesh.

A ball mill cyclone circuit was used with a jig treating the ball mill discharge.

8.3.2 Flotation

Cyclone overflow was treated by flotation to give a concentrate and a discard tailing.

8.3.3 Cyanidation

Flotation concentrate was thickened, aerated and filtered with the filter cake going to batch cyanidation.

Each batch was cyanided in 4 cycles each composed of a 24 hour agitation period followed by filtration.

8.4 Flotation of the Pyrite

This is a step common to the tests on Beaconsfield ore and to practice at Kalgoorlie and Wattle Gully.

Successful treatment of the flotation feed (in the case of Beaconsfield the copper flotation tailing) is vital to the overall gold recovery. The results of this stage of treatment are shown in the following table. In this and other tabulations which follow the Kalgoorlie and Wattle Gully figures were derived from those given in Volume 3 of the Eighth Commonwealth Congress.

Comparison of Flotation Results - Kalgoorlie and Beaconsfield and Wattle Gully.

Property		GMK	Great Boulder	LV & S	North Kal.	Beacons-field average	Wattle Gully
Flotation feed	weight %	100.0	100.0	100.0	100.0	100.0	100.0
	assay Au	8.7	5.4	7.1	7.0	14.1	2.42
	assay S %	2.5	1.8	2.2	2.9	6.9	-
Flotation conc.	weight %	8.0	4.6	5.8	6.3	18.0	2.25
	assay Au	99.5	99.8	110.8	99.5	64.3	98.2
	assay S %	30.0	37.0	29.9	42.0	35.2	-
	distrib. % Au	90.9	85.4	90.9	88.7	82.2	91.3
	S	93.7	85.4	92.5	92.0	92.1	-
Flotation tail	weight %	92.0	95.4	93.2	93.7	82.0	97.75
	assay Au	0.86	0.81	0.69	0.84	3.05	0.21
	assay S %	0.17	0.3	0.15	0.25	0.66	-
	distrib. % Au	9.1	14.6	9.1	11.3	17.8	8.7
	distrib. % S	6.3	14.6	7.5	8.0	7.9	-

Notes.

1. Gold assays are expressed as grammes/tonne.

2. Kalgoorlie and Wattle Gully weight % figures have been calculated from the gold assays.

3. All figures relate to the pyrite flotation stage only.

Features of interest in the comparison are:

a. the Beaconsfield flotation feed was much higher in gold and sulphur assay than in the other cases,

b. owing to the higher sulphur assay of the feed the production of flotation concentrate was also higher in the case of Beaconsfield,

c. The gold and sulphur assays of Beaconsfield flotation tailing were considerably higher than for Kalgoorlie and Wattle Gully flotation tailings.

The figures show that the averaged results of the tests on Beaconsfield ore were poorer than the Kalgoorlie, Wattle Gully, results. In other words the indications are that the Beaconsfield ore tested was more difficult to treat by flotation than the other ores.

8.5 Cyanidation

The results of cyanidation of flotation products are given in the tabulation following

Cyanidation Result

Property	Cyanidation Feed	Assay Au Grammes/tonne		
		Feed	Tail	Bulk Reject
GMK	Pyrite Concentrate	102.6	24.5	
GMK	Calcine Kalgoorlie	30.6	3.8	
GMK	Calcine Perth	32.1	2.3	0.83
North Kalgoorlie	Te concentrate	183.7	45.9	
North Kalgoorlie	Pyrite concentrate	99.5	30.6	
North Kalgoorlie	Calcine	29.1	4.3	0.64
Great Boulder	Calcine	124.0	6.0	0.77
Lake V & S	Calcine	110.2	3.7	0.61
Beaconsfield Av.	Calcine	88.0	23.5	
Beaconsfield Av.	Flotation tail	3.05	2.1	5.1
Wattle Gully	Pyrite concentrate	98.2	11.5	0.47

Note.

1. The Kalgoorlie bulk final reject figure would be a little higher than shown above as no account has been taken of the loss of weight during roasting.

2. The bulk reject consists of both cyanidation tailing and flotation tailing.

3. Later figures from Wattle Gully show considerably improved results.

The averaged Beaconsfield test results gave a much higher gold assay in the bulk reject than that obtained at Kalgoorlie and Wattle Gully.

It must be pointed out that Kalgoorlie practice is to carefully control the temperature of the roast to ensure a calcine in which the gold is accessible to the cyanide solution. The temperature is kept at 600°C or less in the early stages of roasting. In the Launceston tests the temperature of 900° is mentioned but whether this temperature applied in the early stages of roasting is not mentioned.

Whether a difference in roasting conditions is the cause of the difference in calcine cyanidation tailing assays at Kalgoorlie and in the Launceston tests is unknown.

8.6 Summary of Comparison with Kalgoorlie and Wattle Gully Ores

The comparisons show that:

a. the major constituents of the Beaconsfield samples and Kalgoorlie ores are of similar minerals in varying proportions,

b. Wattle Gully ore consisted largely of quartz with slate, carbonates were not mentioned,

c. of the minor constituents the content of pyrite, chalcopyrite and arsenical pyrite are much higher in the Beaconsfield samples than in the other ores,

d. part of the gold in the three ores was recoverable by gravity - amalgamation methods,

e. the three ores contained part of their gold content as minute inclusions within pyrite,

f. when given a similar degree of grinding the pyrite flotation with Beaconsfield ore did not result in assays for gold and sulphur in the flotation tailing as low as those for the other ores,

g. cyanidation of calcine from the roasting of the Beaconsfield pyrite concentrate and of the pyrite flotation tailing gave cyanide and bulk reject tailing of much higher gold assay than the corresponding rejects from the other ores, and

h. with the existing information it cannot be expected that the gold recovery from Beaconsfield ore will approach the 90-95% range obtained in Kalgoorlie and Wattle Gully.

8.7 Cyanidation of Concentrate

Direct cyanidation of concentrate was practiced at Wattle Gully and in one test on Beaconsfield ore.

At one period at Wattle Gully a sulphide concentrate produced by strakes and tables and assaying about:

55 grms/tonne Au
7 per cent As
10 per cent S

was treated by amalgamation, using litharge and cyanide solution, in grinding pans followed by batch cyanidation using decantation methods (3 stages of washing).

An extraction of 92% of the gold was obtained with the tailing assaying 4.6 grms Au/tonne.

No particular difficulties were encountered.

When the plant was reconstructed and flotation used for the concentration of the sulphides cyanidation of the concentrates presented some difficulties. The treatment previously outlined was then adopted.

In the case of the pyrite concentrate from tests N8 and 9 on Beaconsfield ore the concentrate contained

52.6	grms/tonne	Au
0.35	per cent	Cu
2.06	per cent	As
40.7	per cent	S

Direct cyanidation of this concentrate compared with that of the Wattle Gully flotation concentrate thus

	Beaconsfield	Wattle Gully
Flotation conc. Au grms/tonne	52.6	98.2
Cyanide tailing Au grms/tonne	7.4	11.5
Gold in solution %	86.0	88.3

Unfortunately tests N8 and 9 were the only tests which gave sufficient concentrate to enable a direct cyanidation test to be made and it is unknown how the pyrite concentrates from the other tests would react to direct cyanidation.

As the cyanidation tailing from tests N8 and 9 using direct cyanidation were lower in gold assay than the calcine cyanidation tailing produced from the same concentrate, it may be asked to what degree gold recovery would be affected.

The following figures illustrate this.

Tests N8 and 9

Product	Weight %	Au Assay	Distr. % Au (orig. Head)
Calcine	19.47	76.0	42.0
Calcine Cy tail	19.47	15.5	8.6
Recovered by Cy.			33.4
Pyrite conc.	28.17	52.6	42.0
Pyrite Cy. tail	28.17	7.4	5.9
Recovered by Cy.			36.1

The gain in overall recovery was thus 2.7%

Possible Treatment Methods

As the only complete figures for the series of tests at Launceston employed roasting of the pyrite concentrate and cyanidation of the calcine much of the previous discussion has centred around the recovery of gold by cyanidation of the calcines.

Because of the arsenic content of Beaconsfield ore a process using roasting would be very undesirable from the environmental point of view as well as because of the operating complications involved.

Reference to the literature has indicated that clean arsenopyrite is inert in cyanide solution. Therefore roasting is not needed because of the presence of such arsenic unless the gold is so intimately associated with the arsenopyrite that economical grinding will not expose the gold to the cyanide solution. The same remarks apply to pyrite.

It has also been shown at Wattle Gully that concentrate containing a considerable arsenic content can be successfully treated by cyanidation.

Further the one test made by direct cyanidation of a Beaconsfield pyrite concentrate showed that this sample could be successfully cyanided. This has not been demonstrated for pyrite concentrates of other grades but a reasonable assumption is that a method suitable for the direct cyanidation of Beaconsfield copper flotation tailing or pyrite concentrate can be developed.

A pre-requisite to such a process is to reduce the amount of copper mineral escaping to the copper flotation tailing.

Having assured that this can be done the next query would be should cyanide treatment be given to:

a. the copper flotation tailing, or

b. a pyrite flotation concentrate, with or without

c. the pyrite flotation tailing.

The gold associated with pyrite often needs intensive treatment for successful solution by cyanide and most centres appear to follow the practice of separating the sulphides from the non-sulphides for such treatment. The advantages are:

a. the intensive treatment is given to a comparatively small tonnage of pulp, with smaller plant requirements,

b. the gold not extracted by cyanide is contained in a small tonnage of solids of higher gold assay than the tailing from treatment of the total pulp, this could be an advantage for possible future treatment and

c. the amount of cyanide tailing for storage is reduced.

The last item does not apply if the flotation tailing is also cyanided.

