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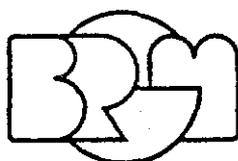
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FEASIBILITY STUDY OF CORKLEIGH CREEK

KUNSTEN MINE (AUSTRALIA)

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REVISED BY SEREM (AUSTRALIA) PTY. LTD.



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SUMMARY AND CONCLUSIONS

On the basis of the present state of the exploration work, it is possible to form the following hypothesis concerning reserves :

- . minimum hypothesis (1) : 86,900 tonnes containing 1.23% WO_3
- . reference hypothesis : a 'reasonable' additional amount of 55,600 tonnes, assumed to contain 1.33% WO_3 , brings the total reserves to 142,500 tonnes with a grade of 1.27% WO_3 .

As we cannot expect a 100% efficiency in mining, we have reduced these figures to 122,700 tonnes of workable reserves with a grade of 1.17% WO_3 .

These tonnage figures should not be considered maximums.

The project presented in this report has the following characteristics :

- . mine capacity : 24,000 tonnes of ore per year
- . value of ore content : A\$100.60/tonne taking into account an 86% recovery at the mill and counting on the value of US \$122/mtu (price US \$142-149 at March 1979)
- . operating cost : varies from \$39.17 to 42.27/tonne depending on the level the ore comes from (280 to 160 m levels)
- . initial investment : \$2,415,118 (+ \$241,000 contingencies) of which some \$669,000 are already spent or committed.
- . payback : 2.47 years.

(1) The terms used here i.e. 'minimum hypothesis' and 'reference hypothesis' for the reserves may seem questionable. They are used here so as not to create a confusion with the terms proven, probable and possible reserves as calculated by SEREM AUSTRALIA.

The reserve factor is of great importance for the present study and the reader may consult chapter 4 for the exact interpretation of these terms.

Under present conditions, there is a high probability of profit and a low risk of loss.

With reserves of 142,500 tonnes the project is definitely profitable. In relative terms it is only slightly vulnerable to possible variations in costs and investments. It becomes marginal but does not show a loss in the case of the most pessimistic hypothesis for ore content value.

On the basis of the least favourable and very pessimistic hypothesis, (additional reserves of 55,600 tonnes are not confirmed and the value of the contents is \$75.45/tonne rather than \$100.60/tonne) the operation would show a loss of \$604,000.

A lower yield per tonne of ore, due to lower wolfram prices or lower grades, is quite possible, but it is very unlikely that the additional reserves of 55,600 tonnes will not be confirmed.

As a matter of fact, it is quite probable that more reserves will be found below the 160 m level, which will make the project even more attractive.

Even without the additional reserves, the minimum reserves already guarantee a reasonable return of the proposed investment.

Under these conditions, it seems appropriate to construct the mill and start mining operations immediately.

CHAPTER 1

LOCATION AND ACCESS

The Oakleigh Creek mine, previously known as Mount Pelion, is located on the right bank of the Forth River 70 km south of Devonport.

In the upper part of its course the Forth River cuts deeply through the glacial plateau of central Tasmania whose altitude ranges from 1,000 to 1,200 m while the valley bottom at the latitude of the mine is 200 m above sea level. (1)

At this point the valley bottom on the right side of the river is narrow. At the slope bottoms, however, there is a terrace of glacial formations which on the other side is bounded by a steep slope dominating the river by about fifteen metres. This terrace is very important for the setting up of the mine installations. It has a width of about forty metres in the vicinity of the mine works and a width of 150 to 200 m a little further north.

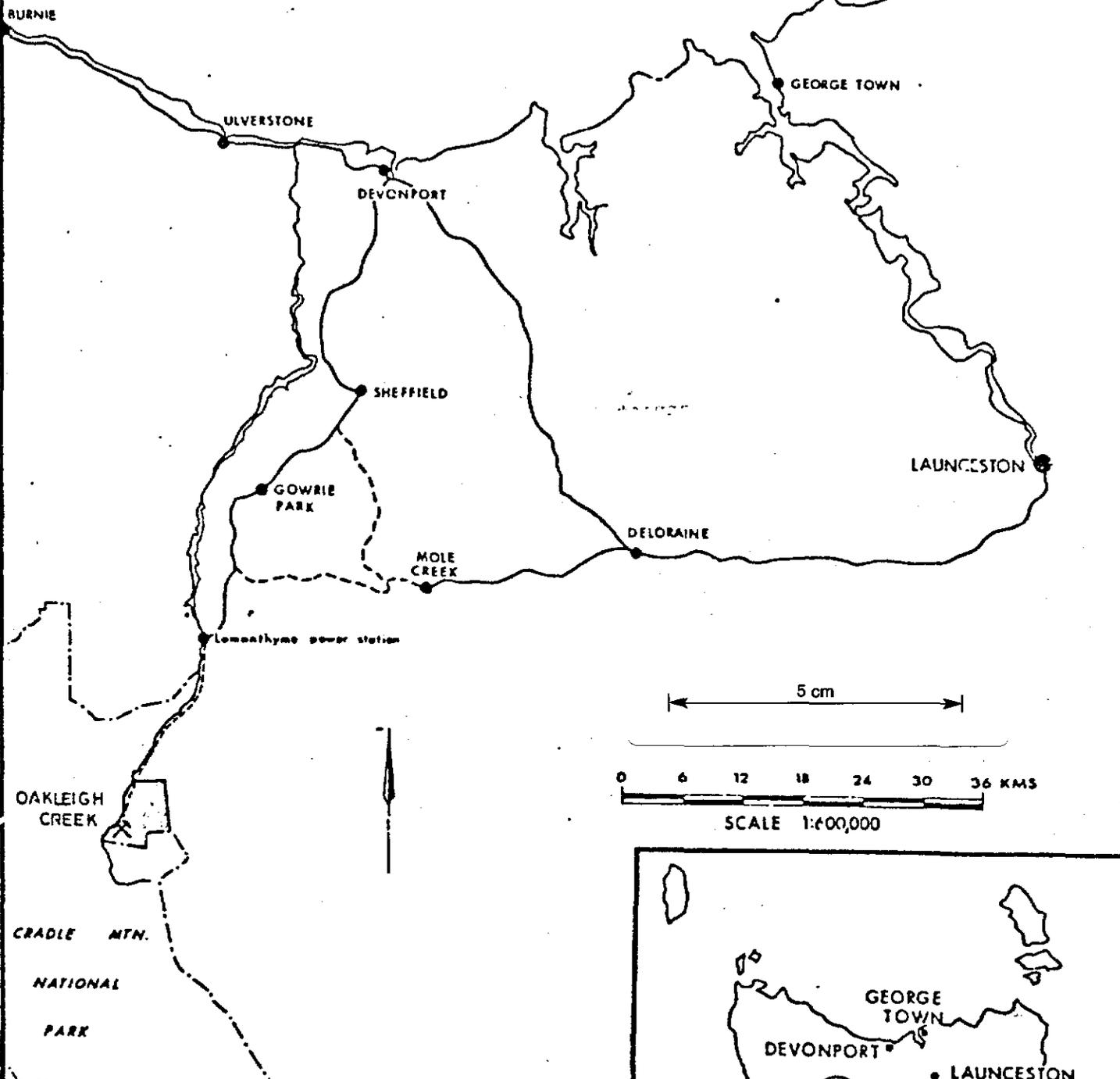
The climate is cold and very humid with precipitation of approximately 2,500 mm per year. This rainfall made possible in the valley bottom the development of rain forest which progressively gives way to eucalyptus forest on the slopes and further downstream.

To the west and on the opposite bank of the Forth River begins the Cradle Mountain National Park which is one of the most popular in Australia. It should be noted that the river itself is inside the park. It appears that the boundaries of the national park were drawn taking into account the existence of the mine.

(1) Levels 240 and 280 which will be discussed below correspond to arbitrary conventional elevations and not to altitudes above sea level.

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Bass Strait



LOCALITY MAP

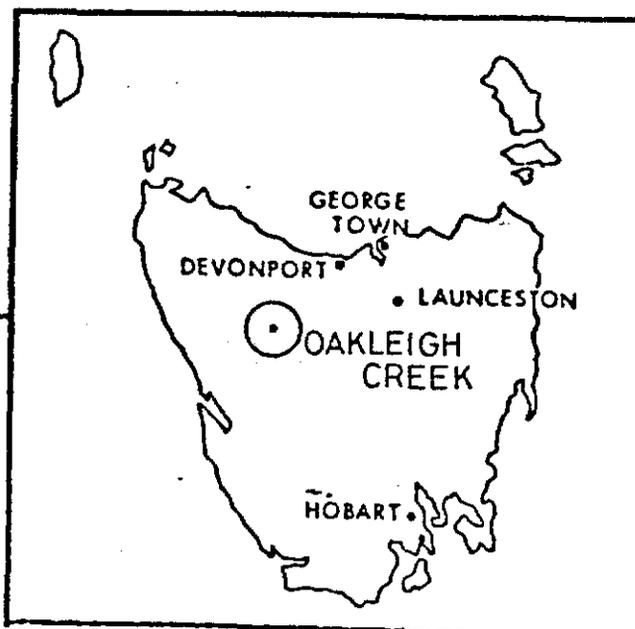


FIGURE I

Access to Oakleigh Creek is easiest through Devonport. This city is 99 km away by road and the last 26 km consist of forest track. Devonport is a port through which the mine equipment can be shipped.

Other nearby cities are as follows :

- ++ Launceston (150 km by road) is the main city in the north of Tasmania and the administrative centre ;
- ++ Burnie (151 km by road) is a small industrial city and ore port where the concentrates from the mine can be loaded.

Smaller towns connected with the project are as follows :

- ++ Sheffield is 66 km by road from Oakleigh Creek and is the closest village in which the families of mine executive could live. It is also planned to set up secretarial services here.
- ++ Gowrie Park is a group of houses 52 km from the mine where the Sheffield township has offered as a campsite for the joint venture. Although this would have several advantages this offer was not accepted because a trip of about 1 hour 20 minutes is involved.
- ++ At the locality called Lemonthyme, 26 km from the mine, there is an uninhabited hydroelectric plant which marks the end of the asphalt road and the closest location at which it is possible to link up with the electrical supply network.
- ++ Deloraine is the township upon which the project will depend administratively since the mine is located on its territory.

Lastly, it should be noted that throughout the northern coast of Tasmania there are many small workshops and construction firms capable of carrying out the kind of work which is usually subcontracted.

CHAPTER 2

LEGAL ASPECTS OF PROJECT

2.1. Project ownership

Participants in the joint venture, which will be operated through a trustee company, *CENTRAL TASMANIAN TUNGSTEN PTY. LTD.*, are as follows :

. TRIAKO MINES N.L.	33 1/3 %
. BUKA MINERALS N.L.	33 1/3 %
. SEREM (AUSTRALIA) PTY. LTD.	33 1/3 %

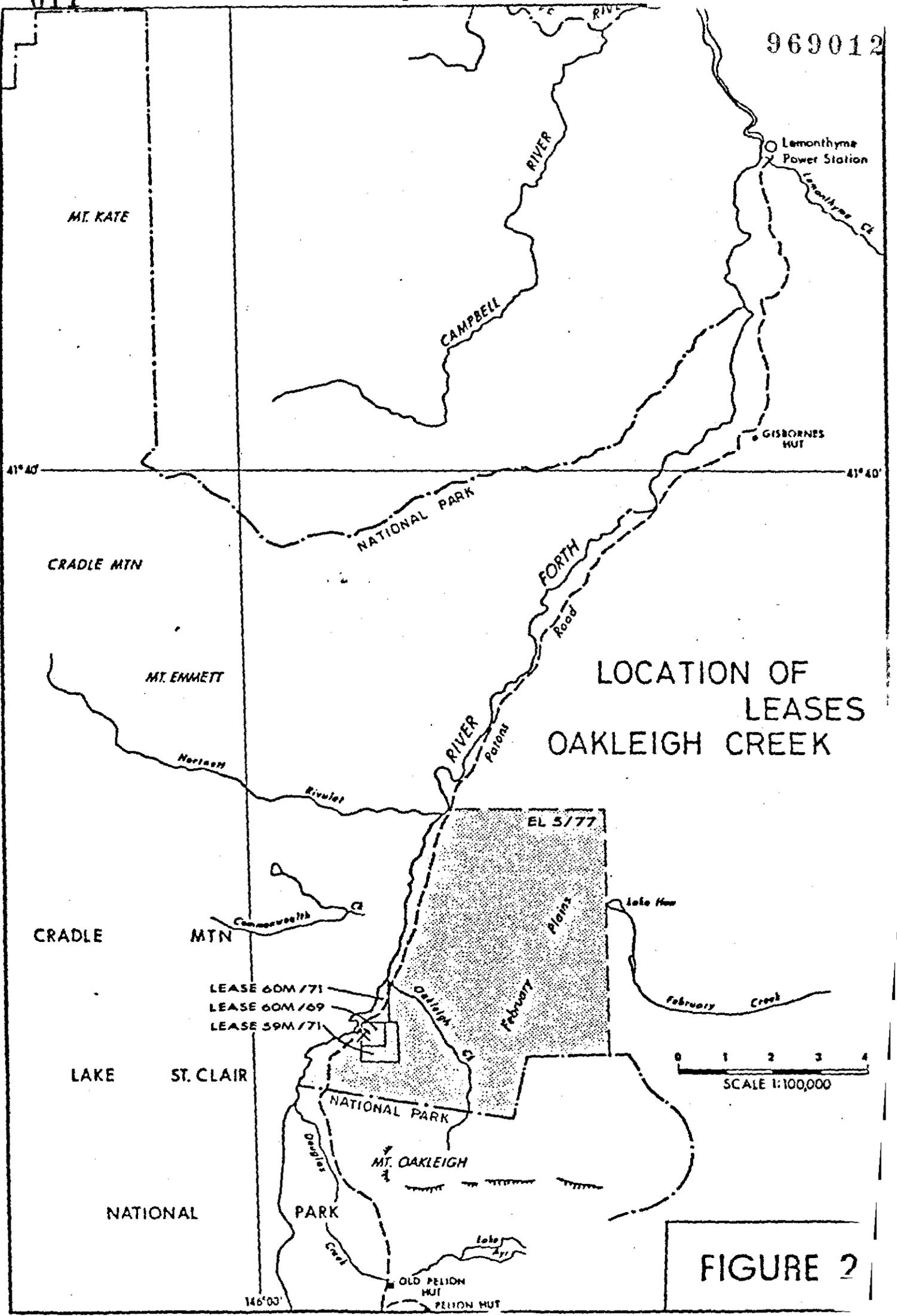
SEREM (AUSTRALIA) PTY. LTD. is a wholly owned subsidiary of *Soci t  d'Etudes, de Recherches et d'Exploitations Mini res (S.E.R.E.M.)* which is itself a wholly owned subsidiary of *Bureau de Recherches G ologiques et Mini res (B.R.G.M.)* a mining and geological agency of the French Government.

