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TASMANIA

OPEN CUT.

ABERFOYLE TIN LIMITED

ORE POTENTIAL OF THE ABERFOYLE VEIN SYSTEM

SOUTH OF BRANDON SHAFT

by

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TASMANIA DEPARTMENT OF MINES

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1.0 INTRODUCTION

Following the treatment of approximately 1900 tonnes of -2 inch material from the Brandon Shaft "mullock" heap which yielded a weighted average head grade of 0.92% Sn and 0.16% WO_3, an investigation was launched as to where this high grade ore came from.

It appears that the source was the developed 4 level connection from Brandon Shaft to Spiers Shaft. Horizontal development totalled 245 m (800 ft).

A sample of the -2" material on the north side of the shaft and alleged to be from the Brandon Shaft sink assayed 0.35% Sn and 0.08% WO_3.

Subsequent investigation of available data indicates that the Aberfoyle vein system persists as an intense swarm of narrow mineralized veins 183 m (600 ft) south of Brandon Shaft that would be amenable to open cut mining.

This report discusses the proposal and recommends some surface drilling to test the southern extent of the swarm as well as quartz content. If warranted, ore grade indications could then be achieved by closer spaced drilling.

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2.0 GEOLOGICAL SETTING OF THE ABERFOYLE DEPOSIT

The Aberfoyle vein system occurs as an intense parallel swarm of quartz - cassiterite - wolframite veins trending north south and dipping 50° to 60° to the west. Down to the 5 level horizon, 153 m (502 ft) below the surface, this intense vein swarm occupies a zone 61 m (200 ft) wide and at least 550 m (1800 ft) long. Below this level the system gradually broadens out with fewer but wider veins, occurring within a zone over 152.4 m (500 ft) wide. The lowest level of development into the Aberfoyle veins is on 11 level - 336 m (1101 ft) below the surface.

The interbedded quartzites and argillites of the Silurian-Devonian Mathinna Group are host to the Aberfoyle deposit.

The quartz-cassiterite-wolframite veins appear to be genetically related to a cupola of aplite which is presumably an apophysis of the main Ben Lomond granite mass which outcrops 1.5 km to the south west.

The original top of the Aberfoyle Vein deposit has been eroded off, and has since been overlain unconformably by flat dipping sediments of Permian Age. In the mine area approximately 12 m to 18 m (40 ft to 60 ft) of Permian mudstone and conglomerate overlies the deposit.

Figure 1 outlines the regional geological setting of the Aberfoyle deposit and Figure 2 outlines in cross section the relationship of the vein system to the aplite cupola.

3.0 QUARTZ VEIN MINERALOGY

The quartz veins vary in width over a large range of 1mm through to 1.5 metres, with no particular width being prevalent. A general widening of vein width occurs with depth and this is also accompanied by a decrease in the number of thin stringer veins. Therefore below about 5 level, the deposit is comprised of much fewer but wider veins.

Mineralogically the vein demonstrates both vertical and lateral zonation.

Vertical zonation takes the form of variations in the proportion of ore minerals with depth. Although wolframite and cassiterite are found at all levels, visible inspection of all levels and mine production statistics show the ore to be richer in wolframite near the cupola and in cassiterite near the surface.

Overall combined ore grades are highest in the upper levels of the mine, reaching a maximum about 4 level and then gradually decreasing with depth. This can be related to the type of veining emplaced as a result of the changing sequence of mineralization.

Edwards and Lyon (1957) deduced a paragenetic sequence of vein mineralization which may be broadly summarized in three major overlapping stages.

Early : Cassiterite and wolframite accompanied by halide - bearing minerals such as fluorite, apatite, topaz and triplite.

Intermediate : Sulphides, chiefly arsenopyrite, pyrite, pyrrhotite, chalcopyrite, sphalerite and stannite accompanied by quartz.

Late : Minor sulphides, chiefly galena, with scheelite accompanied by carbonates and some fluorite.