A paragraph from the paper by Clarke, Jackson and Woodcock, which concerned a plant of about the same tonnage capacity as presently proposed for Beaconsfield is of interest. They stated:

"Secondly, in a number of cases there is the possibility of treating certain products either by flotation (with cyanidation of flotation concentrate) or by direct cyanidation. In each case, flotation was selected for more detailed examination, although direct cyanidation extracted up to 0.2 dwt. per ton more gold. There were two main reasons for this decision. First to cyanide all the ore requires a higher capital investment in agitators, washing thickener, filters and precipitation equipment, than in flotation cells plus sufficient cyanidation equipment to handle 2 per cent of the ore, particularly when some concentrate cyanidation equipment is already at the mine. Secondly, Wattle Gully is in a grazing district on the watershed of the Loddon River, and tailing disposal is a problem. It seems certain that the company would not be permitted to stack large tonnages of residues containing cyanide, and would have to first destroy the cyanide. This would involve additional capital costs and also substantially increased operating costs for labour and reagents. These operating costs along were estimated to be in excess of the maximum of 3 shillings per ton saved by additional extraction in cyanidation. Hence flotation was preferred."

Much remains to be done to establish the most promising of the treatment systems for Beaconsfield ore when representative samples become available. The main features to be investigated would be:

a. methods to increase the amount of gold recovered by gravity - amalgamation methods,

b. means of improving the recovery of copper in the copper concentrate,

c. the improvement of the sulphur and gold recoveries in the pyrite concentrate produced by flotation treatment of the copper flotation tailing,

d. cyanidation of the pyrite flotation tailing,

f. cyanidation of the copper flotation tailing,

g. the optimum degree of grinding, and the power needed for such grinding, for all systems, and

h. environmental effects of the storage of cyanide tailings.

With the knowledge thus developed a sound choice of the treatment method would be possible.

Yours faithfully,

(sgd.) K. Parsons.

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75-1108
VOL.2

FIGURE 3/3

ALLSTATE EXPLORATIONS N.L.

BEACONSFIELD GOLD PROJECT

MILL FLOW SHEET – INCLUDING CYANIDE CIRCUIT FOR PYRITE TREATMENT

PREPARED BY K. PARSONS – CONSULTANT

DATE 4-12-74

NOTE
THE 3 FILTERS COULD BE SERVED BY THE ONE VACUUM PUMP

511315


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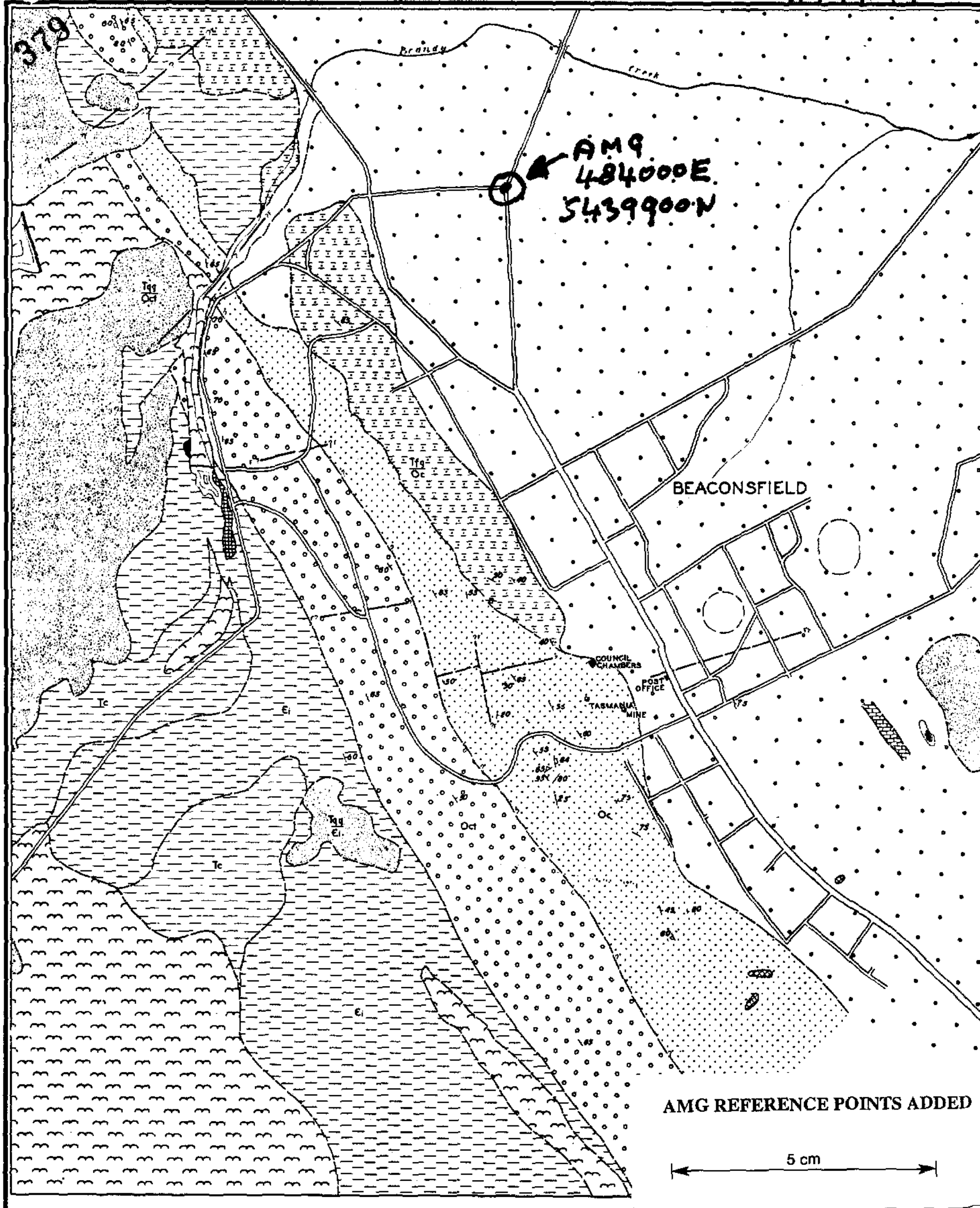
PART 'B' REF. 4/1
ALLSTATE EXPLORATIONS N.L.
BEACONSFIELD
MAP SHOWING PRINCIPAL MINES WORKINGS AND STRATA IN WHICH THEY ARE SITUATED
PLAN OF STRATA TAKEN ON PLANE OF TASMANIA MINE MAIN ADIT
5 cm
TO YORK TOWN ROAD
PROBABLE FAULT
WELD
ALLUVIAL GRAVELS CLAYS AND SANDS OF THE DEEP LEAD
SANDSTONE
SLATE
LIGHT SANDSTONES
DARK SANDSTONE
WHITISH SANDSTONE
BLACK SANDSTONE
DARK SANDSTONE
WHITISH SANDSTONE
BLACK SANDSTONE
LIGHT AND DARK GRITS AND SANDSTONES
GRITS
DARK BLUISH SLATE
FAULT
PROBABLE FAULT
100 ac granted to Robert De Little
ROAD
SANDSTONES
GRITS & SANDSTONES
William Bowen
J B Curran
H Masters & A Wilson
511346



AMG REFERENCE POINTS ADDED

5 cm

GEOLOGICAL MAP

BEACONSFIELD AREA

500 0 500 1000 1500 FT

LEGEND

GEOLOGICAL

QUATERNARY
- BATTERY SANDS
- RECENT ALLUVIUM

TERTIARY
- SILICEOUS CONGLOMERATE
- SILICEOUS CLAY
- Tqg QUARTZ GRAVEL
- Tfg FERRUGINOUS GRAVEL
- RESIDUAL SOILS
- Tc CLAY

SILURIAN?
- LIMESTONE RUBBLE

ORDOVICIAN
- Oct CABBAGE TREE HILL CONGLOMERATE
- Oc CAROLINE CREEK SANDSTONE

CAMBRIAN
- Ci ILFRACOMBE SLATE

STRUCTURAL
- FAULT
- STRIKE & DIP OF BEDDING
- STRIKE DIP OF FOLIATION
- STRIKE & DIP OF JOINTING
- VERTICAL JOINTING

TOPOGRAPHICAL
- ROAD
- DAM
- SWAMP

A. J. NOLDART
GEOLOGIST 1963

TRUE MAG

Part 'B' Ref 4/2

DEPT OF MINES 2212-30 Allstate Explorations N.L.

COMMONWEALTH OF AUSTRALIA

DEPARTMENT OF NATIONAL DEVELOPMENT

BUREAU OF MINERAL RESOURCES, GEOLOGY AND GEOPHYSICS

Part 'B'
Reference
4/3

RECORD No. 1965/139

BEACONSFIELD DEEP LEAD GRAVITY SURVEY,

TASMANIA 1964

by

A.W. HOWLAND-ROSE

511349

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CONTENTS

ILLUSTRATIONS

Note. This Record supersedes Progress Report No. 1964/16.

1965/139

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SUMMARY

A gravity survey was made over the gold-bearing deep lead at Beaconsfield, north-east Tasmania, in order to locate the lead and determine its vertical section.

All traverses showed gravity minima, which are approximately coincident with the deepest part of the lead or slightly displaced to the south-west. The shape of the anomaly indicates that the eastern bank of the lead is steeper than the western bank.

It appears that deep weathering, which may be associated with the continuation of the auriferous lead, is present north-west and south-east of the town.

1. INTRODUCTION

The past prosperity of the Beaconsfield gold field, in the north-east of Tasmania, was based on the Tasmania reef, which was discovered in 1877. Until work ceased in 1914, about 854,600 oz of gold were won from 1,067,556 tons of ore. In addition to the reef gold, alluvial gold was also worked down to a false bottom at 112 feet, but it appears that the quantity of alluvial gold recovered was of relatively minor importance. At the request of the Tasmanian Mines Department, the Bureau of Mineral Resources made a gravity survey, between 19th March and 10th April 1964, over the gold-bearing deep lead at Beaconsfield in an attempt to delineate the course of the lead.

The survey was made by the author and one field assistant, and the traverses were pegged and levelled by D.J. Sheaves and two chainmen from the Department of the Interior, Canberra.

2. GEOLOGY

The general geology of the area and the positions of the auriferous quartz reefs and the deep lead are shown in Plate 1.