BUKA MINERALS N.L. and *TRIAKO MINES N.L.* are both listed on the Australian Stock Exchange, but *TRIAKO* owns 58 % of *BUKA*.

Recently, *AQUITAINE (AUSTRALIA AND NEW ZEALAND) LIMITED* acquired 42 % of the shares in *TRIAKO*, and has an option to increase this to 60 %. The parent company of *AUSTRALIAN AQUITAINE* is *SOCIETE NATIONALE ELF AQUITAINE (SNEA)*, 92 %, with the remainder held by Australian institutions and investors.

2.2. Agreements and tenement situation

SCAMANDER MINING CORPORATION N.L. and *LOUISA MINING CORPORATION N.L.*, by virtue of an agreement between them dated June 30, 1976, hold three Tasmanian Mining Leases covering 80 ha. (210 acres), which are valid for 21 years, over the old Mt. Pelion lodes : No. 60M/69 (granted 1.4.69), 60M/71 and 59M/71 (granted 1.10.71). These leases were the subject of an Option Agreement between these companies and *KIBUKA MINES PTY. LTD.* (acting on behalf of *TRIAKO* and *BUKA*, and being a wholly owned subsidiary of the latter) dated June, 2, 1977.



LOCATION OF
LEASES
OAKLEIGH CREEK

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FIGURE 2

5 cm

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proportion to their interest.

SEREM is the mining and exploration operator responsible to a Management Committee, which comprises representatives of SEREM, TRIAKO and BUKA, voting in accordance with the percentage share of each party.

KIBUKA have entered into an option agreement dated 26/6/78 with L.M. GRAUE and W.P. KIDD over Mineral Lease 24M/60 which covers the All Nations Mine, Moina. This is a free option with an exercise date of 26/6/79 and a purchase price of \$ 26,000. As it lies within the area of common exploration interest referred to previously, SEREM has been given the right to earn 33 1/3 % interest in the property. Although this operation is managed by the joint venture the expenditure involved will not be taken into account in the present feasibility study because this is an additional investment which must be justified on its own merits. (1)

(1) A maximum of \$ 15,000 may be spent on All Nations during the period of the option.

CHAPTER 3

GEOLOGICAL BACKGROUND

3.1. General geological background

The Oakleigh Creek mine is located in a Pre-Cambrian metamorphic series consisting essentially of quartzites and micaschists. The trend of the beds in the neighbourhood of the mine ranges from 82° to 109° and their dip from 15° to 27° north.

1,600 m to the northeast of level 240 outcrops the small granitic intrusion of Birthday which traverses the Pre-Cambrian series. There is another granitic outcrop of the same type at Lone Pine 3 km further north. It is believed that these granites are the origin of the tungsten tin and copper occurrences found in this region.

3.2. The main vein

The only ore body of economic interest which is known up to the present time is a quartz vein traversing Pre-Cambrian quartzites. Its trend varies from 170° to 175° and its dip is 75° east. At the outcrop and over the first 70 m of level 240 there is only a single vein with a thickness varying from 30 to 40 cm. Further south it frequently breaks up into a bundle of two or three parallel veinlets with an accumulated quartz thickness of between 30 and 50 cm while the total thickness including the intercalated gangue ranges from 1 m to 2 m.

In addition to the quartz the minerals usually observed in the vein are as follows : tourmaline, wolframite, muscovite, goethite, siderite, fluorite, pyrite, mispickel, bismuthinite and cassiterite. Sphalerite, chalcopyrite and pyrrhotite are observed more rarely. Tourmaline is found mainly on the edges of the vein and is very abundant in the walls. It should be noted that scheelite is practically absent.

This mineralogical composition resembles that at Storey's Creek and Rossarden. Cassiterite and fluorite are, however, less abundant and

no concentrations of muscovite or pyrite are observed along the walls.

To the north of the portal of level 240 the vein disappears under glacial formations. Hall and Relph (1970) as well as Fleming (1977) state that the extension of the vein was found on the western bank of the Forth River in the National Park. This statement could not be confirmed by the SEREM geologists and should be considered very hypothetical. The extension of mining works to the south is examined in detail at the beginning of Chapter 4.

3.3. Other known tungsten occurrences in this zone

About fifty metres to the east of the portal of level 240 along the track leading to level 280 outcrops a quartz vein with a thickness of about ten centimetres which contains wolframite.

Several other veins containing tin and wolfram were indicated by authors such as Hitchcock, (1920), and Thomson, (1950). The most interesting of these outcrop along Reid Creek at distances of between 180 m and 300 m to the southeast of the portal of level 240. One of these veins has a thickness of 23 cm.

Other tin and wolfram occurrences have been indicated in the neighborhood of the granitic outcrops of Lone Pine and Birthday which were mentioned above.

There has been no recent prospecting work on these occurrences but most of the available data on this subject has been collected by Lawrence, (1977 b).

CHAPTER 4

RECONNAISSANCE WORK AND RESERVES

4.1. Reconnaissance work

4.1.1. Work preceding exercise of option

Without reviewing the history of the mine the main events before 1977 will be mentioned.

* A trench stripped the vein outcrop over 137 m. The thickness of the vein amounted to an average of 34 cm. Twenty samples were selected by the Department of Mines. The weighted average of the grades was 1.68 % WO_3 and 0.22 % Sn.

This trench has now partly caved in and no recent observations were made.

* At level 240 the vein was traced over 151 m. The average thickness of the vein was estimated at 30 cm and partial samplings gave values ranging from 0.56 to 1.03 % WO_3 .

These results are approximate indications but they cannot be used for computing reserves because the method used for sample collection is not clearly defined. In addition, recent campaigns are much more significant because of the extent of the areas sampled and the number of samples. However the relatively high tin grade of the samples of the Department of Mines should be noted. This value corresponds well with the observed frequency of cassiterite in the surface samples. This mineral is much rarer in the underground works.

4.1.2. Diamond coring

In November 1977 SEREM carried out a campaign of four core drillings in order to explore the continuity of the vein.

The most northerly of these drillings was unable to traverse the glacial formations. The three others actually cut through the vein. They are plotted in the plane of the vein in figure 3.

The results are as follows :

	Thickness of quartz vein (after correction for angle of incidence of drilling)	Wolframite and Sn mineralization
MP 1	20 cm	traces of Sn
MP 2	48 cm	very abundant wolframite
MP 3	45 cm + 27 cm separated by 35 cm of quartzite	rare wolframite and Sn

The fact that drilling MP 1 did not find wolframite is not significant since in the known part of the vein a drilling has only a 40 % chance of finding this mineral.

4.1.3. Recent mining work

The following works were carried out from May 1978 to March 1979-:

- . 153 m of adit were cut along level 240 bringing the total length driven to 304 m ;
- . 163 m were cut on level 280 ;
- . 39 m of raise were cut between the two levels.

All these works are within the vein. The results given below are based upon the works described above.

4.2. Sampling in underground works

Two types of sampling were carried out.

4.2.1. Bulk sampling

27 samples of about 2 tonnes distributed over the first 150 m of the 240 level were collected by blasting in the adit roof. The samples thus consisted of vein rock and wall rock. After crushing and successive reductions each sample was chemically analysed. There was then hand selection of a sufficient quantity of quartz and quartzite grains in order to estimate the proportion of each of these two phases. It was possible to arrive at the grade of the quartz phase (or vein grade).

The average for these grades is 3.17% WO_3 (1) for the entire group of 27 samples, as compared to 2.96% WO_3 obtained by point counting for the same section of the adit.

More bulk sampling was done on the 240 m level between 181 and 250 m.

After each round two scoop loads were set aside and samples were transported to Burnie for crushing and assaying by the Government Laboratory in Launceston.

Taking into consideration the width of the vein and the dimensions of the adit, the bulk samples should have contained theoretically 18.3% of quartz. However by counting the quartz in samples representative for the different bulk samples, it was found that the average quartz content was only 13.4%. As the real width of the quartz vein can be measured without difficulties, it seems reasonable to assume that the bulk samples were not quite representative. By assuming this, the correlation between bulk sampling and point counting was again quite remarkable.

Average grade bulk samples	:	0.49 % WO_3
Average width vein	:	0.40 m
Average grade vein by point counting	:	4.10 % WO_3
Average grade vein as calculated by measured quartz content	:	4.18 % WO_3

(1) The figure of 3.15 % WO_3 which was given at first was corrected by the supervisors of this operation.

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4.2.2. Point counting

This classical technique for quartz and wolframite veins was the one which was the most usually used.

For each punctual sample the respective quartz and wolframite lengths i.e. Lq and Lw are measured in a direction perpendicular to the vein plane. If various parallel veins are present the lengths are accumulated.

The punctual grade for W₃ is then deduced according to the following formula :

$$\% W_3 = 0.76 \frac{7.25 L_w}{(7.25 \times L_w) - (2.65 L_q)}$$

For each series of measurements a punctual measurement was made every metre. Several series of measurements with different starting points were made in certain sections of the adit.

The following data is given below :

- . firstly the comparison of several successive series of measurements of the same section ;
- . secondly the breakdown of the last series of measurements for adit sections in order to indicate lateral variations.

Table 1 : CRUDE SAMPLING RESULTS

Character and date of measurements	Location of measurements	Length sampled	Average thickness	WO ₃ grade of vein	WO ₃ grade over 1 m	
Bulk sampling (November 1977)	Level 240 1 m - 150 m	150 m	35.33	3.17	1.12	
August 1977	Level 240 1 m - 150 m	150 m		2.96	1.05	
July 1978	Level 240 1 m - 150 m	150 m	37.77 ⁽¹⁾	3.76	1.42	
July 1978	Level 240 1 m - 168 m	168 m	39.96	4.11	1.64	
September 1978	Level 240 1 m - 168 m	168 m	39.45	3.75	1.48	
Point counting	September 1978	Level 240 1 m - 30 m	30 m	33.50	3.29	1.10
		31 m - 60 m	30 m	35.33	2.62	0.93
		61 m - 90 m	30 m	39.89	2.97	1.18
		91 m - 120 m	30 m	42.60	3.80	1.62
		121 m - 150 m	30 m	41.37	4.61	1.91
		151 m - 168 m	18 m	47.50	5.42	2.57
	October 1978	169 m - 203 m	35 m	41.8	2.05	0.86
		204 m - 252 m	49 m	38.1	5.29	2.02
	March 1979	253 m - 304 m	52 m	30.06	3.31	0.99
	September 1978	Level 280 1 m - 32 m	31 m	53.5	5.09	2.72
	October 1978	33 m - 71 m	39 m	49.0	2.90	1.42
	November 1978	72 m - 89 m	18 m	56.0	1.96	1.10
	March 1979	90 m - 154 m	65 m	44.62	3.47	1.55
	July 1978	raise 1 m - 18 m	18 m	35.8	2.96	1.06

(1) Between August 1977 and July 1978, level 240 was enlarged and several vein branchings were discovered which had not been measured previously.

Examination of this table shows the following :

- . firstly there is satisfactory agreement between successive samplings over the same sections ;
- . secondly the "grade x thickness" value is relatively constant (grade for one metre).

4.3. Geological reserves

For the evaluation of the ore reserves, which was carried out by SEREM AUSTRALIA geologists, the deposit was generally divided into 50 m x 40 m blocks. The thickness and grade of each block were computed on the basis of the grades of the nearest mine works. A density of 2.70 was applied for the computation of the tonnages.

Three categories of reserves were defined :

- A - proven reserves - these are entirely surrounded by mining works ;
- B - probable reserves 1 - these result from an extrapolation of 40 m around the adits and are almost entirely contained in the polygon defined by drillings MP 1, MP 2, MP 3, the southern end of level 240 and the known outcrops. The existence of the vein in this zone is practically certain. The evaluation of grades by extrapolation is less conventional but is still reasonable.
- C - probable reserves 2 - these result from an extrapolation of 80 m around the adits and are only partially contained within the polygon defined above. It is very probable that the vein exists in the zone but the extrapolation of grades becomes less certain.