The veins occurring in the upper levels are early to intermediate stage veining containing cassiterite and wolframite, apatite, topaz and with sulphides often occurring in the centre of the quartz vein.

In the lower levels of the mine, many of the veins are either intermediate veining with quartz and sulphides barren of any cassiterite and wolframite, or obvious late stage mineralization with only 30 to 50% quartz, the remainder being abundant carbonate, galena and fluorite.

Therefore the first two stages of primary mineral deposition containing the cassiterite and wolframite is dominant in the upper levels of the vein deposit.

4.0 ORE POTENTIAL SOUTH OF BRANDON SHAFT

In the Brandon Shaft area, approximately 7000 to 8000 tonnes of mined material has been stacked on the surface. A sample of -2" material cut from a 24 tonne parcel on the south side of Brandon Shaft assayed 0.71% Sn and 0.16% WO_3 (0.87% CM). A check assay recorded 0.36% Sn and 0.11% WO_3 (0.47% CM). Subsequent sampling and milling of -2" material has substantiated these initial results, with 1895 tonnes of -2" material being milled up to 19.8.1980 with a head grade of 0.92% Sn and 0.16% WO_3 (1.08% CM).

Week ended	Est. Tonnes Milled	Head Grade %Sn	Head Grade %WO_3	
8.7.80	170	0.69	0.13	
15.7.80	525	0.84	0.12	
22.7.80	250	1.05	0.20	
29.7.80	374	1.25	0.24	
5.8.80	151	0.86	0.14	
12.8.80	184	0.92	0.19	
19.8.80	241	0.65	0.12	
Total	1895	0.92	0.16	(1.08% CM)

An investigation into where the high grade material originated was aided by Aberfoyle's underground Shift Foreman Ray Brown. He recalls that most of the Brandon Shaft sink was placed on the north side of the Shaft and a small amount of the shaft sink and connection drives on 4 level from Brandon Shaft through to Spiers Shaft was placed on the south side. A sample of -2" material cut from the north side of the shaft assayed 0.35% Sn and 0.08% WO_3 (0.43% CM). Obviously this area contains some very rich grades.

5.0 BRANDON SHAFT PILLAR

On this basis an investigation was started, looking at the ore potential specifically of the Brandon Shaft Pillar, from the base of the Permian sediments down to 4 level. The pillar extends for 9.14 m (30 ft) either side of the shaft.

Vein information from development openings and diamond drill holes were plotted on to cross section 5100N. Quartz vein true widths were added across the block to determine a percentage quartz content. According to Blisset (1959) quartz grades down to 4 level averaged at least 5% cassiterite. It is important to point out that metal is commonly highly concentrated within the thinner quartz veins, and the expectation of a 5% CM quartz grade is not unreasonable for the proposed ore block, with tin being dominant.

Total tonneage within this block is 91,000 tonnes. Estimated quartz vein width is 7 ft or 4.07% of the block.

∴ Quartz content is 3700 tonnes @ 5% CM.
Total block therefore is 91,000 tonnes @ 0.20% CM.

6.0 OPEN CUT PROPOSAL

The investigation into the Brandon Shaft Pillar highlighted the fact that the Aberfoyle Vein Swarm extends further south of Brandon Shaft to at least section 4500N, a distance of 183 m (600 ft). This is indicated by level development, underground diamond drilling and surface drill hole S8. Being a lateral extremity to the Aberfoyle deposit, the veins begin to thin, and only 1 to 3 veins within the area have been wide enough to be amenable to narrow stope underground mining. The quartz veins recorded vary in width from 1 cm to 25 cm with the majority between 5 cm and 10 cm. The quartz veins down to 4 level occur within a tightly sheeted swarm 61 m (200 ft) wide with veining gradually becoming less concentrated outside this zone. The zone dips at 60° to the west.

It is considered that such an ore block may be shown to be economically amenable to open cut mining and as a result the area was studied in closer detail.