The main feature of the area is Cabbage Tree Hill, along the south-western side of the town of Beaconsfield. It is composed of rocks of probably Ordovician age; the crest consists of coarse-grained grits or fine quartz conglomerate alternating with beds of hard metamorphic sandstone, known collectively as the Cabbage Tree Hill Conglomerate (Noldart, 1964), which is conformably overlain on the north-eastern flank of the hill by members of the Caroline Creek Sandstone.

To the north-east of Cabbage Tree Hill are the Tertiary deposits of the deep lead followed by Tertiary residual soils. Beneath these Tertiary deposits are rocks of Lower Silurian or Ordovician age, consisting of beds of Gordon Limestone, beneath the deep lead, and layers of slate, sandstone and limestone, which overlay the Gordon Limestone to the north-east (Montgomery, 1891). The exact positions and extents of these rocks are not known.

The Ordovician and Ordo-Silurian rocks form a conformable series, the Cabbage Tree Hill Series, striking approximately north-west and dipping to the north-east at between 45° and 65° (Hughes, 1953).

The auriferous quartz reefs at Beaconsfield are thought to consist of two distinct reefs: the Tasmania reef, which strikes at about N50°E, dips south-east, and lies within the Caroline Creek Sandstone; and the Moonlight-cum-Wonder reef, which strikes about N55°W, dips south-west, and lies within the Cabbage Tree Hill Conglomerate. The richer of the two reefs was the Tasmania reef, which caries from a few inches to about twenty-five feet in width, is about 1300 feet long, and was mined to a depth of 1500 feet. Both reefs were rich in the upper portions, but especially the Tasmania reef, where, from the 400-ft level to the surface, values averaged about 25 dwt per ton.

The deep lead is probably in an old river valley that was formed in Tertiary times when the area was at a higher elevation (relative to sea level) than it is today. From borings and old workings, unweathered bedrock is known to be at a depth greater than 270 feet below the present sea level. Where the positions and depths of bedrock are known, they have been marked in Plate 1. However, as many of the workings had long been abandoned even in 1891 (Mongomery, 1891), the actual sites are only inferred.

The channel was eroded through auriferous Ordo-Silurian rocks, and a large portion of the Tasmania reef has been eroded and sluiced into the old channel. As the Tasmania reef was being eroded, so were other reefs on Cabbage Tree Hill, and this gold also found its way into the old river. As more recent gravels derived from Cabbage Tree Hill have been found to be auriferous, it is thought that the lead must contain rich gravels as more sorting and concentration would have taken place (Montgomery, 1891).

The lowest alluvial workings were 112 feet below ground level, on a false bottom of black ligneous clay, which contained plant remains of Upper Tertiary age. It appears that no deeper workings in the deep lead were attempted, although the Ophir Company put down two drill holes, 286 feet and 375 feet to bedrock, and Montgomery (1891), commenting on evidence from the drill holes, concluded: "If these results are reliable the richness of the lead would be phenomenal". Montgomery also states that the false bottom contained "a good deal of gold", and the "high reef" (the bedrock forming the sloping sides of the old channel) were "fairly payable".

Unfortunately, no records are available now regarding later work on the alluvial deposits, and it seems likely that, as there were rumours about possible salting of the Ophir bores (Montgomery, 1891), little work was done after this time. Montgomery, however, concludes that "there are good a priori reasons for thinking that the lead ought to be richly auriferous".

3. METHOD AND EQUIPMENT

The gravity method detects variations in the Earth's gravitational field due to variations in density of the rocks at depth. Thus if the densities of the rocks are known, then their relative positions may be indicated provided the density contrasts are sufficiently great.

Because the unconsolidated material filling the lead channel is less dense than the unweathered bedrock, the position of the lead will be shown up by a gravity 'low', the extent and magnitude of which should indicate approximately the width and depth of the lead.

A Sharpe gravity meter, No. 145, having a sensitivity of 0.10637 milligals per scale division, was used in the field work.

4. REDUCTION

Gravity readings were corrected for drift in the usual manner, by taking readings at a base station at intervals of not more than one hour, and using the results to establish a drift curve.

The elevation correction corrects for natural decrease in gravity with increase in elevation above sea level. The correction is proportional to height above a given reference level (in this case mean sea level) and the density of the intervening material; the density chosen was $2.2 g/cm^3$.

The latitude correction takes into account the increase in gravity from the equator to the pole. The latitude correction was taken from the "International Tables" (Nettleton, 1940) and amounted to 1.295 milligals per mile. The base station used for latitude correction was zero grid position on Traverse D, where the latitude is approximately 41°12'.

5. INTERPRETATION

Bouguer anomaly contours are shown in Plate 2 and Bouguer anomaly profiles in Plate 3.

The main feature of the results is a continuous gravity 'low', corresponding roughly in position with the Gordon Limestone. As this is the densest rock occurring in the area, the gravity 'low' must be due to a buried valley or zone of deep weathering. In general, the gravity minima correspond reasonably closely with the position of the lead as mined.

The Bouguer anomaly profiles for Traverses D and L, C, G, and H were compared with theoretical profiles (Plates 4-7), which were computed from assumed geological cross-sections. The sources of the geological information used in these assumed cross-sections were Montgomery (1891), Hughes (1953), and Noldart (1964).

The instrument used to compute the gravity profiles due to the assumed geological cross-sections was a vertical section integrator (Olbrich, in preparation). An average density was assumed for each group of strata. The densities used are as listed in Table 1, and are based on densities determined from samples collected in the field.

TABLE 1

Densities of rocks in the Beaconsfield area

Rock type	Density in g/cm^3
Cabbage Tree Hill Conglomerate	2.35
Caroline Creek Sandstone	2.25
Gordon Limestone	2.70
Shales	2.35
Weathered shales	1.95
Deep lead gravels, sands, clays, etc.	1.95

The dip of the strata was taken as 55°E and the width of the limestone was taken to be 300 to 400 feet.

The constructed gravity effects due to assumed geological cross-sections were compared with the observed gravity profile for each traverse and the geological configuration progressively adjusted until a reasonable conformity was obtained.

Traverses D and L

The first assumed geological section (Plate 4) was based on Montgomery's section across Cabbage Tree Hill (Montgomery, 1891). The bedrock at the deepest part of the lead was assumed to be approximately 370 feet below the surface. Of necessity, the section was simplified, but it can be seen that the eastern portion gives good agreement with the observed gravity profile.

Section 2 was constructed by changing the configuration of the western slope of the lead and also the width and position of the limestone. This gives better agreement, but the negative portion at 450W is still not explained completely; it may be due to a fault that is undetectable on the surface.

Traverse C

The maximum depth to bedrock assumed in this case was 400 ft. Progressive adjustments were made to the cross-section (Plate 5), and it will be seen that Section 3 gives a better conformity with the observed gravity profile than Sections 1 and 2.

Traverse G

On this traverse, there are no outcrops in the vicinity of the lead; therefore the positioning of the lead is rather more uncertain than for Traverses C and D. Two sections (Plate 6) were drawn up with maximum depths to bedrock of 420 and 490 feet. The anomalies indicate that the correct geological structure is probably somewhere between the two assumed sections. The negative anomaly between 1200W and 1400W could be explained by a depression in the bedrock as shown in Section 2.

It appears that the lead at this point has a more steeply dipping eastern bank, and is probably deeper than on Traverses C and D.

Traverse H

The positioning of the geological strata for this cross-section (Plate 7) is more uncertain. The lead channel probably occurs at about 1600W or 1800W. Caroline Creek Sandstones crop out at 2100W and conglomerate of density 2.6 g/cm^3 crops out at 2600W. It seems likely from the observed profile that there may be a slight depression in the bedrock between these two outcrops.

Section 2 gives reasonable agreement with the observed gravity, and it appears that the bedrock in the vicinity of this traverse is not so deep (about 250 feet), and the banks of the channel not so steep, as further south.

Traverses A, J, and K

These traverses were placed south-east of the town in an area where it was thought the Gordon Limestone would not be far below the surface, and where there was no previous evidence of the presence of the lead. However, the observed profiles over these traverses are similar to those in the town area. As the limestone has not actually been seen in outcrop on Traverses A and J, it seems likely that deep weathering has occurred in the limestone. This may be associated with an auriferous lead.

6. CONCLUSIONS

The gravity 'low' shown in the Bouguer anomaly contour map (Plate 2) agrees with the position of the deep lead as indicated by Montgomery (1891) and shown in Plate 1. Montgomery shows the lead to have its greatest width near Traverse F, whereas the gravity results indicate the greatest width of the channel to be at Traverse E.

From the comparison of observed and theoretical gravity profiles (Plates 4 to 7) and from drilling information on the Ophir bores on Traverse F (Plate 1), it appears that the deepest part of the bedrock is not quite coincident with the lowest value on the gravity profile, but is displaced slightly to the north-east, and that the lead has a steeper slope on its eastern bank than on its western bank.

389

In addition it appears probable that the lead continues to the north-north-west and south-south-east of the town as indicated in Plate 2.

Drilling to the lead gutter down-stream from the Tasmania reef (i.e. to the north-west) is recommended. Montgomery in 1891 thought further investigations for alluvial gold would prove successful, but there is no evidence of any further investigations having been made.

7. REFERENCES

HUGHES, T.D. 1953 The Beaconsfield and Lefroy Goldfields. In GEOLOGY OF AUSTRALIAN ORE DEPOSITS. Aust. Inst. Min. Metall.