Oakleigh Creek : reserves situation at 1-3-1979

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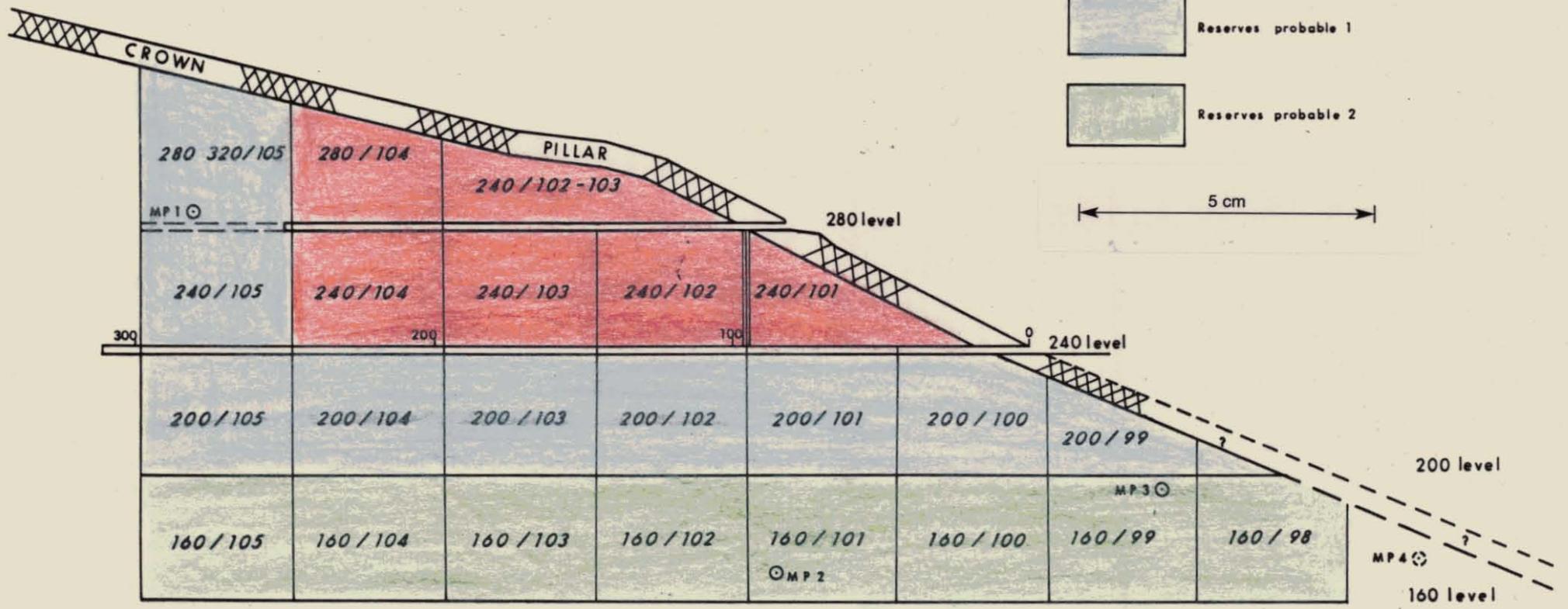
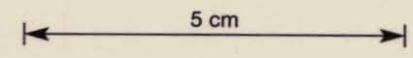
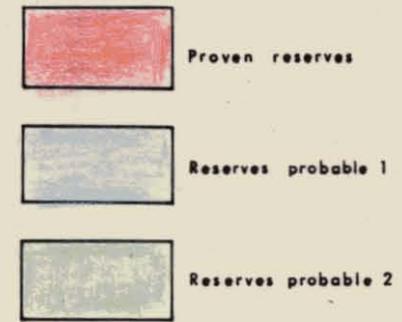


Figure 3

BLOCK (cf. figure 3)	Vein width	Vein grade % WO 3	Grade working width 1.20m	Tonnage working width 1.20m	Tonnage ex adits, raises & crown pillar
<u>Proven reserves</u>					
280/102-103	0.52	2.72	1.18	9050	5300
280/104	0.41	3.95	1.35	7200	5400
240/101	0.37	3.14	0.97	7650	5300
240/102	0.47	4.07	1.59	6500	5950
240/103	0.49	3.27	1.33	6500	6100
240/104	0.40	3.39	1.13	6500	6100
Total			1.26	43,400	34,150
<u>Probable reserves 1</u>					
280-320/105	0.30	2.00	0.50	9150	7850
240/105	0.33	4.99	1.37	6500	6100
Total			0.88	15,650	13,950
Total proven + probable +240			1.15	59,050	48,100
<u>Probable reserves 1</u>					
200/100	0.50	3.14	1.31	6500	6300
200/101	0.40	3.14	1.05	6500	6500
200/102	0.47	4.02	1.57	6500	6500
200/103	0.49	4.01	1.64	6500	6500
200/104	0.40	3.01	1.00	6500	6500
200/105	0.33	4.99	1.37	6500	6500
			1.32	39,000	38,800
Total proven + probable 1			1.23	98,050	86,900
<u>Probable reserves 2</u>					
200/99	0.50	3.14	1.31	5200	3900
160/98	0.50	3.14	1.31	6500	6200
160/99	0.50	3.14	1.31	6500	6500
160/100	0.50	3.14	1.31	6500	6500
160/101	0.48	3.14	1.26	6500	6500
160/102	0.48	4.07	1.63	6500	6500
160/103	0.48	3.27	1.31	6500	6500
160/104	0.40	3.39	1.13	6500	6500
160/105	0.33	4.99	1.37	6500	6500
Total probable reserves ²			1.33	57,200	55,600
Total reserves			1.27	155,250	142,500

4.4. Workable reserves

The following corrections are made in order to transform geological reserves into workable reserves.

4.4.1. Reduction of tonnage taking into account the rate of recovery characterizing the working method.

On the basis of the pillars shown in figure 4 (Chapter 5) the rate of recovery is 86%.

However if it would be possible to recover the pillars in the stopes a recovery grade of 93.72% can be expected.

In the following calculations only the lower figure of 86% recovery has been taken into consideration.

4.4.2. Hypotheses concerning extension of vein

In view of the configuration indicated by the reconnaissance work we believe that it is definitely too pessimistic to suppose that the deposit consists only of the proved tonnage.

In order to compute profitability the three following hypotheses will therefore be considered :

- hypothesis A : this is the most pessimistic of those considered and supposes that the deposit is limited to the blocks classified as "proved" reserves + "probable reserves 1" ;
- hypothesis B includes the "proven reserves" + "probable reserves 1 and 2".

4.4.3. Working grades

In order to simplify the computations it will be supposed that the working grade is uniform.

Since the grades for "probable reserves 2" are based upon data which are not sufficiently significant, only the grades of the proven reserves and "probable reserves 1" will be used as a basis. The weighted average for these reserves is 1.23 % WO_3 .

This base grade is considered as related to a working width of 1.20 m and an additional reduction of 5 % will also be applied to cover waste due to working inefficiency as follows :

$$1.23 \times 0.95 = 1.17 \text{ \% } WO_3$$

It is supposed that this grade is a good characterization of the ore covered by hypothesis A (with a relative error of less than 15 %).

The extrapolation of this ore grade to the ore of probable reserves 2 is reasonable but it is impossible to establish a confidence interval for this extrapolation.

For the financial computations this value for contents is varied ± 25 % (cf. paragraph 14.5).

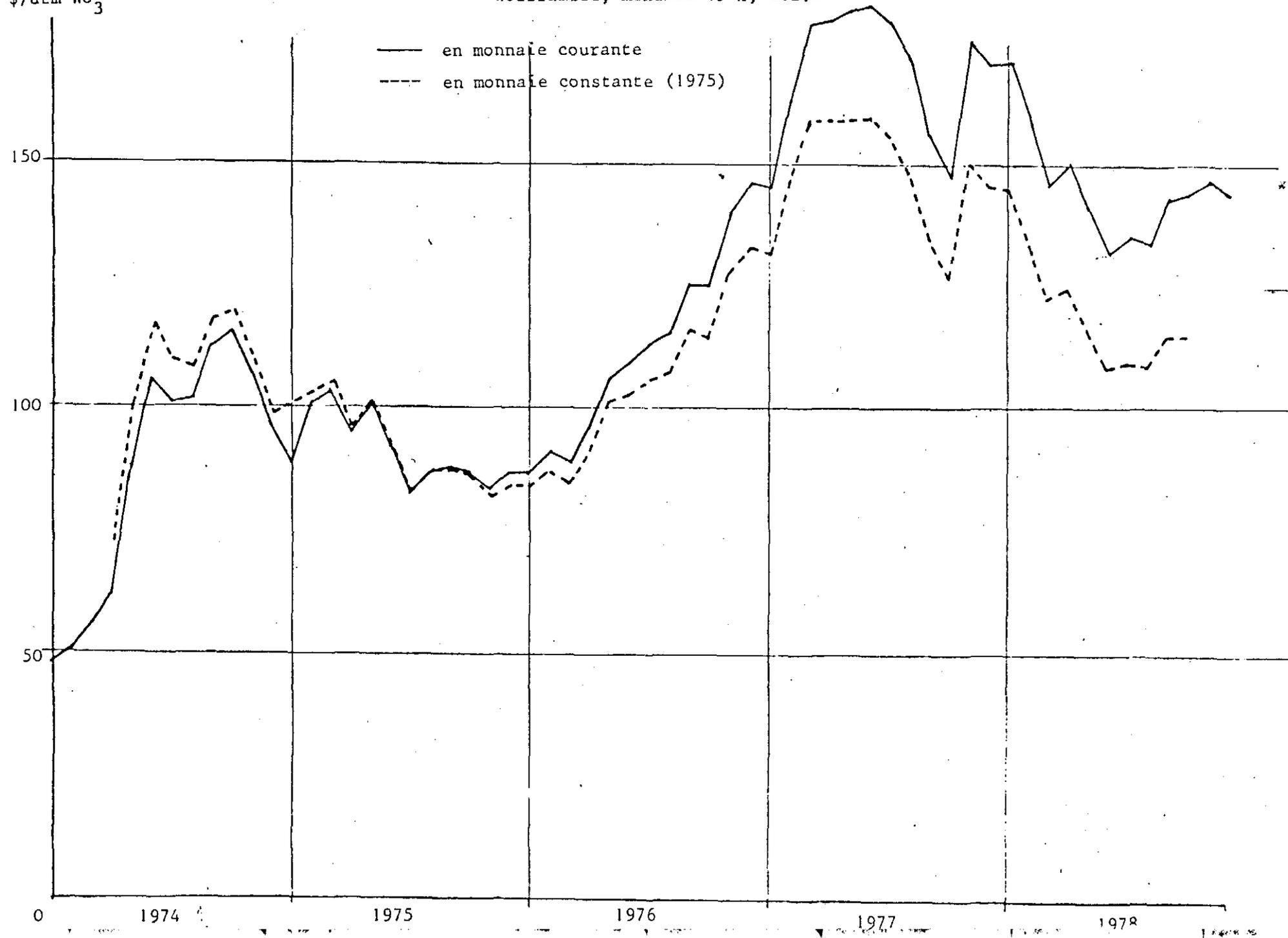
The results are shown in table 3 below.

SG/PMP

Evolution des prix du tungstène de 1974 à 1978

Wolframite, minimum 65 %, c.i.f.

\$/utm WO_3



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Table 3 : WORKABLE RESERVES

Average stope width and grade	Vein Extension	Workable Tonneages			
		Above level 240	Between 200 and 240	Between 200 and 160	Total
1.20 m 1.17 %	HYP. A	41,400	33,400	-	74,800
1.20 m 1.17 %	HYP. B	41,400	36,800	44,500	122,700

CHAPTER 5

UNDERGROUND MINING

5.1. Mining method

5.1.1. Choice of mining method

The geometrical and geotechnical characteristics on the basis of which a working method for Oakleigh Creek can be chosen are as follows :

- . subvertical and regular vein with working widths of between one and two metres ;
- . walls with very good stability.

Under these conditions, two working methods were considered :

- . open overhand or underhand stoping ;
- . shrinkage stoping.

The two methods have comparable costs and each has advantages and disadvantages.

Open underhand stoping was considered in the first studies. The principal argument in its favor was that it eliminated the problem of blockage during drawing from a one meter wide stope which is the major disadvantage of shrinkage stoping.

However after further examination, shrinkage stoping was chosen for the following reasons :

- . recent exploration work showed that the average stope width would often be greater than one meter (see § 5.1.2). Under these conditions, drawing becomes much easier ;
- . personnel and the Department of Mines would at the present time be unlikely to accept the work and safety conditions involved in open stoping ;
- . local miners are much better acquainted with shrinkage stoping methods.

Once shrinkage stoping is decided upon, other choices must be made.

5.1.2. Haulage adit in the vein ore in the foot-wall

This is a classical alternative and it is not necessary to insist upon the advantages of an adit gallery in the foot-wall parallel to the vein. It should only be noted that in the case of shrinkage stoping, this solution would make it possible to eliminate shutes and this facilitates extraction.

The adit in the vein is more profitable in the short term because it can be constructed more quickly and has a lower operating cost. This solution will therefore be used for starting up the mining operations. For the sake of simplification, this is the only solution which will be considered in this report. The haulage adit in the foot-wall could however be more profitable in the long term and a test could be made when a new stope is to be prepared.

Both solutions are adaptable to rail as well as truck haulage.

5.1.3. Geometry of standard mining panel (cf. figure 4)

a) *distance between levels* : 40 m

b) *standard length of panels* : 50 m

These two parameters were chosen so as to give a reasonable mining time. Working with one shift, 7.5 months of working are necessary before beginning to empty the stopes.

c) *average stope width* : 1.20 m

This parameter is obviously not chosen but imposed by the mineralization (cf. § 4.4). During mining operations an effort should be made to keep this width as narrow as possible except for the base sublevel where a minimum of 1.20 m should be kept in order to avoid blockage.