The initial parameters studies are listed below -

1. Dimensions of ore block.
2. Tonneage of ore block.
3. Open pit slope and resultant mullock to ore ratios.
4. Quartz content.
5. Quartz grade.
6. Ore block grade.
7. Methods of upgrading the ore before milling.
8. Mill capacity and recovery.

6.1 Dimensions of ore block

The ore block defined is 61 m (200 ft) wide, and extends between sections 5100N and 4500N. Brandon Shaft lies 4 metres south of section 5100N.

Section 4500N was chosen as the southern limit, as underground drilling on 4 level and surface drill hole S8 both show that the vein system still persists as an intense swarm of narrow veins at this point. Surface drill holes S7 and S6 indicate that veining extends further south of section 4500N, but due to incomplete drill logs and the loss of the core, the true nature of the veining is not known.

In the vertical direction, the ore block extends from the base of the Permian, to 4 level, a down dip distance of 122 m (400 ft). The base of the Permian Sediments varies from 12 m (40 ft) to 25 m (80 ft) below the surface, depending on the surface topography.

The 4 level horizon has been chosen as the base of the ore block, because below this level the vein system begins to broaden out with fewer wider veins.

6.2 Tonneage of ore block

The tonneage contained in the ore block as described above is :

$$\frac{400 \text{ ft (height)} \times 200 \text{ ft (true width)} \times 650 \text{ ft (length)}}{13.6 \text{ (density)}} = 3.82 \text{ million tonnes}$$

With an open cup slope of 45°, the veining would also extend to the north of this block, limited by the pit perimeter and the base of the Permian. This block would contain 670,000 tonnes.

This increases the ore block tonneage to 4.49 million tonnes.

Similarly the veining may extend to the pit perimeter south of the described ore block, increasing the overall tonneage to 5.16 million tonnes. However this is currently undefined, and the area south of the ore block for the purpose of this report has been treated separately.

6.3 Open pit slope and mullock to ore ratio

As mentioned above the overall slope of the open pit has been taken to be 45°. This makes the total tonneage for the open pit in the order of 20.78 million tonnes

less ore block of 4.49 million tonnes

leaves 16.29 million tonnes of mullock

$\therefore$ mullock to ore ratio of 3.6 : 1

However, if the veining extends to the south as far as the pit perimeter, the mullock to ore ratio is reduced to 3 : 1.

i.e.	Total tonneage of open pit	20.78 million tonnes
	less ore block of	5.16 million tonnes
	leaves	15.62 million tonnes of mullock

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6.4 Quartz content

The limited level development and drilling information available for this block indicates a quartz content across the 200 ft block of 3%.

6.5 Quartz grade

The average quartz grade in the geological Indicated Reserves (includes all categories A,B,C and D) of March 11th, 1980, for level 1 to 4 is 3.2% CM. However these veins represent the remnants left after the high grade veins have been mined.

According to Blisset (1959), down to 4 level, cassiterite forms more than 5% of the vein material.

It is also a common observation that metal is more highly concentrated within the thinner quartz veins.

An expectation of 5% CM quartz grade for the proposed ore block is not unreasonable.

6.6 Ore block grade

The combination of 3% quartz content and 5% quartz grade provides an overall grade for the ore block of 0.15% CM.

Figure 3 shows what overall grade can be achieved with varying quartz content and quartz grade. For example, if the quartz content is 5% then the overall grade could be 0.25%.

Both these parameters are extremly important and need to be more closely defined before any further action can be taken.

A good knowledge of quartz content and an estimate of quartz grade can be determined by diamond drilling 4 holes on 4 cross sections 61 m (200 ft) apart. Each hole would be approximately 150 m (500 ft) in length.

The drill hole layout for each of the four sections is shown on Figures 4,5,6 and 7. The drill holes on the sections have a separation of 30 to 40 m normal to the holes at the toe of the ore block.