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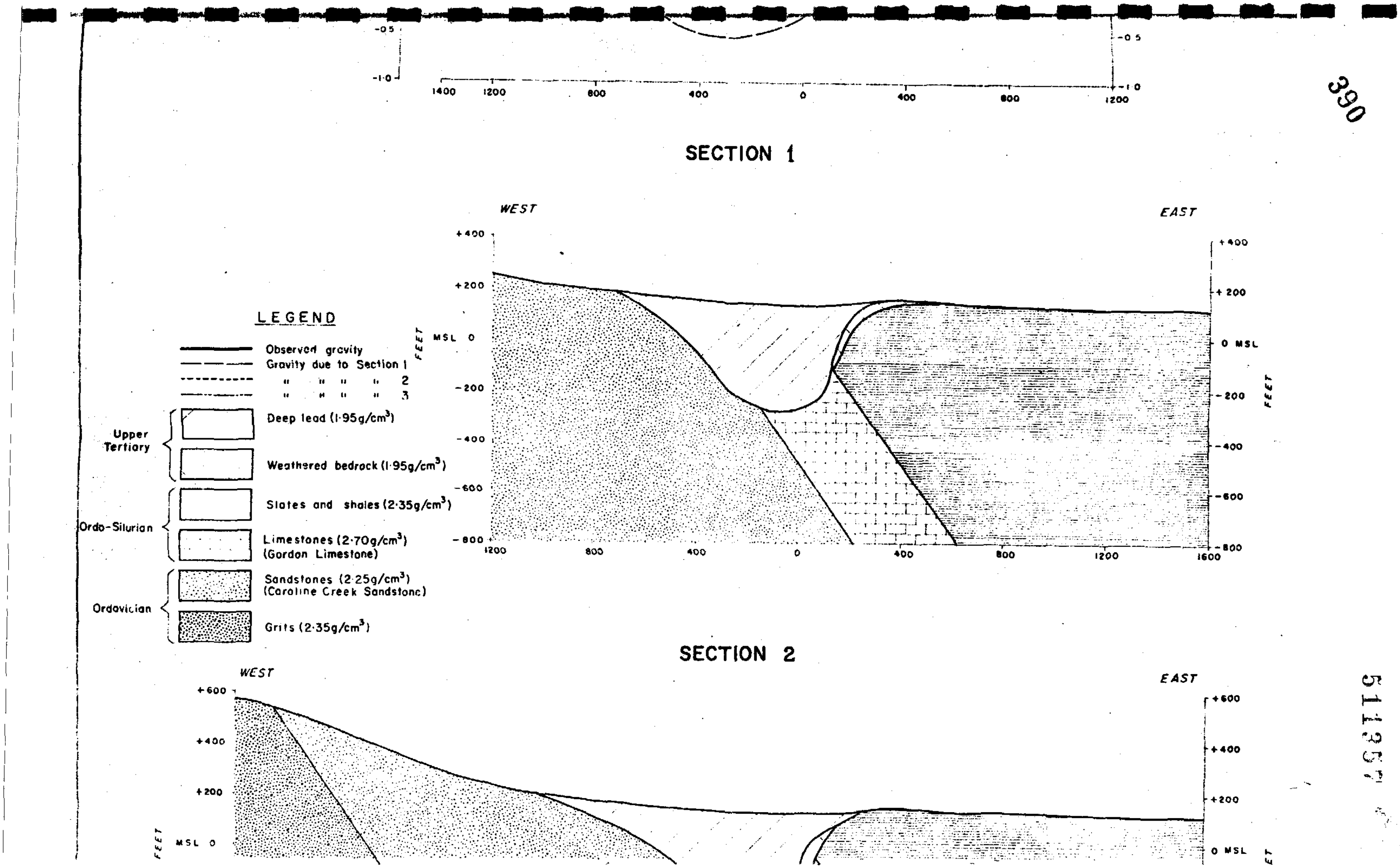
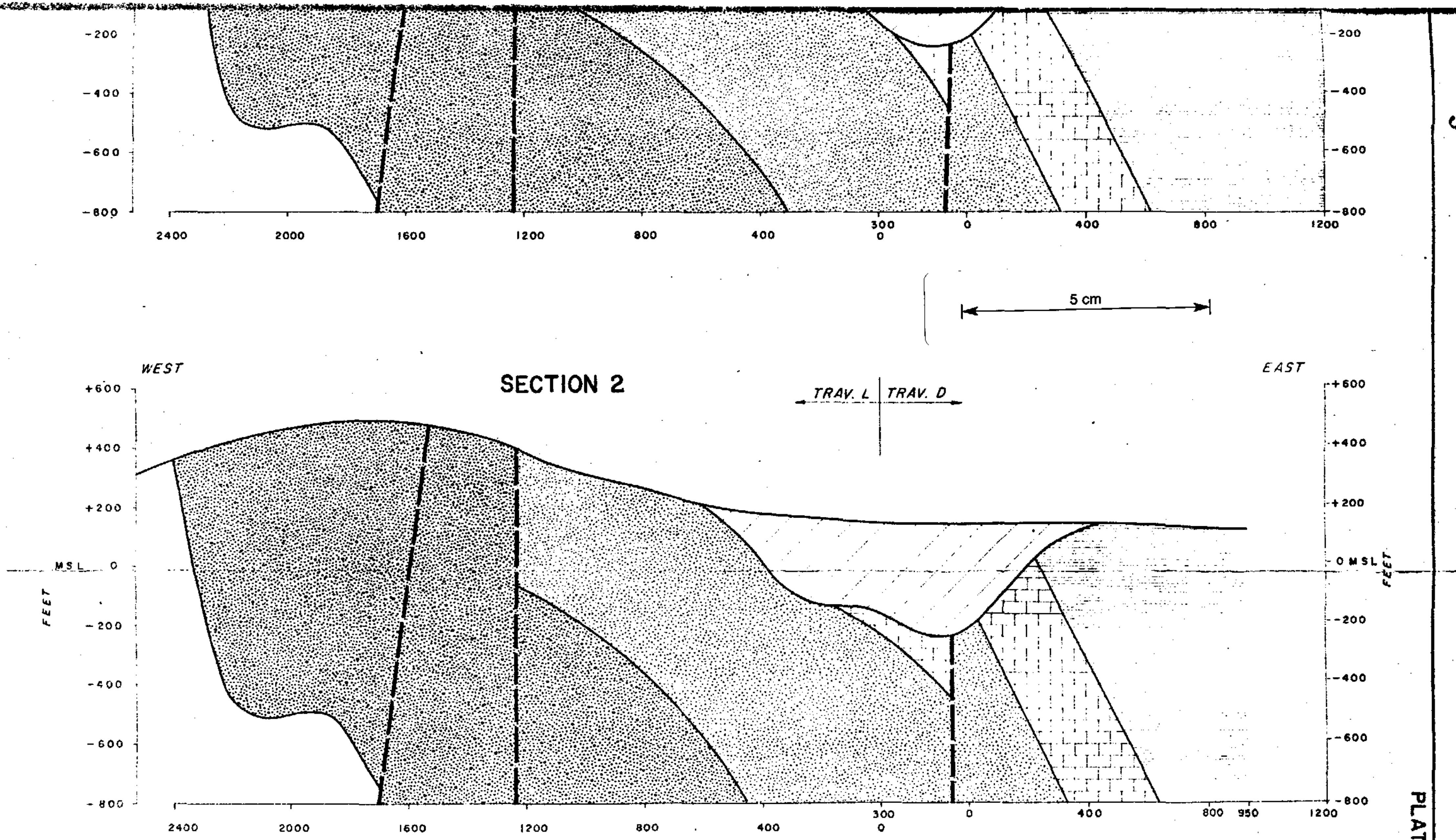
NOLDART, A.J. 1964 Notes on auriferous deposits, Beaconsfield Goldfield. Dep. Min. Tas. Tech. Rep. No. 8, 1963.

OLBRICH, W. A vertical section integrator for the computation of gravity anomalies. Bur. Min. Resour. Aust. Rec. (in preparation).


SECTION 1
WEST
EAST
FEET
MSL 0
0 MSL
LEGEND
Observed gravity
Gravity due to Section 1
" " " " 2
" " " " 3
Upper Tertiary
Deep lead (1·95g/cm³)
Weathered bedrock (1·95g/cm³)
Ordo-Silurian
Slates and shales (2·35g/cm³)
Limestones (2·70g/cm³) (Gordon Limestone)
Ordovician
Sandstones (2·25g/cm³) (Caroline Creek Sandstone)
Grits (2·35g/cm³)
SECTION 2
WEST
EAST



FEET

400 0 400 800 1200

Vertical scales as shown

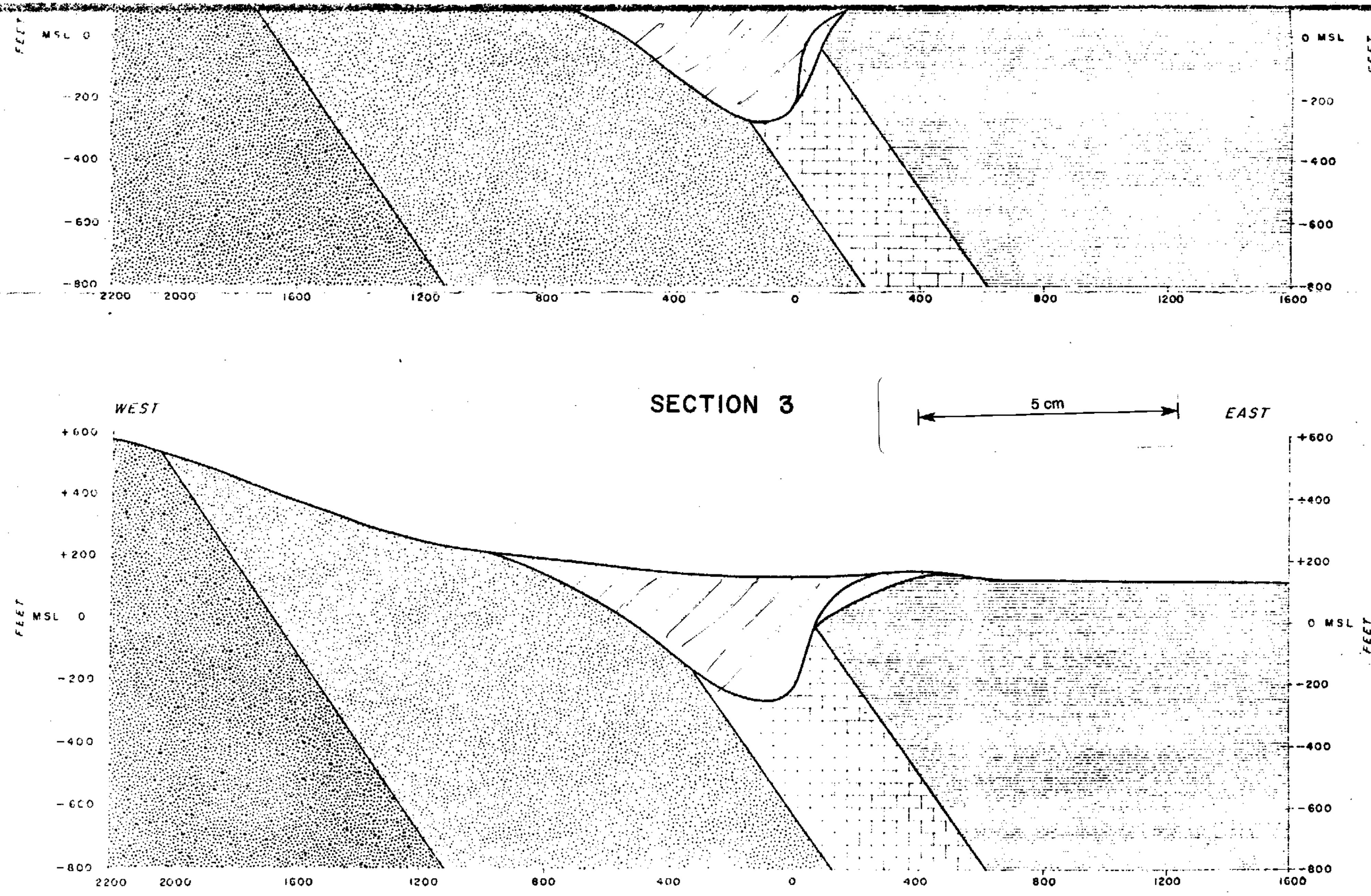
COMPARISON BETWEEN OBSERVED AND THEORETICAL GRAVITY PROFILES ON TRAVERSES D AND L