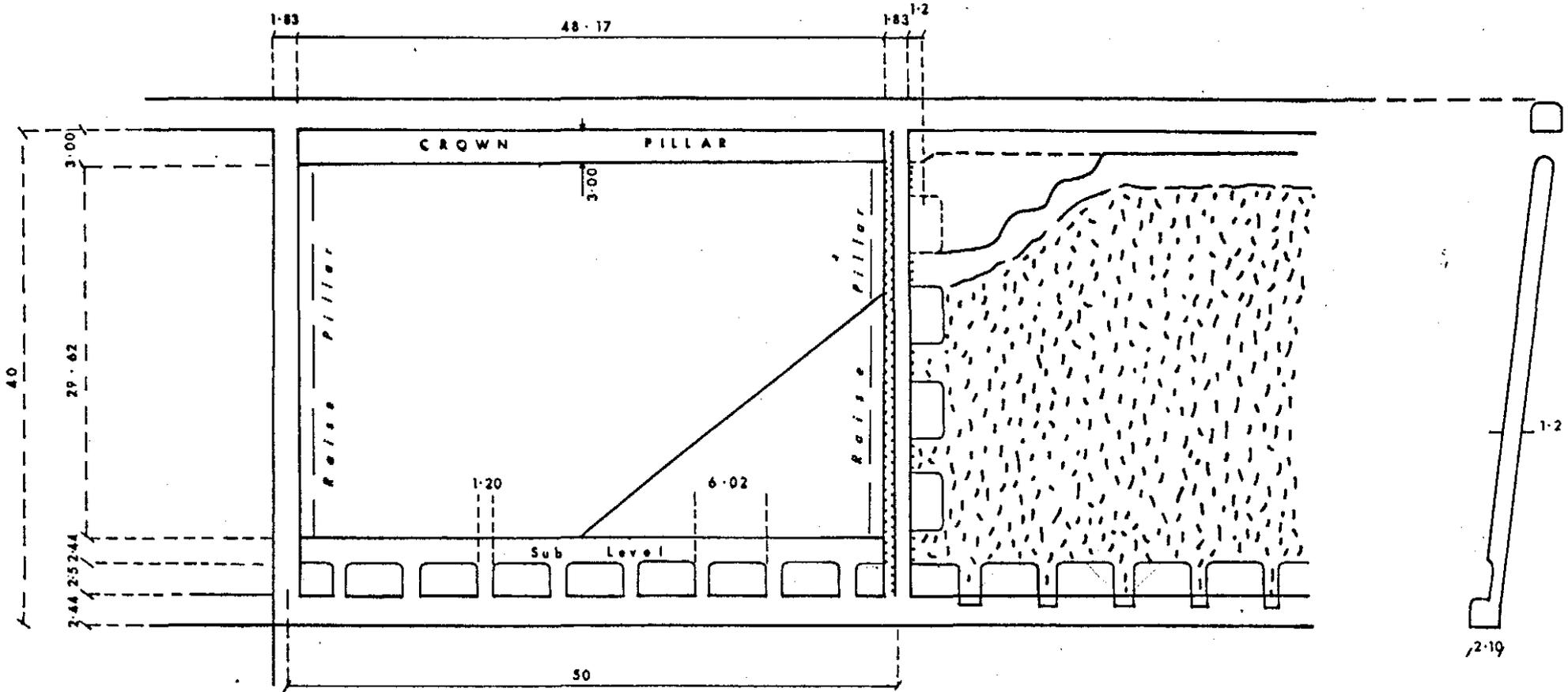
d) *base adit with 2.25 x 2.40 section.*

The height of 2.40 m is imposed so as to allow ducts (ϕ 500) to be laid down easily. The width of 2.25 is an average. There is a width of 2.10 m at level 280 since the hauling will be very short. The width will be slightly greater at levels 200 and 160 where hauling will be trackless.

e) *main raises : section 1.83 x 1.20 m*

The length of 1.80 m makes it possible to divide the raise if necessary into an ore compartment and a passage compartment. These passages will serve as means of access to the stope face. A timber partition will have to be built between the passage and the stope filled with blasted ore as the latter fills up.

It is planned to leave protective pillars on the ore compartment side. These pillars can partly be recovered at the time the raise is abandoned and the possibility of recovering half of these pillars is taken into consideration in this report.



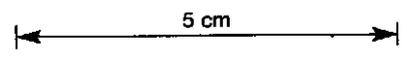
50 METRE STOPE BLOCK DIMENSIONS

STOPE BLOCK	Dimensions	Volume	tonnes
STOPE BLOCK ex main drive	50 x 37.56 x 1.2 x 2.7	6085	tonnes
SUB LEVEL DRIVE	48.17 x 2.44 x 1.2 x 2.7	381	"
MAIN RAISE (1/2 each end of block)	37.56 x 1.2 x 1.83 x 2.7	223	"
SILL PILLAR	48.17 x 2.5 x 1.2 x 2.7	390	"
CHUTE RAISE	8 x 1.20 x 1.20 x 2.5 x 2.7	78	"
CHUTE CONES	8 x 1 x 1 x 1.20 x 2.7	26	"
ADJUSTED SILL PILLAR	390 - 78 - 26	286	"
CROWN PILLAR	48.17 x 2.7 x 1.2 x 3	468	"
PART RAISE PILLAR (half one each end)	1.0 x 1.2 x 29.62 x 2.7	96	"
ORE IN STOPE	29.62 x 1.2 x 46.17 x 2.7	4431	"

ORE DISTRIBUTION in TONNES per 50 METRE STOPE BLOCK

	DEVELOPMENT	IN SITU	RUN OF MINE
SUB LEVEL DRIVE	381		
MAIN RAISE (2 x 1/2)	223		
CHUTE RAISE (8)	78		
CHUTE CONE (8)	26		
ADJUSTED SILL PILLAR		286	
CROWN PILLAR		468	
PART RAISE PILLAR (2) remains		96	
PART RAISE PILLAR (2) extracted			96
ORE IN STOPE			4431
TOTALS	708	850	4527

TOTAL STOPE BLOCK 6085
 tonnage factor 2.7 tonnes per cubic metre.
 above figures exclude main level drives.



SCALE 1:500

Figure 4

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f) *safety pillars at base level*

These safety pillars are shown in fig. 4. Their recovery will be rather hypothetical as the hauling in the vein solution is chosen. Obviously if the haulage adit in the foot-wall had been chosen, no safety pillars would be left above the base level.

5.1.4. Back filling of empty stopes

Since the safety pillars are considered sufficient to protect the levels it is not planned to back fill the empty stopes. However if circumstances make this necessary it would be easy to dump jig waste into the chambers from the upper level.

The corresponding increase in the cost price is \$2-3 per cubic metre of back fill depending upon the transport distance. This operation will however be exceptional and its cost is included in "adit maintenance and miscellaneous".

5.2. Mine organization

The underground workforce of 19 and individual labour costs are as follows :

			<u>Cost per Shift</u>	
<u>Mining</u>	Shift boss	1	1 x 90 x 1.3	117
	Miners	6	6 x 70 x 1.3	546
	Timbermen	2	2 x 70 x 1.3	182
	Steel Sharpener	1	1 x 60 x 1.3	78
	Rail Haulage	1	1 x 60 x 1.3	<u>78</u>
				1001
<u>Decline</u>	Development miners	2	2 c. 70 x 1.3	182
<u>Maintenance</u>	Fitters	3	3 x 55 x 1.3	216
	(1½ in Mill, 1½ underground)			
<u>Absenteeism</u>	Spares	3	3 x 50 x 1.3	105

Camp costs per man are additional at the rate of \$6 per man per day.

5.2.1. Stoping

Normally two men would work in the one stope, one man at each end. At a production rate of 1.25 firings per shift, each man should break 13 tonnes.

Broken ore would be drawn off through the chutes at the bottom of each raise.

Because of the shrinkage factor of 35%, the ore drawn off each stope is expected to be :

$$2 \times 13 \times 35\% = 9 \text{ tonnes per shift}$$

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5.2.2. Development workings

Sub-levels will be commenced from the raise at each end of the block to facilitate ventilation. A double-drum winch and narrow scraper will be used for clearing, with ore being passed down the raise for loading.

Chute raises, of which there are eight in a 50 metre block, will raise up 2.5 metres to the floor of the sublevel.

Chute cones will be cut at each angle point of each chute raise where it meets the sub-level.

Raises will follow the vein and extend vertically from one level to another. Height is approximately 37.5 metres. Raises serve as ore passes, ventilation shafts and access for men, air and water pipes.

Chutes will have air operated doors supported in a timber framework.

Dimensions of these development works and unit costs are shown in the accompanying table. These figures and rates are derived from monitoring the costs and performance of the contractor, and from a detailed study of Central Tasmanian Tungsten's operations September 1978 to January 1979, with particular emphasis on Period 1, 1979.

		Adits	Main Raises	Sub-levels	Chute Raise	Chute Doors
Section		2.29 x 2.44 m	1.83 x 1.2 m	2.44 x 1.2m	1.20 x 1.20	-
Advance/round		1.8	1.8	1.8	1.25	-
Tonneage/metre		8 min. + 7 waste	5.9	7.9	3.9	-
Advance/day		1.6	1.0	1.6	4.0	-
Tonnes of ore/day		12.8	5.9	12.6	15.6	-
* Costs \$/m	Labour	179	260	179	146	\$800 each
	Supplies & Transport	187	158	159	143	
	Total	366	418	338	289	
* Costs \$/day	Labour	286	260	286	NA	
	Supplies	300	158	255		
	Total	586	418	541		

* Does not include Local and Sydney overheads.

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5.3. Loading and hauling

5.3.1. Choice of equipment

The problem was studied and the decisions were as follows :

- . to purchase a 1 cy EIMCO 911 LHD loader (1) and a 3.5 t capacity EIMCO 964 dumper (1) for the exploration work ;
- . before starting production, operations will be executed to install rails at level 240 and to purchase a battery locomotive, 8 side tipping cars and a SCOMA C 150 (1) loader on rails.

The material from level 280 will be loaded in the 964 dumper and dumped in a raise and then reloaded in the rail-cars at level 240.

A 2 cy loader may be used in the decline.

The 911 will serve as reserve loader.

In order to remove the material from levels 160 and 200 in one shift, a second dumper of 4.5 cu. yd. capacity may be necessary. A third dumper is foreseen as spare unit ; the latter can be bought second-hand.

The installation of rails at level 240 was discussed for a long time because it seemed at first that the trackless equipment (which would have to be bought in any case for working the lower levels) could also be used at level 240. The rails were however chosen for the following reasons :

- . the operating cost for rails is lower. For a distance of 250 m it was estimated that hauling with a dumper would cost substantially more. Nevertheless this saving on the operating cost alone would hardly pay for the investment of \$42,000 necessary for installing

(1) This equipment has already been bought.

raills at level 240. The elements given below also played an important role in the dicision ;

- . the purchase of a second dumper can be put off for one year ;
- . ventilation problems would be made easier ;
- . the section for level 240 can be kept narrower.

5.3.2. Organization

As long as level 200 has not begun to produce, one man can handle hauling for levels 240 and 280 but two will be necessary afterwards.

5.4. General mine services

5.4.1. Ventilation

Primary ventilation for levels 240 and 280 will be produced by a small fan at the portal of level 280 which should be closed with a door. No ventilation raise will be necessary. In order to produce $5 \text{ m}^3/\text{s}$ (about 10,000 cfm) a 1.50 kW fan will be more than sufficient.

Primary ventilation for levels 200 and 160 will require the boring of ventilation raises with a $1.83 \times 1.2 \text{ m}$ section and which vents into the adit above it. A fan circulating the air of the mine will be located at the top of these raises. An approximate computation shows that in order to produce $15 \text{ m}^3/\text{s}$, a 10 kW fan will be necessary.

Naturally all dead end galleries in which equipment is moving will need secondary ventilation and $\varnothing 500 \text{ mm}$ ventubes if 2 cy loaders are not used and $\varnothing 600$ if they are used. The installed power of these secondary ventilators will be about 10 kW per level.

5.4.2. Dewatering

It is obviously impossible to know whether there will be water inflows. However at the present time there is no reason to expect that they will be of great extent.

An arbitrary hypothesis was used as a basis supposing that there would be a water inflow of 5 l/s at level 200 and 15 l/s at level 160. The following was deduced :

	level 200	level 160
. effect on operating cost in \$ for supplies per day	5	40
. investment (equipment, installation and sumps)	15,000	25,000

Obviously the investments for levels 200 and 160 should be added.

The 240 and 280 levels are graded such that water flows by gravity to the adit openings.

5.4.3. Compressed air

A 25 m³/mm (900 cfm) compressor unit will be necessary, and the existing 20 m³/mm (700 cfm) can be used as a reserve compressor unit.

The larger unit can be found in good condition for \$30,000.

5.4.4. Electrical energy

As long as there is no need for pumping, the ventilators will be connected to the 69 KVA generator. Additional energy requirements can be obtained from the Mill Power Station.

5.4.5. Miscellaneous work

It is planned to use two men for installing shutes, maintaining galleries and rail track, laying pipes and other miscellaneous work.

5.4.6. Maintenance

One man will be occupied completely with steel sharpening and maintenance of drills.

The rest of the maintenance can be carried out by a mechanic (fitter) plus one half electrician (shared with the mill).

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5.4.7. Supervision

The mine captain alone is sufficient for supervision of the underground work.

5.5 Mine operating cost

Daily expenditure for extraction of 100 tonnes per day.

5.5.1. If all production comes from above the 240 level

Mining	per shift	
. gelignite	157	
. detonators	15	
. steels consumption	76	
. energy	5	
. compressed air	29	
. supplies (drills)	11	
. other supplies (consumables)	20	313
<hr/>		
<u>Rail Haulage</u>		
. loco battery charging	4	
. loco and rolling stock, maintenance	17	21
<hr/>		
<u>Underground Services</u>		
. vent ducts	8	
. air and water pipes	20	28
<hr/>		
<u>Rubber-Tyred Equipment</u>		
. 911B \$13.92 @ 8 hrs	111	
. 964 \$10.15 @ 8 hrs	81	192
<hr/>		
<u>Electric Power 69 KVA</u>		
. cost per hour @ \$10.15 / hr	19	19
<hr/>		
<u>Labour</u>		
. mining (11) includes supervision	1001	
. maintenance (1½)	108	
. Absentee spares (3)	195	1304
<hr/>		
<u>Camp Cost</u>		
. 15½ @ \$6	93	93
<hr/>		
TOTAL COST/SHIFT		\$ 1970
		=====

5.5.2 Additional cost during decline development

<u>Mining</u>		
. gelignite	53	
. detonators	5	
. steels consumption	25	
. energy	5	
. supplies (drills)	10	
. other supplies (consumables)	20	118
<hr/>		
<u>Underground services</u>		
. vent ducts and air/water pipes	14	14
<hr/>		
<u>Labour</u>		
. development miners 2	182	182
<hr/>		
<u>Camp cost</u>		
. 2 x \$6	12	12
<hr/>		
TOTAL COST PER SHIFT		\$ 326

5.6 Investment for mining equipment

At 1st January 1979, the following had been purchased and were in use at the mine :

. Eimco 911B LHD loader	51,369
. Eimco 964 truck	33,573
. Drills and accessories	13,000
. Miscellaneous compressed air equipment	10,000
. Compressor Atlas Copco 700 cfm + freight	11,600
. Scoma loader on rails + freight	4,240
. Miscellaneous equipment and installation	<u>14,914</u>
	<u>\$138,696</u>

Equipment still to be purchased is as follows :

. Drills and accessories, additional	10,000
. second compressor 900 cfm s/h	30,000
. battery loco with spare battery s/h	16,000
. 8 side tipping car (900 l. capacity)S/H	3,200
. double-drum winch and scraper	6,000
. ventilators and ducts	3,000
. initial spares stock	6,000
. freight on items	4,000
. compressor shed	8,000
. Kibuka workshop	4,000
. rails and installation	20,000
. site works	10,000
. miscellaneous	<u>10,000</u>
	<u>\$130,200</u>

CHAPTER 6

UNDERGROUND MINING DEVELOPMENT

Before the starting up of mine production an exploration programme was planned in order to further explore levels 240 and 280 as well as a minimum amount of preparation for the mining panels.