Emphasis should be placed on drilling with equipment designed to obtain maximum core recovery.

Logging of quartz thickness for each hole would provide a knowledge of quartz content throughout the block and also the variation of quartz content across and along strike.

An estimate of quartz grade and of the overall ore block grade can be achieved by visual grading the core, and assaying either half or whole core.

The costs involved are listed in Appendix A.

Similar geological situations occur in the Mt. Charlotte gold mine near Kalgoorlie, and in the M^{t}. Carbine wolfram deposit of North Queensland. Copies of papers describing both these deposits are included as Appendix B.

6.7 Methods of upgrading the ore

A method successfully employed to upgrade the ore to the mill both underground and currently on the reclamation of the Brandon Shaft material is to pass the broken dirt through a suitable sized grid. Underground a 6 inch grid is used. Using a 2" grid for the material from Brandon Shaft, allows two thirds of the dirt to be rejected as oversize mullock and allows the remaining third, composed of fines containing the mineralized quartz, to be transported to the mill.

The quartz veins are rarely frozen to the wall rocks but have a sharp free boundary, often with a mica selvage between the quartz and country rocks. When blasted, the quartz shatters along pre-existing irregular internal fractures. The mullock being less brittle and less fractured remains as larger size material.

This method is expected to be as equally successful on thinner quartz veins, but will require testing.

A two third rejection of oversize mullock will upgrade the ore delivered to the mill to 1.48 million tonnes @ 0.45% CM.

6.8 Mill capacity and recovery.

Present milling capacity on a continuous basis would be 187,000 tonnes per annum. With the introduction of a heavy media cyclone (H.M.C.) circuit rated at 50 t.p.h. mill throughput on a continuous basis would be 350,000 to 400,000 tonnes per annum. Assuming some high grading by sorting out oversize material this would be more than enough to treat the proposed open cut production. However existing screen capacities and the crushing circuit would need checking.

The Jig Tailings project (see previous reports) to treat accumulated jig tailings provides a rationale and justification for the H.M.C. installation.

Delivered head grade would probably be less than the present mine grade and this would detrimentally affect metal recovery, however, recovery of around 60% to 70% would be expected.

7.0 CONCLUSIONS AND RECOMMENDATIONS.

The preliminary data available - pertaining to the open cut proposal gives the following indications.

Total tonneage of the open pit (45^{o} slope) - 20.78 million tonnes.

Ore block tonneage 3.82 million tonnes
+ Veining in the northern face of the pit 0.67 million tonnes - 4.49 million tonnes.

Mullock to ore ratio.	3. 6 : 1
Quartz content	3%
Quartz Grade	5%
ore block grade	0.15%

Upgrading by oversize mullock rejection will deliver 1.48 million tonnes @ 0.45% CM to the Mill.

The initial ore block defined by sections 4500N and 5100N have an open pit outline which encroaches upon Spiers Shaft and the Aberfoyle Mill area (Figure 9)

As this is most likely an unsatisfactory situation, an ore block of similar dimensions has been defined a further 61m (200 ft) south (Figure 10). The second situation will require sectional drilling on cross section 4300N rather than 5100N.

It is recommended that the important parameters of quartz content and grade be determined by the diamond drilling of 4 fan holes on 4 cross sections (4300N, 4500N, 4700N, 4900N) each 61m (200 ft) apart. Each hole would be approximately 150m (500 ft) in length. Budgeted drilling costs are outlined in Appendix A.

If the results from this are favourable, it should be followed up by defining the southern extent of the vein swarm. A better delineation of grade can then be achieved by closer spaced drilling.

An initial small scale open pit will provide a test of grade in the ground, the upgrading technique by size sorting and a test parcel for treatment purposes.

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APPENDIX A.

Brandon Shaft Area Open Cut Drilling Costs.