PLATE 4

Geophysical Branch, Bureau of Mineral Resources, Geology and Geophysics

TO ACCOMPANY RECORD No 1965/133

K 55/B 7-95



FEET

400 0 400 800 1200

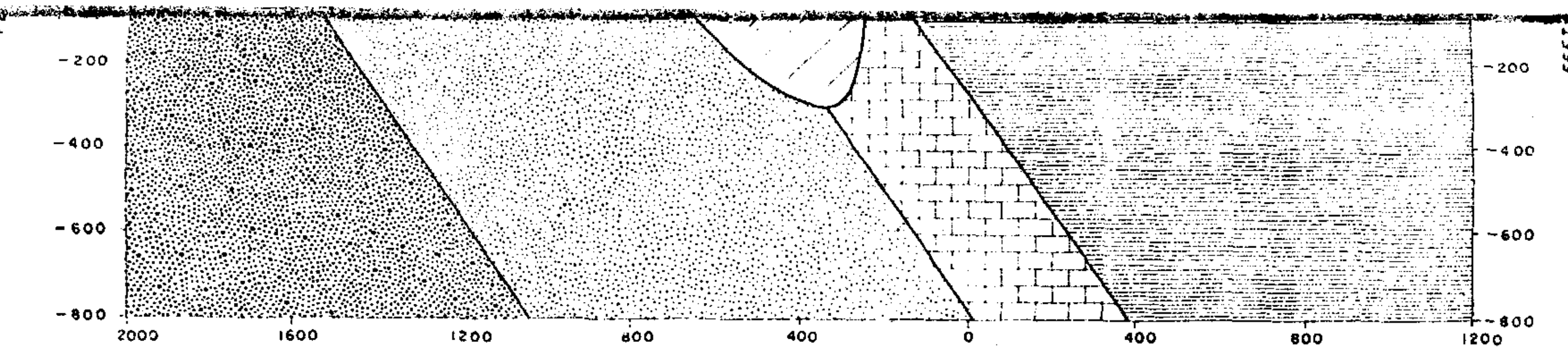
Vertical scales as shown

COMPARISON BETWEEN OBSERVED AND THEORETICAL GRAVITY PROFILES ON TRAVERSE C

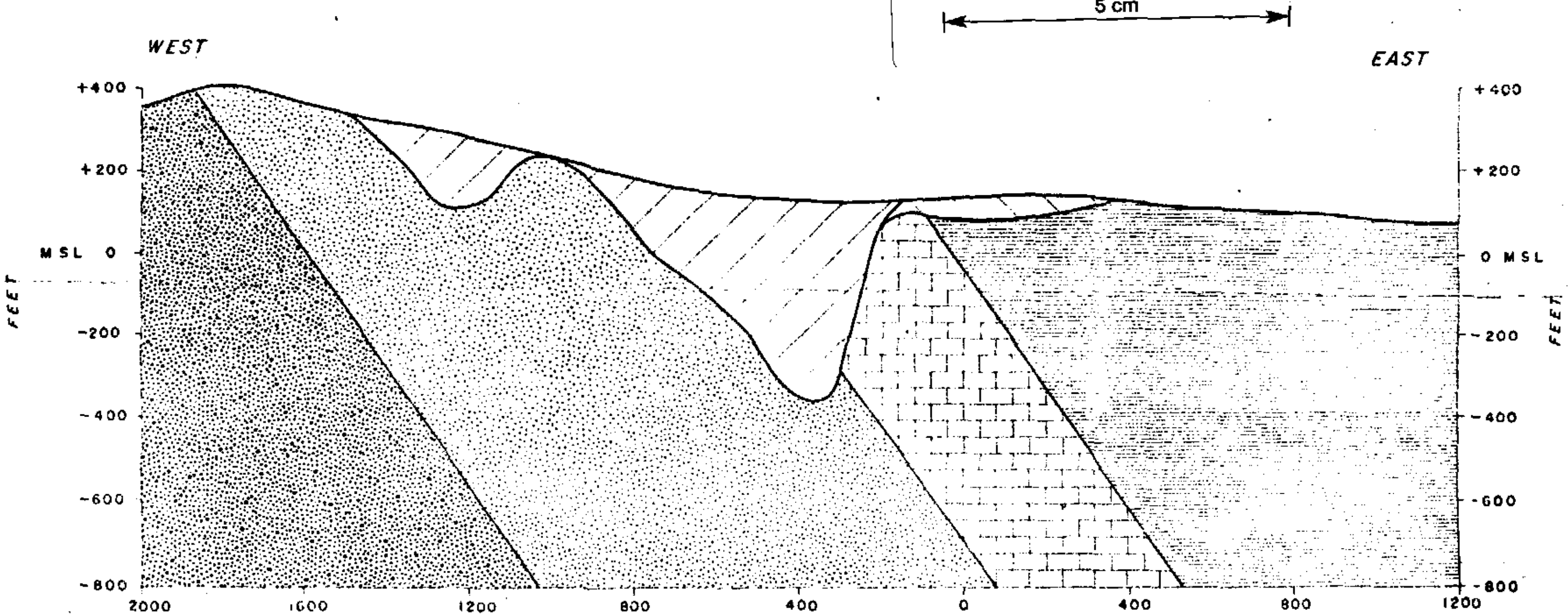
PLATE 5

K55/B7-96

TO ACCOMPANY RECORD No 1965/139



SECTION 2



COMPARISON BETWEEN OBSERVED AND THEORETICAL GRAVITY PROFILES ON TRAVERSE G

FEET
400 0 400 800 1200