After mine production has started up, a development programme is planned for levels 200 and 160. The execution of this programme will start in the beginning of 1980.

6.1. Development before starting of mine production

6.1.1 Completed work

Between 5th April 1978 and 17th October 1978 a local contractor carried out the following :

- . stripping of the first 150 m of level 240 to 9' x 8' ;
- . levelling of level 240 ;
- . digging of 100 metres of adit at level 240 on section of 8' x 7'6". At the end of the contract, the total length of the adit reached 252 m.
- . digging of 100 m of adit at level 280 on scale of 8' x 7' ;
- . digging of a 37.7 m raise between levels 240 and 280.

The execution of this contract along with some associated works cost \$143,895 (paid to the contractor) plus about \$15,425 of supplies paid directly by the joint venture.

Between November 1978 and March 1979 more development work was carried out by Serem :

- 52 m digging in the 240 m adit
- 65 m digging in the 280 m adit

6.1.2. Work to be carried out

At the beginning of March 1979, the driving on the 240 level had been terminated at approximately 1695N, whilst the 280 level was driving ahead at 1756N. The ore on the existing No. 1 stockpile is estimated at 8,000 tonnes at the end of the exploration period.

6.1.2.1. Mining Programme 1979

The 1979 programme is aimed at providing four stopes ready for production at 1st January, 1980.

These are, in order of preparation :

Block 240/101	Stope length	29 m
Block 240/103	Stope length	50 m
Block 240/105	Stope length	50 m
Block 280/104	Stope length	50 m

During the preparation of these stopes, narrow width (1.2 metre) development ore will become available from raises, sub-levels and chute raises, and will be stocked on a separate No. 2 stockpile.

Initially the mill's requirement of 100 tonnes per day will be met as shown on the following Production Plan Summary, i.e.

Daily stope production	44 tonnes
No. 1 stockpile, existing	25 tonnes
No. 2 stockpile, 1.2 development	<u>31 tonnes</u>
	100 tonnes

The 1979 programme, made up of exploration and development works, still to be completed after 1st March, is set out below to show costs and yields of development tonnage.

Exploration

Order	Works	Cost	Development Tonneages	
			Existing Stockpile No 1	Narrow Width Stockpile No2
1	280 L. Complete 50 m to co-ordinate 1705N, rate \$366	18,300	751	-
Development				
1	Prepare Block 240/101	37,000	-	659
2	Construct four raises 240 L. to 280 L. Coordinates 1705N 1755N, 1805N, 1855N	71,312	-	892
3	Prepare Blocks 240/103 and 105 two @ \$30,066	60,132	-	970
4	Prepare Block 280/104	65,178	-	982
*	Total development cost	233,622	Total No 2 Stockpile	3,503 t.

* Does not include Local and Sydney overhead.

Detailed dimensions, costs and estimated times are set out at the end of this section.

The Raise Programme will take approximately 25 weeks with one party, and it will be necessary to introduce a second mining party about May 1979, and a third party about July 1979.

However, it may be possible to reduce this to seven weeks by the use of a raise boring contractor, and this is set out as an alternative in the detailed part at the end of this section.

6.1.2.2. Details of stope development1. Block 240/101

Raise	Height	23 m @ 1.2 m advance/shift	=	20 shifts
Sub-level drive (top)	Length	29 m @ 1.6 m advance/shift	=	19 shifts
Sub-level drive (bottom)	Length	20 m @ 1.6 m advance/shift	=	19 shifts
Chutes 5 x 2.5m	Total	12.5m @ 4.0 m advance/shift	=	4 shifts
TOTAL TIME				13 weeks = 62 shifts

Costs

Raise	1935N	23 m @ \$418	=	9,614
Sub-level drive (top)		29 m @ \$338	=	9,802
Sub-level drive (bottom)		29 m @ \$338	=	9,802
Chute raises 5 x 2.5		12.5m @ \$289	=	3,613
Chute Cones 5		16.2m @ \$9.85	=	160
Chutes 5		5 @ \$800	=	4,000
				<u>36,991 say \$37,000</u>

Tonneages from development

Raise	1935N	23 x 1.2 x 1.83 x 2.7	=	136 tonnes
Sub-level drive (top)		29 x 2.44 x 1.2 x 2.7	=	229 tonnes
Sub-level drive (bottom)		29 x 2.44 x 1.2 x 2.7	=	229 tonnes
Chute raises		5 x 1.20 x 1.20 x 2.5 x 2.7	=	49 tonnes
Chute cones		5	=	<u>16 tonnes</u>
				659 tonnes

2. Raise Programme

Requirements 4 Vertical Raises @ 37.56 metres from 240L to 280L
Position 1795N, 1755N, 1805N, 1855N

CTT Personnel (\$ 418 per metre), plus contract drilling - pilot hole,

Cost	4 x 37.56 x 418 =	62800
Contract Drilling	4 x 37.56 x \$ 50 =	7512
Mobilization Driller		= 1000
		<u>\$ 71,312</u>

Estimated Time	37.56 @ 1.2 m. per shift =	31.3 shifts
	Four raises =	125.2 shifts
		= 25 weeks (one party)

Should time become a factor in the 1979 Mining Programme, the possibility of carrying out the Raise Programme by contractor using raise boring equipment was investigated.

R.V.C (Pacific) Pty Ltd was contacted and could supply a machine and an operator, with the additional labour and support facilities being provided by C.T.T.

The time estimate is 7 weeks and costs are estimated from R.V.C's quotation as follows:

Contract Raise Boring: R.U.C. (Pacific) Pty. Ltd.,

Time	7 weeks 2 60 hrs x \$ 35.88 per hour =	15069
Metres	4 x 37.56 x \$ 302 per metre =	45372
Mobilization		= 12000
Power Requirement	265 KVA	
Hire of Generator	7 weeks x 265 x \$ 3 per KVA p.w =	5565
Hire of Compressor	7 weeks x 7 days x \$ 85 per day =	4165
Hire of Crane	Arrival/Unload	
	Departure/Load	400
Pad Preparation	4 x \$ 100	400
		<u>\$ 82971</u>

Plus CTT labour and Facilities

3. Standard 50m Block Details
Blocks 240.103 and 105

Item	Length	Rate/m	Cost/\$	Tonnes	Time/shifts
Sublevel Drive	4817	\$ 366	17630	381	30
Main Raise	Shown in Raise Programme				
Chute Raise	8 x 2.5m	\$ 289	5780	78	5
Chute Cones	8 x 3.5 tonnes	9.85 per/tonne	256	26	5
Chutes	8 @ \$800		6400		with cones
Total for one block			30,066	485	40
Total for two blocks			60,132	970	80

Time:

For Block 240.103 40 shifts = 8 weeks
 For Block 240.105 40 shifts = 8 weeks

4. Block 280/104

For ventilation purposes, raises of 45 m at 1755 N and 36 m at 1805 N will need to be constructed.

Costs

Raise 1755 N	48 m @ \$ 418	20,064
Raise 1805 N	36 m @ \$ 418	15,048
Sublevel Drive	48.17m @ \$ 366	17,630
Chute Raise	8x2.5m @ \$ 289	5,780
Chute Cones	8@3.25 tonnes @ 9.85	256
Cost of Chutes	8 @ \$ 800	6,400
		<u>\$ 65,178</u>

Development Tonnes

Raise 1755 N	48 x 1.2 x 1.83 x 2.7	284 tonnes
Raise 1805 N	36 x 1.2 x 1.83 x 2.7	213 "
Sublevel Drive	48.17 x 2.44 x 1.2 x 2.7	381 "
Chute Raise	8 x 120 x 120 x 2.5 x 2.7	78 "
Chute Cones	8 x 1 x 1 x 1.20 x 2.7	26
		<u>\$ 982 tonnes</u>

Time

Raise 1755N	48m @ 1.2m advance per shift	= 40 shifts
Raise 1805 N	36 m @ 1.2m " "	= 30 "
Sublevel Drive	48.17 @ 1.6 m " " "	= 30 "
Chute Raise	20 m @ 4 m " " "	= 5
	Total Time 21 weeks=	<u>105 shifts</u>

6.2 Mining Programme after 1979.

The general plan of development over the life of the mine is set out in Appendix 1-2, Stope Tonnes per Day - 50 metre blocks.

This shows that the decline should commence at the end of 1979 to enable blocks below the 240 level to be mined by mid 1981.

However, the drilling planned in 1979 to test below the 160 level will determine this programme.

A feature of the general plan is the need to supplement stope production from the narrow width development ore in Stockpile No.2 to meet the mill's requirement over the greater part of the mine's life.

This problem arises because of the difficulty in securing sufficient working faces, a problem common to all narrow vein mines, with limited extensions and acces.

For the time being it is assumed that the digging of the decline to the 160 m level will cost \$350,000 and that \$110,000 of supplementary equipment is needed to carry out the work.

These amounts are considered as capital expenditure and will be spaced over the three years following the beginning of work on the decline :

\$150,000 first year
\$220,000 second year
\$ 90,000 third year

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PRODUCTION PLAN SUMMARY.TONNES PER DAY.

PERIOD	1980													1981												
	1	2	3	4	5	6	7	8	9	10	11	12	13	1	2	3	4	5	6	7	8	9	10	11	12	13
STOPES	44	44	44	44	44	44	44	44	44	44	61	61	61	61	61	78	78	78	78	78	78	78	78	78	78	78
Nº1 STOCKPILE EXISTING	25	25	25	25	25	25	25	25	25	25	25	25	25	-	-	-	-	-	-	-	-	-	-	-	-	-
Nº2 STOCKPILE NARROW WIDTH DEVEL. ORE	31	31	31	31	31	31	31	31	31	31	14	14	14	39	39	22	22	22	22	22	22	22	22	22	22	22
TOTAL	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100

APP. 1-1

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CHAPTER 7

THE MILL

7.1. Mineral processing tests on ore

Only tests made after January 1978 will be discussed since these are the most significant ; besides, the tests made previously had comparable results.

27 samples of about 2 t distributed over 150 m were obtained by blasting in the adit roof. The samples thus consist of vein and wall rocks.

After crushing and successive reductions a composite sample, reduced to - 9.5 mm, containing 0.83 % WO_3 and 0.033 % Sn, was obtained. All the tests described below were made with this sample.

7.1.1. Mineralogical study (Amdel report n° 2)

A sample was classified in seven grain size fractions. With each fraction there was dense medium separation with four liquids with different densities.

The main results are as follows :

- . good liberation of wolframite at - 6.4 mm (80 % liberated in fraction - 6.4 + 3.2 mm) ;
- . very good liberation of wolframite at - 0.5 mm (95 % liberated in fraction - 0.5 + 0.25 mm ; 98 % liberated in fraction - 0.25 + 0.125 mm) ;
- . very favorable characteristics for gravimetric preconcentration with fraction - 9.5 + 0.5 mm (for a liquid of specific gravity 2.8 the floats of this fraction in relation to run-of-mine represent 77.7 % of the weight and 1.6 % of the WO_3 content).

7.1.2. Heavy media cyclone tests (Amdel report n° 3)

The tests concerned the - 9.5 +0.5 mm fraction (78.5 % of run-of-mine) with several specific gravities. In the best test, floats were obtained which in relation to run-of-mine contained 76.1 % of weight and 3 % of the WO_3 content. The sinks had a grade of 15.6 % WO_3 .

7.1.3. Low intensity magnetic separation tests (Amdel report n° 4)

A disk magnetic separator was used to process the concentrate from the preceding test. The results were as follows :

	Weight %	Grade		Distribution %	
		WO_3	Sn	WO_3	Sn
Mags	42.1	34.8	0.055	93.7	6.0
Non-mags ,	75.9	1.69	0.63	6.3	94.0
Feed	100	15.6	0.39	100	100

An attempt was made on the non magnetic fraction to concentrate the tin by floating the sulphides and then applying heavy medium separation. The results were not very good and it was concluded that tin recovery was not economically feasible with the tin content of the sample.

The magnetic concentrate was analysed chemically and mineralogically. Mineralogically, the following were found : 45 % wolframite, 30 % tourmaline, 5 % quartz, 3 % muscovite, 3 % siderite, etc. The results of chemical analysis include the following :

WO_3	34.9 %
As	0.125 %
S	1.76 %

The other impurities for which there are penalties are in very small quantities.

The opinion of specialists is that there will be no difficulty in obtaining a marketable concentrate from a product of this kind by simple tabling. Sulphur and arsenic contents could however be slightly high to get a first quality product (< 0.75 % S and < 0.2 % As). In this case some flotation of the sulphides from the concentrate could be considered.

7.1.4. Wet high intensity magnetic separation tests (Readings)

Two tests were carried out involving passing run-of-mine reduced to - 0.5 mm through a wet high intensity magnetic separator. The first magnetic fraction was passed twice.