16 Holes each 155 m in length - Total 2480m

Costs.		$
Drilling 2480m @	\$ 60/m	148800
Core trays and storage	\$ 2/m	4960
Site preparation		500
Geology (core logging, drill supervision)		2000
Survey		300
Sn + WO_3 assays @ \$15/m - 976m		14640
Sundries (drafting etc.)		200
		171400

+Administration

The above budget is for contract diamond drilling.

A more likely, and certainly more economic, approach would be to use our own Diamec 250 drill and drillers. The unit can immediately be transferred to surface from 13 Level Lutwyche.

The area is within easy reach of the mine air and water supply. Drilling with the Diamec would require the purchase of an AW barrel, bits, reamers and casing to allow hole collars to be cased before drilling TT46 into solid rock.

QUARTZ GRADE % CM

ARTZ NTENT %		2%	3%	4%	5%	6%
	2%	0.04	0.05	0.08	0.10	0.12
	3%	0.06	0.08	0.12	0.15	0.18
	4%	0.08	0.12	0.16	0.20	0.24
	5%	0.10	0.15	0.20	0.25	0.30
	6%	0.12	0.18	0.24	0.30	0.36
	7%	0.14	0.21	0.28	0.35	0.42
	8%	0.16	0.24	0.32	0.40	0.48
	9%	0.18	0.27	0.36	0.45	0.54
	10%	0.20	0.30	0.40	0.50	0.60

FIGURE 3

APPENDIX

B

TABLE 1

Chemical losses and gains in alteration of the Elizabeth Creek Granite

	Unaltered granite Analysis	Unaltered granite mg/cc	Sericitized granite Analysis	Sericitized granite mg/cc	Change mg/cc	Muscovite-rich greisen Analysis	Muscovite-rich greisen mg/cc	Change mg/cc	Quartz-rich greisen Analysis	Quartz-rich greisen mg/cc	Change mg/cc
SiO_2	75·73	1999·27	76·20	2049·78	+50·51	84·23	2156·29	+157·02	95·64	2352·74	+353·47
TiO_2	0·05	1·32	0·07	1·81	+ 0·49	0·06	1·54	− 0·22	0·05	1·23	− 0·09
Al_2O_3	12·25	323·40	12·34	319·61	− 3·79	6·89	176·38	−147·02	1·17	28·78	−294·64
Fe_2O_3	0·17	4·49	0·54	13·99	+ 9·50	0·38	9·73	+ 5·24	0·25	6·15	+ 1·66
FeO	1·13	29·83	1·41	36·52	+ 6·69	2·64	67·58	− 37·75	1·09	26·81	− 3·02
MnO	0·08	2·11	0·10	2·59	+ 0·48	0·27	6·91	+ 4·80	0·14	3·44	+ 1·33
MgO	0·23	6·07	0·29	7·51	+ 1·44	0·19	4·86	− 1·21	0·17	4·18	− 1·89
CaO	0·76	20·06	0·65	16·84	− 3·22	0·29	7·42	− 12·64	0·06	1·48	− 18·58
Na_2O	2·15	56·76	0·87	22·53	−34·23	0·24	6·14	− 50·62	0·24	5·90	− 50·86
K_2O	5·16	136·22	4·49	116·29	−19·93	1·76	45·06	− 91·16	0·32	7·87	−128·39
P_2O_5	0·02	0·53	0·02	0·52	− 0·01	0·02	0·51	− 0·02	0·02	0·49	− 0·04
H_2O+	1·53	40·39	2·31	59·83	+19·44	2·48	63·49	+ 23·10	1·07	26·32	− 14·07
H_2O-	0·96	25·34	0·73	18·91	− 6·43	0·54	13·82	− 11·52	0·42	10·33	− 15·01
CO_2	0·22	0·53	0·03	0·78	+ 0·25	0·01	0·26	− 0·27	0·00	0·00	− 0·53
S.G.	2·64		2·59			2·56			2·46		
Gain mg/cc					88·80			228·13			356·46
Loss mg/cc					67·61			314·71			527·10

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MOUNT CARBINE TUNGSTEN REEF SWARM

by C. L. PLUMRIDGE[1]

INTRODUCTION

The Mount Carbine deposit is at lat. 16°31'S, long. 145°7'E 36 km on a bearing of 083° from Port Douglas.