Vertical scales as shown

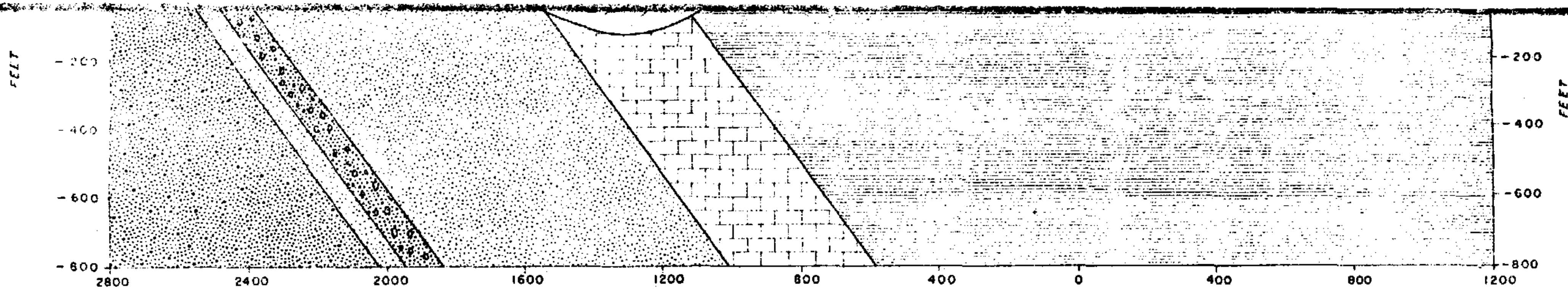
PLATE 6

Geophysical Branch, Bureau of Mineral Resources, Geology and Geophysics

K55/B7-104

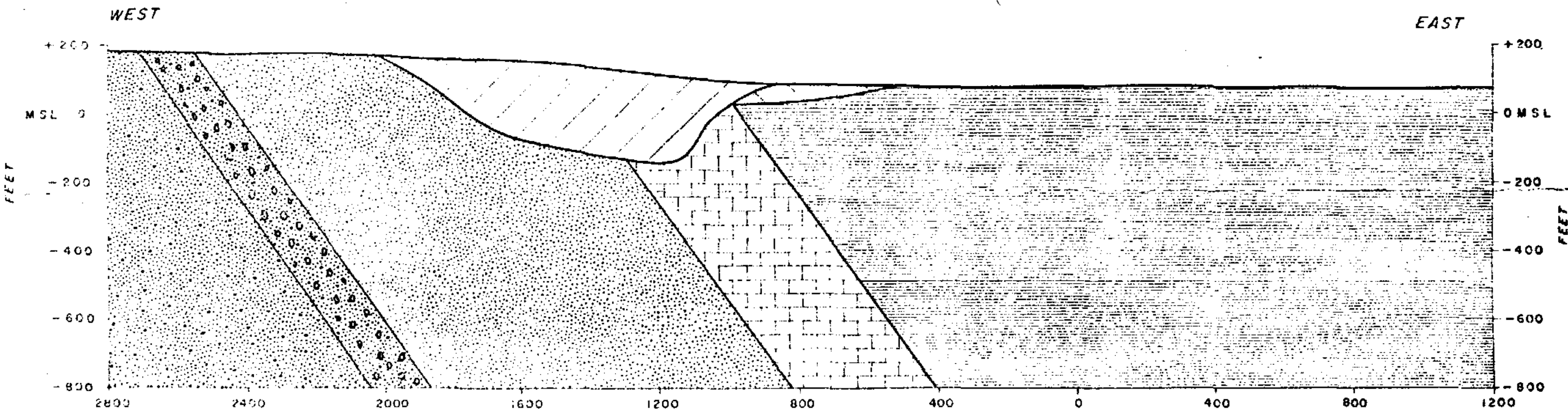
TO ACCOMPANY RECORD No 1965/139

511350



5 cm

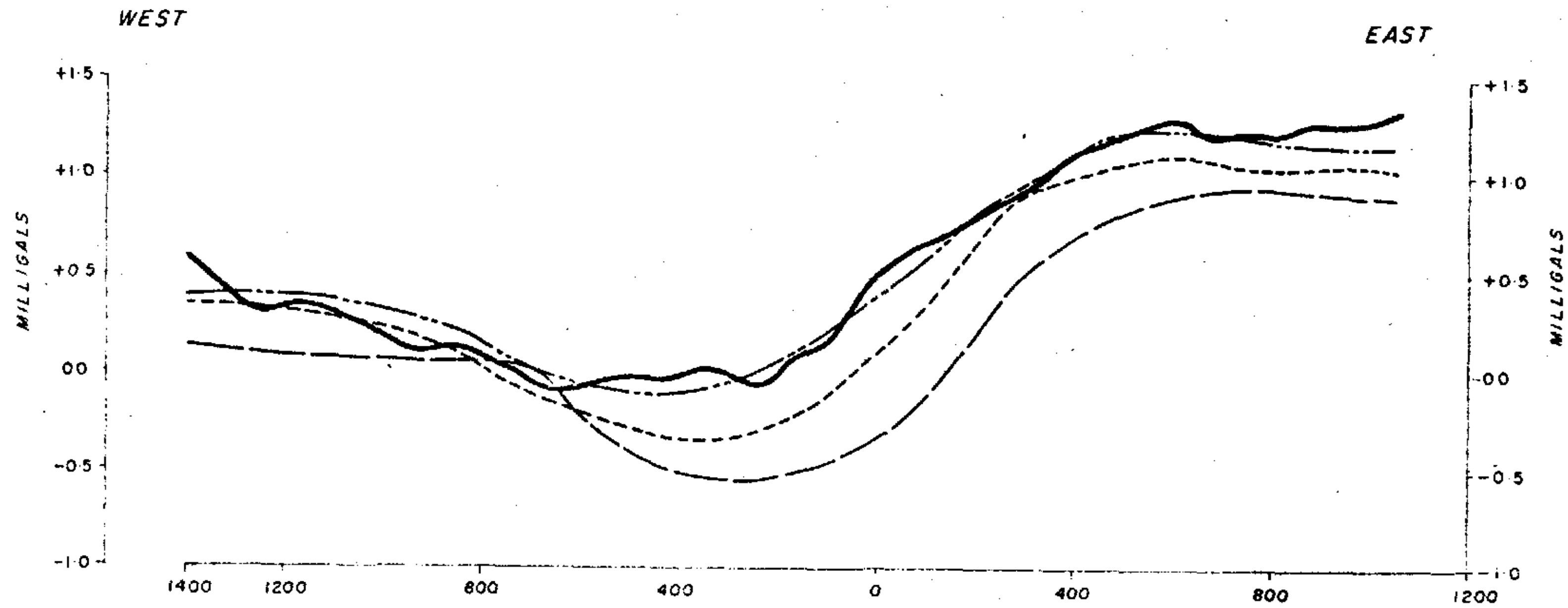
SECTION 2



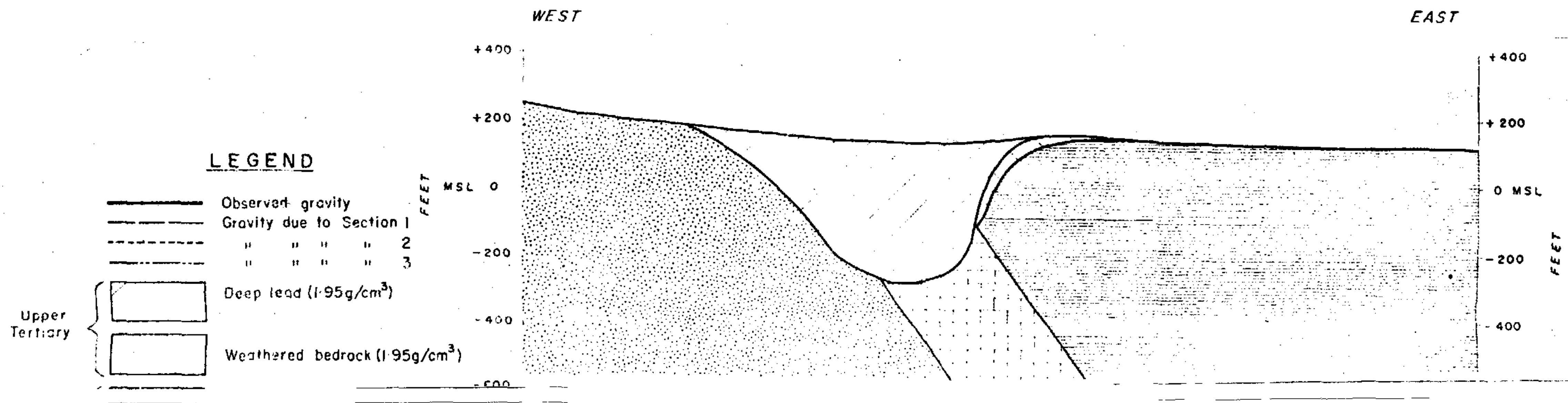
COMPARISON BETWEEN OBSERVED AND THEORETICAL GRAVITY PROFILES ON TRAVERSE H

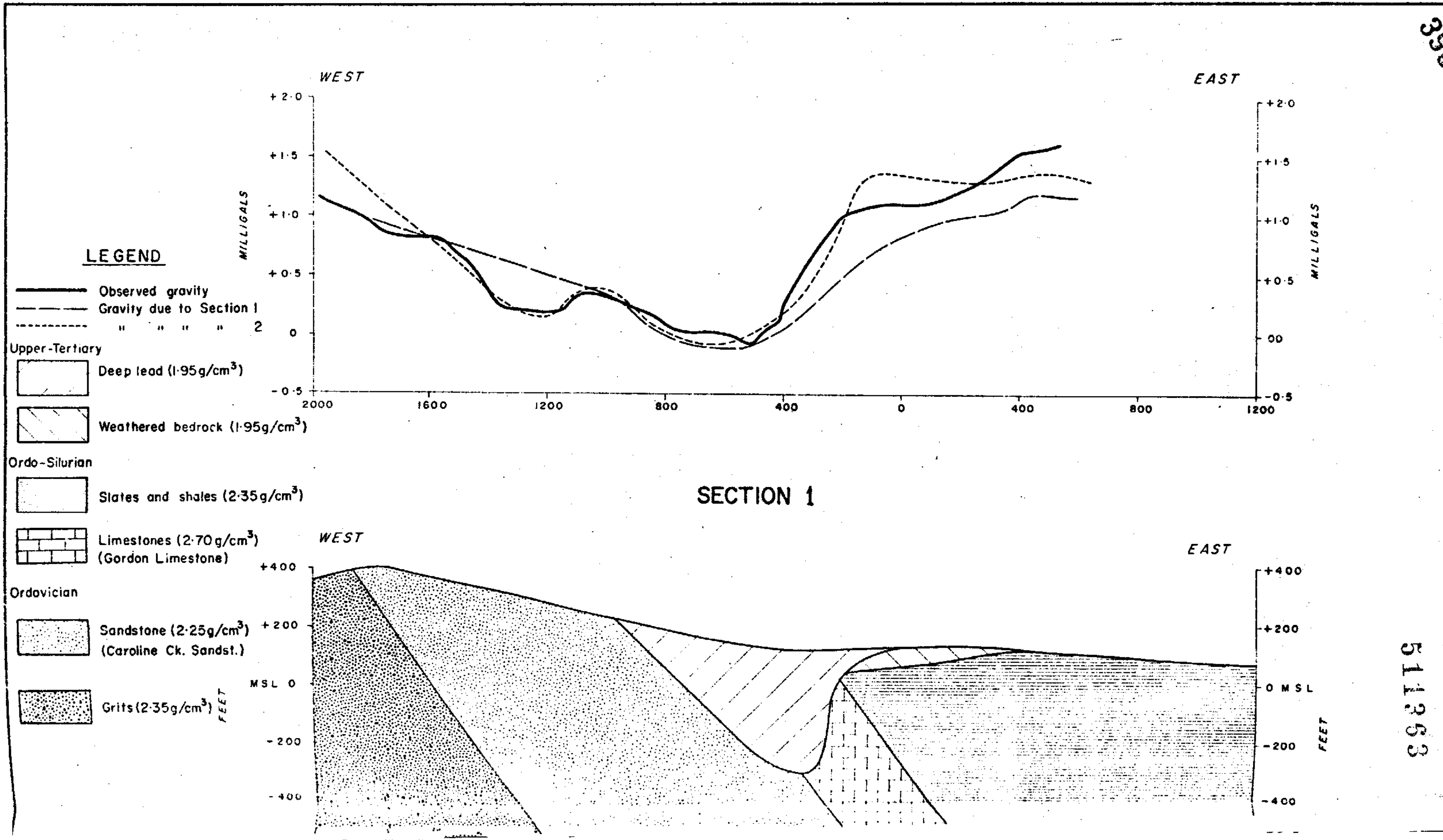
FEET
400 0 400 800 1200
Vertical scales as shown