In both cases a preconcentrate with excellent recovery was obtained. The results of the second test were as follows :

	% weight	WO ₃ grade	WO ₃ distribution
Mags	4.7	18.1	96.8
Non-mags	95.3	0.03	3.2
Feed	100	0.88	100

It was not possible to increase the WO₃ grade of the magnetic fraction because of the abundant tourmaline. However by applying heavy liquid separation of density 4, a concentrate with the following composition was obtained :

	%
WO ₃	71.9
As	0.31
S	0.56
Bi	0.12
Zn	0.17
Mn	4.7
Fe (in form of Fe ₂ O ₃)	18.3
P	0.03
Sb	< 0.01
Pb	< 0.1
Cu	< 0.06
Mo	< 20 ppm

These results confirm the low intensity separation tests. The arsenic and sulphur grades are near the penalty limits but they can easily be lowered by flotation of the sulphides. The other impurities are in small quantities.

Conclusions to be drawn from tests

The Oakleigh Creek ore is easily concentrated by a combination of gravimetric and magnetic methods. There are several possible methods all of which should give good results.

7.2. Engineering studies

7.2.1. Rough estimates

Several engineering firms have given rough estimates based only on summary flow sheets which differed in each case. Without reviewing all of them it will however be noted that the range was very wide i.e. from \$ 383,000 to \$1,700,000.

The first estimate from Dravo for a gravimetric mill (jigs + tables) was \$ 736,000.

7.2.2. Dravo conceptual engineering report

In June 1978 Dravo was asked to prepare a draft project for the mill. It was understood that the cost should be estimated in terms of a turn-key, type operation for which Dravo would be the main contractor.

At the beginning the flow-sheet used was the conventional gravimetric mill with jigs and tables. Later because of the excellent results of WHIMS (1) two other designs were also studied :

"ALTERNATIVE 1" Preconcentration with jigs of fraction - 6 + 0.5 and then grinding of jig concentrates to - 0.5 mm. This ground concentrate with the natural - 0.5 mm fraction is passed through the WHIMS followed by tabling.

"ALTERNATIVE 2" Grinding of run-of-mine to - 0.5 mm followed by passage through WHIMS and tabling.

(1) WHIMS : wet high intensity magnetic separation

The results of the study were as follows :

	Investment	Operating cost	Expected recover
Gravimetric mill	\$ 1,715,114	9.78	80-85 %
"Alternative 1"	\$ 1,593,483	8.44	85-90 %
"Alternative 2"	\$ 1,645,984	8.06	90-95 %

The method was chosen on the basis of this comparative study and the gravimetric mill which is the least interesting from all points of view was rejected. Alternative 2 which is interesting because of the high recovery level was rejected because this method produces only fine tailings, while in Alternative 1, 75 % of the tailings are in the form of gravel which is much easier to store at the Oakleigh Creek site. The latter method was therefore chosen.

The estimate for Alternative 1 was considered much too high and was not accepted by the joint venture. Dravo later proposed a simplified version of Alternative 1 called Alternative 1A with the following breakdown :

. main equipment	328,000
. civils	51,000
. steel and assembly	82,000
. building	89,000
. conveyors	41,000
. piping	28,000
. electricity	70,000
TOTAL DIRECT COST	\$ 689,000
. engineering	156,000
. field supervision and expenses	199,000
. overhead	87,000
. contingencies	22,000
. profit	34,000
. miscellaneous	26,000
	\$ 1,213,000

with a guaranteed maximum of \$ 1,286,000.

This offer was also rejected by the joint venture.

7.2.3. Anderson study now in progress

The SEREM asked the Anderson company to prepare another draft project based upon the following hypotheses :

- the selected flow-sheet is the Dravo "Alternative 1A" although it may contain some minor modifications especially as concerns the addition of a second jig in series with the first.
- The metal structures, the arrangement of the conveyors and the building will be completely revised. Large savings are possible in relation to the Dravo design.
- The general organization of the construction will be as follows :

- . Anderson will be responsible for the engineering design ;
- . SEREM will be responsible for the detailed planning of operations, ordering of the equipment and the choice of second hand equipment in association with Anderson and, if necessary, with the aid of consulting engineers ;
- . SEREM will select a Tasmanian main contractor who will be responsible for subcontractors ;
- . the principal contractor will supervise the works on the site and a consulting engineer may also participate in some phases of the work ;
- . SEREM will accept the finished work and start up operations with the aid of Robertson Research.

ANDERSON had finished the draft project by the end of January, and at this date a detailed cost estimate was available.

7.3. Proposed flow-sheet

The flow-sheet of Dravo "Alternate 1A" presented in appendix 2-1 will not be described in detail and only the main characteristics will be given.

7.3.1. Processing capacity

Nominal capacity : 15 t/h i.e. 100 t/shift.

Since most of the equipment will be overdimensionned so that it will be possible to increase this capacity by 20 or 30 % by reinforcing the screening which will probably be the bottle neck.

7.3.2. Feed particle size

The primary ore bin will have a 25 mm grizzly (primary crusher 10" x 24").

7.3.3. Quality of final concentrates

Dravo estimates that the concentrate after flotation could have a grade of approximately 70 % WO_3 which will be considered 65 % in order to reduce the possibility of error. If necessary, one small flotation cell will be added to the circuit in order to eliminate arsenic and sulphur from the table concentrate.

7.3.4. Recovery

Dravo expects recovery of between 85 % and 90 % for "Alternative 1" and "slightly lower" recovery for "Alternative 1A". 86 % will be used in the economic computations.

7.3.5. Tailings

The jigs will reject about 11 t/h of + 0.5 - 6 mm gravel.

The magnetic separator will reject 3.6 t/h of - 0.5 mm silty sands.

The ultrafine and sulphide tailings from flotation will be negligible (less than 100 kg/h).

7.3.6. Installed power

The total installed power is 240 kW.

A 350 KVA generator will produce surplus power of about 50 kW which will be available for the mine.

7.3.7. Anderson study

The flow-sheet of ANDERSON is presented in Appendix 2-2.

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7.4. Problems linked to site

7.4.1. Location of the mill

The mill will be located on the glacial terrain terrace in the immediate vicinity of the portal of level 240 (cf. sketch of general location - Appendix 7).

7.4.2. Water supply

Reid Creek can cover the water needs of the mill most of the time. However its flow is insufficient in the summer months, and provision has been made for the tails water to be returned to storage. Additional make-up water will be provided by electric pump and pipeline from the Forth River.

7.4.3. Storage of tailings

The choice of methods and sites for the storage of tailings has been made with the aid of the WLP firm which is specialized in this field.

An area located 100 m to the NE of the mill was proposed for the silty sands (25 % of the total). With an initial investment of \$ 15,000 it will be possible to build a dam covering 3.3 years of needs which will have a minimal environmental effect. This dam could be extended to the east but this would affect the forest. A later increase in dam capacity, covering four additional years, would have a reasonable effect on the environment. Its cost is estimated at \$ 20,000.

Gravel storage is much simpler since no dam is necessary. The gravel will be loaded on trucks and dumped. The end of the platform on which the mill is to be built could hold more than three years of production and the gravel would later be dumped in the low swampy zone located about 300 m to the SE of the mill. A dump height of about 20 m will be available there, and a practically unlimited storage capacity.

The gravel produced will be of excellent quality. Because of transport cost it cannot be marketed but it can be used in situ for many purposes i.e. dam construction, road maintenance and filling of mine stopes if necessary.

7.5 Operating cost

These costs are mainly taken from the Dravo report.

Cost per day (in \$)

. supervision	80	
. production labour (4 men)	280	
. maintenance labour (1.5 men)	108	
A TOTAL LABOUR	<u>\$468</u>	
. generator	124	
. spare parts	100	
. maintenance	110	
. tailings disposal	80	
. share of camp costs	45	
B TOTAL MISCELLANEOUS	<u>\$459</u>	
TOTAL A + B		\$927
. modifications		\$100
TOTAL		<u>\$1027</u>
		<u>=====</u>
Cost per tonne		\$10.27

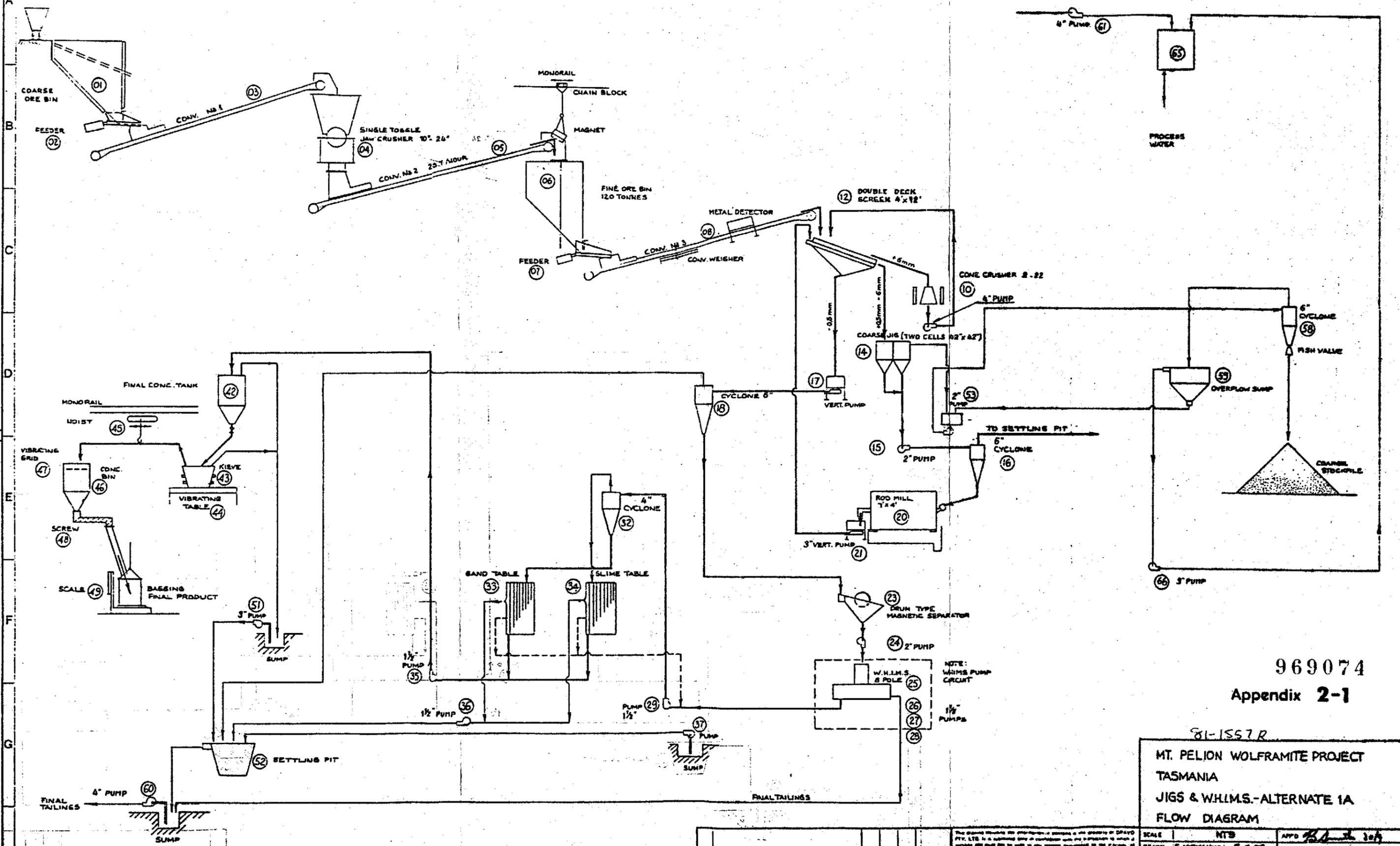
7.6 Investment

At the present time, we can make the following estimate of the cost of the mill based on Anderson's conceptual engineering study :

. direct cost of equipment (transport included)	355,220
. civil works and construction	259,860
. electrical, piping, etc.	153,160
	<hr/>
Total	768,240
. design, engineering, purchasing	81,000
. supervision of construction, purchasing, etc.	71,040
	152,040
	<hr/>
	920,280
. tailings disposal	15,000
. studies and tests	43,887
	<hr/>
Grand Total	\$ 979,167
	<hr/> <hr/>

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Appendix 2-1

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MT. PELION WOLFRAMITE PROJECT
TASMANIA
JIGS & WHIMS.-ALTERNATE 1A
FLOW DIAGRAM

This drawing shows the proposed construction and operation of DRAYO PTY. LTD. It is a preliminary drawing and is subject to change without notice. It is not to be used for any purpose other than that for which it was prepared. The drawing is not to be copied, reproduced or otherwise used without the written consent of DRAYO PTY. LTD. The drawing is not to be used for any purpose other than that for which it was prepared.

SCALE	NTS	APPD	<i>[Signature]</i>
DRAWN	E. WILKINS	5-9-78	
CHECK'D	AF	14-9-78	
SECTION	MECH	7136	
Dravo		DRAWING NUMBER	7136/22
		REV	0

REV.	DESCRIPTION	BY	CHK	AP	DATE	ISSUE No.	DATE PRINTED

073

CHAPTER 8

INFRASTRUCTURE

076

8.1. Access road

The Oakleigh Creek mine is linked to the Lemonthyme powerstation by 26 km of forest road. The first thirteen kilometers are regularly used and are maintained by lumber companies. Their quality is considered acceptable for limited traffic.

Two contracts have already been signed in order to improve the quality of the last thirteen kilometers :

- . a \$ 35,000 contract which was executed before the exercise of the option and was paid by SEREM. (It will therefore not be taken into account in the investment calculations.)
- . a \$ 27,428 contract which was executed in June-July 1978.

These two contracts had effective results and now the road can be used under all weather conditions. The main work can be considered completed. Taking into account some additional detail work and maintenance during the development period, for an amount of \$6,000, the total investment paid by the joint venture for the road construction will amount to \$33,428.

Further, \$10,000 will then be allowed on the operating cost for road maintenance and various types of earthworking.

8.2 CAMP

The major part of the campsite has been cleared and at 1st March, 1979, two units ex: Kibuka and a caravan have been installed together with a septic tank.