The lode consists of wolframite and scheelite in a swarm of nearly vertical parallel quartz reefs which strike around 135°. The vein swarm covers an area of approximately 375 m by 610 m with lesser veined material extending to the north. The depth has been tested by diamond drilling and reefs are known to persist to at least 300 m.

Within the 23 hectare area described above, 9·8 per cent of the volume of the rock mass is reef or vein material.

1 Geologist, North Broken Hill Ltd.

The deposit was noticed by early settlers before the turn of the century. Development was started by small companies in 1911 but was suspended by 1920. Small operations have been carried on by gougers since then, the intensity of work varying with the price of tungsten. The deposit was described by Ball (1915). In 1969 North Broken Hill Limited drilled three angled diamond drill holes perpendicular to the strike of the reefs. Total distance drilled was 818 m.

GEOLOGY

The deposit occurs in an area of tightly folded Mid Devonian/Lower Carboniferous Hodgkinson Formation comprised mainly of argillaceous sediments with minor basic volcanics. These have been intruded by large and small bodies of Carboniferous Mareeba Granite. Ages and names of rock units are taken from 1:250 000 Mossman Geological Sheet of the Bureau of Mineral Resources.

The deposit is 1 km from a large granite body and is about 4 km from an area intruded by minor bodies of the same type of granite.

The stratigraphic position of the deposit does not appear to have been a decisive factor in its localization as petrological evidence indicates that the host rock is a metamorphic derivative of the surrounding rock types (Whittle, 1969).

The host rocks to the reefs (Fig. 1) were originally micaceous siltstones, feldspathic sandstones and greywackes, on which were superimposed the effects of pneumatolytic tourmalinization and silicification. Structurally, the mineralized zone is a complex combination of brecciated relatively competent arenites and flow-recrystallized highly boundinaged relatively incompetent argillites (Whittle, 1969).

There is little evidence of the present structure of the original bedding. However, the geometry of the metamorphic units is considered more important. Fig. 1 shows that the host rock is bounded by rocks of lower metamorphic grade on all sides except the north-west, where the boundary is not detectable.

The reef swarm is coincident with the highest grade metamorphic effects and a 20° change in strike of the schistose cleavage of the host rock.

The quartz reefs are rarely frozen to the wall rock but have a sharp free boundary, in places with a thin layer of mica between quartz and wall. A typical reef is 15 to 50 cm wide. The width of a vein remains remarkably constant even where the reef buckles and turns. This indicates that the quartz has filled joints rather than faults. Relative movement of wall rocks in buckled areas should produce swelling and pinching due to displacement of matching wall contours; this, however, does not happen. Further evidence for lack of relative wall rock movement is the lack of macroscopic brecciated wall rock in the quartz.