PLATE 7

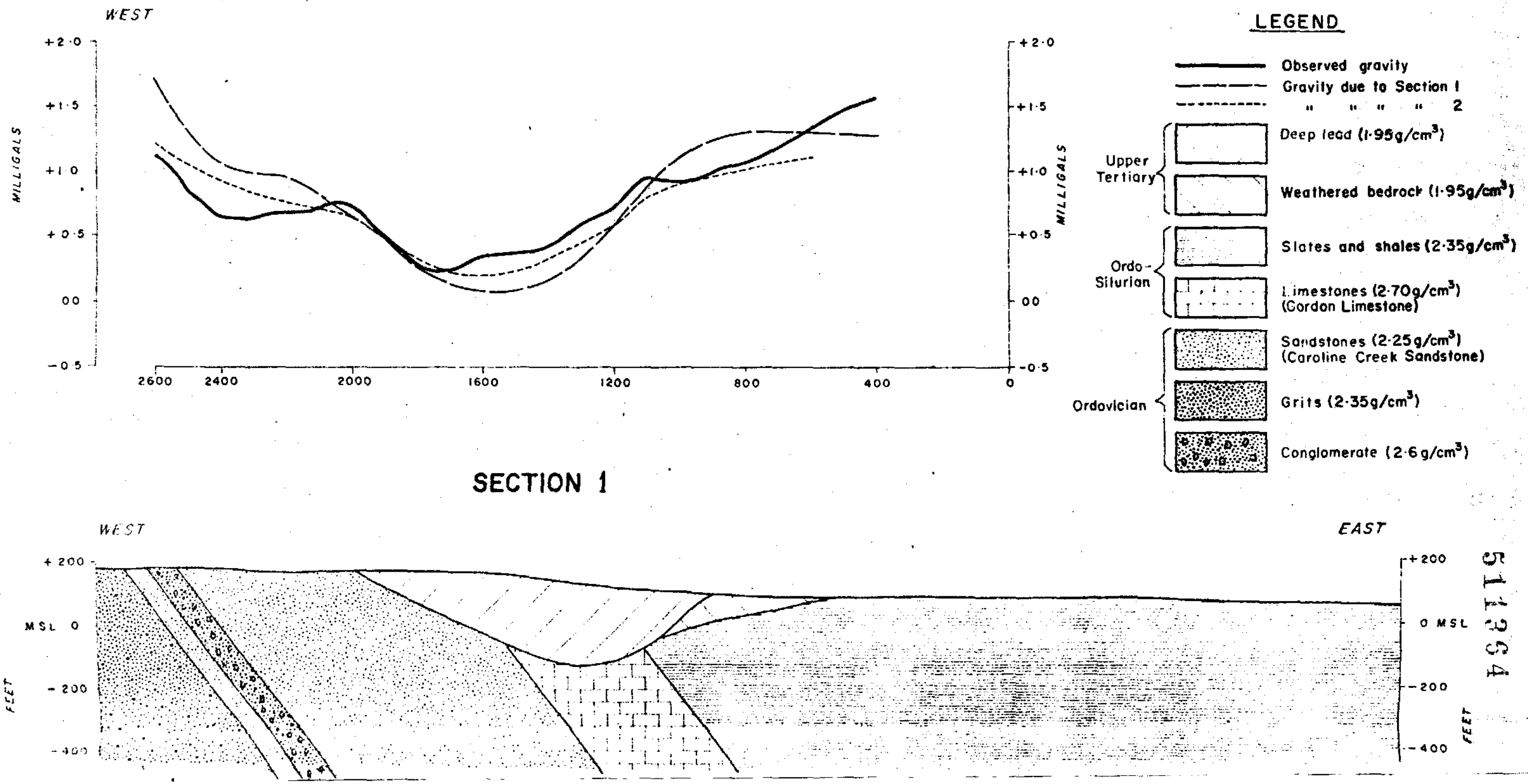

WEST
EAST
+1·5
+1·0
+0·5
00
-0·5
-1·0
MILLIGALS
1400
1200
800
400
0
400
800
1200

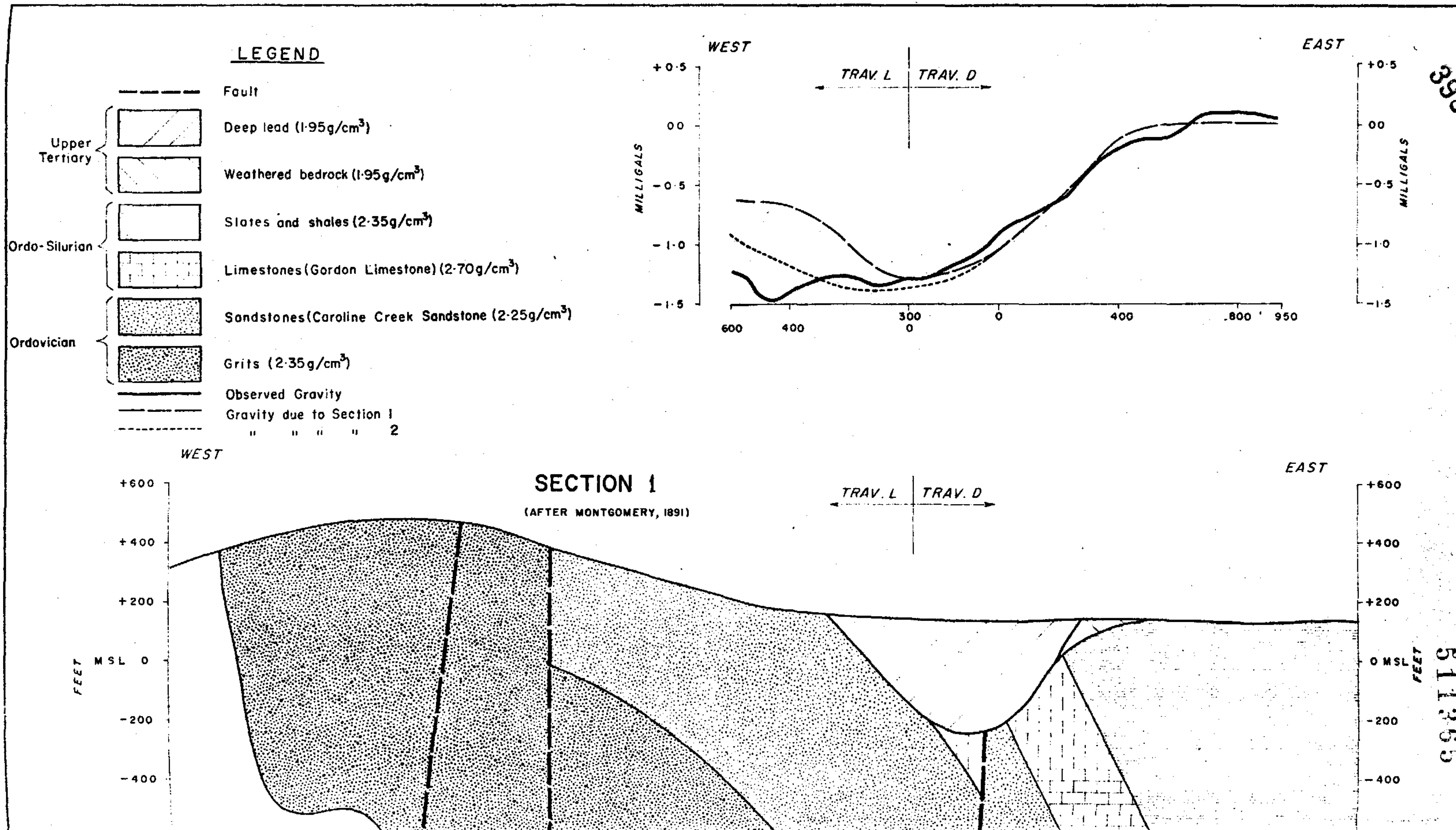
SECTION 1


WEST
EAST
+400
+200
MSL 0
-200
-400
-500
FEET
0 MSL
LEGEND
Observed gravity
Gravity due to Section 1
" " " " 2
" " " " 3
Upper Tertiary
Deep lead (1·95g/cm³)
Weathered bedrock (1·95g/cm³)


WEST
EAST
MILLIGALS
+2·0
+1·5
+1·0
+0·5
0
-0·5
2000
1600
1200
800
400
0
400
800
1200
LEGEND
Observed gravity
Gravity due to Section 1
" " " " 2
Upper-Tertiary
Deep lead (1·95g/cm³)
Weathered bedrock (1·95g/cm³)
Ordo-Silurian
Slates and shales (2·35g/cm³)
Limestones (2·70g/cm³)
(Gordon Limestone)
Ordovician
Sandstone (2·25g/cm³)
(Caroline Ck. Sandst.)
Grits (2·35g/cm³)
SECTION 1
WEST
EAST
+400
+200
MSL 0
-200
-400
FEET
0 MSL

51363


WEST
MILLIGALS
+2·0
+1·5
+1·0
+0·5
00
−0·5
2600
2400
2000
1600
1200
800
400
0
LEGEND
Observed gravity
Gravity due to Section 1
" " " " 2
Upper Tertiary
Deep lead (1·95g/cm³)
Weathered bedrock (1·95g/cm³)
Ordo-Silurian
Slates and shales (2·35g/cm³)
Limestones (2·70g/cm³) (Gordon Limestone)
Ordovician
Sandstones (2·25g/cm³) (Caroline Creek Sandstone)
Grits (2·35g/cm³)
Conglomerate (2·6 g/cm³)
SECTION 1
WEST
EAST
FEET
+200
MSL 0
−200
−400
0 MSL


LEGEND
Fault
Upper Tertiary
Deep lead (1·95g/cm³)
Weathered bedrock (1·95g/cm³)
Ordo-Silurian
Slates and shales (2·35g/cm³)
Limestones (Gordon Limestone) (2·70g/cm³)
Ordovician
Sandstones (Caroline Creek Sandstone (2·25g/cm³)
Grits (2·35g/cm³)
Observed Gravity
Gravity due to Section 1
" " " " 2
WEST
EAST
TRAV. L
TRAV. D
MILLIGALS
+0·5
0·0
−0·5
−1·0
−1·5
600
400
300
0
400
800
950
SECTION 1
(AFTER MONTGOMERY, 1891)
+600
+400
+200
MSL 0
0 MSL
−200
−400
−600
FEET