Final costing is estimated as follows:

	Installed	Still to come
1 x 5 Unit Bunkhouse ex Kibuka	4000	
4 x 6 Unit Bunkhouse, de Jong		39,800
1 Caravan	4200	
1 Wash car		20,000
1 Kitchen-Dining Room ex Kibuka	4000	
1 Recreation Room		15,000
1 Works Barracks		5,000
1 Office		8,000
Transport of Unit 8 @ \$500		4,000
		<hr/>
Total Units Delivered	12,200	91,800
69 KVA Generator	5327	
Miscellaneous Equipment		7,000
Site Work:		12,000
		<hr/>
	17,527	110,800
Total costs camp		128,327

8.3 VEHICLES

Two vehicles and a tractor have been purchased, whilst at 1st March, 1979, the vehicle position is as follows:

	Purchased	Still to come
2 4 x 4 Subaru Cars	6,500	6,500
1 Holden Truck	4,000	
1 10 Ton Truck S/H		10,000
1 F.E. Loader S/H (2 c.y.)		18,000
1 Tractor	1,000	
1 Ambulance S/H		5,000
<hr/>		
1 4 x 4 Toyota S/H		5,000
1 Bus S/H		10,000
	<hr/>	<hr/>
	11,500	54,500
	<hr/>	<hr/>
Total costs vehicles		66,000

The loader and 10 t truck will be used mainly for removal of jig tailings and will not be operated for more than one hour per day.

8.4. Electrical energy

The solution consisting of hooking up with the Lemonthyme power station has not been studied in detail but it seems unlikely that the cost can be amortized. It is supposed that power will come from two electrical generating units :

- . 1 unit with a capacity of 69 KVA for the camp. Its cost is included in that of the camp (cf. § 8.2)

- . 1 unit with a capacity of 400 KVA for the mill, work site and mine ventilation (cf. § 7.3.6). A new generator of this type and the installed cost is estimated to be \$ 65,000.

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8.5. Total infrastructure investment

. access road	33,428
. camp	128,327
. vehicles and motorized equipment	66,000
. generator	65,000
TOTAL	<u>\$ 292,755</u>

CHAPTER 9

GENERAL EXPENSES

082

9.1. Expenses included in operating costs

Daily expenses can be broken down as follows :

. mine manager	92	
. mine clerk	52	
. cook	78	
. miscellaneous	12	
. supervision by project manager	27	
. exploration and grade control	23	284
TOTAL LABOUR		<hr/>
. staff catering (\$6/man/day)	18	
. vehicles	60	
. supplies	12	
. freight and cartage	17	
. travel and accomodation	35	
. rents (executive housing at Sheffield)	20	
. stationery and communications	25	
. miscellaneous	32	219
TOTAL		<hr/> \$ 503 <hr/>

9.1.2. Transport of concentrates to Burnie

It is necessary to allow an average of \$ 43/day for the transport and packing of 12 t of concentrates every eight working days.

9.1.3. Exploration

Allowance should be made essentially for surface reconnaissance and diamond drilling, of \$ 25,000/year i.e. \$ 100/day.

More extensive workings have to be justified by their own profitability, and these will be paid for by investment credits which are not taken into account here.

9.1.4. General expenses Sydney

The daily expenses break down as follows :

a) SEREM administration expenses

. managerial	65
. accounting	48
. secretarial, draughting and miscellaneous	28
TOTAL labour	141
. travel, accomodation	33
. communications and stationary	30
. miscellaneous (insurance, lease fees, etc.)	45
TOTAL supplies and services	108

TOTAL SEREM

249

b) TRIAKO-BUKA administration expenses

25

TOTAL a + b

\$ 274

The royalty to be paid to Scamander is included in operating cost although it is only paid for four years i.e. \$ 23,333 per year, say \$ 100 per day.

The total per day is as follows :

. administration expenses	274
. royalty	<u>100</u>
<u>TOTAL GENERAL EXPENSES SYDNEY</u>	<u>\$ 374</u>

9.2 Expenses included in capital cost

The expenses given below include expenditure either incurred or committed up to the present, plus proposed expenditure from the present up to the starting of production which is planned for 1/1/80.

9.2.1. On site general expenses

(Expenses connected with the construction of the mill are excluded.)

. camp operation expenses	6,000	
. cook (six months)	9,360	
. maintenance of vehicles	6,000	
. miscellaneous	3,000	
TOTAL		24,360

9.2.2. Technical services

. mine manager	16,200	
. project manager	6,000	
. supervisor for maintenance	16,000	
. surveyor	2,000	
. other expenses	2,000	
TOTAL		42,200

9.2.3. Local administrative expenses

. mine clerk	10,500
. travel, accommodation	1,000
. rents	4,500
. communications and supplies	2,000
. miscellaneous	1,120
TOTAL	19,120

9.2.4. General expenses Sydneya) Joint venture administration (1)

1978 administrative expenses	27,984
1979 payments to Scamander/Louisa	20,000
1979 other expenses Serem/Amdex	28,000
TOTAL	75,984

b) Technical expenses

Labour :	managerial (2)	9,600
	technical	9,600
	travel, accommodation	9,200
	draughting	4,000
	miscellaneous	2,000
TOTAL		34,400

Total joint venture administrative and general expenses included in capital cost : \$ 196,064

- (1) It is agreed that for operating costs and investments SEREM will charge only for actual expenses and services without any additional amounts as a management fee.
- (2) The cost of the project manager is distributed between the technical services and SEREM administrative expenses depending upon whether the activity is specifically technical or is linked to managerial functions.

9.3. Cost of delayed starting up

The general expenses to be included in the investment which are estimated in § 9.2 are based upon the hypothesis that production will begin on 1/1/79. It is easy to calculate the additional cost for each additional month of delay apart from all additional work :

. technical services	5,500
. mine operating expenses	3,600
. local administrative expenses	950
. SEREM administrative expenses Sydney	2,950
. TRIAKO-BUKA administrative expenses	<u>500</u>
TOTAL	§ 13,500

CHAPTER 10

LABOR

089

10.1. Labor cost10.1.1. Social security

a) <u>social security payments</u>	
. "worker"s compensation"	12 %
. pay roll tax	<u>5 %</u>
TOTAL	17 %

b) <u>payed holidays</u>	
. workdays per worker	215
. public holidays	15
. annual vacations	20
. sick leave	<u>10</u>
PAYED DAYS	260

. Days payed/days worked 1.209

Total overheads ratio : $1.17 \times 1.209 = 1.415$

This ratio is pessimistic since the pay for holiday is lower than pay for work days and accordingly a figure of 1.30(labor) or 1.35(staff) has been used.

10.1.2. Manshift cost

On the basis of available data for Tasmanian mines the gross 1979 salary for an 8 h shift has the following range :

. workers with fixed salaries	\$ 28- 35
. drillers with fixed salaries	\$ 35- 50
. miners on contract	\$ 40-100

The workers which already have been hired at the mine earn \$ 4 per hour.

According to this data and taking into account an increase for the geographical isolation of the mine the following table was prepared :

	Average gross salary per 8 h shift	Average cost per 8 h shift
Mine captain		
Miners	\$ 70	\$ 100
Assistant miners		
Haulage men		
General underground services	\$ 56.5	\$ 80
underground maintenance		
Mill head		
Mill operators	\$ 49.5	\$ 70
Maintenance per day		
Driver and cook		

The salaries given above are definitely higher than those usually paid in Tasmania except as concerns the more highly paid miners working on contract. However, in this case the norms are much above the norms adopted in this report. If contracts of this kind are possible, all involved will profit.

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10.2. List of personnelUnderground

. blasting (3 teams)	6	
. development work	2	
. haulage (rail and trackless)	1	
. general mine services	1.5	
. mine maintenance	2	
. sharpening drill steels	1	
. supervision	1	
	<hr/>	
TOTAL UNDERGROUND		14.5

Mill

. operators	3	
. dam and pumps	1	
. maintenance	1.5	
. supervision	1	
	<hr/>	
TOTAL MILL		6.5

Driver-warehouseman	1	
Cook-camp maintenance	1	
Mine clerk	1	
Mine manager	1	
	<hr/>	

TOTAL STAFF PRESENT		25
Absenteeism		3
		<hr/>
TOTAL STAFF		28

10.3. Availability of labor

It was at first feared that it would be difficult to find personnel ready to work without their families in as isolated location as Oakleigh Creek but the requests for employment already received by our mine manager are encouraging.

The fact that labor is relatively easy to find is explained by several factors favoring the project:

- . the north of Tasmania is an economically depressed area;
- . this region always supplied manpower to the large mines of western Tasmania. Most of these workers keep their homes and families in Northern Tasmania and return for weekends. Under these conditions a job at Oakleigh Creek which is much closer to their homes may seem to be an interesting alternative.
- . many young Tasmanians like life in the bush.

There is still a problem in finding labor which is at once qualified and reliable.

CHAPTER 11

ENVIRONMENTAL PROBLEMS

094

This project has especially complex environmental problems because Oakleigh Creek is located near the Cradle Mountain National Park which is one of the most popular in Australia.

The local population and the Department of Mines are definitely favorable to the project because of the depressed economic situation in northern Tasmania.

The Department of the Environment has received a report on the environmental impact of the mine. Up to the present time the contacts of the joint venture and this department have generally been positive. The department does not seem to wish to penalize systematically the project but it will certainly follow its development very attentively.

At this stage it can be supposed that the problems related with the environment will be solved provided that all reasonable precautions are taken i.e. appropriate dam construction, correct routes for roads and tracks, water recycling, sound proofing for compressors and generators, etc.

It is certainly necessary to satisfy the Department of the Environment, but the main point is to continue to present a good image to public opinion whose reaction could be the decisive factor.

CHAPTER 12

GENERAL PROJECT DEVELOPMENT PROGRAM

The program given below shows one of the ways the erection of the camp, mining work and the construction of the mill can be coordinated so as to begin production on 1/1/80.

Paragraph 6.1.2.e presented the hypotheses selected for mill capacity during the first months of production.

This program seems technically feasible in view of the situation in Decembre 1978. It is clear, however, that its completion will depend on several conditions, especially the following :

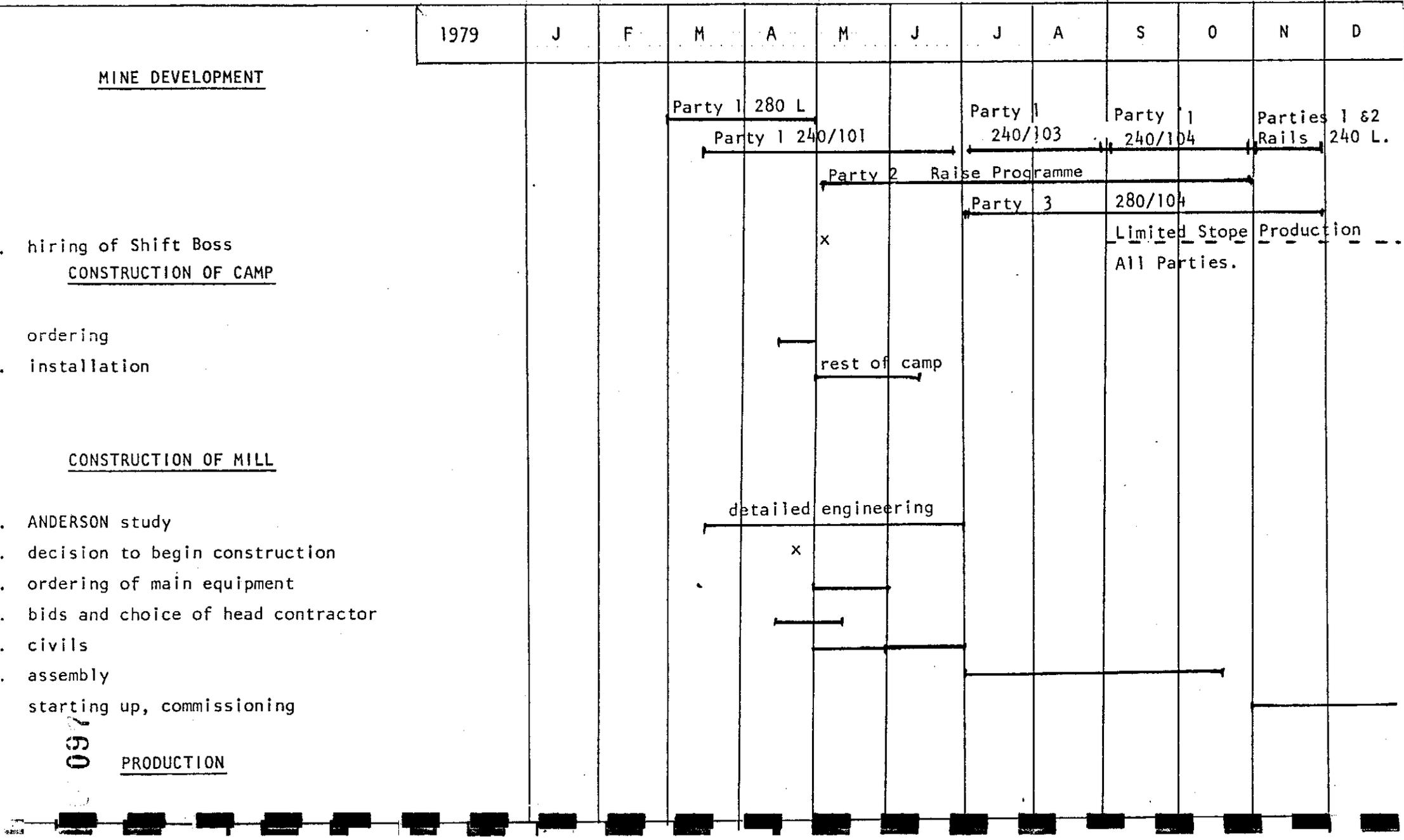
- . the decision to construct the mill which must be made before the end of April, 1979 ;
- . the various contractors must commit themselves as to the planned completion times. This seems possible but must be confirmed.