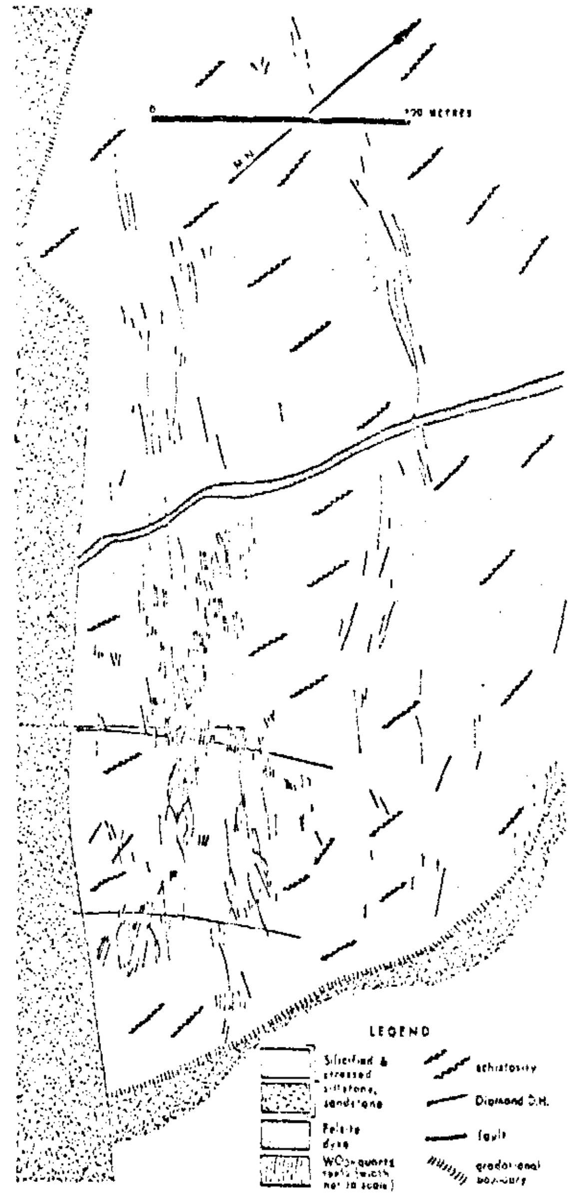

FIG. 1—Surface map, Mt. Carbine tungsten reef swarm.

The appearance of the reefs is that of white glassy typical reef-type quartz. However, the quartz does contain 5 to 10 per cent potash feldspar (Ball, 1915) which shows a tendency to occur with the tungsten. Under the microscope, the quartz appears stressed. The tungsten occurs almost entirely as wolframite and scheelite, wolframite being twice or three times as common as scheelite. The crystals and blocks of crystals of wolframite and scheelite occur in sizes ranging from sub-megascopic up to about 1 m. A common crystal is around 5 cm long.

Other minerals present are calcite, apatite, pyrite, marcasite, chalcopyrite, pyrrhotite, molybdenite and arsenopyrite. None of these reach sufficient abundance to be of economic interest.

The distribution of tungsten in the quartz is patchy and unpredictable. Because of this patchy nature and a coarse crystal size, the grade is difficult to assess.

The deposit crops out well, forming a prominent hill 120 m above the Quartenary alluvium of the Mitchell River valley. The outcropping quartz reefs break into coarse scree which has covered the hill. Wolframite has been won from the scree and soils and has provided much of the tungsten won by gougers.

GENESIS

The deposit occurs in the centre of a concentration of stress and pneumatolytic effects. The tungsten occurs in quartz reefs which have filled open fissures. The deposit is considered to be a product of the advance of the Mareeba Granite.

THE OREBODY

Mount Carbine has been mined in narrow stopes. The quartz reefs were mined and the host rock was left standing like irregular pillars. This intricate mining left large amounts of tungsten behind in reefs too narrow for extraction. The alternative of bulk mining the deposit in an open cut would recover nearly all the tungsten, but the grade appears insufficient to justify this.

Drilling was insufficient to allow accurate tonnage estimates. However, probably 15 to 25 million tonnes of about 0·09 per cent WO_3 grade material lies between surface and 150 m depth. The deposit continues below this depth.

The bulk grade is not accurately known but the average of the 818 m of drill core gave 0·09 per cent WO_3. Unfortunately, due to uncertainty of assays, this grade must remain as an approximation.

ACKNOWLEDGEMENT

The writer wishes to thank the Management of North Broken Hill Limited for permission to publish this article and also Dr. A. W. G. Whittle for permission to quote from his reports on Mt. Carbine.