398

511365


6665
AIR SHAFT
PHOENIX SHAFT
APPROX. SURFACE PROFILE
GOLDEN GATE SHAFT
MAIN SHAFT
FLORENCE SHAFT
HARTS SHAFT
LEFROY SHAFT
GRUBB SHAFT
Level Main Shaft 0'
"MAIN CROSS COURSE"
"DEEP LEAD" CHANNEL
DDH B4 COLLAR RL-53'
DDH A6 COLLAR RL-45'
DDH A3 (III) RL-60'
No1 LEVEL TASMANIA (GOLDEN GATE)
No6 LEVEL FLORENCE
No 3 LEVEL 494'
No 4 LEVEL 601'
No 5 LEVEL 715'
No 6 LEVEL 815'
No 7 LEVEL 915'
1000' LEVEL
1100' LEVEL
1200' LEVEL
1250' LEVEL
1370' LEVEL
1500' LEVEL
LIMIT OF STOPING
-500'
-1000'
-1500'
-2000'
-2500'
-3000'
3500'
IDEALISED GEOLOGICAL INTERPRETATION (Diagrammatic)
MAJOR THRUST
Core logging starting point
Hole A3 (II) wedged off A3 (I)
Massive shear breccia zone
Quartz veining some Py
Hole A3 (III) wedged off A3 (II)
Shear - breccia zone, carbonate veining, minor Py
Shear zone
Massive quartz veining
Shearing, some quartz calcite veins
D.D.H. A3 (III) 3790'
19 ft.
@ 59·3 dwt / tonne Au
4·7 dwt / tonne Ag
1·06 % Cu
23 ft.
@ 41·4 dwt/tonne Au
6·6 dwt/tonne Ag
0·91 % Cu
11·75 ft.
@ 26·4 dwt / tonne Au
11·0 dwt / tonne Ag
1·1 % Cu
14 ft.
@ 23·1 dwt / tonne Au
7·2 dwt / tonne Ag
0·54 % Cu
8·17 ft
@ 8·35 dwt / tonne Au
0·7 % Cu
D.D.H. B4H 1733'
D.D.H. B4 1805'
D.D.H. B4B 1874'
D.D.H. A7 1920'
D.D.H. A6 2040'
REFERENCE
LITHOLOGIES
TERTIARY
"Deep Lead" Material
Clays, lignite, silts, sand, gravels.
ORDOVICIAN
"Grubb Beds"
Black shales, siltstone, minor limestone.
Gordon Limestone Equivalent (?)
Limestone, minor shales, sandstone
"Transition Beds"
Sandstone, calcareous sandstone, limestone shales, decreasing lime content with depth.
Black quartzite, minor pebble bands.
Pebbly sandstone, grit, sandstone, conglomerate, increasing coarse content with depth
Tasmania Lode
Brecciated quartz, ankerite lode with part assimilated country rock, auriferous sulphides and occasional gold specks
Quartz - carbonate veins with minor, pyrite in parts
Zone of intensive shearing.
Dip of bedding.
1100' level Tasmania Lode
1370' level Tasmania Lode
1500' level Tasmania Lode
POSSIBLE ORE LIMIT
2000' lode contour
2500' lode contour
2700' lode contour
GRUBBS SHAFT
B4
A6/7
A3
LINE OF SECTION
217° mag ← → 37° mag
VERTICAL PROJECTION OF DIAMOND DRILL HOLES AND TASMANIA LODE CONTOURS
SCALE 1"=100'
SCALE
0 100 200 300
FEET
5 cm
ALLSTATE EXPLORATIONS N.L.
BEACONSFIELD GOLD FIELD.
75-1108
VOL. 2.
Part "B" Reference 4/4
TRICENTROL AUSTRALIA LIMITED
CROSS SECTION AT 217° MAGNETIC
THROUGH D.D.H A6 COLLAR
SHOWING PROJECTIONS OF D.D.H. B4, D.D.H. A3
AND TASMANIA LODE WORKINGS
BEACONSFIELD, TASMANIA.
DATE: NOVEMBER, 1974
AUTHOR: T.W.M.
PLAN No.
SCALE: 1"= 100'
DRAFTING: L.R.F.
511366
6665


511367
75-1108
VOL. 2
Part 'B' Ref. 4/5
DEVELOPMENT OF TASMANIAN GOLD MINE
ex. Old Company Records
TASMANIA G. M. AND Q. C. Co. REG
PLAN
0 25 50 75 100 FEET
5 cm
Florence Shaft
Main Shaft
Allstate Explorations N.L.
Beaconsfield Gold Mine
P 11
6666


75-1108
VOL. 2.
Part 'B' Ref. 4/6
THE TASMANIA GOLD MINE, LTD.
LONGITUDINAL SECTION
Scale 100 feet to 1 inch.
WEST
EAST
5 cm
Approximate Surface Profile
HART'S SHAFT
GRUBB'S SHAFT
MAIN SHAFT
HART SHAFT
GOLDEN GATE SHAFT
LEEROY SHAFT
No 1 Florence
No 2 Florence
No 3 Florence
No 4 Florence
No 5 Florence
No 6 Florence
No 8 404 Level
No 4 601 Level
REEF MEASUREMENTS & VALUES.
RESUME
Treated
Stocks
Total
These Blocks stoped in 1914.
level not driven to its extreme
MAIN WATER BURSTS
ALLSTATE EXPLORATIONS N.L.
BEACONSFIELD GOLD MINE
511368
6667

75-1108
VOL 2

PART 'B' REF. 4/7

THE TASMANIA G. M. L^TD

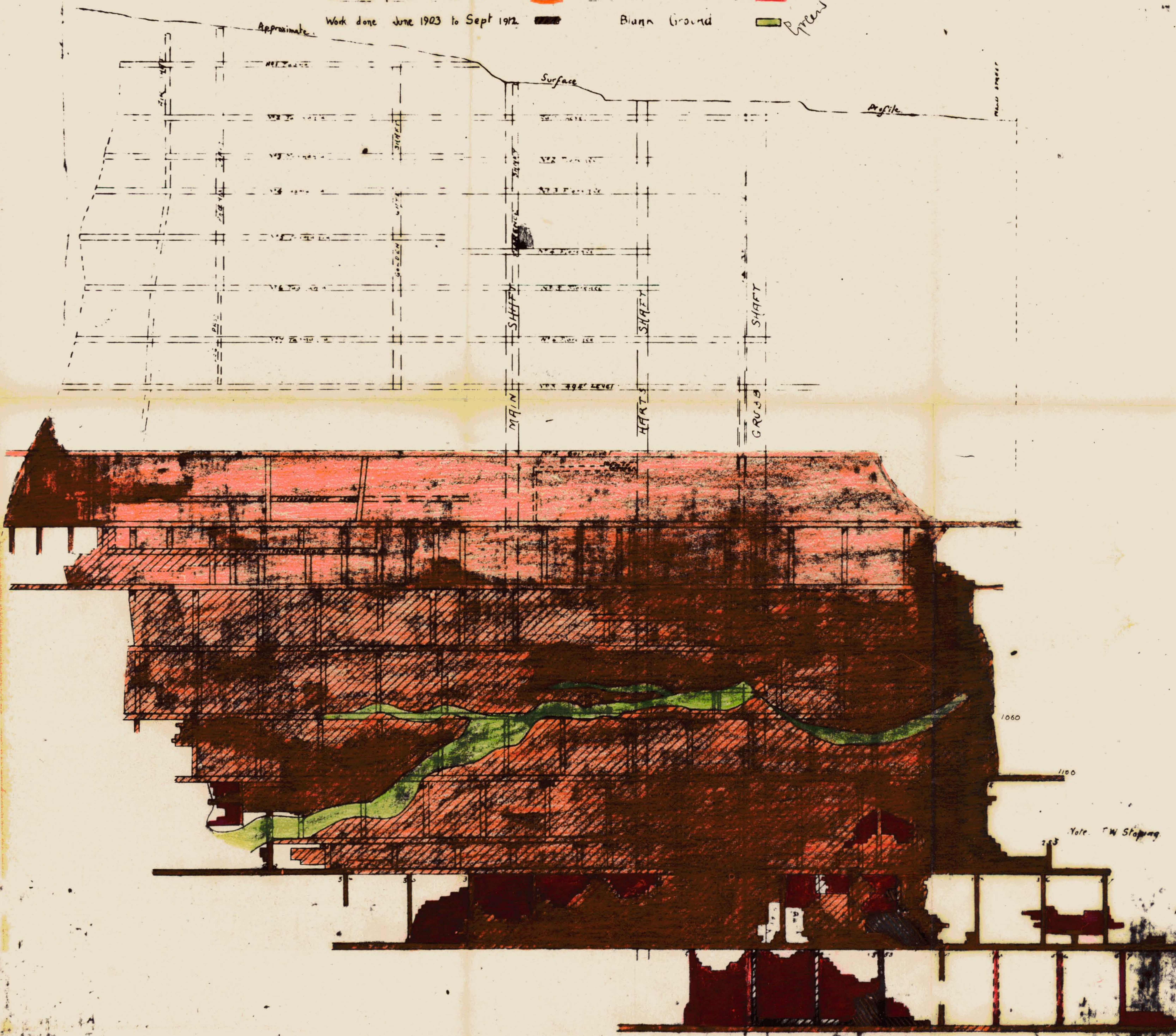
LONGITUDINAL SECTION SHOWING STOPING EXTRACTION — TASMANIAN GOLD MINES - OCT. 1913

Scale 100 F^T to 1 Inch

5 cm

Work done previous to June 1903 — Work done 1^st Oct 1912 to 30 Sept '13

Work done June 1903 to Sept 1912 — Blank Ground (Green)



511369

W.A.T. 8/10/13

ALLSTATE EXPLORATIONS N.L.
BEACONSFIELD GOLD MINE
REF. 4/7

6668