One of the main roles of the project manager is to draw up a detailed schedule for the whole operation and revise it in terms of events.

1860696

OAKLEIGH CREEK : SIMPLIFIED PROJECT DEVELOPMENT SCHEDULE

THIS SCHEDULE IS PROVISIONAL AND MAY BE CHANGED AS A RESULT OF AGREEMENTS WITH CONTRACTORS



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CHAPTER 13

SALE OF CONCENTRATES

13.1. Marketing

Wolframite concentrates can be sold either directly to consumers or to dealers.

It is usually the large mines which sell directly to the consumers and as a rule, the small mines sell their production to dealers. Realistically it should be supposed that Oakleigh Creek will use this second method.

In this case marketing is easy but the prices are slightly lower than those published in Metal Bulletin. There are slight differences between prices offered by the various dealers. This aspect must be explored thoroughly but it has no practical effect at the feasibility study level.

Contacts were made with two dealers who are very interested in buying concentrates :

- the first proposed a contract for 200 t of concentrates delivered to the port of Burnie at the minimum Metal Bulletin price less 2 - 5% (1).
- the second suggested contracts covering one year of production but the price was not clearly defined. It was however indicated that the price would necessarily be lower than any CIF price in a European port by US \$ 2.60/mtu, to account for freight from Tasmania to Europe.

For the present computations the first proposal will be used since it was clearly defined.

It was stated in paragraphs 7.1.3. and 7.1.4 that it can be hoped that concentrate quality will be sufficient so that there will be no penalties. These penalties are however applied only if the market is depressed.

(1) It should be noted that this dealer indicates a penalty for arsenic of US \$ 0.05/mtu for each 0.01 % As in the interval 0.11 % < As < 0.40 % while most conventional contracts apply penalties at contents higher than 0.20 % As.

13.2 Sales price

Forecasts of specialists concerning the average tungsten price for the next few years cover a broad range from maintenance of the present price (approximately US \$ 140/mtu) for the optimistic, to less than US \$100/mtu for the most pessimistic.

In the present report the most probable forecast will be considered to be a value slightly higher than the average of the two prices mentioned above i.e. US \$ 122/mtu.

Taking into account a reduction of 3% mentioned in the preceding paragraph, the freight from Tasmania to Europe of about US \$3.00/mtu and the exchange rate of US \$ 1.15 for A \$1, the result is a price for concentrates delivered in Europe of A \$ 100/mtu.

In order to study the sensitivity of the project to the tungsten price it is reasonable to consider a variation of $\pm 20\%$ in relation to the figure given above i.e. a range of US \$ 94-141/mtu.

CHAPTER 14

ECONOMIC EVALUATION

All prices and costs in this report are given in Australian 1979 dollars.

The financial calculations given in this chapter are in constant 1979 dollars.

It should be noted that the consumer price index has been rising especially fast in Australia. There has been an average increase of 13% per year between 1972 and 1977 but this rate of increase slowed down between 1977 and 1979.

14.1. Operating cost (per tonne of run-of-mine ore) - Summary

14.1.1. If all production comes from above level 240

. underground work	19.70	(cf. breakdown	\$ 5.5.1)
. mill	9.27	" "	\$ 7.5)
. modifications	1		
. local general expenses	5.03	" "	\$ 9.1.1)
. transport of concentrates	0.43	" "	\$ 9.1.2)
. general expenses Sydney	<u>3.74</u>	" "	\$ 9.1.4)
TOTAL	\$39.17		

=====

14.1.2. Increases for starting production at lower levels

At the start of production at level 200	\$ 1.50
At the start of production at level 160	\$ 3.10

14.2. Capital cost14.2.1. Capital and other costs preceding production

	Acc. cost 1978	Estimate 1979	Total
1. Joint venture administration	27,984	48,000	75,984
2. Exploration	267,790	92,340	360,130
3. Capital costs installation			
3.0 Mine Equipment	138,696	130,200	268,896
3.1 Mill	43,887	979,167	1,023,054
3.4 Infrastructure	52,049	240,706	292,755
4. Operations (pre-production)			
4.0 General expenses	-	120,080	120,080
4.1 Mine	3,509	233,622	237,131
4.2 Mill	-	20,000	20,000
4.3 Environmental	4,088	8,000	12,088
5. Product Sale	-	5,000	5,000
Total	538,003	1,877,115	2,415,118
10 % contingencies			241,000
GRAND TOTAL			2,656,118

Of this total the amount spent on 31 December 1978 was \$520,292 and the amount spent or committed on 7th March 1979 was \$670,503. The breakdown of this amount will be found in appendix 4.

14.2.2. Capital cost following starting of production

This investment amounts to \$460,000. Its nature as well as the scheduling are shown in paragraph 6.2.

It is not known exactly when the digging of the decline will begin, but for the computations it will be supposed that the expenditure will begin to be committed at the beginning of the first year of mining operation.

14.2.3. Estimation error

A relative variation of $\pm 10\%$ will be considered possible for the entire capital cost.

14.3. Financial effects of opening and closing mine

14.3.1. Opening of mine

It is estimated that a minimum of two months will be necessary, after starting up production, for the results of the first ore sales to be banked. (1)

A working capital estimated at \$160,000 must therefore be mobilized from other sources. When this amount is added to the initial capital cost the maximum negative cash flow for the project is obtained.

In the tables for the computations, this lag is simulated by subtracting two months from the sales for the first year and adding them to the last year.

In the first year of mining operations the grade of the run-of-mine will be lower than the standard grade (1.17%) because 8,000 t⁽²⁾ of ore from the first driving operations will have to be absorbed with a grade of some 0.6% WO₃ (2.25m of opening). This low-grade ore from the existing stock-pile will be absorbed at a faster rate than the 25T/day scheduled, to make up for the lower grade. To simplify the calculations, we do not take in consideration the extra tonnage, nor the theoretical lower grade.

14.3.2 Closing of the mine

The closing of the mine involves specific expenditure but also certain savings or recoveries of invested amounts.

(1) This period is relatively short for a mining project but seems to be possible because of the way the trade in tungsten is organised.

(2) Taking in account that the material taken out during the widening of the first 150m of the 240 level has been stock-piled separately and is considered as waste, it is estimated that the stockpile of the other material taken out of the 240 level the 280 level and the first raise will be some 8000 t at the end of the exploration programme.

Debit

. compensation for termination of employment (four month salaries)	160,000
. miscellaneous closing expenses	40,000
TOTAL	<u>\$ 200,000</u>

Credit

The rest of the amount from two months of concentrate sales mentioned previously appears under "sales", but the following has also to be taken into account:

. residual value of equipment (supposed to have been auctioned off)	100,000
. savings due to decrease of preparation work (1)	<u>100,000</u>
TOTAL	<u>\$ 200,000</u>

It can be seen that the debit and credit balance approximately.

(1) Theoretically once all the panels have been prepared, the mine can operate for a certain time without preparation work. There is thus a saving or rather repayment of the capital advance for preparation work. In the present case it was estimated that this capital amounts to \$ 200,000. In reality all this capital will not be recovered since additional prospection work will be carried out with the available labor but not up to the last moment. It is therefore estimated that it will be possible to recover half of this capital i.e. \$ 100,000.

14.4. Value of contents per tonne of ore

The value of the contents per tonne of ore is the product of three factors :

- . the grade of the ore expected to be 1.17% W_3 (1) with the relative range estimated at $\pm 15\%$:
- . the rate of mill recovery which is expected to be 86% with the relative range of $\pm 2.5\%$:
- . the price of the metric unit of W_3 which is expected to be A\$100/t at Burnie with a relative range estimated at $\pm 20\%$.

Since these three parameters affect the computation of the return in exactly the same way, variations in their product i.e. the value of the contents will be studied. Its average value is considered to be A\$ 100.60/t, with the relative range of $\pm 25\%$. (2)

(1) "Probable 2" reserves were given the same grade as the average "proven + probable 1" reserves, though the former may have a higher grade due to the expected increase of vein-width at depth.

(2) It can be shown in statistics that if s , y and z are independent variables and p is their product, their variances are related by the following formula:

$$\frac{\sigma_p^2}{p^2} = \frac{\sigma_x^2}{x^2} + \frac{\sigma_y^2}{y^2} + \frac{\sigma_z^2}{z^2}; \text{ this formula can be}$$

extended to the relative ranges and this makes it possible to define the variability of the value of the contents.

14.5. Financial results

The group of hypotheses which seemed the best founded were chosen for the definition of the "reference case". Considering this reference case as an average, the various parameters were varied within ranges mentioned in the preceding paragraphs and "extreme cases" were thus defined.

The common hypothesis for all cases is a production capacity of 100 t/d i.e. 24,000 t/y.

For all cases the following were calculated;

- . accumulated non-discounted cashflow of the project (before financial charges and taxes)
- . life-span of project;
- . payback (counted from starting up of production);
- . average return on investment.

Financing arrangements and taxes are taken into account separately by the joint venture participants. They cannot therefore be dealt with in connection with the computation of the technical profitability of the project.

As an example, appendix 5 shows the computation for the reference case.

14.5.1. The reference case

Hypotheses

Workable reserves	: 122,700 t with 1.17% WO_3 (hypothese B of § 4.4.3)
Operating cost	: varies from \$39.17 to \$42.27/t according to § 14.1.1)
Investment	: initial investment of \$2,656,118 (+160,000\$ operating capital), then \$ 460,000 spread over three years
Value of contents	: \$100.60/t of run-of-mine ore
Stope opening	: 1.20m

14.5.3. Variations for operating costs and investmentsHypotheses

Reserves, value of contents and stope opening	: same as reference case
Operating cost	: + 8 % in relation to reference case
Investment	: + 10 % in relation to reference case

Results

Investment		Operating cost		
		Reference - 8 %	Reference	Reference + 8 %
Reference - 10 %	Accumulated cash- flow Payback Return		4,375,000 2.41 34%	
Reference	Accumulated cash- flow Payback Return	4,502,000 2.35 31%	4,093,000 2.47 28%	3,695,000 2.62 26%
Reference + 10 %	Accumulated cash- flow Payback Return		3,811,000 2.57 24%	

For all the cases above, the span-life is 5.11 years.

Conclusions

* There is a good probability that the project will be profitable.

* If several unfavourable hypotheses were to coincide the risk of loss is not nil but this risk remains very low in comparison with the probability that the project will be profitable.

* The risk of loss is linked mainly to the fact that the proven reserves are rather small. It might be advisable to carry out an additional exploration campaign in order to reduce this risk. However the work carried out at levels 240 and 280 has now confirmed sufficient reserves above the 240 level to justify the development of the mine, the construction of the mill and the mining camp. The boring of a decline to level 200 and the driving of this level for 300 m would however cost at least \$ 500,000 if this work were carried out without support from an operating mine. This expenditure is excessive in relation to the cost of the delay of the project for at least another year;

Under these conditions, the most logical course seems to be to decide immediately to build the mill, to establish the mining camp and to prepare the mine above the 240 level.

Breakdown of sums spent or committed

1 - 3 - 1979

	Incurred as of 31/12/78	Incurred or committed as of 1-3-79	TOTAL
1. Joint-venture administration (1)	27,984	32,500	60,484
2. Exploration (2)	267,790	30,000	297,790
3. Capital costs installation			
3.0 Mine equipment	138,696	5,000	143,696
3.1 Mill	43,887	20,000	63,887
3.4 Infrastructure	52,049	20,000	72,049
4. Operations (pre-production)			
4.0 General expenses	-	10,000	10,000
4.1 Mine	3,509	10,000	13,509
4.2 Mill	-	-	-
4.3 Environmental	4,088	3,000	7,088
5. Product sale		2,000	2,000
Total	538,003	132,500	670,503

This breakdown is in accordance with the accountancy of the joint venture and does not exactly coincide with the breakdown used in this report:

- (1) the management account item roughly corresponds to what is called SEREM administrative expenses (cf. paragraph 9.2.4)
- (2) the exploration item includes the following: mine development on site running expenses, local administrative expenses and technical services.

OAKLEIGH CREEK - CASH-FLOW COMPUTATION

Reference case
 Variation in value of contents,
 mineable reserves 122,700 t
 Life-span 5.11 years
 Initial investment: 2,656,118
 + working capital 160,000

Hypotheses: Annual production: 24,000t
 Value of contents: $vm = A\$100.60/ \& \text{ of ore} \pm 25\%$

Year	1	2	3	4	5	6	7	8	Total	
Tonnage processed	-	-	24,000	24,000	24,000	24,000	24,000	2,700	122,700	
Tonnage marketed	-	-	20,000	24,000	24,000	24,000	24,000	6,700	122,700	
Operating cost/tonne			39.17	39.17	40.67	41.20	42.27	42.27		
Grade % W ₃			1.17	1.17	1.17	1.17	1.17	1.17		
<u>In A\$ 1000</u>										
Sales	-	-	2012	2414	2414	2414	2414	674	12,342	
Operating costs			940	940	976	989	1014	114	4,973	
Mine Profit			1072	1474	1438	1425	1400	560	7,369	
Investment	(538)	(2278)	(150)	(220)	(90)	-	-	-		
Cash flow	(538)	(2278)	922	1254	1348	1425	1400	560	4,093	
Cash flow cumulated	(538)	(2816)	(1894)	(640)	708	2133	3533	4,093		P.B 2.47 return 28%
Value of contents										Payback & average retu
1.25 vm = \$ 125.75/t	Sales		2515	3017	3017	3017	3017	842	15,425	P.B. 1.75
	Cash flow	(538)	(2278)	1425	1857	1951	2028	728	7,176	return 50%
0.75 vm = \$ 75.45/t	Sales		150.9	1810	1810	1810	18180	506	9,255	P.B. 4.23
	Cash flow	(538)	(2278)	419	650	744	821	392	1,006	Return 7%

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APPENDIX 5

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BUKA-SEREM-TRIAKO JOINT VENTURE

**OAKLEIGH CREEK WOLFRAMITE PROJECT
DETAILED LOCATION PLAN**



REPORT: 79/SYD/03
DATE: FEBRUARY 1979

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