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MAUREEN URANIUM FLUORINE MOLYBDENUM PROSPECT, GEORGETOWN

by P. J. O'ROURKE[1]

INTRODUCTION

The "Maureen" prospect is a unique deposit enclosed within the basal sedimentary portion of a Palaeozoic acid volcanic sequence. Uranium-fluorine and molybdenum mineralization occur together as lenticular stratiform bodies conformable within the conglomerate-sandstone-shale-siltstone sequence. Higher grade zones form "mantos" elongated down dip generally within the lower grade stratiform mineralization. The deposit is spatially related to the Precambrian-Palaeozoic unconformity and is thought to be metasomatic.

The "Maureen" prospect, at lat. 18°00'S long. 143°26'E, is about 290 km by air south-west of Cairns and 35 km by air north-north-west of Georgetown (Fig. 1). The deposit occurs in a semi-arid region subject to monsoonal wet season rain averaging about 76 cm per year most of which falls during the period December to March. Summer temperatures are often in the 38 to 45°C range.

During 1969 the writer carried out an appraisal of the Georgetown area to ascertain the potential for economic deposits of base metals. Geological field investigations during 1969-1971 located occurrences of base metal mineralization. As a result of exploration on these occurrences an area of about 400 square miles was selected as having favourable geology for economic deposits of base metals and uranium. Airborne radiometric and magnetic surveys carried out in July 1971 resulted in the discovery of radioactive outcrops. Detailed investigations including 6 700 m of drilling were carried out during 1972. Further drilling

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ESTIMATION OF GOLD ORE RESERVES AT MT. CHARLOTTE, KALGOORLIE, WESTERN AUSTRALIA

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ABSTRACT

Gold occurs at Mt. Charlotte associated with a stockwork of pyritised quartz veins in Unit 8 of the Golden Mile Dolerite. The orebody is massive, 200m long and 40m wide and has been proved to over 600m below surface.

Mining is by sublevel ring drill stoping with the ore recovered by diesel powered equipment.

Ore reserves are based on traverses of diamond drill holes oriented to produce an equal intersection width on the two principal sets of quartz veins. Assay and geological boundaries are marked on each drill section and this boundary is transferred to mine plans and cross sections.

Ring-drilling patterns are based on the cross sections and the resulting mining outline is transferred back to the drilling sections.

The mine ore reserve is calculated from the mining outline.

INTRODUCTION

The Mt. Charlotte gold mine is situated towards the northern end of the Kalgoorlie Goldfield, Western Australia. Mt. Charlotte is a low hill rising to 378m above sea level and some 50m above the surrounding plain level.

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The mine is located close to the site of the discovery of gold by Hannan at Kalgoorlie in June, 1893 and it is most probable that Hannan's gold originated in the Mt. Charlotte orebody. Mt. Charlotte provided the location of Kalgoorlie's first gold and its first mine, and in 1979 is the only mine surviving in a field which has yielded approximately 1.25 million kg (40 million ounces) of gold from 100 million tonnes of ore. Of this tonnage 8.8 million tonnes from the Mt. Charlotte mine have yielded 38 600 kg (1.24 million ounces) of gold, an average recovered grade of 4.39 g/tonne.

Since the re-opening of the mine in 1964 8.5 million tonnes of ore have yielded 36 400 kg (1.17 million ounces) of gold, a recovered grade of 4.28 g/tonne; the head grade has averaged 4.79 g/tonne.

Mining commenced at Mt. Charlotte in 1893 by open cutting methods and by 1916 2 200 kg (71 000 ounces) of gold had been recovered from 315 000 tonnes of ore. Rapidly declining grades as mining operations entered unoxidised pyritic ore forced the cessation of operations.

Western Mining Corporation Ltd investigated the orebody by surface drilling in 1938 and 1939 but abandoned its purchase options on the outbreak of World War II.

From 1948 to 1950 The Wiluna Gold Mines Ltd carried out extensive testing of the orebody by driving, crosscutting and winzing on the 91m, 152 and 213m horizons. A number of randomly oriented horizontal diamond drill holes were also drilled. An ore potential of

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